# FINAL TECHNICAL REPORT

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# EXECUTIVE SUMMARY

The project objective was to demonstrate scaled production of high purity rare earth oxides (REO), nominally exceeding 90% grade, from coal refuse sources using innovative technologies that reduce cost and improve environmental outcomes relative to traditional rare earth processing technologies. The project utilized a critical material pilot plant constructed and tested as part of a previous U.S. Department of Energy project. Target performance criteria was a 50% reduction in production cost based on previous optimum values, 150% increase in recovery and greater than 2% concentrates of rare earth oxides, cobalt and manganese. Concentrate production goals were to produce a rare earth mix oxide product at a rate of 200 grams per day having a minimum purity of 50% as well as products of cobalt and manganese having a minimum purity of 2%. Project partners included the University of Kentucky, University of Utah, Virginia Tech, Alliance Coal, Kentucky River Properties and Mineral Separation Technologies.

The project was performed over two budget periods. The research performed during Budget Period 1, which occurred between 2019 October 1 to 2020 March 31, involved laboratory tests to identify the optimum parameters for the targeted technologies including the use of bio-oxidation reactors for acid production. Following a Go/No-Go decision based on a set of performance criteria, the technologies were installed and tested in the pilot plant as part of Budget Period 2 activities, which occurred from 2021 April 1 to 2022 June 2022.

Acid cost reduction was a major target using bio-oxidation reactors to produce sulfuric acid using naturally occurring pyrite existing in coal. Initially, laboratory tests were conducted to assess acid production potential and collect scale-up information from the bio-oxidation reactors using coal-based pyrite as the primary feed. Ferrous oxidation rates were measured to be around 2.0 grams/liter/hr, which consistently produced acidic solutions having a pH of around 1.2 and an equivalent molar acid concentration of 0.2 as determined by titration. Subsequently, a bio-oxidation circuit was designed for the pilot plant to produce 7.5 l/min of acid using two 11-$m^3$ (3000 gallon) reactors. Pyrite feed for the pilot scale bioreactors was prepared using a combination of screening, classifying cyclone, and spiral concentrators. In addition to the pyrite concentrate, a high-grade calcite product was also generated as a byproduct, which was later used in the bench-scale precipitation study for the recovery of REEs and other CMs. The pilot-scale demonstration of bio-acid production revealed that acid concentration equivalent to as high as 1.0 M sulfuric acid could be continuously produced. However, high acid strength bio-acid contained exceptionally high iron concentrations, which complicated downstream processing of the pregnant leach solution (PLS). A Techno-Economic Analysis of the bio-oxidation circuit showed that the cost of the sulfuric acid produced from a coal pyrite source was approximately $0.13 per kg acid equivalent if produced using a four-day retention time in the reactors. This value represents nearly a 48% reduction from that of purchased bulk sulfuric acid.

Calcination (or roasting) studies were conducted on coarse refuse from West Kentucky No. 13 and Fire Clay coal seam sources using a thermogravimetric analyzer equipped with a digital scanning calorimeter. The test results revealed the potential to increase recovery by nearly 100% using temperatures between 500°C to 700°C with the most significant impact being realized for the light rare earth elements. Further increase in the calcination temperature under oxidizing atmosphere did not benefit the REE recovery. It was identified that roasting at 400°C converted pyrite in the West Kentucky No.13 coal to a magnetic iron form, which could be removed using magnetic

separation. However, due to the accompanying loss of rare earth elements, this avenue was not explored. Interestingly, acid-baking of the calcined product using sulfuric acid at 250°C increased heavy rare earth recovery from around 40% to 80% while decreasing the acid requirements by over 50 percent. This improvement in light and heavy REE recovery was attributed to decomposition of clays and REE-bearing minerals such as monazite, xenotime, zircon and other crandallite group minerals. A systematic parametric study on the acid baking process revealed that acid concentration, treatment time, acid-to-solids ratio and baking temperature significantly impacted the REE recovery.

Leach tests were performed using the bio-oxidation solution (or bio-acid) as the lixiviant followed by precipitation tests. Leach recovery values were found to be around 35% after 50 minutes but decreased to 20% if the leach time extended to five hours. This finding was likely the result of gypsum formation as indicated by a corresponding drop in Ca recovery. The formation of gypsum is believed to be due to the high ionic strength of the bio-oxidation solution. Tests were performed to evaluate the impact of reducing the ionic strength by dilution which revealed that the formation of gypsum could be prevented, thereby restoring the rare earth recovery values to around 35 percent.

A systematic precipitation study using NaOH and $Na_2CO_3$ revealed that the former reagent was more selective towards the precipitation of REE and other critical metals (CMs). Use of sodium carbonate resulted in significant contamination of RE-CM cake due to elevated precipitation of calcium. As such, NaOH was employed in further testing. An investigation using synthetic REE and contaminant solutions revealed that the presence of contaminants in the solution significantly impacted the REE precipitation behavior. In the absence of any contaminants in a sulfuric acid solution, REEs precipitated between pH 7.5-9.0. However, the addition of the contaminants decreased the REE precipitation to 4.5-6.0 range. Interestingly, elevated iron concentrations in the range of 10,000 ppm, symbolic of the bio-acid solution, caused REE precipitation around pH 2.5. Furthermore, oxidation treatment significantly impacted the precipitation behavior of iron. Ferric iron was effectively separated from the solution at pH 3.3, however, most of the ferrous iron remained in the solution until circum-neutral pH range. As ferrous iron caused downstream processing complications, $H_2O_2$ was employed to oxidize iron to its ferric form. A fundamental study on the impact of iron and aluminum concentration in the leachate solution revealed that elevated iron concentrations caused co-precipitation of REEs through adsorption, resulting in substantial losses. As such, separate precipitation stages for iron and aluminum were vital to maximizing REE recovery.

Other cost reduction opportunities investigated included: 1) use of naturally occurring calcite in coal as a base material for pH adjustments and 2) aluminum stripping from the rare earth cake prior to oxalate precipitation. Calcite addition was found to be an effective pH modifier for conventional pregnant leach solutions. However, calcite dissociation-initiated gypsum formation when used to adjust the pH of the bio-oxidation solution. Additional tests found that dilution with water or a sulfuric acid solution eliminated gypsum formation. Aluminum stripping using 1 M NaOH was found to remove more than 90% aluminum from the rare earth cake. An economic assessment revealed that implementing this step provides a significant reduction in operating cost (over $100 per kg of REO in some cases). Additionally, the used stripping solution could be employed in the subsequent tests for contaminant removal. However, pilot scale testing revealed

that filtration of the washed cake would require a prohibitive number of filter presses due to high cake resistance.

A parametric study on the oxalic acid precipitation behavior revealed that the REE recovery and purity was significantly impacted by oxalic acid dosage, iron concentration, temperature and solution acidity. Furthermore, bench-scale studies using a PLS obtained from the leaching of coal refuse material generated Ni and Co compounds with 94% and 99% purities, respectively, through a combination of precipitation, leaching, and solvent extraction techniques. It was determined that the metal sulfide precipitation stage using sodium sulfide, crucial for the recovery of CMs, was sensitive to the presence of iron and aluminum in the system. It was concluded that the hydrolysis of aluminum and iron sulfide precipitation significantly increased chemical dosage requirements.

The pilot plant was upgraded for the pilot scale demonstration of REE and CM recovery. A new automation system was installed with the ability to continuously record, monitor and adjust the process parameters such as pump speed, solution pH, Eh and temperature. Furthermore, an interactive HMI system was built within the pilot scale facility to provide operators flexibility to adjust the process parameters based on needs of each experiment. In terms of pilot-scale testing, five different process flowsheets using three different materials were tested. The primary feedstocks were West Kentucky No.13 and heap leach PLS whereas a secondary feedstock was a lignite waste material from a sand operation. Findings from the bench scale tests in conjunction with previous pilot scale studies were employed to design the experiments.

Based on the findings from bench-scale results, it was determined that the precipitation of contaminants such as Al and Fe was crucial for maximizing the product purity and recovery while reducing the cost of the process. As such, all pilot scale experiments started with the PLS generation through leaching followed by iron and aluminum removal, nominally at 3.3 and 4.5 pH. Leaching lixiviants used for the test were industrial grade sulfuric acid or bio-acid generated at the pilot scale facility. For most of the tests, the solid feed rate was 200 lb/hr whereas lixiviant was added at 2 gpm to provide an optimal residence time of 45 minutes. Following the contaminant removal step, several different process schematics were tested with an aim to maximize the REE recovery and product purity.

In the first test, direct processing of aluminum precipitation raffinate for REE recovery using oxalic acid at pH 1.5 was investigated. It was determined that overall REE recovery was approximately 45%. Unfortunately, elevated calcium content in the PLS significantly impacted the RE-Oxide product grade. Similarly, high calcium content decreased both the product purity and grades of CM products. As such, a new flowsheet was tested with the same initial process schematic but different precipitation stages for REEs and CMs at pH 6.0 and 9.0. It was noted that the overlapping precipitation behavior of Co, Ni and Zn with REEs limited the applicability of this process schematic. While this change increased the RE-Oxide grade from 36% in the first test to 87%, the loss of critical metals to the REE cake and bypass of the REEs to the CM cake significantly impacted the recovery of both REEs and CMs. Therefore, the modified process flowsheet combined oxalic acid precipitation stage raffinate and redissolved CM cake filtrate to maximize both the recovery and purity of the products. Consequently, a RE-Oxide product with 85% purity and CM cakes with over 19% Co, 38% Ni, 14% Zn and 9% Mn content were generated with significantly higher recoveries compared to the previous test.

While the modified process flowsheet improved the recoveries and grades, elemental losses were observed in separate precipitation and redissolution losses inspired the adaptation of a single precipitation stage at pH 9.0 for both REEs and CMs. This change was anticipated to maximize the REE recovery while minimizing the costs associated with separated redissolution and processing stages. As expected, REE recovery in this new circuit arrangement was over 56% with a product grade of over 87% RE-Oxide content. Similarly, Co, Ni, Mn, and Zn recoveries of 54%, 40%, 67%, and 66%, respectively, were achieved. Unfortunately, the elevated calcium content present in the solution due to its precipitation at pH 9.0 caused a decrease in the CM cake quality. Therefore, the final process flowsheet involved the addition of calcium oxalate precipitation following the oxalic acid precipitation stage, which effectively eliminated calcium contamination of the CM products. Finally, the pilot scale experiments conducted using bio-acid demonstrated comparable REE leaching recoveries to the conventional sulfuric acid leaching. Elevated iron concentration in the solution caused the co-precipitation of REEs with the iron and aluminum cake, resulting in the REE losses. Furthermore, elevated iron content bypassing the iron and aluminum precipitation stages contaminated the metal sulfide and manganese cake, respectively.

A techno-economic analysis was performed based on a commercial facility capable of treating 500 tph of coal-based material. The production cost for West Kentucky No. 13 coarse refuse material ranged from approximately $500-$700/kg of total rare earth oxide whereas the lignite source had significantly lower production cost of $100-$300/kg. The significant difference in cost was due to the easier leaching characteristics of the lignite material. The West Kentucky No.13 material required thermal treatment to realize acceptable recovery values, which added additional costs to process. The modest recovery values were due to the micro-dispersed nature of the REE minerals (i..e., crandallite-group minerals, monazite, xenotime) within the refuse. In contrast, the REEs in the lignite source are associated with the humic acid in the organic matric and thus leachability using relatively weak acid solutions. Furthermore, the REE and CM contents in the lignite material were nearly an order of magnitude greater than the West Kentucky No. 13 coarse refuse.

In general, roughly 50% of the OPEX and CAPEX cost in all tests was associated with the leaching process. The TEA analysis showed that, based on the recoveries and production costs associated with the different process approaches tested, no scenario resulted in a positive net present value (NPV). An in-depth analysis of the results revealed that considerably lower REE and CM recoveries were achieved in the pilot-scale testing relative to the bench-scale tests. As such, another set of TEA analyses were performed on the lignite material using the bench-scale leach recovery values combined with the pilot scale performance for the downstream processes. For the lignite feedstock, all of the process scenarios using the same process flowsheets as the original pilot plant tests yielded a net positive value. Furthermore, the production cost decreased from $100-$300 $/kg to less than $150/kg of total rare earth oxide for all process flowsheet scenarios.

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# 1.0 INTRODUCTION

## 1.1 OBJECTIVES

The primary objective of this project was to demonstrate scaled production of high purity rare earth oxides (REO), nominally exceeding 90% grade, from coal refuse sources using innovative technologies that will reduce cost and improve environmental outcomes relative to traditional rare earth processing technologies. In a prior DOE study (DE-FE0027035), the recipient successfully designed, constructed, and operated a pilot-scale rare earth processing facility that primarily used conventional approaches to rare earth element (REE) extraction and recovery, including acid tank leaching and solvent extraction. Operational data from this facility has successfully demonstrated the ability to produce REO concentrates at grades exceeding 70% to 90% and at production rates of 10 to 100 g/day. Efforts in this project seeked to leverage this existing infrastructure and improve the technical and economic performance of the process circuit by integrating several innovative process elements that include: advanced sorting for REE pre-concentration, pre-leach calcination to improve leachability, bio-oxidation for autogenous acid production, post-leach selective precipitation and re-dissolution to improve leach solution concentration and purity, and innovative process circuity to reduce overall chemical consumption. In addition, a critical materials recovery circuit was added to the downstream processing units to allow co-production of other critical materials, namely cobalt and manganese alongside the rare earth elements including scandium. These process improvements were expected to improve the pilot-plant capacity to 200 g/day of mixed rare earth oxides with a minimum rare earth elemental concentration of 50 wt% and potentially higher than 90 wt%, while significantly reducing overall project costs. To verify these improvements and streamline commercialization activities, the project integrated technical, economic, and environmental systems analyses concurrent with pilot-scale operation.

## 1.2 APPROACH

The project approach involved the development, design, optimization, integration, and operation of innovative process technologies for producing high-purity rare earth oxides from coal refuse sources at a nominal feed rate of 250 lb/hr reporting to the hydrometallurgy circuit. The two primary sources were coarse refuse materials generated from the processing of West Kentucky No. 13 and Fire Clay seam coals. Approximately 2600 tons of West Kentucky No. 13 coarse refuse and 100 tons of lignite was processed through the REE recovery circuit during the project whereas 100 tons of the Fire Clay reject material was treated through an optical sorter.

The project leveraged the recipient's existing pilot-scale rare earth processing facilities and incorporated new process equipment specifically chosen to improve the plant capacity and reduce process operating costs. For management and reporting purposes, the project was subdivided into 18 distinct tasks organized into two distinct phases of work delineated by a go/no-go decision point. Budget Period 1 focused on development and integration of the new process technologies to:

- Optimize the operational parameters, design criteria, and performance benchmarks for the new processing technologies through detailed laboratory-scale investigations.
- Collect, prepare, and characterize sufficient quantities of pilot-plant feedstock to include coal refuse from industrial production facilities assaying greater than 300 ppm total REE on an elemental basis;

- Fabricate, construct, assemble, and install new process units into the existing pilot plant circuitry;
- Perform a detailed systems safety analysis, and associated health and safety training, required to ensure proper startup, operation and shutdown of the modified pilot-scale facility; and
- Initiate commercialization and technology transfer activities to address technical and economic process performance, workforce readiness, project financial requirements, and overall feasibility.

Budget Period 2 focused on the deployment, operation, and continuous improvement of the overall pilot-plant circuitry. Tasks in this project period were undertaken to:

- Continuously operate the pilot plant circuitry while evaluating and optimizing key unit operations through parametric experimental studies;
- Conduct ongoing operational cost analyses and laboratory-scale troubleshooting to resolve plant issues and improve overall performance;
- Perform detailed environmental monitoring of the pilot-scale facility to establish mitigation strategies and emission controls necessary to fully protect water/air resources and ensure no new solid wastes are generated beyond those already present at the industrial mining sites;
- Conduct additional pilot-scale testing to evaluate the extraction of rare earth elements from alternate feedstocks, such as mine influenced waters and refuse acid drainage (RAD), available at the two industrial mining sites;
- Provide NETL the single sample that reflects the highest achieved REE concentration generated during conduct of project effort;
- Develop and validate mathematical process models and flowsheet simulation routines that can be used to optimize existing process plant performance and design new/improved plant flowsheets for future installations of commercial rare earth recovery systems; and
- Finalize the technical, economic, commercialization and market analyses required to promote the effective transfer of technology and know-how related to the recovery of rare element elements from coal-based sources to the industrial partners supporting this initiative.

## 1.3 SCHEDULE & MILESTONE STATUS

### *1.1.1 Budget Period 1 Schedule*

Figure 1.3.1 shows the project schedule for Budget Period 1. The official start date of the project was October 1, 2019 as indicated by the schedule. However, official approval for federal expenditures to occur was received on November 8, 2019. As such, initiation of the project work tasks was delayed by approximately one month. Due to delays associated with COVID-19, a request was submitted for a 3-month no-cost time extension. The request was approved which resulted in a new end date of March 31, 2021 for Budget Period 1. The ‘x’ marks in Figure 1.1 reflect the time when COVID-19 was having the most significant effect on the progress of the project.

Budget Period 1 primarily involved the performance of laboratory tests to: 1) optimize the operation and collect scaling data for the biooxidation reactor; 2) evaluate the leaching efficiency

achievable by acid solution generated by the biooxidation reactor; 3) study the solution chemistry of the pregnant leach solution (PLS) generated from leaching coal-based materials using the biooxidation acid product and the effect on downstream processes including selective precipitation, solvent extraction and oxalic acid precipitation; 4) fundamental assess the calcination effects on leachability and optimize the performance; and 5) develop a flowsheet and scale processes to economically produce high grade rare earths, scandium, cobalt and manganese.

COVID-19 closed the access to university laboratories in April 2020 which housed the bio-oxidations reactors. As such, no bio-oxidation acid product could be produced during the period, which was needed to assess leachability characteristics, precipitation behavior, solvent extraction performance, precipitate settling kinetics and filtration rates. As such, flowsheet development was delayed which also led in a delay in equipment procurement. Starting June 2020, laboratory access was provided and activities re-started.

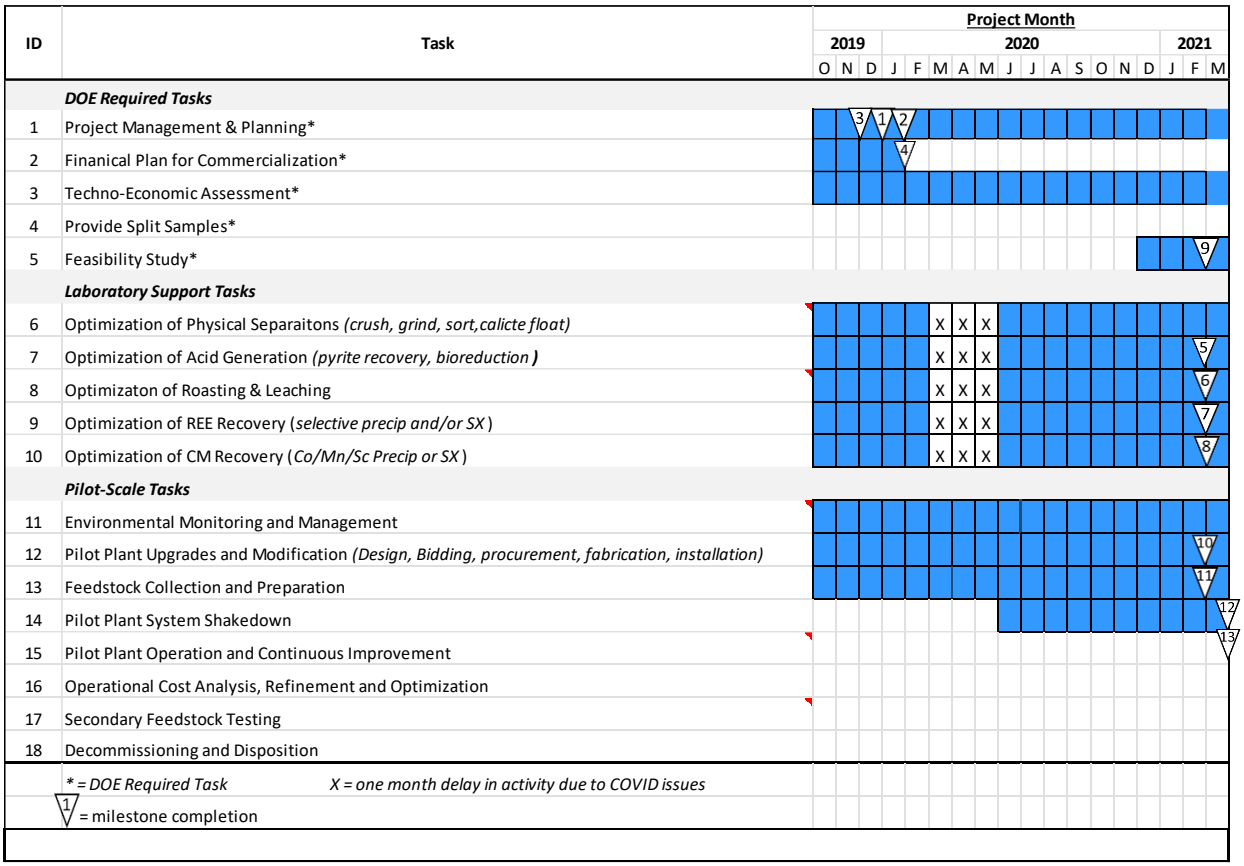



Figure 1.3.1. Project schedule and progress overview.

### *1.1.2 Budget Period 2 Schedule*

The start date for Budget Period 2 was April 1, 2021 with a completion date of December 31,2022. The project schedule for BP 2 is shown in Figure 1.3.2. There were a number of delays in the project due to major failures associated with the bioreactors, which resulted in a six-month, no-cost extension of the project. A thorough discussion of the events causing the delays is provided in the technical discussions about the pilot plant operations.

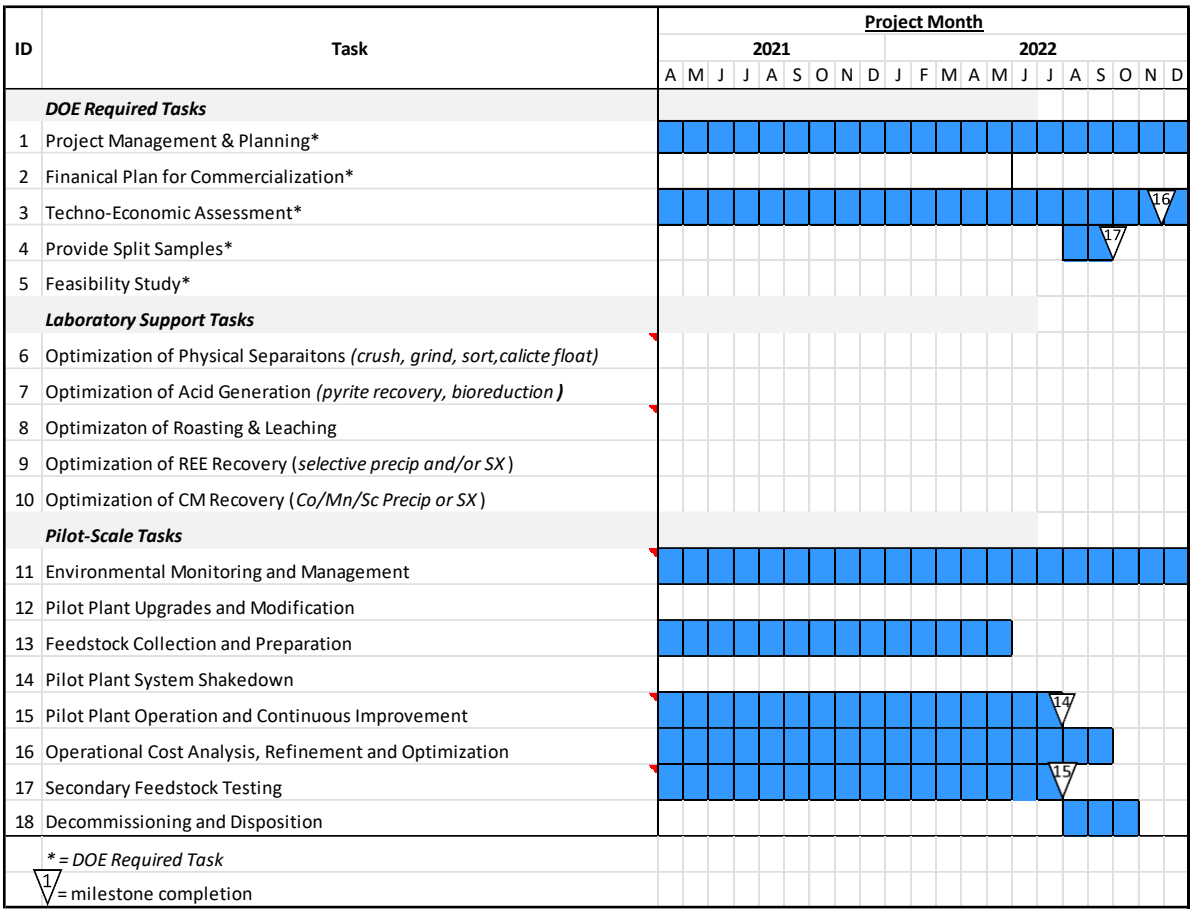



Figure 1.3.2. Project schedule for Budget Period 2.

### *1.1.3 Project Milestones*

Table 1.3.1 provides a list of project milestones with the corresponding task numbers, planned completion dates and verification methods.

Table 1.3.1. Project milestone status.

| Task/ Subtask | MS No. | Milestone Title & Description | Completion Date | | Comments on Status |
|---|---|---|---|---|---|
| | | | Plan | Actual | |
| 1.1 | 1 | Submission & approval of modified PMP | 12/15/2019 | 1/28/2020 | PMP was submitted to the Project Manager for review and approval. |
| 1.2 | 2 | Submission & Approval of TMP | 1/31/2020 | 9/16/2020 | Completed and submitted. |
| 1.3 | 3 | Submission & approval of Work Readiness Plan | 11/31/2020 | 1/28/2020 | Completed and submitted. |
| 2.0 | 4 | Submission & approval of Financial Plan for Commercialization | 1/31/2020 | 2/3/2020 | Completed and submitted. |
| 3.0 | 16 | Techno-Economic Analysis report completed and submitted | 5/31/2022 | 03/28/2023 | Completed. |
| 4.0 | 17 | Split samples provided to NETL | 5/31/2022 | 01/13/2023 | Completed and submitted. |
| 5.0 | 9 | Feasibility study completed and submitted as part of Go/No-Go decision process | 2/28/2021 | 3/1/2021 | Completed and submitted. |
| 7.0 | 5 | Bio-oxidation & Acid generation process optimized | 2/28/2021 | 3/31/2021 | Completed |
| 8.0 | 6 | Roasting & leaching process optimized. | 2/28/2021 | 3/31/2021 | Completed |
| 9.0 | 7 | REE concentration process optimized. | 2/28/2021 | 3/31/2021 | Completed |
| 10.0 | 8 | CM concentration process optimized. | 2/28/2021 | 3/31/2021 | Completed |
| 12.0 | 10 | Major Equipment items received and installed | 2/28/2021 | 6/30/2021 | Completed |
| 13.0 | 11 | Feedstocks collected & characterized | 2/28/2021 | 1/15/2022 | Completed |
| 14.0 | 12 | Pilot plant circuit shakedown test completed. | 3/31/2021 | 6/30/2021 | Completed |
| 14.0 | 13 | Safety Management Plant submitted | 3/31/2021 | 1/8/2021 | Completed |
| 15.0 | 14 | Testing of primary feedstocks completed. | 4/30/2022 | 7/15/2022 | Completed |
| 17.0 | 15 | Testing of secondary feedstocks completed. | 4/30/2022 | 5/6/2022 | Completed |

# 2 MAJOR ACCOMPLISHMENTS

## 2.1 GO/NO-GO CRITERIA

- Techno-economic analysis (TEA) indicates greater than 50% cost reduction per kg of REO produced by 50% primarily due to acid production using biooxidation reactors and the use of an alkaline strip of the rare earth cake to remove aluminum prior to oxalic acid precipitation;
- Concentrate purity of rare earth oxide concentrates greater than 50% and often greater than 90% was achieved. Scandium products from a solvent extraction circuit was around 2% purity. Other metal oxide products included 15% cobalt oxide purity, 11% nickel oxide and greater than 40% manganese oxide. All exceeded the 2% requirement;
- TREE/CM recovery increased by a factor of 3.0 as a result of roasting coal-based material prior to leaching which meets the target of 1.5 multiplier established for the Go/No-Go decision; and
- pH of the barren leach solution (BLS) produced by the bioreactor equaled approximately 1.2 which is significantly lower that the target of 2.0. More importantly, the bioreactor product has an quivalent 0.2 M H2SO4 strength as determined by titration. Recent data indicated the potential to produce even stronger acid solutions with a pH as low as 1.0.

## 2.2 TECHNO-ECONOMIC ANALYSIS

- An integrated techno-economic process model was developed to assess the performance and costs of the process technologies and flowsheets developed in this project.
- A preliminary techno-economic assessment was conducted to evaluate a 500 st/hr commercial scale plant treating refuse from the West KY No. 13 seam. This preliminary analysis only included REE production; however, Sc and other CMs will be added in a future revision.
- The analysis showed that the total capital cost for this plant was $122 million, with approximately 35% attributed to the biooxidation circuit.
- The total operating costs for the plant were determined to be $288 /kg REE with more than half attributed to chemical costs.
- A detailed assessment of the biooxidation circuit showed that the cost sulfuric acid produced from a coal pyrite source was approximately $0.05 per kg acid equivalent. This value represents nearly a 75% reduction from that of purchased bulk sulfuric acid.
- Comparative analysis between the TEA results and pilot scale data of an older iteration of the flowsheet confirm that the biooxidation process and the use of coal calcite can reduce the process chemical costs by >50%, thus meeting the go/no-go criteria.

## 2.3 OPTIMIZATION OF PHYSICAL PROCESSES

- A mobile screening unit having the capability of processing 20 tph of material was installed at the x-ray sorter location to screen out the plus 2-inch and minus ½-inch. Conveyors with weigh belt scales were installed on the feed, ejected and passed through process streams of a dual x-ray transmission sorter material to promote higher throughput with fewer labor requirements. The results demonstrated approximately 84

percent rejection of waste material while upgrading the REE content in the product by more than 130 percent relative to the feed material.

- A rougher-cleaner spiral circuit was installed in Alliance Coal's River View preparation plant to treat the underflow of the StackSizer underflow stream. The results obtained from mineralogical content by XRD as well as ash and total sulfur contents showed the pyrite was concentrated into the rougher-cleaner tailings stream.
- A calcite concentrate of greater than 60% purity was achieved by size separation in a pilot scale continuous circuit for pH adjustment purposes. The undersize stream was processed through magnetic separator and classifying cyclone to produce a pyrite concentrate that was adequate for sulfuric acid production using a biooxidation reactor.

## 2.4 OPTIMIZATION OF BIO-OXIDATION AND ACID GENERATION

- Automatic operation have been accomplished for the two bioreactors of 1.5 L and 3 L volumes.
- The bacteria are adapted to the bioleaching solution containing pyrite concentrate, with the ferrous oxidation rate of up to 4 g/L/hr observed.
- Complete reaction of the pyrite consumption was confirmed with characterization of the feeding and effluent solids with multiple techniques, e.g., SEM-EDS, XRD, and ICP.
- Elemental sulfur analysis confirmed the most conversion of sulfur to sulfate for acid generation.
- The effect of processing parameters such as the gas flow rate and stirring speed, were studied.
- Optimization on the operating parameters were conducted with the response surface experimental design method.
- We can often sustain a rate of 3.5 grams of ferric ions produced/liter/hour. Accordingly,

If 15 moles of ferric ions are needed per mole of pyrite, a reactor with a residence time of 96 hours would theoretically produce:

$$[H^+] = \frac{\frac{3.5 gFe^{3+}}{\frac{liter}{hr}}}{\frac{55.85 gFe^{3+}}{moleFe}} \frac{moleH^+}{15 molesFe} 96hr = 0.40 MH^+$$

A reactor with 6 % feed concentrate containing 75 % pyrite with a enough time to oxidize nearly all of it (usually around 96 hours) can produce:

$$[H^+] = \frac{60 g\ concentrate}{l} \frac{0.80\ g\ pyrite}{g\ concentrate} \frac{1 molepyrite}{119.98 g\ pyrite} \frac{1 moleH^+}{mole\ pyrite} = 0.40 MH^+$$

This is even matching of bacteria capacity with optimal concentrate feed if no acid consuming minerals are present.

- Our concentrate contains close to 20 wt % acid consuming oxide/carbonate minerals such as $CaCO_3$

For the reaction of $CaCO_3 + 2H^+ \rightarrow Ca^{2+} + H_2O + CO_2$, the potential consumption of acid is:

$$[H^+] = \frac{60 g\ concentrate}{l} \frac{0.20\ g\ calcite}{g\ concentrate} \frac{1 mole\ calcite}{100\ g\ calcite} \frac{2\ mole\ H^+}{mole\ calcite} = 0.24\ M\ H^+$$

Our observed apparent acid consumption is generally about 0.2 M $H^+$, so about half of the acid the bacteria produce is consumed.

However, it is noteworthy that the release of $CO_2$ from the decomposition of carbonates such as calcite in acid helps the bacteria to grow faster, so some calcite is good, but usually only 1-2 wt % is likely to be useful.

- Ferrous oxidation rate and oxygen consumption rate were measured and used as the indicators of the microbial activity and the optimization tests.
- Bacteria can consistently generate approximately 0.2 M net acid using a 6 wt % pyrite concentrate containing 75 % pyrite and up to 20 % acid consuming minerals using a 4-day residence time (or significantly more acid production with reduced acid consuming minerals such as 1-2 wt % calcium carbonate).
- Air flow rate and stirring speed need to be maintained at optimum levels
- Temperature should be maintained near 35ºC
- Nutrient additions can be reduced significantly from traditional 9K medium values if appropriate minerals are present.
- Jarosite formation tests provided the conditions for Fe removal with precipitation.
- REE bioleaching tests were conducted with the bioleaching solution and confirmed the feasibility of REE leaching with the bioleaching solution.
- The bench-top (3 liters) data was used for scale-up (10,000 liters)
  Tip Speed: 1.28 meters/sec maximum
  Gas Flow Rate: 2,000 liters per minute minimum
  Residence Time: 3 days minimum
  Solids Concentration: 5 wt % minimum
  Nutrients: (1.6 g/l ammonium sulfate, 0.8 g/l dipotassium phosphate)
  Target pH < 1.5 (1.8 maximum)
  Temperature: 35ºC +/- 1 ºC
  Heat removal estimate: 10 kW (35K BTU/hr) for cooling from reactions
  12 week scale up
  Start with inoculum from lab scale collection, scale to 800-liter reactor
  Target of more than 600 mV Eh reading, 0.5 g/l/hr oxid. r. before continuous feeding
  Scale from 800-liter reactor to 10,000-liter reactor
  Target of more than 600 mV Eh reading, 0.5 g/l/hr oxid. r. before continuous feeding
- The scale up to large reactors was very successful and produced more acid (up to 0.5 M) than expected.

## 2.5 OPTIMIZATION OF ROASTING & LEACHING

- Thermal treatment of coal coarse refuse material significantly improved the rare earth element recovery due to 1) Decarbonization of coal, 2) Dehydroxylation of clays, 3) Liberation of REE-bearing minerals, 4) Subsequent thermal decomposition of the released REE-containing minerals.
- Testing on pure clay minerals confirmed the hypothesis of clay dehydroxylation driving the improvement in REE recovery.

- It was determined that the calcination of 2.20 sink density fraction of Western Kentucky No. 13 material at 600 ºC relative to 400 ºC improved the LREE recovery by 30 absolute percentage points while decreasing the Fe content by 68 percent.
    - Alternatively, roasting the material at 400ºC converted pyrite into a magnetic iron oxide which was susceptible to magnetic separation and could effectively remove approximately 80 percent of the total iron content in the feed.
- Increasing the roasting temperature above 800 ºC in ambient atmosphere resulted in the sintering of clays, which adversely impacted the REE recovery.
    - However, thermal treatment conducted in an inert atmosphere in the same temperature range removed the adverse effect of high temperature calcination.
- Direct acid baking at 250 ºC the West Kentucky 13 and Fire Clay coarse refuse recovered 50% HREEs, significantly higher than the recoveries obtained via simple thermal treatment.
- Roasting at 600 ºC followed by acid baking at 250ºC further improved the LREE and HREE recovery from both sources to approximately 80 percent using 50% less acid compared conventional processing methods.
    - Parametric study on the acid baking process demonstrated that acid baking time, baking temperature, acid to solids ratio and acid concentrations significantly impacted the REE recovery.
    - Furthermore, it was determined that leaching conducted at ambient temperatures provides higher REE recovery relative to 75ºC leaching operations.
- Using the bioleach solution, peak leaching recovery of TREEs occurred at 1 hour of leaching time on roasted west Kentucky No. 13 material with top particle size of 2 mm.
- Reduction of particle size and prolonged roasting duration improved the leaching reaction rate and recovery of TREEs to 38% within 15 minutes of leaching time.
- Leaching recovery slowly reduced with time after reaching the peak recovery due to the coprecipitation of light REEs (i.e. La and Ce) with the precipitation of jarosite.
- Co-precipitation of light REEs (Ce and La) occurred during leaching with the formation of Na-jarosite confirmed by XRD analysis. Recovered value of REEs and critical elements increased with prolonged leaching time.
- Sc and all the heavy REEs concentration reached plateau and remained unchanged after 1 hour leaching despite the change of Eh and pH. Nearly 100% of thorium was removed though co-precipitation with Fe.
- Applying counter current leaching circuit, the TREE leaching recovery was improved from 24.4% after one stage 90 minutes leaching to 31.8% after 3 stage of 30 minutes leaching per stage.
- A systematic characterization study on the secondary feedstock i.e., Shakespeare material, revealed that the more than 90% of the REEs in the material could be recovered using only 0.5 M sulfuric acid.
    - The leaching temperature did not have a significant impact on the REE recovery from this material.
- Internal tank clarifier was successfully implemented to the leaching circuit to allow simultaneous leaching and solid/liquid separation.

## 2.6 OPTIMIZATION OF RARE EARTH RECOVERY FROM PLS

- A systematic study on use of NaOH and $Na_2CO_3$ as a precipitant revealed that using sodium hydroxide provided better selectivity for contaminant removal and RE-CM precipitation. Using $Na_2CO_3$ caused overlapping REE and Al precipitation behavior as well as significant calcium precipitation before pH 9.0 setpoint for combined precipitation.
- Iron in the freshly generated PLS was present in ferrous form. As such, hydrogen peroxide ($H_2O_2$) was employed to oxidize $Fe^{+2}$ to $Fe^{+3}$, which resulted in sharp iron precipitation in the acidic pH (~3.3) range.
- Modeled precipitation test results showed that without contaminants rare earth elements precipitate at pH 8.5 in sulfate system. With increasing concentration of Al and Fe, the precipitation pH of rare earth elements shifts to pH 6.5-7.0. When the Fe concentration is extremely high in RE leachate solution (i.e 7000 ppm), which is similar to bioleach solution system, the precipitation pH of rare earth elements shifted to pH 3-4.
- Using NaOH solution as pH modifier, the precipitation test on the RE bearing bioleach acid leachate shows that the precipitation pH of REs was at around 5.5 with 7000 ppm of Fe in leachate solution. With prolonged leaching time, the Fe concentration reduced, the precipitation pH of REEs increased to around pH 6.5.
- In sulfuric acid leaching system, calcite can be used to replace NaOH solution for Fe and Al removal process, which reduced the chemical usage significantly in the overall circuit.
- Introducing calcite into the undiluted bioleach PLS system with 7000 ppm of Fe, the precipitation pH of REs shifted to below pH 3, which created difficulty to separate Fe and REEs. However, lower the ionic concentration of bioleach PLS by dilution can restore the selectivity of REs from Fe and Al precipitation.
- A precipitation cake of over 7% TREE grade was achieved by implementing Al wash on the crude RE precipitation cake. The Al:TREE ratio reduced from 12:1 to 2:1, which will significantly reduce the downstream cost for RE redissolution and oxalate precipitation.
- A parametric study on the oxalic acid precipitation revealed that solution pH, contaminant concentration, oxalic acid dosage and solution temperature significantly influenced the REE recovery and purity. Findings from the study were employed on the pilot-scale to maximize REE recovery as well as product purity.
- A pilot-scale optimization study on the oxalic acid precipitation circuit revealed that the recirculation of PLS improved the REE recovery due to seeding.
- Use of flocculants in the contaminant and RE-CM precipitation stages significantly improved the settling velocity of the precipitates through agglomeration which mitigated the filter blinding, increasing the filter cycle time as well as percent solids obtained from solid-liquid operation.
- Steel wool was effectively employed for the reduction of ferric iron into its ferrous form, which raised the selectivity of organic extractant in the solvent extraction circuit.

## 2.7 OPTIMIZATION OF CRITICAL MATERIAL RECOVERY

- A process consisting of staged precipitation, re-dissolution, sulfide precipitation, sulfide calcination and redissolution, solvent extraction, and selective precipitation was developed. High-grade Ni and Co compounds, with a purity of around 94% and 99%, respectively, were obtained from a leach solution of coal refuse through the process.
- The partitioning behavior and mechanisms of REEs during precipitation in acid mine drainage were investigated. It was found that the presence of aluminum species in the solution reduced the adsorption of REEs on the ferric precipitates at pH below 4-5, while the presence of ferrous species reduced the adsorption of REEs on the aluminum precipitates at pH below 6-7. Competitive adsorption is the major cause of the phenomena.
- The effects of major contaminants existing in the leach solutions of coal refuse were investigated. It was found that the simultaneous hydrolysis of aluminum and sulfide and the precipitation of iron sulfide increase the consumption of sulfide in the precipitation of Ni, Co, and Zn.
- The mineralogy of REEs in coal was systematically investigated by conducting SEM-EDS and pH-dependent leaching tests on the low temperature plasma ash of Fire Clay and Western Kentucky No. 13 coals. Many different types of REE-bearing minerals, such as monazite, xenotime, zircon, and crandallite group minerals, were found in the ash materials. In addition, significant portions of REEs in the ash materials were leached under weakly acidic conditions. This phenomenon indicated some REEs were atomically dispersed in the organic matrix or complexed with the organic matter in coal.

## 2.8 ENVIRONMENTAL MONITORING AND MANAGEMENT

- Environmental control plan was amended based on the new circuit. Air emission was addressed by using central ventilation and dust collector.
- Wastewater disposal was monitored periodically for toxic elements discharges. TCLP test was carried out periodically for solid waste disposal management.
- Calcite generated from the pilot scale pyrite concentration circuit was mixed with leaching solid residue generated from the experimentation in an effort to neutralize the acidity of the material.
- Continuous measurements for radioactivity demonstrated absence of any radiation, indicating minimal concentrations of the radio active elements.

## 2.9 PILOT PLANT UPGRADES AND MODIFICATION

- Updated plant layout design
- Process flowsheet designed based on bench top laboratory results as well as data obtained from testing of the processes and circuits from a pilot plant as part of the activities associated with a previous project.
- Design of a bio-oxidation reactor circuit with P&ID drawings as well as equipment scaling, construction and procurement.
- Procurement and evaluation of pre-used belt filter to allow an elevation in the throughput capacity of the system.
- Piping and instrumentation diagrams for the entire process flowsheet completed.

- Upgrade of the automation and control system to allow better pH and Eh measurements, operational control and data collection.
- Compressed air system upgrade to provide sufficient air supply to the increased number of air diaphragm pumps that will be used in the circuit for slurry transport and tank mixing applications.

## 2.10 FEEDSTOCK COLLECTION, PREPARATION, AND CHARACTERIZATION

- The feedstock collected from Western Kentucky No. 13 was re-evaluated due to the change of the physical properties of the material after weathering. The feed material to chemical extraction circuit was upgraded by eliminating the coarser fraction by screening.
- XRD and quantitative analyses performed on different size fractions and the handpicked signature material from the weathered pile showed that the rejected coarser fraction contains significant amount of Fe (12.9%) in form of siderite which was easy to be leached (66% recovery) into solution.
- The leaching recovery of TREE improved by more than three times after roasting.

## 2.11 PILOT PLANT OPERATION AND CONTINUOUS IMPROVEMENT

- Multiple pilot scale experiments were conducted using various process flowsheets in an effort to optimize the recovery and purity of the final products.
- High purity (>80%) RE-Oxide product was consistently generated in the pilot scale testing.
- Circuit arrangement 1 provided promising means to enhance the recovery of rare earth elements and other critical metals while effectively reducing the cost of the processes. However, due to the high calcium content of the material, the purity of the final products was considerably impacted.
- Circuit arrangement 2 tested the efficiency of the separate precipitation stage in an attempt to increase both purity and recovery of the final products. The results indicated that the separate precipitation stages were advantageous to the purity of the products as the final products with more 80 percent purities were obtained.
- Circuit arrangement 3 investigated the modified circuit arrangement to maximize the REE and CM recovery. It was determined that granted strict process controls, high purity of REEs and CMs can be achieved. However, mixing the RE and CM cake solution would be required to achieve a high recovery of REEs.
- Circuit arrangement 4 examined the potential of combined precipitation of REEs and CMs at pH 9.0. It was determined that high product recovery and purity could be obtained while decreasing the CAPEX and OPEX of the operation.
- Circuit arrangement 5 added a calcium precipitation stage in the previous process flow in an attempt to improve the purity of the metal sulfide cake. It was found that the overall product grade increased. However, since the PLS generated from the Baker seam material was rich in iron concentrations, its precipitation in the sulfide cake did not produce the anticipated improvement in the grade.
- A comparative analytical investigation on the ICP analysis between three different labs demonstrated accuracy of the ICP data meeting commercial lab standards.
- SEM-EDX study on the RE-Oxide samples generated from the pilot scale processing of the Shakespeare and PLS material identified Ca, Si, and Fe as the primary

contaminants. Trace concentrations of Na, K, Cl, Fe and S also reduced the overall purity of the oxide product.

- Similarly, XRF analysis on the select metal sulfide products revealed Ca and Fe contamination of sulfide product.
- Samples from select tests were collected and sent to a commercial laboratory to determine the concentration of environmentally sensitive elements such as As, U, Th and Hg. The results demonstrated that As and Th were fairly concentrated in the RE-Oxide product whereas elevated U content was identified in the Mn-cake. Mercury analysis was inconclusive due to its <0.3 ppm concentration.
- Pilot-scale demonstration of bio-acid production through bacterial decomposition of pyrite generated bio-acid up to 1.0 M sulfuric acid.
    - However, higher strength bio-acid contained iron concentration as high as 20,000 ppm, which complicated the downstream processing.
- Bench scale solvent extract tests using a 50:1 aqueous to organic (1%v/v Cyanex 272 in Orform) ratio achieved 85.3% and 5.6% scandium and iron loading, respectively, from acid mine drainage runoff obtained from the Dotiki West Kentucky-13 coal seam refuse pile.
- Stripping efficiency of scandium increased as the organic to aqueous ratio decreased. A 1:1 organic to aqueous ratio using 2 M sulfuric acid achieved 89.1% scandium stripping efficiency.
- A small-scale continuous solvent extraction process with 2 stage loading and stripping significantly reduced the iron to scandium ratio in solution.
- Investigation of post-stripping cleaner stages showed that scandium and iron loading efficiency increased with pH. Scandium and iron loading efficiencies in the cleaner stage were 72% and 18%, respectively, using 50 wt% sodium hydroxide to increase the pH to 0.5.
- Comparison of sodium hydroxide and sodium phosphate precipitation revealed a distinct separation between iron and scandium and aluminum precipitation behaviors, but was not economically enough to implement at plant scale.
- Steel wool was shown to be an effective means of reducing ferric iron to ferrous. The Eh of a ferric sulfate solution decreased from 670 to 300 mV with a 4.6 min residence time, but increased the iron concentration in solution from 6000 to 9000 ppm.
- A solvent extraction circuit capable of generating > 2% scandium product from leachate was demonstrated, producing 11.81 g of 0.77% scandium product.

# 3 RESULTS AND DISCUSSION

## 3.1 PROJECT MANAGEMENT & PLANNING (TASK 1)

### *3.1.1 Availability and Accessibility of Training Programs*

The technology developed in this project included innovative process equipment and flowsheets to enable the economic and environmentally-responsible production of rare earth oxides and critical materials from coal preparation plant refuse. In a commercial setting, this technology would be deployed as a supplemental processing facility co-located onsite with an existing coal mining and processing facility. The workforce needed for this facility is expected to include the following labor categories (provided with an indicative list of potential occupation codes per the US. Bureau of Labor Statistics):

1. *Unskilled laborers*. Helpers, plant operators and tenders, warehouse personnel (BLS Codes: 51-9011, 47-3013, 47-5081, 51-9198). Anticipated need: 5 per operating shift; 15 total.
2. *Skilled laborers*. Plant electricians, maintenance specialists, system operators, chemical technicians, and direct labor supervisors (BLS Codes: 51-9012; 51-1011; 51-8091, 49-9041, 47-1011, 47-2111, 19-4031). Anticipated need: 7 per operating shift; 21 total.
3. *Technical and Engineering Staff*. Metallurgical engineers, chemical engineers, health and safety specialists, environmental specialists, automation and control specialists (BLS Codes: 17-2041, 17-2081, 17-2151, 17-2071, 17-2131). Anticipated need: 3 per operating shift; 9 total.

The first labor category, unskilled labor, largely includes lower skill positions where higher training certificates and educational programs are not required. These positions will only require a high school diploma or equivalent.

The skilled labor category includes positions that will require some certification and/or considerable work experience in a similar field. For example, mining electricians (including plant electricians at processing operations) are required to meet various state-specific training and certification requirements. Training programs for this certification are offered by several state agencies. For example, in Kentucky, the electrical training class is offered free of charge at all Mine Safety Branch Offices throughout the year. Since the project technology largely includes unit operations and equipment already in use in the mining industry, further project-specific training is not anticipated.

The technical and engineering staff will include positions that require a four-year engineering degree from an institution accredited by the Accreditation Board for Engineering and Technology, Inc. (ABET). The U.S currently has 13 ABET accredited programs in Mining Engineering and 8 accredited programs in metallurgical engineering[1]. Many of these programs offer advanced graduate-level training in mineral processing and extractive metallurgy. The generational decline in mining and mineral processing programs is a well-noted issue and has been documented in

[1] Accredited Programs Search. ABET. Available: <https://amspub.abet.org/aps/category-search>

several formal studies. See external sources for more detail[2,3]. Since the technology will likely be deployed on active mining sites, all site personnel will be required to have extensive health and safety training per Part 46 regulation of the Mining Safety and Health Administration (MSHA). These courses are offered in house my many major mining companies and several third-party training centers are located throughout the coal mining regions. Online options are also available.

### *3.1.2 Collaborations with Education and Training Providers*

The project was a multi-university collaboration that includes three premier ABET accredited minerals engineering schools: the University of Kentucky, Virginia Tech, and the University of Utah. These programs have a long commitment to the development and training of technical labor needed to successfully transfer state-of-the-art research form the laboratory to the field. This project has directly contributed to the development and training of 1 post-doctoral associate, 2 graduate students, and 14 undergraduate students at the University of Kentucky. Six other individual were employed at the pilot plant in various roles including a purchasing agent, production quality engineer, and mechanic. In addition, five graduate students and 10 undergraduate students were involved in the research program at Virginia Tech and the University of Utah combined. Moreover, the faculty at these institutions have strong connections with other national and international mining engineering programs.

With regard to the skilled and unskilled labor categories, the current project also includes collaboration with several mining and processing equipment companies. Throughout the project, individuals from these entities will have the opportunity to collaborate with project personnel to learn the operational details of the technology. This interaction and collaboration will be used to assess the suitability of the existing labor force for these duties. Additional training materials will then be developed in-house as necessary.

### *3.1.3 Identify Any Necessary Certificates, Certifications or Other Educational Attainment Involved in this Technology/Activity*

The training certifications and requirements for each unique labor category are described in detail above. For unskilled and skilled labor categories, the project technology will leverage the existing workforce in the individual mining communities. The U.S coal mining districts already have suitably trained electrical and mechanical specialists as well as the training infrastructure needed to sustain this workforce. Given the recent decline of the U.S coal industry, many of these people are currently under-employed, and the commercial implementation of the proposed technology will bring substantial benefit. For the engineering and technical labor, specialized training in rare earth element processing is essential for the successful implementation of the technology. Recent federal research support provided by the US Department of Energy to the universities is helping to provide this training to a new generation of engineers.

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[2] Poulton, M. "The Ruptured Pipeline: Analysis of the Mining Engineering Faculty Pipeline." *AGU Fall Meeting Abstracts*. 2011.

[3] National Research Council. Emerging workforce trends in the US energy and mining industries: a call to action. National Academies Press, 2015.

### *3.1.4 Identify Any Economically Distressed Communities or State or Federal Designated Opportunity Zones or Other Geographically Defined Empowerment Zones Where This Activity May Occur:*

The technology may be commercially deployed throughout the coal mining districts in Appalachia. These areas include several "Qualified Opportunity Zones" (QOZs) as defined by the US Internal Revenue Service (IRS). The Table 3.1.1 below includes a list of QOZs with active coal production in Kentucky and West Virginia. Data assimilated from various sources[4,5,6].

Table 3.1.1 Counties with active coal production and also designated as qualified opportunity zones in Kentucky and West Virginia.

| Kentucky | West Virginia |
|---|---|
| Bell | Boone |
| Boyd | Greenbrier |
| Breathitt | Harrison |
| Clay | Kanawha |
| Floyd | Marion |
| Harlan | Marshall |
| Hopkins | Mason |
| Jackson | Mercer |
| Johnson | Mingo |
| Knott | Monongalia |
| Knox | Nicholas |
| Laruel | Ohio |
| Leslie | Raleigh |
| Letcher | Randolph |
| Magoffin | Taylor |
| Martin | Upshur |
| Morgan | Wayne |
| Muhlenberg | Wyoming |
| Owsley | |
| Perry | |
| Pike | |
| Pulaski | |
| Union | |
| Webster | |
| Whitley | |
| Wolfe | |

[4] Opportunity Zones Resources. U.S. Department of The Treasury, Communicty Development and Financial Institutions Fund. Available: < https://www.cdfifund.gov/Pages/Opportunity-Zones.aspx>

[5] Severance Tax by County. Kentucky Coal Education. Available; < http://www.coaleducation.org/Ky_Coal_Facts/economy/severance_tax.htm>

[6] Coal Facts, 2019. West Virginia Coal Association. Available: < https://www.wvcoal.com/resources/coal-facts/5176-coal-facts-2019>

## 3.2 FINANCIAL PLAN FOR COMMERCIALIZATION & TECHNO-ECONOMIC ASSESSMENT (TASK 2 & 3)

To assess the economic feasibility and future commercial viability of the process technologies developed in this project, a detailed techno-economic analysis (TEA) was conducted. The primary input data for this analysis consisted of experimental testing conducted at both the bench-scale and pilot-scale as well as cost models for capital, labor, consumables, and power. Key output metrics included project capital costs, REE production cost ($/kg), 20-year project NPV, rate of return, and payback period. TEAs were conducted continuously throughout the project, and the models were periodically updated as new information was obtained. Ultimately, numerous scenarios were evaluated in the model to assess multiple pathways to commercialization.

A spreadsheet-based economic modeling tool was developed and utilized to facilitate the techno-economic assessment. The spreadsheet tool, denoted *REE-Econ* (ver 5.4), was originally developed under another Department of Energy (DOE) sponsored project entitled, "*Pilot-Scale Testing of an Integrated Circuit for the Extraction of Rare Earth Minerals and Elements from Coal and Coal Byproducts Using Advanced Separation Technologies*" (DE-FE0027035). The REE-Econ was upgraded from its original form to meet the unique aspects of the current project. Elements of the user interface and calculation algorithms were enhanced to improve robustness and ease of use. It should be noted that earlier versions of this tool were used in various other DOE projects, including DE-FE0029956 and DE-FE0031525. As this software and methodology have been used in previous reports, namely the guiding assumptions and cost estimation methodology, portions of this report have been duplicated from those prior investigations. Specific work completed under this project includes the moderate updates to the user interface and calculation tools, the adaptation of the software to the specific flowsheets being evaluated in this study, and the subsequent analysis and interpretation of the results.

This *REE-Econ* software integrates both technical factors (e.g., process feed rates, unit recovery values, reagent consumption rates, scale-up criteria, etc.) and economic factors (e.g., process capital costs, reagent costs, finance and tax structure, etc.), while outputting key economic indicators, such as line-item operating cost estimates, equipment and installed capital cost projections, a pro forma cash flow, net present value *(NPV)*, internal rate of return *(IRR*), and capital payback period. For commercial-scale analyses, this assessment represents an AACE Class 4 estimate (-30% to +50%), as it relies on equipment scaling factors and parametric models for the determination of unit costs.

By utilizing the *REE-Econ* software, the following objectives were completed under this task. Further detail on each objective is provided below.

- Initial screening TEA of all pilot-scale test scenarios (n = 11)
- Supplemental TEA of SHXP tests using alternate test results (n = 4)
- Detailed analysis and sensitivity analysis of top two scenarios.

**Initial Screening:** TEAs were conducted using the raw experimental data from the various pilot test programs. The pilot test program included two heap leach scenarios, five SHXP scenarios, and four WK scenarios. Experimental data from the pilot plant, including mass flows, recoveries, reagent consumption rates, filtration rates, and others, were scaled to a commercial plant without any further adjustment or intensification. Outputs, including costs and revenues were analyzed to determine the most viable scenarios suitable for more detailed analysis.

***Supplemental TEA*:** During the pilot testing program, the research team noted that the REE recovery for the SHXP tests did not meet the projections anticipated from the corresponding laboratory-scale studies. To account for this variation, supplemental TEAs were conducted by integrating both the laboratory and pilot scale data as needed.

**Detailed Analysis:** After the initial screening and supplemental analyses, the optimal scenarios from the SHXP and WK trials were identified and subject to more detailed analysis with respect to the breakdown of CAPEX, OPEX, and other responses. This analysis provide insight on the cost drivers and was used to identify commercialization pathways.

**Sensitivity Analysis**: Leveraging the two optimal scenarios, a detailed sensitivity analysis was conducted to determine the most significant input parameters and the potential range of output variables. Feed rate, feed grade, chemical consumption, operational parameters, REE/CM sales prices, and other significant technical and economic variables were evaluated in this study.

### *3.2.1 Spreadsheet Software*

The *REE-Econ* software integrates both technical factors (e.g., process feed rates, unit recovery values, reagent consumption rates, scale-up criteria, etc.) and economic factors (e.g., process capital costs, reagent costs, finance and tax structure, etc.), while outputting key economic indicators, such as line-item operating cost estimates, equipment and installed capital cost projections, a pro forma cash flow, net present value *(NPV)*, internal rate of return *(IRR*), and capital payback period. The current version also integrates equipment-specific design factors, such as tank H/D ratios, heat loss rates, specific settling areas, and others to provide realistic equipment sizing with the proper cost implications.

The software workflow is driven by various worksheet "tabs." For typical use, the flow is left to right across the worksheets and top to bottom within the worksheets. Figure 3.2.1 shows a screenshot of the application interface, while a listing and description of each worksheet tab is shown in Figure 3.2.2. Within each worksheet, the spreadsheet cells are delineated between a printable area, highlighted with a white background, and a nonprintable area denoted by a grey background. The printable area contains key process information and economic indicators, while the nonprintable area contains secondary calculations, explanatory text, and other information. The workbook also includes integrated instructions for use.

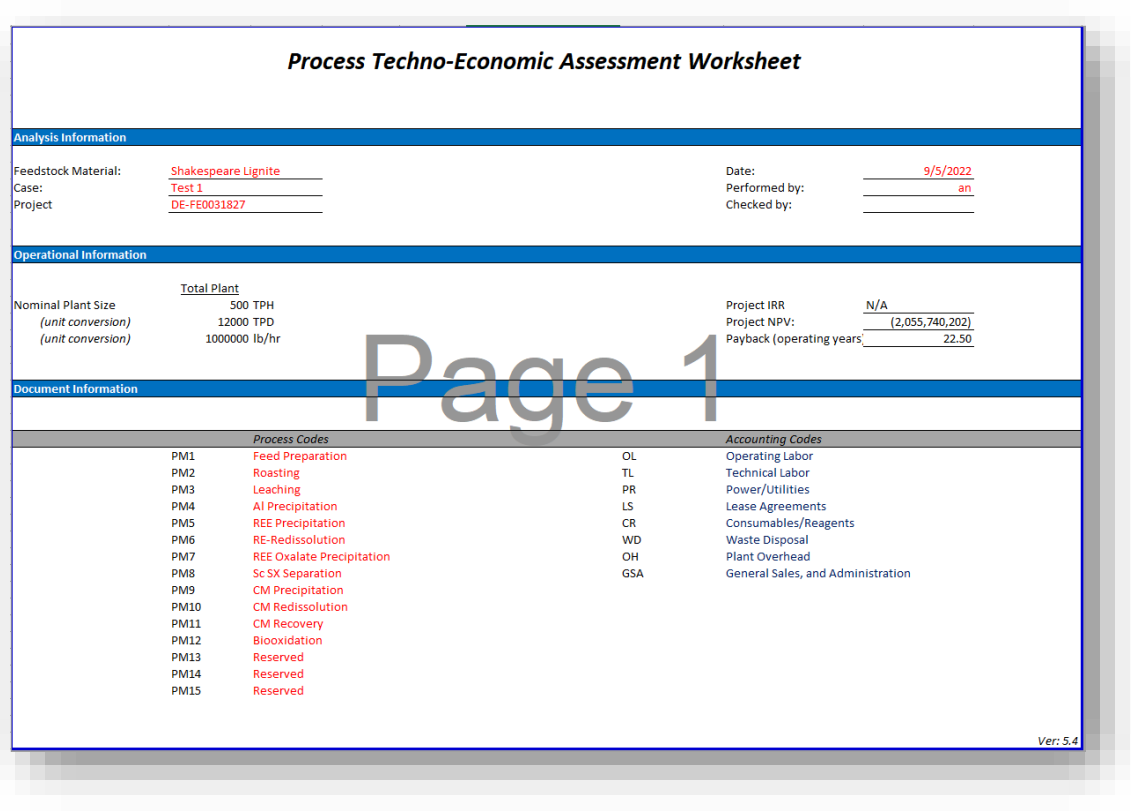



Figure 3.2.1. TEA spreadsheet interface.

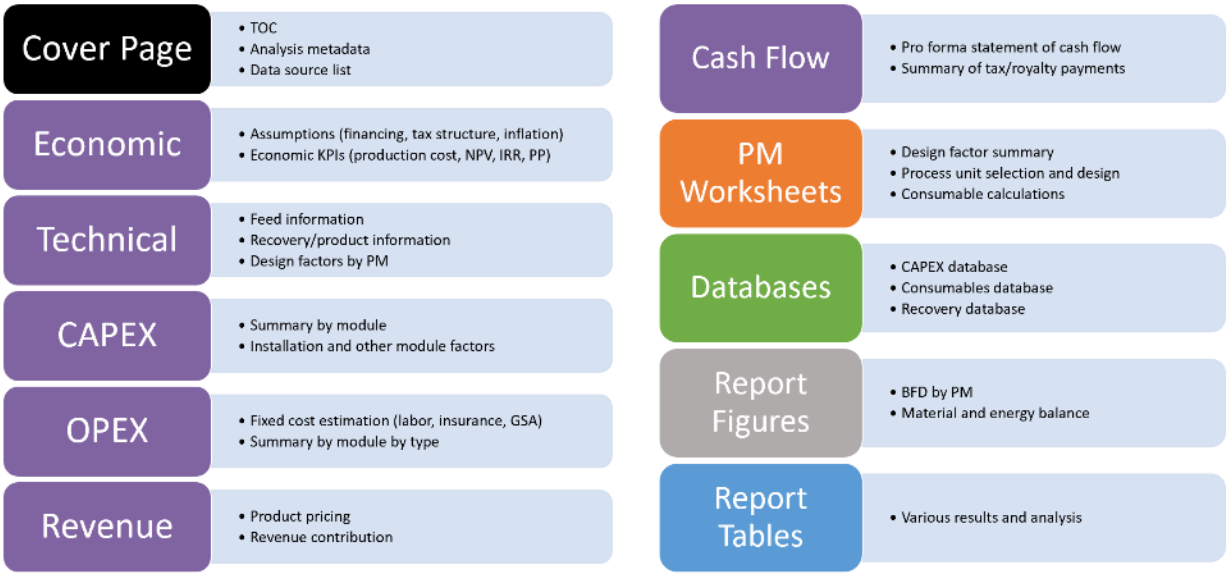



Figure 3.2.2. TEA spreadsheet worksheet structure.

### *3.2.2 Global Economic Assumptions*

Several technical and economic assumptions were required to complete the techno-economic analysis. Regarding technical inputs, the experimental input data used to create each material balance was obtained directly from various project participants. These data were largely acquired from the corresponding pilot-scale test campaigns; however, some laboratory and literature data was used when needed to fill knowledge gaps. The source of all technical input parameters is noted in the TEA spreadsheet alongside the input cell.

Many of the economic assumptions, including those regarding the financing structure, escalation rates, tax calculations, and operating period have been supplied by National Energy Technology Laboratory (NETL) in the *Guidance for Development of Techno-Economic Analyses* document. Key economic assumptions utilized in this study include:

- All amounts are in US dollars.
- The total operational period for the plant is 20 years.
- The plant capacity is defined by a fixed feed rate, rather a fixed product rate (i.e. the feed rate is fixed and the product rate is calculated by considering feed grade and recovery). For standard analyses in the study, the rate was fixed at 500 short TPH (st/hr.).
- The feedstock grade for each case reflects laboratory analyses from material sampled during this study.
- Inflation was applied to sales revenue and operating costs using a fixed rate of 3%.
- Capital costs are spread over a period not to exceed three years, and the allocation between those three years is 10%, 60%, and 30% for years one through three, respectively. Thus, the total analysis period (capital purchase plus operating) is not to exceed 23 years.
- During the capital expenditure period, capital costs escalate at a constant rate of 3.6 percent.
- The project is debt financed for 50 percent of the total overnight capital requirement. The remaining 50 percent is financed by equity.
- The debt repayment terms include: 6% interest rate, 10-year loan period, and no grace period on debt repayment. The repayment uses a standard amortization schedule with constant payments throughout the payoff period.
- Working capital is not included in this estimate and will instead be borne by the operating entity at no cost to the project.
- The combined federal and local tax rate is fixed at 26%.
- All capital is depreciable, using a 150% declining balance depreciation schedule over 20 years. The depreciation method was not changed to straight line when conditions favored the switch.
- The mineral depletion rate for REEs is 14%. Depletion is charged at the appropriate rate times the net sales revenue after deducting royalties and any severance tax,

provided that the total amount calculated by depletion rates does not exceed 50% of the taxable income before depletion.

- The plant is part of a larger commercial entity with sufficient revenue to offset negative taxable income. Thus, losses are not carried forward and are instead calculated as a "negative tax" that indicates the reduction in tax burden required for overall entity.
- All process modules are located at the same facility with no significant transportation cost between modules.
- The plant operation schedule is fixed at eight (8) hours per shift, three (3) shifts per day, 336 operational days per year.
- Equipment costs were determined by scale-factor analysis for primary unit operations. Installation costs were determined by an overall Lang factor.
- Reagents, consumables, power, and labor were determined by itemized cost analysis.
- Other operating costs, including maintenance and supply, QA/QC, sales, admin, property tax, insurance, and plant overhead were all calculated using proportional factors.

For the TEA analysis, costs are categorized according to both a functional accounting code and according to a process module. For operating costs, applicable accounting codes include: operating labor, technical labor, power/utilities, tolling/lease agreements, consumables/reagents, waste disposal, plant overhead, and general sales and administration. Likewise, capital costs are coded as: base equipment; initial reagent fills; ancillary operations; buildings engineering, procurement, construction, and management (EPCM), and contingency.

### *3.2.3 Scale-Up and Cost Estimation Procedures*

A detailed bottom-up assessment was performed to determine the total expected capital expenditures for a hypothetical commercial-scale facility. After completing a mass and energy balance for the process scenario under consideration, the unit capacity (and subsequently the size and number) of each unit operation was determined by a fundamental assessment of the processing needs. Not all unit operation costs scale by simple feed throughput, so a detailed unit-by-unit appraisal was required. For example, the capital cost of roasting operations tends to scale exponentially with respect to heat input in terms of million BTUs/hour (MBtu/hr.), while the capital cost of grinding mills scale is related to power (expressed as horsepower (HP) draw). The engineering team consulted several standard cost engineering reference guides, including the Mine and Mill Estimator's Guide, Matches.com, Gentry and O'Neil (1986), and Garett (1989), to determine the relevant scaling criteria for each equipment type. These scaling parameters were then used to calculate the unit capacity required at the commercial feed rate. Once the capacity was determined, an exponential capital cost model was used to determine the final cost. Details on the scale-up criteria and unit capacity measures are shown in Figure 3.2.1.

Table 3.2.1. Scale-up criteria used in techno-economic analysis

| Process Operation | Cost Model Input | Scale-Up Criteria |
|---|---|---|
| Loaders | Bucket Capacity | Constant TPH per unit (unit size fixed) |
| Crushers | Power Draw [HP] | Constant HP per TPH |
| Screens | Screen area [$ft^2$] | Constant TPH/$ft^2$ screen area for a given screen opening size |
| Conveyors | Throughput [TPH] | Simple throughput |
| Storage Bins | Storage capacity [tons] | Percent of Hourly production surge capacity |
| Mills | Power Draw [HP] | Constant Bond Work Index |
| Storage Tanks | Storage capacity [gallons] | Constant Storage time/retention time |
| Mixers | Power Draw [HP] | Constant Power Intensity (HP/1000 gal.) |
| Pumps | Feed rate [gpm] | Simple throughput |
| Cyclones | Feed rate [gpm] | Simple throughput |
| Thickeners | Thickener area [$ft^2$] | Constant unit capacity ($ft^2$/t/day) |
| Filter Presses | Filter volume [$ft^3$] | Constant cycle time |
| Belt Filter | Throughput [TPH] | Simple throughput |
| Dryers | Water evaporation rate [lb/hr $H_2O$] | Constant moisture reduction |
| Roasters | Heat Input [MBtu/hr] | Constant heat requirement per unit feed |
| Scrubbers | Gas rate [cfm] | Constant airflow per TPH |
| Spray Quenchers | Gas rate [cfm] | Constant airflow per TPH |
| Chillers | Heat Input [MBtu/hr] | Constant heat requirement |
| Leach Tanks | Volume [gal] | Constant Retention Time |
| Solution Heaters | Heat Input [MBtu/hr] | Constant heat requirement |
| Mixer Settlers | Volume [gal] | Constant Retention Time |

For estimation and analysis purposes, all capital costs were determined assuming new equipment, free-on-board *(FOB)* shipping point. Fixed equipment costs for individual process units (*CAPEX*) were determined using a standard power law cost model, given by:

$$CAPEX = \left[\frac{CI_C}{CI_H}\right][K][aX^b]$$

where $CI_C$ and $CI_H$ are the current and historic cost indices, K is the equipment cost ratio (or Lang) factor, X is the unit capacity, and "a" and "b" are fitting constants unique to each equipment item (Gentry and O'Neil, 1986). Initially, the cost model fitting parameters (a and b) were derived from multiple standard equipment costing resources (MAMEC, Gentry and O'Neil, Garett, 1989). These data are shown in Table 3.2.2for the major equipment categories. The cost model data was derived from multiple sources, each with a different reference year; therefore, the Chemical Engineering Plant Cost Index *(CEPCI)* was used to normalize the costs to a single fixed construction year. Historic cost indices were derived from the reference year data used to generate the cost model (see Table 3.2.2), and the current cost index was assumed to be 815.59, which corresponds to a projected 2025 construction year.

Table 3.2.2. Capital cost model parameters used in techno-economic analysis.

| Process Operation | Coefficient, a | Exponent, b | Capacity Unit, X | Reference |
|---|---|---|---|---|
| Front End Loader (2yd) | Constant Cost = $147,400 / unit | | | MAMEC, 2016 |
| Front End Loader (10yd | Constant Cost = $945,700 / unit | | | MAMEC, 2016 |
| VSI Crusher | 3,247 | 0.68 | HP | MAMEC, 2016 |
| Roll Crusher | 1,120 | 0.8484 | HP | MAMEC, 2016 |
| Vibrating Screen | 1,002 | 0.9093 | ft^2 | MAMEC, 2016 |
| Conveyor | 2,092 | 0.5491 | TPH | MAMEC, 2016 |
| Storage Bins | 4,441 | 0.6185 | Tons | MAMEC, 2016 |
| Roaster | 390,000 | 0.48 | MBTU/hr | Garrett, 1989 |
| Gas Scrubber | 7 | 0.9414 | cfm | MAMEC, 2016 |
| Spray Chamber Quenche | 358 | 65000 | cfm | MAMEC, 2016 |
| Chiller | 97,585 | 0.6 | MBTU/hr | Matches |
| Polyethylene Tank | 1 | 0.9492 | gal | MAMEC, 2016 |
| Steel Tanks | 179 | 0.5624 | gal | MAMEC, 2016 |
| Tank Mixer | 10,640 | 0.564 | HP | MAMEC, 2016 |
| Process Pump | 2,152 | 0.3814 | gpm | MAMEC, 2016 |
| Thickener | 280 | 0.8023 | ft^2 | MAMEC, 2016 |
| Filter Press | 6,068 | 0.72 | ft^3 | MAMEC, 2016 |
| Solution Heater | 25,929 | 0.953 | MBTU/hr | MAMEC, 2016 |
| Belt Filter | 207,819 | 0.249152 | TPH | MAMEC, 2016 |
| BioLeach Tanks | 2,405 | 0.4203 | gal | MAMEC, 2016 |
| Blower | 197 | 0.4625 | cfm | MAMEC, 2016 |

After the base equipment prices were established, installed capital costs were determined by applying Lang factors to the direct equipment costs. Typical ranges for Lang factors are specified by Gentry and O'Neil (1986) and are shown in Table 3.2.3. Each individual Lang Factor is categorized as an ancillary cost; a buildings/infrastructure cost; or an engineering, procurement, construction, and management *(EPCM)* cost. An overall 50% capital contingency was also applied to the final base equipment price.

Table 3.2.3. Lang factor estimation for commercial-scale installation.

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| **Total Lang Factor (=1+sum % Base):** 371.95% | | | |
| **Alternatively, a multiplier of:** 3.72 | | | |

The operating expense estimate for the proposed facility was categorized according to both process module as well as a functional accounting code. Accounting codes used in this study include: (i) operating labor, (ii) technical labor, (iii) power/utilities, (iv) lease/toll agreements, (v)

consumables/reagents, (vi) waste disposal, (vii) plant overhead, and (viii) general sales and administration. Similar to the capital cost estimate, a bottom-up approach was used to determine the expected operating costs from first principles. Direct operating costs were specified using designed labor requirements along with publicly available wage and burden rates (Table 3.2.4).

Table 3.2.4. Labor Category Descriptions.

| Labor Category | Code | Labor Rate ($/hr.) | Fringe Benefits | Source/Notes |
|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | May 2021 KY BLS, Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | May 2021 KY BLS, Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | May 2021 KY BLS, Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | May 2021 KY BLS, Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | May 2021 KY BLS, Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | May 2021 KY BLS, Occupation Code: 17-2041 |

*Labor rate data retrieved from: https://www.bls.gov/oes/current/oes_ky.htm*

Power costs were determined via itemized listing of the horsepower requirements for every major unit operation, assuming an industrial electricity cost of $0.0683 /kw-hr and an overall electrical efficiency of 85%. Other direct operating costs included reagent dosing requirements, waste disposal costs, and expected maintenance requirements, which were all itemized and calculated directly. General sales and administrative *(GSA)* costs were specified using standard overhead charge rates. For accounting and analysis purposes, costs were allocated based on cost type (fixed, variable or overhead), and each line item cost was also allocated to a process unit. Details on the specific factors and unit costs used in the analysis can be found in *REE-Econ* printouts included in the *Appendix* to this report.

Revenue for the commercial-scale facility was estimated using market REE prices and the expected REE production for various products. A standard price deck, which has been provided by NETL is shown in Table 3.2.5. All standard prices were then subjected to a fixed reduction/discount. For the mixed REEs in a combined REO product, a 25% discount was used to account for the downstream separation. The selection of a 25% discount was based on the discussion given by Dahlberg et al (2014).

Table 3.2.5. REE Price deck used for commercial-scale economic analysis

| Element | Metal Oxide Price ($/kg) | Metal Oxide Price (Elemental Basis) ($/kg) | Price Source |
|---|---|---|---|
| Scandium | $ 4,200 | $ 6,442 | [1] |
| Yttrium | $ 6 | $ 8 | [1] |
| Lanthanum | $ 2 | $ 2 | [1] |
| Cerium | $ 2 | $ 2 | [1] |
| Praseodymium | $ 52 | $ 63 | [1] |
| Neodymium | $ 42 | $ 49 | [1] |
| Samarium | $ 2 | $ 2 | [2] |
| Europium | $ 34 | $ 39 | [1] |
| Gadolinium | $ 32 | $ 37 | [1] |
| Terbium | $ 400 | $ 470 | [1] |
| Dysprosium | $ 230 | $ 264 | [1] |
| Holmium | $ 53 | $ 61 | [2] |
| Erbium | $ 34 | $ 39 | [1] |
| Thulium | $ - | $ - | [2] |
| Ytterbium | $ 29 | $ 33 | [2] |
| Lutetium | $ 797 | $ 906 | [2] |
| Manganese | $ 0.005 | $ 0.012 | [3] |
| Cobalt | $ 22.00 | $ 22.00 | [3] |
| Nickel | $ 18.00 | $ 18.00 | [3] |
| Zinc | $ 3.02 | $ 3.02 | [3] |

[1] NETL FOA 2003 Price Deck
[2] Chinese Metal Exchange
[3] USGS, 2019

### *3.2.4 Process Descriptions*

To aggregate costs across functional unit operations, the research team identified and defined a series of process modules. A process module is defined as a group of similar unit operations that perform a single process function and can be described by a common feed rate. Each process module has a unique worksheet tab in the spreadsheet TEA model (denoted as PM1, PM2, PM3, etc.), which contains all of the pertinent design and costing data. This approach allows capital and operating costs to be compared between process modules to assess cost drivers and identify opportunities for improvement. A complete listing of the process modules used for this project is shown in Table 3.2.6. Further detail on each individual process module is provided below. Moreover, a flowsheet showing the bounding boxes for reach process module is included in Figure 3.2.3.

Table 3.2.6. Listing of process modules used in economic analysis.

| Module ID | Process | Feed Type |
|---|---|---|
| PM1 | Feed Preparation | Solid |
| PM2 | Roasting | Solid |
| PM3 | Leaching | Solid |
| PM4 | Al Precipitation | Aqueous |
| PM5 | REE Precipitation | Aqueous |
| PM6 | RE-Redissolution | Solid |
| PM7 | REE Oxalate Precipitation | Aqueous |
| PM8 | Sc SX Separation | Aqueous |
| PM9 | CM Precipitation | Aqueous |
| PM10 | CM Redissolution | Solid |
| PM11 | CM Recovery | Aqueous |

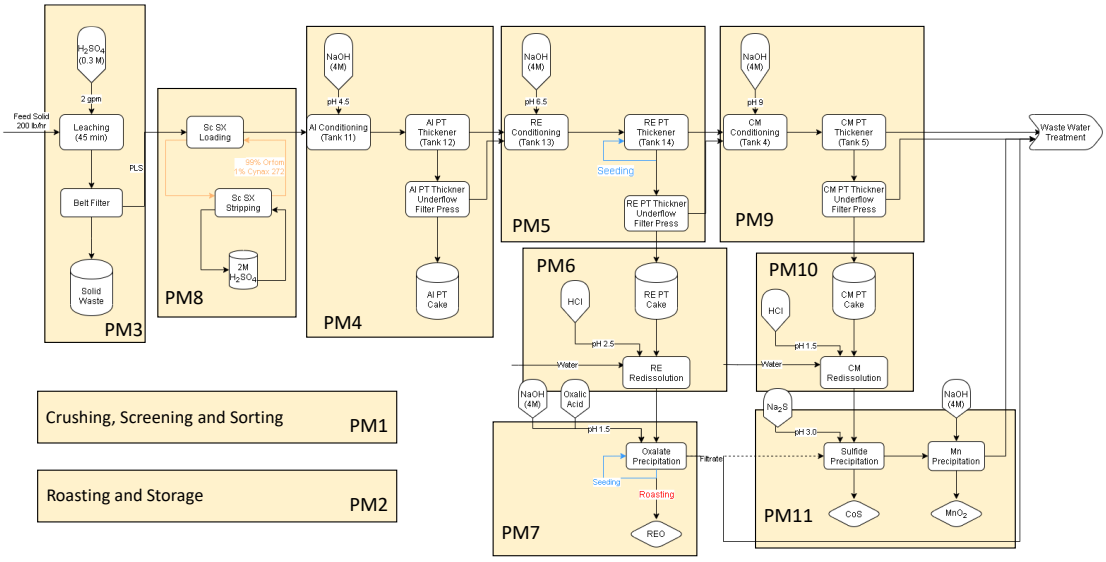



Figure 3.2.3. Process flowsheet showing PM bounding boxes.

***PM1: Feed Preparation***: Raw plant feed consisting of either coarse preparation plant refuse, existing refuse pile material, preparation plant middlings, or similar solid materials are first processed through the feed preparation circuit. This circuit crushes, sizes, and partially enriches the feedstock to generate a material suitable for the downstream roasting and leaching operations.

Based on the business case assumptions, the REE plant is assumed to be co-located at a coal preparation or waste disposal site, and as such, no significant haulage costs are incurred. Rather, the bounds of the overall process start with a 10-$yd^3$ front end loader feeding the plant from the preparation plant refuse pile. For some feedstocks, the material is first passed through a double deck screen to isolate the 2 inch by 0.5 inch material before being processed by an x-ray sorter. Not all feedstocks are amenable to x-ray sorting, and in these cases the screening and sorting steps are bypassed. Following the sorter, the material will be processed through a vertical shaft impactor (VSI) crusher and a roll crusher, each followed by a vibratory sizing screen, to stage crush the material to a nominal 1 to 2 mm product suitable for roasting. The module also includes five conveyor belts for materials handling of the various intermediate and final product streams.

**PM2: Roasting:** Following feed preparation, the crushed solids are directed to a high temperature oxidation/roasting process prior to leaching. In this process module, solid material is conveyed into a rotary kiln where the material is heated to an optimal temperature (nominally 650° C). The thermal-oxidation process dramatically improves the leachability of the REEs and CMs contained in the refuse material. The roasting is fueled by an external natural gas source; however, a significant portion of the required fuel is anticipated to be supplied by the residual carbonaceous material in the coal refuse. This practice is common in the roasting of gold ores, which can contain as little as 3% carbon (Mular et al., 2002).

The roasting operation includes several downstream ancillary/environmental control units to capture any dust or off gasses produced during the roasting process. The roasted product is then directed to a chiller and finally to a product surge bin. This bin provides a consistent feed to the leaching circuit.

**PM3: Leaching:** Following roasting, the feed material is conveyed to a series of cascading rubber lined tanks where leaching is used to extract the REEs from the refuse feedstock. In these tanks, the solid material is mixed with sulfuric acid and or other leaching lixiviants at a moderate solids concentration of 10% to 25% by weight. The retention time in this stage is uniquely calibrated for each feedstock and lixiviant to promote the optimal tradeoff between the recovery of REEs and major metal contaminants such as Al and Fe. A natural gas-powered solution heater is used to maintain an elevated temperature suitable for effective leaching.

After leaching, the material is passed to a thickener, where the pregnant leach solution is separated from the residual solids. The loaded leach solution is passed to a storage tank upstream of the concentration and recovery operations, while the residue reports to a belt filter for further dewatering. The primary effluent of the belt filter is combined with the thickener overflow in the leach solution storage tank, while the purge cycle effluent can be recycled to the leaching feed tank or directed to the wastewater treatment. The dewatered residual solids are discharged from the belt filter and conveyed back to the coarse refuse disposal area at the host site.

**PM4: Leachate Al/Fe Precipitation:** In one embodiment of the process, the PLS after leaching may be initially subjected to Sc recovery (see PM8); however, this step is optional, as scandium recovery may not be warranted for all feedstocks. Moreover, the remaining PLS (or SX raffinate if PM8 is employed) is too dilute for processing with direct oxalic acid precipitation. As a result, an intermediate stage of concentration is needed to increase the concentration of REEs while rejecting gangue metals to the largest extent possible.

Starting in PM4, this preliminary upgrading is achieved by staged precipitation and re-dissolution. During the first precipitation state, the pregnant leach solution is mixed with a base consisting of NaOH and/or calcite recovered from the coal preparation facility. This blend of base reagents raises the pH of the PLS to a point sufficient to precipitate all of the Fe and a significant portion of the Al contamination while retaining the REEs in solution. This pH adjustment process is conducted in a series of rubber lined tanks and/or thickener vessels with separate stages for Fe and Al precipitation. The residual cakes produced from the underflow from these vessels is a process waste that is first dewatered in a filter press and then conveyed to waste storage. The purified overflow product is pumped to PM5.

**PM5: REE Precipitation:** In this stage, a caustic solution is used to raise the pH of the leachate to the point where nearly all of the REEs are recovered into a solid precipitate. As a result, the overflow from this vessel is barren of REEs and sent to either final water treatment or the CM precipitation (PM9) or CM recovery circuit (PM11) if applicable. The REE-enriched precipitate from the thickener underflow is further dewatered in a filter press and then passed to RE-Redissolution stage. As in PM4, the reactions are conducted in rubber lined tanks and/or thickeners. Following this step, the REE-enriched precipitate is anticipated to have a concentration approximately 10 times greater than that of the initial feed that entered the leaching circuit

**PM6: REE Re-Dissolution:** In this module, the enriched REE solid is redissolved into solution using HCl. The reaction is conducted in a series of cascading polyethylene tanks in a manner similar to that of the leaching circuit (PM3). Following re-dissolution, a thickener is used to separate the concentrated REE solution from the residual solids. These solids are then dewatered in a filter press and conveyed to the final waste pile, while the enriched solution is pumped to final REO recovery.

**PM7: REE Oxalate Precipitation:** After selective precipitation and re-dissolution, the enriched REE leach solution is ready for final processing and recovery. In this process module, the REEs are precipitated from the leach solution using oxalic acid. This process step incorporates an oxalic/caustic mix tank, a leaching tank/thickener, a filter press, and a small product roaster. Overall, this process generates a concentrated REE-oxalate product, which is then roasted to generate a final mixed REE oxide with a nominal grade of 95% to 99% REO. These oxides are then suitable for toll refining or final sales, depending on the purchase order requirements and the final product purity. Any residual solution generated in this module is directed to the water treatment process prior to final discharge.

**PM8: Sc SX Separation:** This process module utilizes solvent extraction to directly recover scandium from the initial PLS, while leaving the other REEs in the raffinate. The raw leachate is first pumped to a two-stage loading circuit where the scandium is selectively recovered into the organic phase. To optimize for selective scandium recovery, an Orfom diluent and Cyanex 272 extractant are used in the organic phase. In the loading stage, an O:A ratio of 1:1 is maintained within the mixer; however, the organic is internally recycled in the loading stage to establish an

O:A advance ratio of 1:36. The raffinate from the loading stage is directed to the downstream REE recovery operations, while the loaded organic is pumped to the stripping stage. In stripping, 2 molar sulfuric acid is used as a stripping reagent. Here, an O:A ratio of 1:1 is maintained in the mixer; however, the acidic strip solution is internally recycled in the stripping stage to maintain an O:A advance ratio of 12.5:1. The resulting solution is then precipitated with 2 molar NaOH and filtered to produce a high grade $Sc_2O_3$ product.

**PM9: CM Precipitation:** In this process module, a caustic solution is used to further raise the pH of the Al-Fe-REE-depleted leach solution to the point where nearly all of the cobalt, nickel manganese, and other recoverable CMs selectively precipitate from solution. The CM-enriched precipitate from the thickener underflow is further dewatered in a filter press and then passed to CM-Redissolution stage.

**PM10: CM Redissolution:** Analogous to PM6, the enriched CM precipitate is redissolved into solution using HCl. The reaction is conducted in a series of cascading polyethylene tanks. Following re-dissolution, a thickener is used to separate the concentrated CM solution from the residual solids. These solids are then dewatered in a filter press and conveyed to the final waste pile, while the enriched solution is pumped to final CM recovery.

**PM11: CM Recovery:** The CM-enriched solution is pumped to a precipitation reactor where sodium sulfide ($Na_2S$) is used to selectively precipitate a cobalt-nickel-sulfide product. The solution is remaining solution is then neutralized with 4M NaOH to selectively precipitate manganese oxide. Precipitation contacting is conducted in polyethylene tanks which are followed by a thickener and filter press for solid liquid separation. Any residual solution generated in this module is directed to the water treatment process prior to final discharge.

### *3.2.5 Study Design*

Eleven pilot scale trials were conducted and included as base scenarios in the TEA analysis. These trials reflect five different circuit configurations, three different feedstocks, and various modifications to operational parameters. In addition, five “alternative” scenarios were also evaluated to address deviations between the laboratory and pilot scale data that were identified through the course of the project. As such, a total of 16 scenarios were ultimately assessed in the TEA. Table 3.2.7 summarizes these scenarios, and further detail on the unique attributes is given below. In addition, detailed TEA reports for all 16 scenarios are included as an *Appendix* to this report.

Table 3.2.7. Test matrix utilized for TEA analyses.

| Shakespeare Lignite (SHXP) | |
|---|---|
| C1 | CM recovery from both primary leachate and REOx raffinate with combined RE and CM redissolution leachate. |
| C2 | Simplified flowsheet, direct REOx and CM precipitation |
| C3 | CM recovery from both primary leachate and REOx raffinate |
| C4 | Primary flowsheet: CM recovery from REOx raffinate only |
| C5 | Same as C4, except with bio-acid/$H_2SO_4$ blend. |
| **West Kentucky #13 (WK)** | |
| 1 | Circuit 4 \|\| $H_2SO_4$ \|\| 45-minute leaching time |
| 2 | Circuit 4 \|\| $H_2SO_4$ \|\| 4-hour leaching time |
| 3 | Circuit 4 \|\| 0.3 M bio-acid \|\| 4-hour leaching time |
| 4 | Circuit 4 \|\| 0.5 M bio-acid \|\| 4-hour leaching time |
| **WK Heap Leach (PLS)** | |
| 3 | Circuit 3 approach to CM Recovery |
| 4 | Circuit 4 approach to CM Recovery |
| **Shakespeare Lignite, with improved recovery (SHXP-Alt)** | |
| C1-Alt | *Same as SHXP analysis; however, leach recoveries have been modified to reflect optimal results found during laboratory testing.* |
| C2-Alt | |
| C3-Alt | |
| C4-Alt | |
| C5-Alt | |

Of the 11 baseline scenarios, five utilized a high-grade lignite source (coded SHXP), four utilized an Illinois basin course coal reject (coded WK), and two utilized a heap leach solution generated from the WK material (coded PLS). The SHXP scenarios focused on the evaluation of five distinct circuit configurations representing a different serial arrangement of the various process modules. Figure 3.2.4 through Figure 3.2.6 show block flow diagrams of the various circuits, while Table 3.2.8 summarizes PMs included in each configuration.

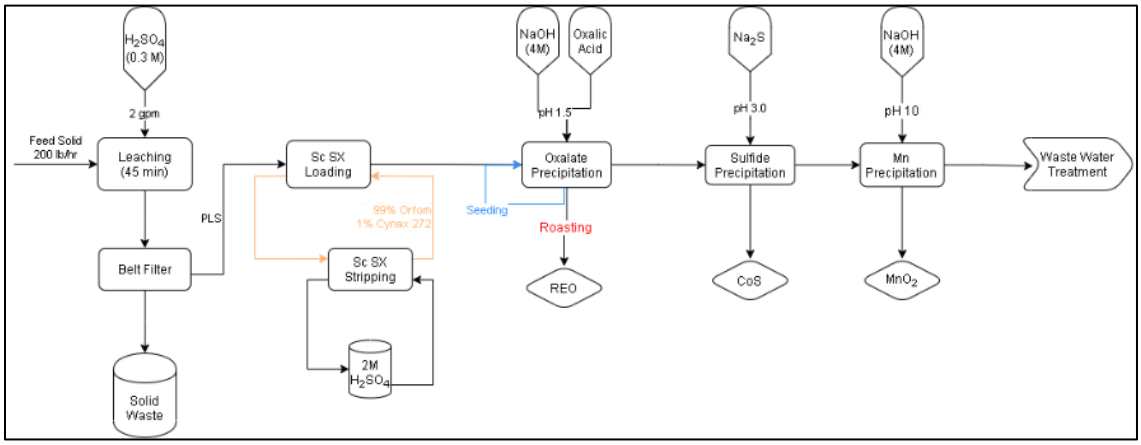



Figure 3.2.4. Circuit 2 block flow diagram.

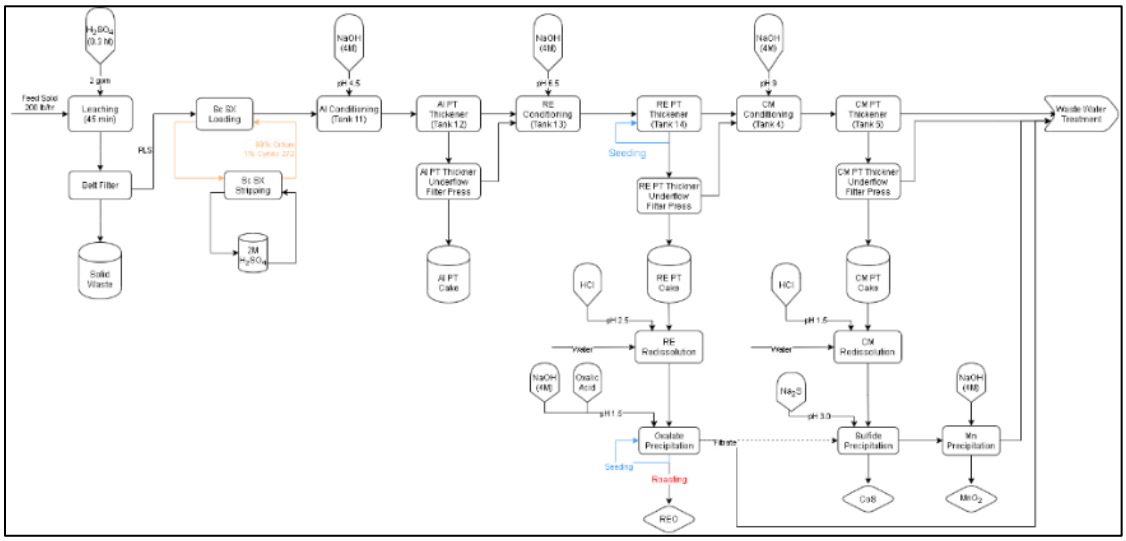


Figure 3.2.5. Circuit 1 and 3 block flow diagram. Note: in circuit 1, the redissolved RE cake and CM cake were combined prior to final product recovery.

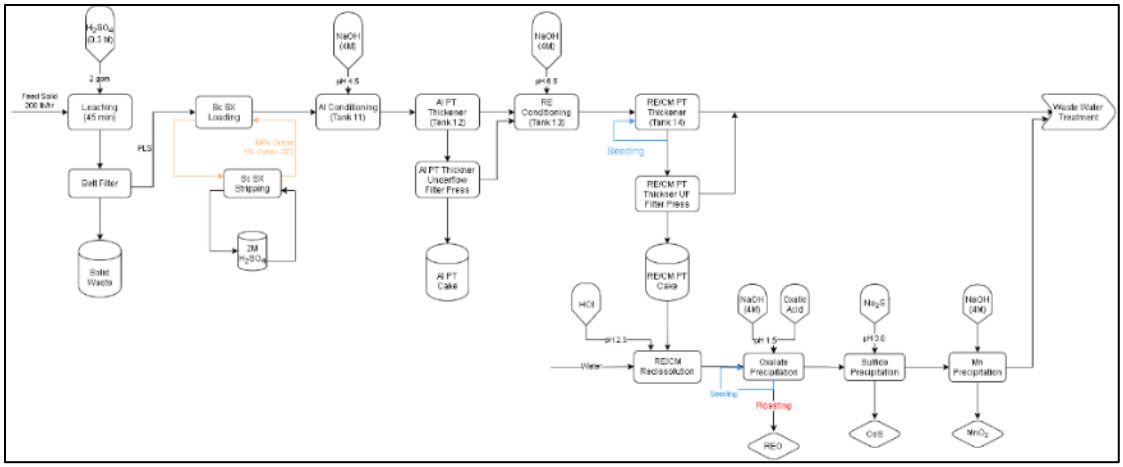



Figure 3.2.6. Circuit 4 and 5 block flow diagram. Note: circuit 5 utilizes bio-acid rather than $H_2SO_4$ in the leaching stage.

Table 3.2.8. Process modules included in various circuit configurations

| PM | Name | Circuit 1 | Circuit 2 | Circuit 3 | Circuit 4 | Circuit 5 |
|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | X | X | X | X | X |
| PM2 | Roasting | | | | | |
| PM3 | Leaching | X | X | X | X | X |
| PM4 | Al Precipitation | X | | X | X | X |
| PM5 | REE Precipitation | X | | X | X | X |
| PM6 | RE-Redissolution | X | | X | X | X |
| PM7 | REE Oxalate Precipitation | X | X | X | X | X |
| PM8 | Sc SX Separation | X | X | X | X | X |
| PM9 | CM Precipitation | X | | X | | |
| PM10 | CM Redissolution | X | | X | | |
| PM11 | CM Recovery | X | X | X | X | X |
| -- | Biooxidation | | | | | X |

Alternatively, the four WK tests focused on different operational conditions with a fixed circuit configuration. Parameters varied during the tests included acid concentrations, acid sources, and leaching retention times.

The two PLS tests focused on different approaches to CM recovery with one test using Circuit 3 and the other utilizing the Circuit 4 approach.

Lastly, during the pilot testing program, the research team noted that the REE recovery for the SHXP tests did not meet the projections anticipated from the corresponding laboratory-scale studies. To account for this variation, supplemental TEAs were conducted by integrating both the laboratory and pilot scale data as needed. Most notably, the pilot scale tests failed to produce the high leaching recoveries observed in the laboratory scale-tests. To mitigate this discrepancy, five "alternative" scenarios were also evaluated for the SHXP circuit configurations. These scenarios combined the laboratory-scale leaching recovery with the other input data collected from the pilot test campaign.

### *3.2.6 Cost Analysis*

Table 3.2.9 through 3.2.12 show the estimated capital expenditures (CAPEX) for each TEA scenario (WK, SHXP, PLS, and SHXP-Alt, respectively). Results show that the WK scenarios will require CAPEX expeditious ranging from $154M to $254M, while SHXP showed CAPEX requirements ranging from $82M to $108M, depending on the specific flowsheet configuration. The additional expense in the WK models was predominantly due to the roasting process used on the samples, with the roasting process accounting for more than half of the difference. WK also used longer leaching times in tests 2 through 4, which increased the required CAPEX. A significant expense in both the SHXP and WK samples was the precipitation steps, specifically the REE precipitation, as both had slow settling times, which would require additional equipment for those steps. SHXP C2 showed the lowest capital requirement of all tests, $82M, owing to the simple flowsheet configuration utilized in that scenario.

Table 3.2.9. WK CAPEX Breakdown

| | WK | | | |
|---|---|---|---|---|
| | **1** | **2** | **3** | **4** |
| Feed Preparation | $9,163,581 | $9,163,581 | $9,163,581 | $9,163,581 |
| Roasting | $72,849,280 | $72,849,280 | $72,849,280 | $72,849,280 |
| Leaching | $22,639,893 | $88,193,822 | $86,595,882 | $88,498,659 |
| Al Precipitation | $13,779,145 | $19,958,194 | $23,516,043 | $21,578,621 |
| REE Precipitation | $15,165,006 | $20,247,545 | $36,489,119 | $44,467,564 |
| RE-Redissolution | $3,036,541 | $4,037,130 | $5,920,978 | $6,277,746 |
| REE Oxalate Precipitation | $2,412,141 | $2,862,478 | $4,334,576 | $3,709,293 |
| Sc SX Separation | $12,480,211 | $12,929,157 | $12,572,021 | $12,571,365 |
| CM Precipitation | $0 | $0 | $0 | $0 |
| CM Redissolution | $0 | $0 | $0 | $0 |
| CM Recovery | $2,505,831 | $3,296,514 | $4,628,636 | $4,943,194 |
| Total: | $154,031,629 | $233,537,701 | $256,070,117 | $264,059,303 |

Table 3.2.10. SHXP CAPEX Breakdown

| | SHXP | | | | |
|---|---|---|---|---|---|
| | C1 | C2 | C3 | C4 | C5 |
| Feed Preparation | $9,163,581 | $9,163,581 | $9,163,581 | $9,163,581 | $9,163,581 |
| Roasting | $0 | $0 | $0 | $0 | $0 |
| Leaching | $33,157,675 | $23,858,076 | $23,499,808 | $26,909,047 | $23,864,858 |
| Al Precipitation | $12,513,307 | $12,433,636 | $15,775,322 | $12,827,895 | $11,310,802 |
| REE Precipitation | $20,234,138 | $0 | $30,528,004 | $21,556,204 | $32,260,412 |
| RE-Redissolution | $205,405 | $0 | $2,719,423 | $4,739,369 | $5,010,629 |
| REE Oxalate Precipitation | $1,987,039 | $9,547,410 | $3,104,282 | $3,790,645 | $2,926,327 |
| Sc SX Separation | $13,139,312 | $12,985,599 | $13,017,937 | $12,846,989 | $12,793,934 |
| CM Precipitation | $9,075,913 | $0 | $7,813,164 | $0 | $0 |
| CM Redissolution | $537,764 | $0 | $118,456 | $0 | $0 |
| CM Recovery | $1,411,945 | $14,336,453 | $2,392,025 | $3,852,515 | $4,150,888 |
| Total: | $101,426,081 | $82,324,754 | $108,132,002 | $95,686,245 | $101,481,430 |

Table 3.2.11. PLS CAPEX Breakdown

| | PLS | |
|---|---|---|
| | 3 | 4 |
| Feed Preparation | $0 | $0 |
| Roasting | $0 | $0 |
| Leaching | $0 | $0 |
| Al Precipitation | $18,348,007 | $13,173,832 |
| REE Precipitation | $20,958,666 | $15,971,566 |
| RE-Redissolution | $1,591,105 | $1,033,442 |
| REE Oxalate Precipitation | $1,554,836 | $1,056,395 |
| Sc SX Separation | $12,577,673 | $12,577,673 |
| CM Precipitation | $7,585,911 | $0 |
| CM Redissolution | $115,703 | $0 |
| CM Recovery | $1,507,323 | $1,024,159 |
| Total: | $64,239,224 | $44,837,068 |

Table 3.2.12. SHXP-Alt CAPEX Breakdown

| | SHXP-Alt | | | | |
|---|---|---|---|---|---|
| | **C1** | **C2** | **C3** | **C4** | **C5** |
| Feed Preparation | $9,163,581 | $9,163,581 | $9,163,581 | $9,163,581 | $9,163,581 |
| Roasting | $0 | $0 | $0 | $0 | $0 |
| Leaching | $33,157,675 | $23,858,076 | $23,499,808 | $26,909,047 | $23,864,858 |
| Al Precipitation | $12,513,307 | $12,433,636 | $15,775,322 | $12,827,895 | $11,310,802 |
| REE Precipitation | $20,234,138 | $0 | $30,528,004 | $21,556,204 | $32,260,412 |
| RE-Redissolution | $205,405 | $0 | $2,719,423 | $4,739,369 | $5,010,629 |
| REE Oxalate Precipitation | $1,987,039 | $9,547,410 | $3,104,282 | $3,790,645 | $2,926,327 |
| Sc SX Separation | $13,139,312 | $12,985,599 | $13,017,937 | $12,846,989 | $12,793,934 |
| CM Precipitation | $9,075,913 | $0 | $7,813,164 | $0 | $0 |
| CM Redissolution | $537,764 | $0 | $118,456 | $0 | $0 |
| CM Recovery | $1,411,945 | $14,336,453 | $2,392,025 | $3,852,515 | $4,150,888 |
| Total: | $101,426,081 | $82,324,754 | $108,132,002 | $95,686,245 | $101,481,430 |

Annual operating expenses (OPEX) were also estimated for each of the flowsheet and test methods used for the WK, SHXP, PLS, and SHXP-Alt scenarios and are shown in Table 3.2.13 through Table 3.2.16. While WK had the higher CAPEX requirement, it generally showed lower annual OPEX when compared to the SHXP process (WK = $262M to $282M; SHXP = $358M to $591M). It is worth noting that WK scenarios 3 and 4 and SHXP scenario C5 utilize bio-acid to lower their OPEX leaching costs; however, the lowered costs come with the tradeoff of increased contaminant treatment costs in the Al or REE precipitation steps. SHXP has a higher RE-redissolution and oxalate precipitation cost due to the higher calcium contamination in the process. As noted prior, SHXP C2 had the lowest CAPEX cost due to its simplistic circuit in comparison to the other circuits but has the highest OPEX expense when compared to the other circuits due to the high contaminant loading in the process.

Table 3.2.13. WK Annual OPEX Breakdown

| | WK | | | |
|---|---|---|---|---|
| | 1 | 2 | 3 | 4 |
| Feed Preparation | $616,370 | $616,370 | $616,370 | $616,370 |
| Roasting | $1,387,991 | $1,387,991 | $1,387,991 | $1,387,991 |
| Leaching | $195,791,662 | $193,965,996 | $131,852,003 | $140,574,247 |
| Al Precipitation | $4,682,365 | $6,071,503 | $21,302,291 | $12,626,100 |
| REE Precipitation | $1,575,560 | $3,773,532 | $5,924,358 | $5,934,525 |
| RE-Redissolution | $5,647,536 | $9,406,357 | $15,480,201 | $17,464,957 |
| REE Oxalate Precipitation | $7,878,039 | $12,437,745 | $22,389,537 | $29,886,107 |
| Sc SX Separation | $3,041,006 | $3,198,659 | $3,072,924 | $3,072,923 |
| CM Precipitation | $0 | $0 | $0 | $0 |
| CM Redissolution | $0 | $0 | $0 | $0 |
| CM Recovery | $5,254,479 | $4,886,005 | $14,687,231 | $18,064,289 |
| Unallocated | $43,960,321 | $47,067,362 | $45,583,634 | $47,547,663 |
| Total: | $269,835,330 | $282,811,518 | $262,296,539 | $277,175,172 |

Table 3.2.14. SHXP Annual OPEX Breakdown

| | SHXP | | | | |
|---|---|---|---|---|---|
| | C1 | C2 | C3 | C4 | C5 |
| Feed Preparation | $616,370 | $616,370 | $616,370 | $616,370 | $616,370 |
| Roasting | $0 | $0 | $0 | $0 | $0 |
| Leaching | $174,174,207 | $307,247,035 | $235,517,492 | $268,586,443 | $222,234,580 |
| Al Precipitation | $39,094,884 | $83,066,183 | $35,643,874 | $92,932,080 | $93,792,355 |
| REE Precipitation | $7,326,823 | $0 | $7,474,734 | $5,722,151 | $11,208,006 |
| RE-Redissolution | $25,653,672 | $0 | $43,640,376 | $18,741,007 | $23,205,877 |
| REE Oxalate Precipitation | $10,225,606 | $59,908,564 | $7,818,207 | $26,148,229 | $22,686,754 |
| Sc SX Separation | $3,192,321 | $3,138,835 | $3,230,084 | $3,169,634 | $3,150,923 |
| CM Precipitation | $7,083,715 | $0 | $5,973,361 | $0 | $0 |
| CM Redissolution | $34,623,958 | $0 | $2,497,763 | $0 | $0 |
| CM Recovery | $1,712,583 | $50,804,720 | $7,700,052 | $12,116,378 | $16,048,248 |
| Unallocated | $54,895,193 | $85,806,220 | $63,017,612 | $74,688,664 | $69,396,705 |
| Total: | $358,599,332 | $590,587,927 | $413,129,924 | $502,720,956 | $462,339,819 |

Table 3.2.15. PLS Annual OPEX Breakdown

| | PLS | |
|---|---|---|
| | 3 | 4 |
| Feed Preparation | $0 | $0 |
| Roasting | $0 | $0 |
| Leaching | $0 | $0 |
| Al Precipitation | $17,488,588 | $12,286,993 |
| REE Precipitation | $5,580,292 | $8,738,193 |
| RE-Redissolution | $4,828,563 | $31,256,024 |
| REE Oxalate Precipitation | $6,268,521 | $900,589 |
| Sc SX Separation | $3,074,905 | $3,074,905 |
| CM Precipitation | $966,730 | $0 |
| CM Redissolution | $9,420,360 | $0 |
| CM Recovery | $4,361,992 | $524,015 |
| Unallocated | $16,222,984 | $15,584,909 |
| Total: | $68,212,936 | $72,365,630 |

Table 3.2.16. SHXP-Alt Annual OPEX Breakdown

| | SHXP-Alt | | | | |
|---|---|---|---|---|---|
| | C1 | C2 | C3 | C4 | C5 |
| Feed Preparation | $616,370 | $616,370 | $616,370 | $616,370 | $616,370 |
| Roasting | $0 | $0 | $0 | $0 | $0 |
| Leaching | $174,174,207 | $307,247,035 | $235,517,492 | $268,586,443 | $222,234,580 |
| Al Precipitation | $39,094,884 | $83,066,183 | $35,643,874 | $92,932,080 | $93,792,355 |
| REE Precipitation | $7,326,823 | $0 | $7,474,734 | $5,722,151 | $11,208,006 |
| RE-Redissolution | $25,653,672 | $0 | $43,640,376 | $18,741,007 | $23,205,877 |
| REE Oxalate Precipitation | $10,225,606 | $59,908,564 | $7,818,207 | $26,148,229 | $22,686,754 |
| Sc SX Separation | $3,192,321 | $3,138,835 | $3,230,084 | $3,169,634 | $3,150,923 |
| CM Precipitation | $7,083,715 | $0 | $5,973,361 | $0 | $0 |
| CM Redissolution | $34,623,958 | $0 | $2,497,763 | $0 | $0 |
| CM Recovery | $1,712,583 | $50,804,720 | $7,700,052 | $12,116,378 | $16,048,248 |
| Unallocated | $58,180,651 | $87,933,796 | $65,482,080 | $77,158,157 | $72,366,707 |
| Total: | $361,884,789 | $592,715,503 | $415,594,391 | $505,190,449 | $465,309,821 |

### *3.2.7 Summarized Results*

Table 3.2.17 and Table 3.2.18 show the summarized technical and economic results from the 16 scenarios. Generally, the lower starting feed grade and lower recoveries associated with the WK scenarios prompted lower REE production rates and higher production costs when compared to the SHXP scenarios. As shown, the feed grade for the WK materials varied from 247 to 329 ppm depending on the test scenario, while the SHXP scenarios were evaluated at a fixed feed grade of 1250 ppm, as was determined during sample analysis. When combined with the changes in recovery, the net results were a significant change in total production rate (1.13 to 1.74 tpd for WK, 3.6 to 12.2 tpd for SHXP, and 7.0 to 27.5 tpd for SHXP-Alt. These results clearly show the impact of higher recovery in supporting superior technical and economic outcomes.

Table 3.2.17. Technical Indicators, by TEA Scenario

| Scenario | Feed (tph) | REE Grade (ppm) | Production Rate (tpd) | | | | Recovery (%) | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | MREO | Sc2O3 | CM Product | Total Product | REE | Sc | Mn | CM Sulfide |
| WK-1 | 500 | 274 | 0.37 | 0.02 | 0.74 | 1.13 | 13% | 9% | 20% | 8% |
| WK-2 | 500 | 283 | 0.42 | 0.01 | 1.31 | 1.74 | 14% | 7% | 33% | 14% |
| WK-3 | 500 | 306 | 0.47 | 0.03 | 0.65 | 1.15 | 14% | 14% | 19% | 6% |
| WK-4 | 500 | 329 | 0.36 | 0.03 | 0.85 | 1.24 | 10% | 14% | 10% | 19% |
| SHXP-C1 | 500 | 1250 | 1.49 | 0.05 | 2.09 | 3.64 | 11% | 15% | 8% | 22% |
| SHXP-C2 | 500 | 1250 | 6.05 | 0.13 | 12.51 | 18.68 | 44% | 39% | 24% | 27% |
| SHXP-C3 | 500 | 1250 | 3.86 | 0.11 | 6.04 | 10.01 | 28% | 33% | 14% | 39% |
| SHXP-C4 | 500 | 1250 | 4.28 | 0.14 | 7.82 | 12.24 | 31% | 37% | 9% | 38% |
| SHXP-C5 | 500 | 1250 | 0.86 | 0.12 | 4.48 | 5.45 | 6% | 29% | 33% | 29% |
| SHXP-Alt-C1 | 500 | 1250 | 3.49 | 0.31 | 3.25 | 7.04 | 26% | 87% | 14% | 33% |
| SHXP-Alt-C2 | 500 | 1250 | 9.70 | 0.28 | 17.56 | 27.54 | 71% | 87% | 33% | 38% |
| SHXP-Alt-C3 | 500 | 1250 | 6.37 | 0.30 | 8.65 | 15.32 | 47% | 87% | 20% | 56% |
| SHXP-Alt-C4 | 500 | 1250 | 7.13 | 0.33 | 11.05 | 18.50 | 52% | 87% | 13% | 54% |
| SHXP-Alt-C5 | 500 | 1250 | 1.77 | 0.35 | 6.29 | 8.42 | 13% | 87% | 44% | 42% |

Table 3.2.18. Economic Indicators, by TEA Scenario

| Scenario | Revenue (MM$/yr) | OPEX (MM$/yr) | OPEX ($/kg) | CAPEX (MM$) | CAPEX Intensity (MM $/(t/d)) | NPV (MM$) | Product Basket Price ($/kg) |
|---|---|---|---|---|---|---|---|
| WK-1 | $ 43.23 | $ 269.84 | $ 2,021.36 | $ 154.03 | $ 136.3 | $ (1,701) | $ 324 |
| WK-2 | $ 32.51 | $ 282.81 | $ 1,910.70 | $ 233.54 | $ 134.3 | $ (1,928) | $ 220 |
| WK-3 | $ 67.49 | $ 262.30 | $ 1,541.53 | $ 256.07 | $ 222.4 | $ (1,558) | $ 397 |
| WK-4 | $ 77.52 | $ 277.18 | $ 2,099.74 | $ 264.06 | $ 212.1 | $ (1,598) | $ 587 |
| SHXP-C1 | $ 130.15 | $ 358.60 | $ 254.29 | $ 101.43 | $ 27.9 | $ (1,674) | $ 92 |
| SHXP-C2 | $ 328.27 | $ 590.59 | $ 236.79 | $ 82.32 | $ 4.4 | $ (1,895) | $ 132 |
| SHXP-C3 | $ 296.37 | $ 413.13 | $ 231.68 | $ 108.13 | $ 10.8 | $ (899) | $ 166 |
| SHXP-C4 | $ 352.68 | $ 502.72 | $ 323.96 | $ 95.69 | $ 7.8 | $ (1,122) | $ 227 |
| SHXP-C5 | $ 270.47 | $ 462.34 | $ 628.02 | $ 101.48 | $ 18.6 | $ (1,418) | $ 367 |
| SHXP-Alt-C1 | $ 701.53 | $ 361.88 | $ 108.84 | $ 101.43 | $ 14.4 | $ 2,535 | $ 211 |
| SHXP-Alt-C2 | $ 698.28 | $ 592.72 | $ 148.19 | $ 82.32 | $ 3.0 | $ 800 | $ 175 |
| SHXP-Alt-C3 | $ 724.98 | $ 415.59 | $ 138.98 | $ 108.13 | $ 7.1 | $ 2,326 | $ 242 |
| SHXP-Alt-C4 | $ 782.16 | $ 505.19 | $ 194.01 | $ 95.69 | $ 5.2 | $ 2,129 | $ 300 |
| SHXP-Alt-C5 | $ 787.00 | $ 465.31 | $ 311.09 | $ 101.48 | $ 12.1 | $ 2,439 | $ 526 |

To explore the natural correlations and trends in the overall economic results, additional analyses were conducted comparing the various technical and economic output parameters. Figure 3.2.7, for example, shows the distribution of production costs ($/kg total) and CAPEX intensity (MM $/(t/d)) for the various test cases. The SHXP, and particularly SXHP-Alt scenarios, produced a lower capital and operating cost when normalized to a standard unit of production.

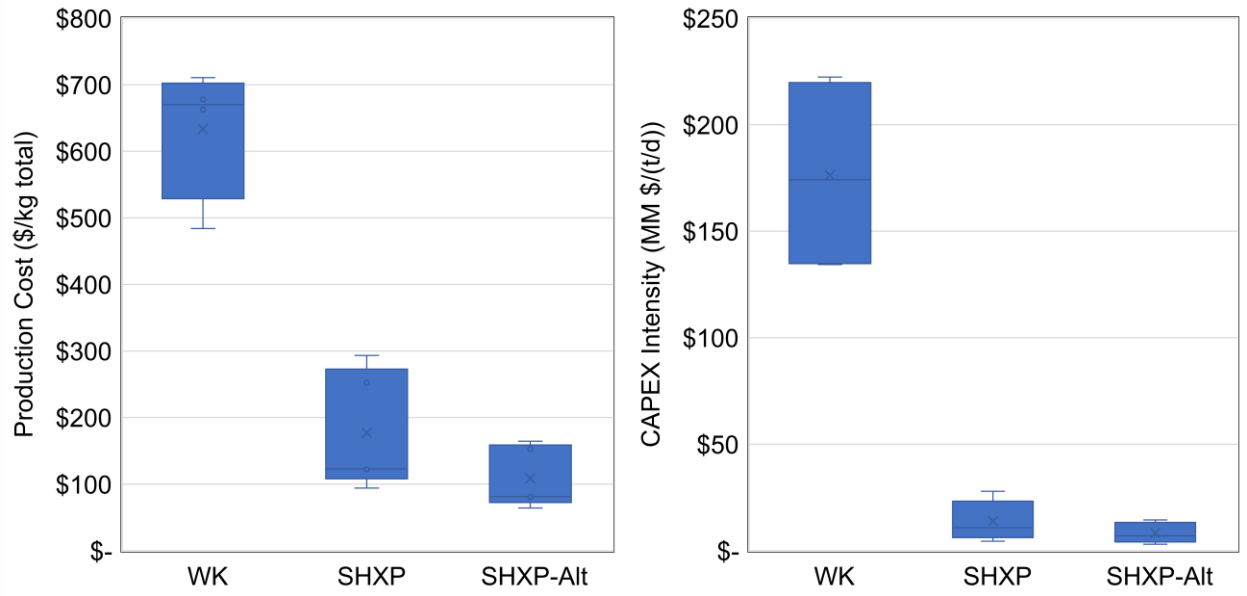



Figure 3.2.7. Distribution of Production Cost (left) and Capital Intensity (right) for TEA scenarios.

In addition, Figure 3.2.8 through Figure 3.2.11 show several correlations between production costs, production rate, recovery, capital intensity, and project NPV for the various TEA scenarios. Figure 3.2.8 shows the influence of production rate on total production costs. As anticipated, this follows an exponential trend with exceedingly high production costs below a production rate of 2 to 5 tpd. Since production rate is governed by feed grade, feed rate, and recovery, this graph shows the critical need for commercial scenarios to exceed 15 to 20 tpd of product to reach a point where the production costs best benefit from economy of scale.

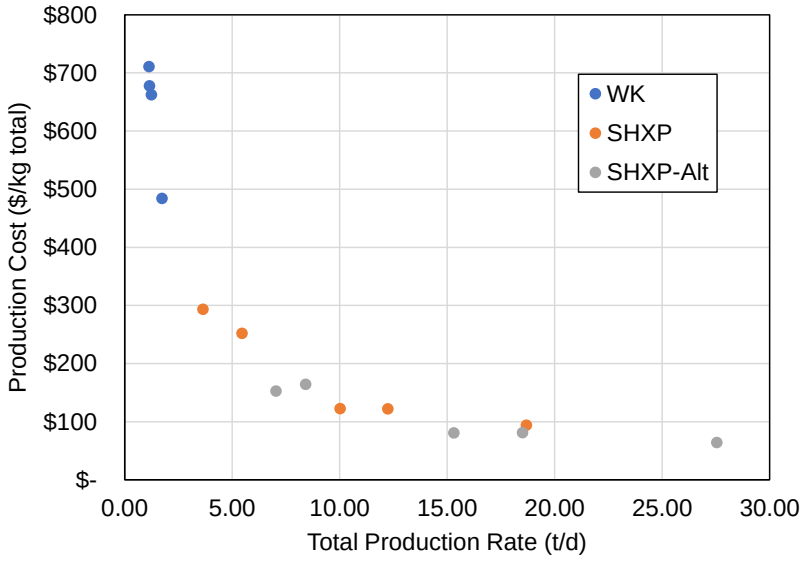



Figure 3.2.8. Production cost as a function of production rate for TEA scenarios.

Similarly, Figure 3.2.9 shows capital intensity as a function of REE recovery, which also shows an exponential trend. Note that this graph only includes the SHXP data, as the capital intensity for the WK scenarios was exceedingly high (500 to 700), which distorts the trends shown in this plot.

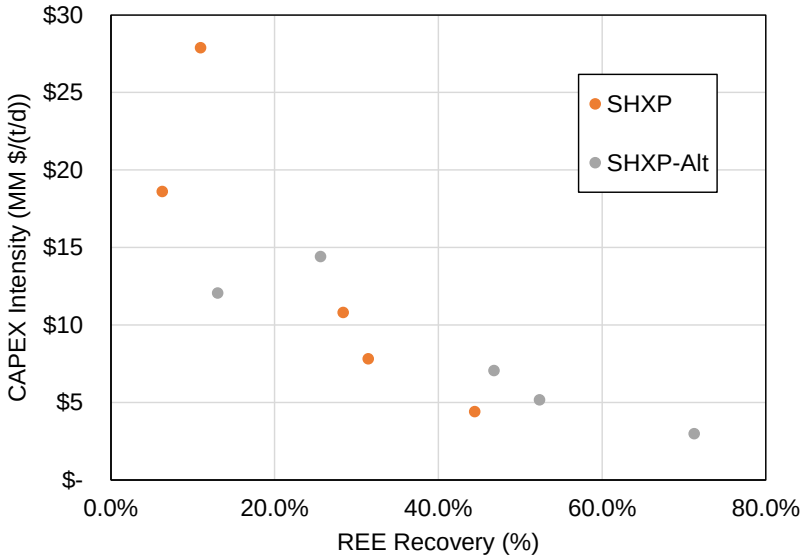



Figure 3.2.9. Capital intensity as a function of REE recovery for SHXP scenarios.

Figure 3.2.10 shows the relationship between project NPV and production cost for the TEA scenarios. As shown, only the SHXP-Alt cases produced positive NPVs, and notably, all of these scenarios also had low production costs, typically < $150/kg.

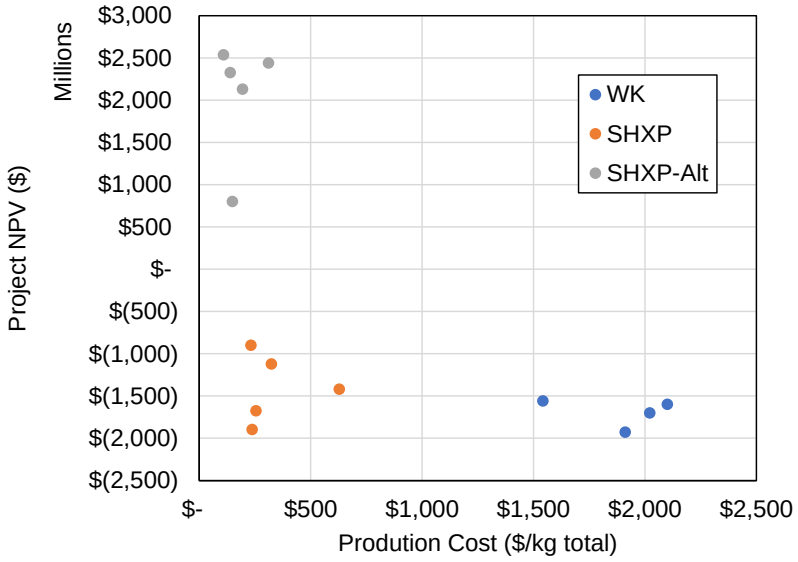



Figure 3.2.10. Influence of production cost on project NPV for TEA scenarios.

Lastly, Figure 3.2.11 shows the strong correlation between scandium production and annual revenue, which is consistent for all TEA scenarios. In all cases evaluated, scandium represented approximately 90% of the project revenue, which in turn yields the strong positive correlation in this plot. As a result, the sensitivity to scandium price was carefully assessed in the detailed scenario analysis.

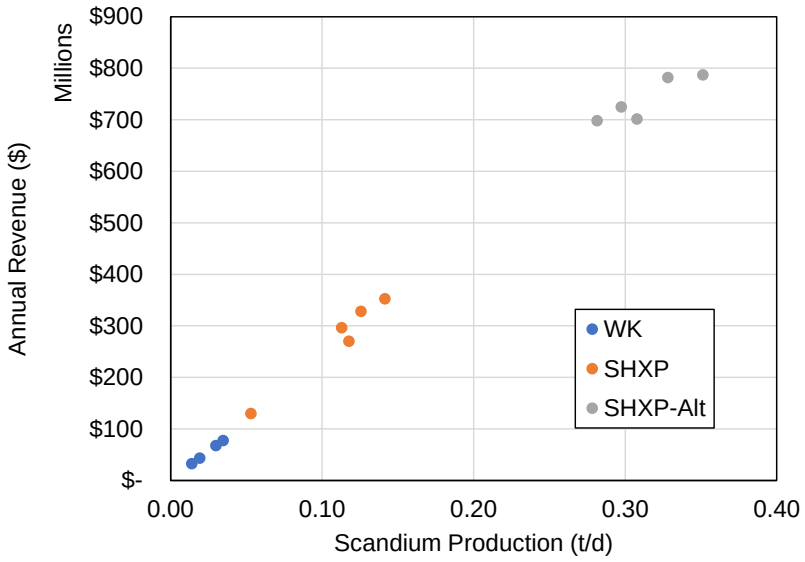



Figure 3.2.11. Influence of scandium production rate on annual revenue.

### *3.2.8 Detailed Analysis of WK4*

To demonstrate the relationship between the mass balance and economic assessment, a detailed assessment of the WK4 is presented herein. Similar data for the other scenarios is included in the *Appendix* to this report. While WK4 did not prompt an economically favorable outcome, it was selected for this detailed analysis to show possible pathways for improvement and ultimately to commercial viability. Moreover, the material balance and chemical consumption values in WK4 have been largely validated through pilot-scale testing.

For the WK4 Scenario, coarse refuse from the West Kentucky No. 13 seam in western Kentucky, was identified as the designated feedstock for the REE recovery process. This feedstock is readily available and was used for laboratory and pilot studies. For this scenario, the nominal feed rate was fixed at 500 st/hr, with the REE grade and distribution shown in Table 3.2.19. Given these feed rate and grade values, the total mass flow of REEs into the process from the feed solids is calculated to be 150 kg/hr, while the total rate of non-REE CM was determined to be 142 kg/hr.

Table 3.2.19. Feedstock REE/CM Concentration used in WK4 Analysis

| Element | Feed Conc. (mg/kg) | Distribution (%) |
|---|---|---|
| Scandium | 22.69 | 6.89 |
| Yttrium | 25.76 | 7.82 |
| Lanthanum | 54.70 | 16.60 |
| Cerium | 125.57 | 38.12 |
| Praseodymium | 14.63 | 4.44 |
| Neodymium | 49.05 | 14.89 |
| Samarium | 10.95 | 3.32 |
| Europium | 1.88 | 0.57 |
| Gadolinium | 10.40 | 3.16 |
| Terbium | 0.25 | 0.08 |
| Dysprosium | 3.67 | 1.11 |
| Holmium | 0.82 | 0.25 |
| Erbium | 3.89 | 1.18 |
| Thulium | 0.56 | 0.17 |
| Ytterbium | 3.60 | 1.09 |
| Lutetium | 0.99 | 0.30 |
| Manganese | 142.00 | -- |
| Cobalt | 21.70 | |
| Nickel | 46.20 | |
| Zinc | 102.40 | |
| **Total REE** | **329.42** | **100.00** |
| **Total Non-REE CM** | **312.30** | -- |

To determine the overall recovery of REE/CMs to the product stream, element-by-element recovery values from individual unit operations were combined in a simple multiplicative manner. Table 3.2.20 shows these recovery values, itemized by individual REE and unit operation. For this current scenario, these recovery values were determined from continuous pilot scale testing of the integrated process.

To support the economic analysis, these recovery values and other technical factors and reagent consumption rates were used to generate simplified material balances for each individual process module. These material balances not only show major solid mass and liquid flow, but they also show the balance of REEs throughout the circuit as well as the dosing rates for major consumables. Figure 3.2.12 through Figure 3.2.20 show these material balances for process modules PM1 through PM11, respectively (PM 9 and PM 10 are not utilized in this circuit configuration).

Table 3.2.20. Element-by-element recovery values by unit operation used in techno-economic analysis

| REE | Recovery in Various Unit Operations | | | | | | | | Overall Recovery | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Leaching | Al/Fe Precipitation | REE Precipitation | RE Re-Dissolution | REOx Precipitation | Ca-Ox Precipitation | Co Precipitation | Mn Precipitation | MREO Product | Co-Ni Product | Mn Product |
| Scandium | 14.2% | 25.1% | 100.0% | 73.9% | 3.6% | 27.8% | 1.5% | 80.5% | 0.0% | 0.0% | 0.0% |
| Yttrium | 23.7% | 51.3% | 100.0% | 64.1% | 59.2% | 98.0% | 26.2% | 60.3% | 4.6% | 0.0% | 0.0% |
| Lanthanum | 42.7% | 51.3% | 100.0% | 74.2% | 77.4% | 98.3% | 86.1% | 28.7% | 12.6% | 0.1% | 0.0% |
| Cerium | 41.3% | 43.9% | 100.0% | 73.3% | 84.4% | 83.4% | 5.2% | 5.8% | 11.2% | 0.0% | 0.0% |
| Praseodymium | 39.7% | 41.4% | 100.0% | 71.0% | 87.0% | 88.2% | 52.2% | 27.3% | 10.1% | 0.1% | 0.0% |
| Neodymium | 45.8% | 42.1% | 100.0% | 73.3% | 90.5% | 95.8% | 63.6% | 79.6% | 12.8% | 0.0% | 0.0% |
| Samarium | 40.3% | 44.6% | 100.0% | 70.8% | 93.7% | 87.0% | 1.3% | 100.0% | 11.9% | 0.0% | 0.1% |
| Europium | 34.2% | 45.1% | 100.0% | 72.5% | 93.2% | 100.0% | 0.0% | 0.0% | 10.4% | 0.0% | 0.0% |
| Gadolinium | 28.6% | 41.7% | 100.0% | 67.9% | 88.7% | 80.4% | 25.5% | 38.5% | 7.2% | 0.0% | 0.1% |
| Terbium | 41.1% | 64.8% | 100.0% | 87.3% | 83.5% | 0.0% | 0.0% | 2.2% | 19.4% | 0.0% | 0.1% |
| Dysprosium | 36.9% | 49.7% | 100.0% | 64.6% | 75.3% | 85.1% | 0.0% | 12.4% | 8.9% | 0.0% | 0.1% |
| Holmium | 25.8% | 55.4% | 100.0% | 56.7% | 60.7% | 77.7% | 0.0% | 3.2% | 4.9% | 0.0% | 0.0% |
| Erbium | 15.2% | 47.8% | 100.0% | 62.9% | 53.9% | 73.7% | 0.0% | 5.3% | 2.5% | 0.0% | 0.0% |
| Thulium | 12.5% | 53.5% | 100.0% | 62.5% | 48.4% | 80.4% | 4.4% | 100.0% | 2.0% | 0.0% | 0.4% |
| Ytterbium | 13.7% | 45.6% | 100.0% | 59.8% | 36.9% | 77.3% | 16.4% | 11.0% | 1.4% | 0.1% | 0.0% |
| Lutetium | 5.8% | 54.0% | 100.0% | 62.5% | 37.7% | 80.4% | 53.4% | 100.0% | 0.7% | 0.1% | 0.1% |
| Manganese | 61.3% | 76.4% | 90.4% | 85.8% | 0.6% | 12.6% | 67.2% | 92.6% | 0.2% | 21.2% | 9.6% |
| Cobalt | 36.3% | 69.5% | 97.5% | 87.5% | 3.8% | 25.9% | 84.7% | 100.0% | 0.8% | 13.0% | 2.4% |
| Nickel | 52.2% | 68.3% | 98.4% | 80.9% | 2.7% | 18.2% | 77.6% | 83.0% | 0.8% | 17.5% | 4.2% |
| Zinc | 61.3% | 71.1% | 99.1% | 85.4% | 6.4% | 39.4% | 98.8% | 86.5% | 2.4% | 20.6% | 0.2% |

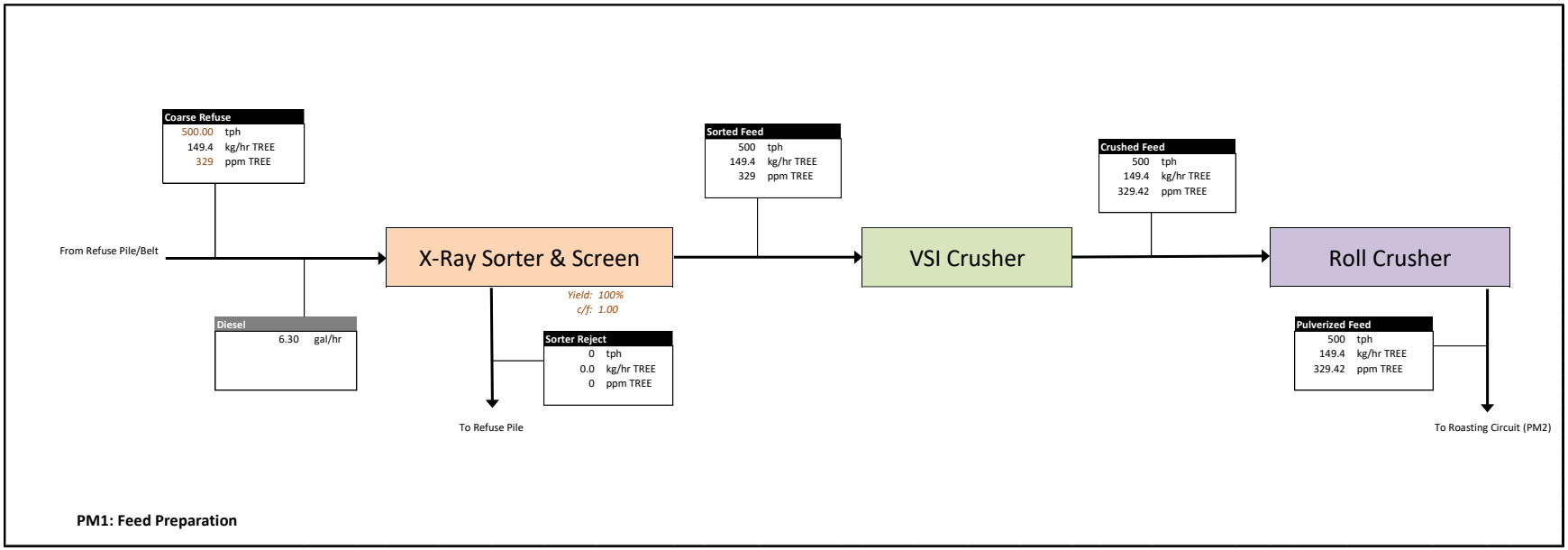



Figure 3.2.12. Block Flow Diagram for PM1: Feed Preparation (WK4).

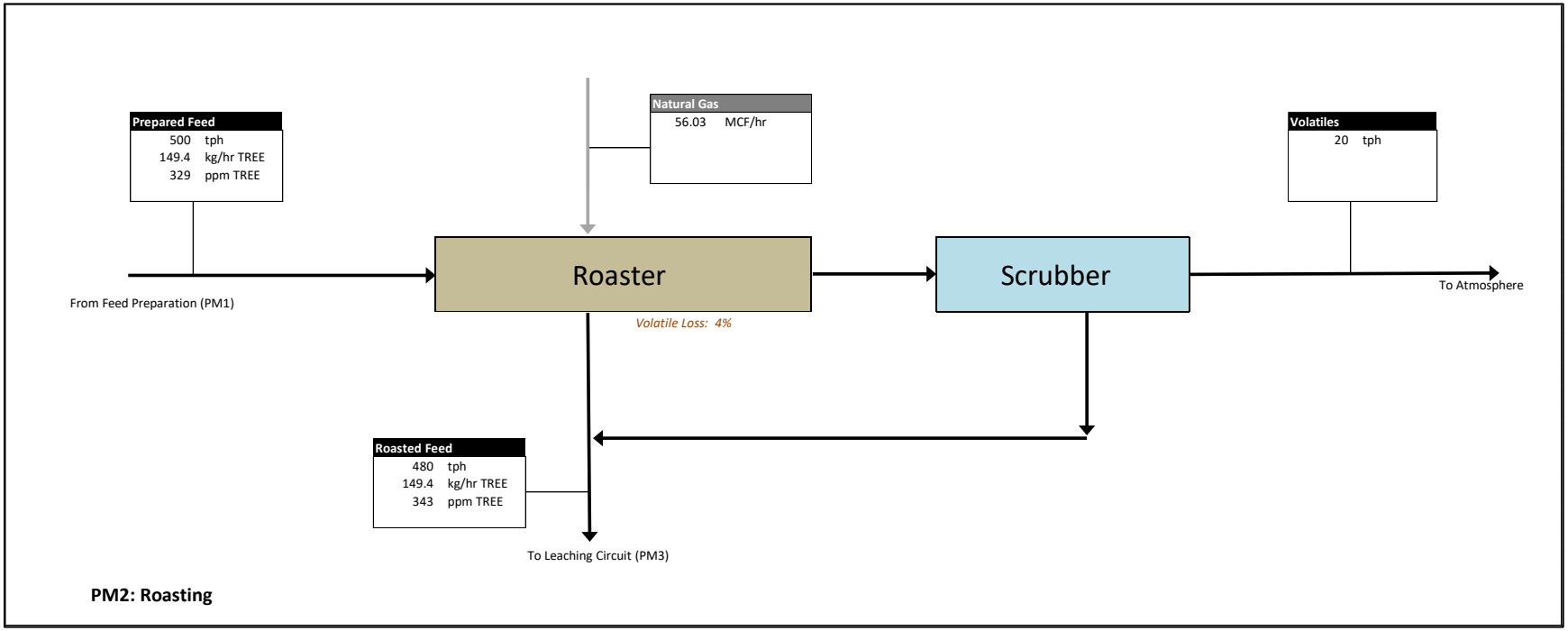



Figure 3.2.13. Block Flow Diagram for PM2: Roasting (WK4).

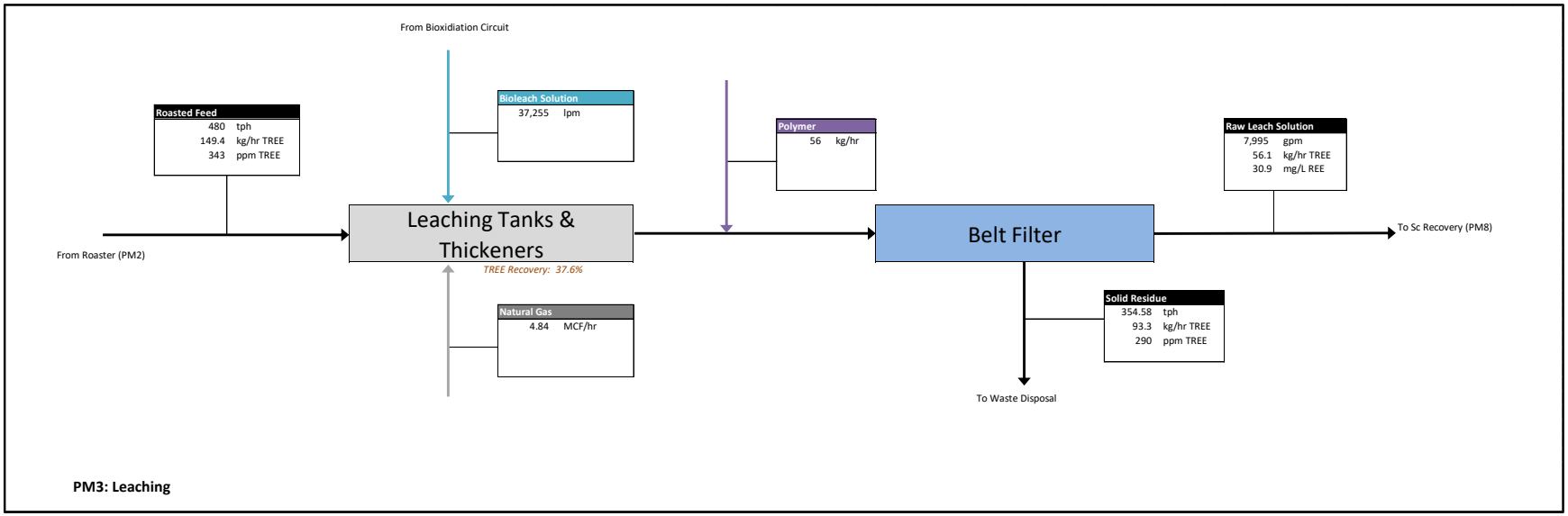



Figure 3.2.14. Block Flow Diagram for PM3: Leaching (WK4).

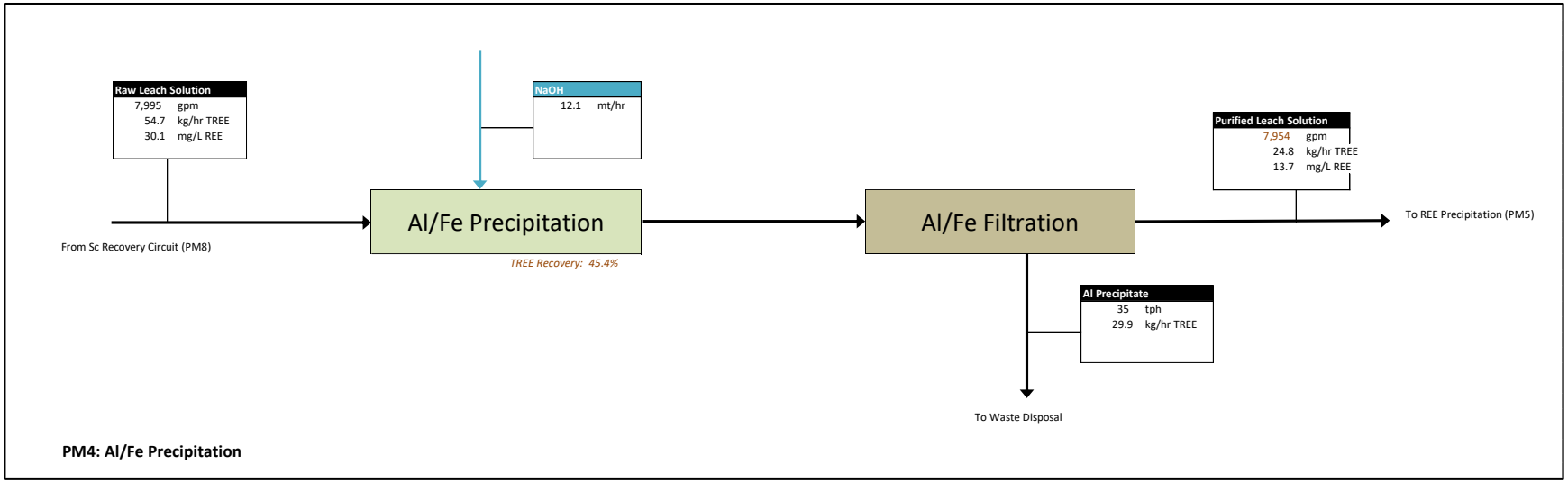



Figure 3.2.15. Block Flow Diagram for PM4 Al/Fe Precipitation (WK4).

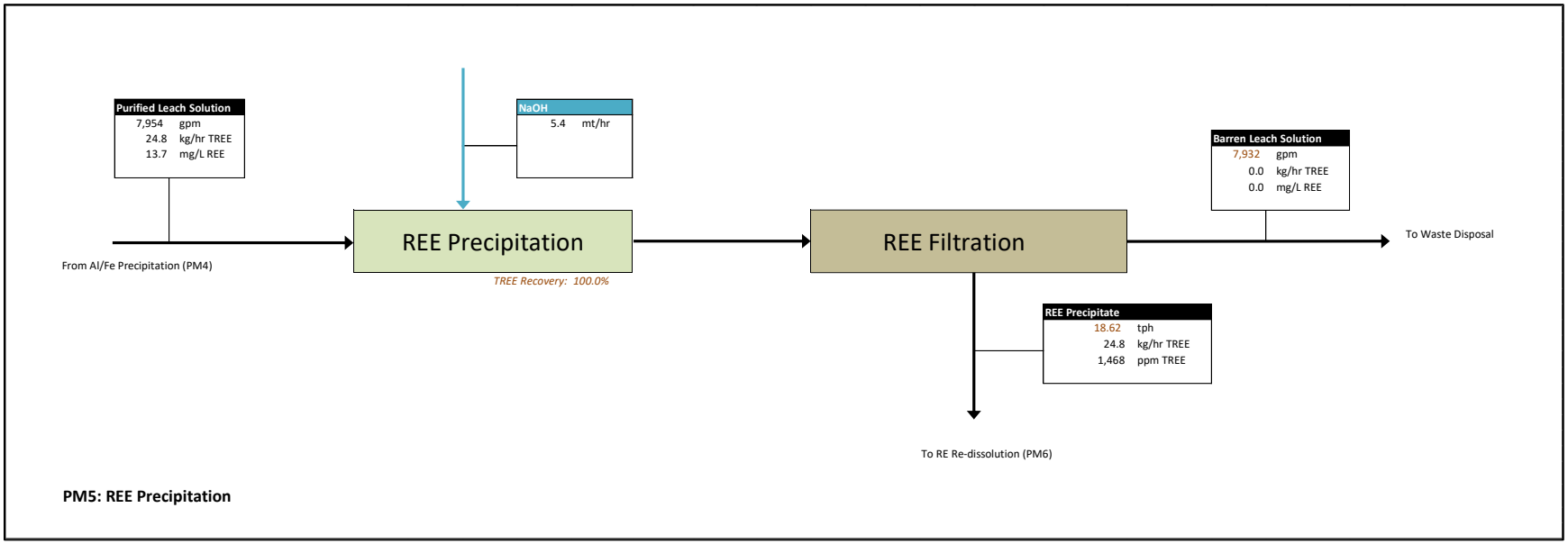



Figure 3.2.16. Block Flow Diagram for PM5: REE Precipitation (WK4).

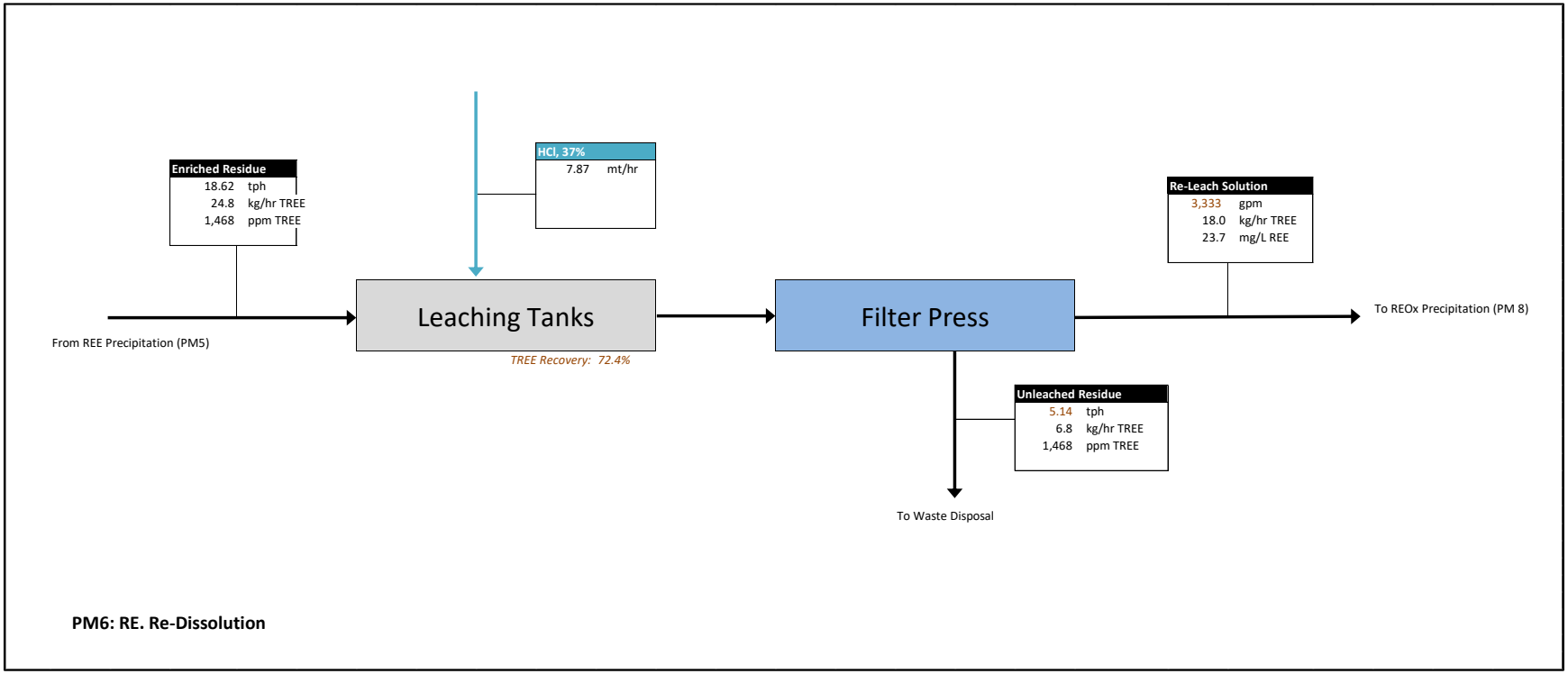



Figure 3.2.17. Block Flow Diagram for PM6: RE Re-dissolution (WK4).

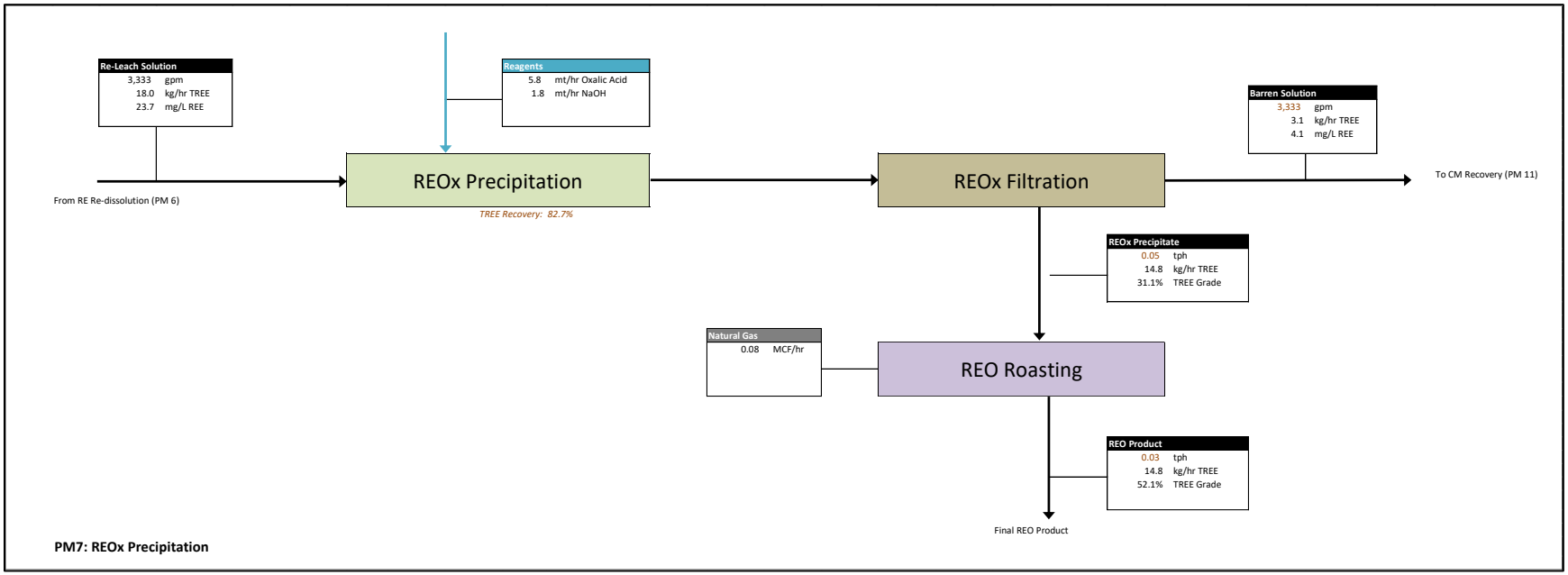



Figure 3.2.18. Block Flow Diagram for PM7: REOx Precipitation (WK4).

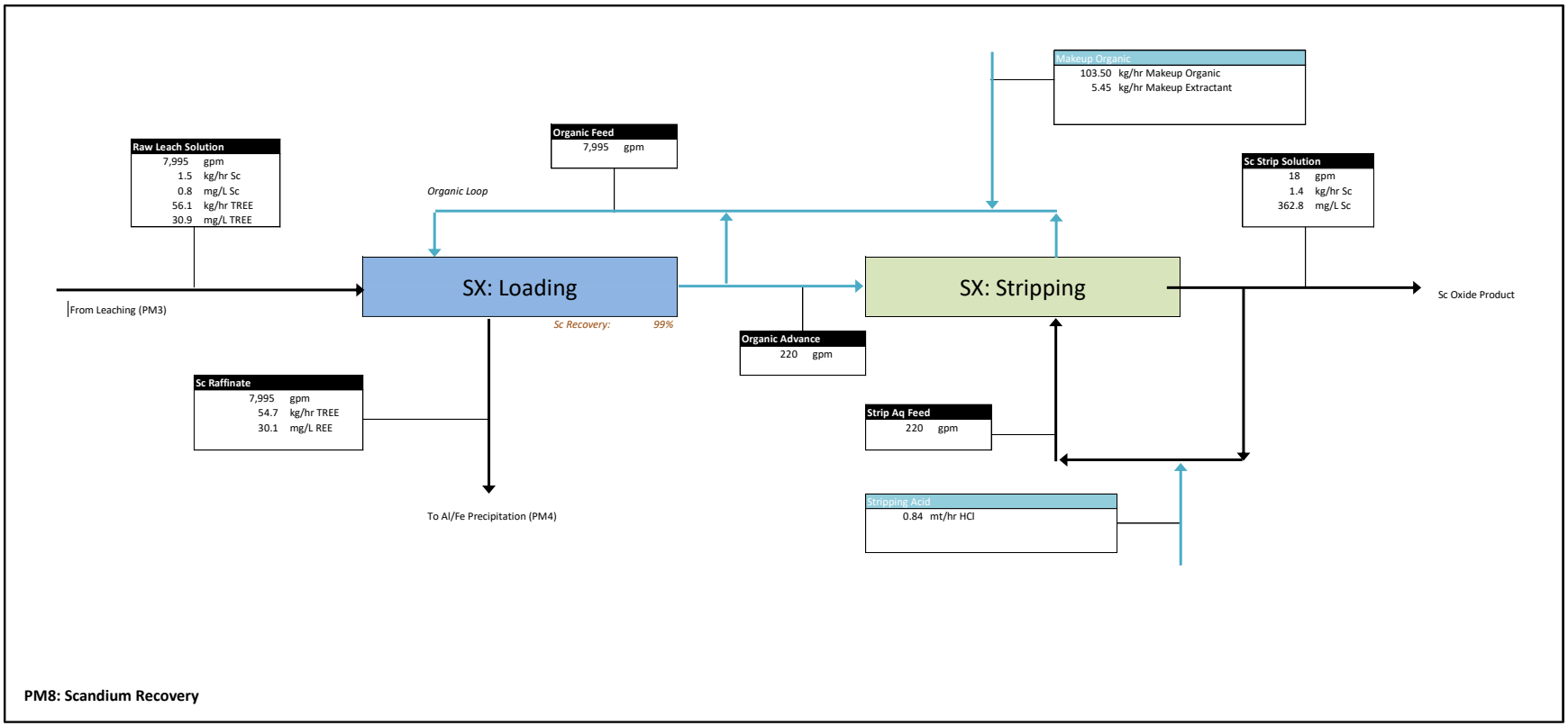



Figure 3.2.19. Block Flow Diagram for PM8: Sc Recovery (WK4).

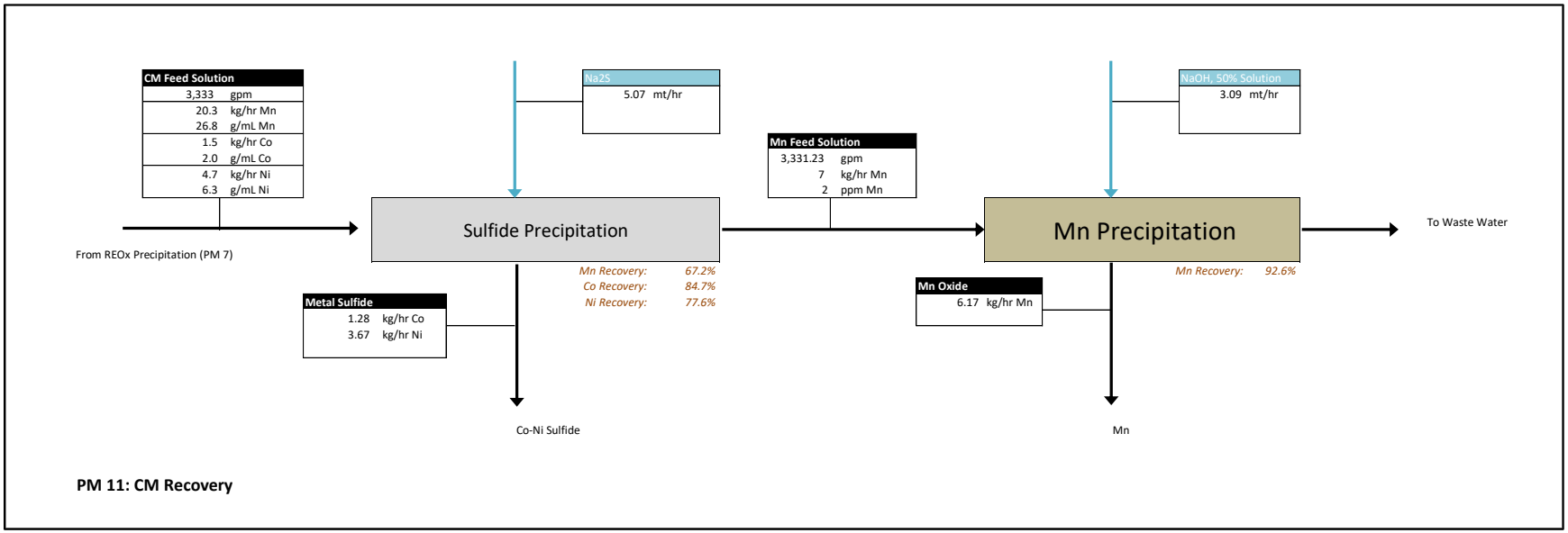



Figure 3.2.20. Block Flow Diagram for PM11 CM Recovery (WK4).

Using these material balances as well as the cost models described above, the total capital cost for the 500 st/hr plant was determined to be approximately $264 million. Table 3.2.21 shows this cost itemized by process module and capital cost category, while Figure 3.2.21 shows a graphical breakdown of this same data. This analysis shows that the roasting circuit and the leaching circuit are the most significant capital costs, each contributing approximately 30% of the total.

Table 3.2.21. Module-level capital cost for commercial scale plant, WK4 (500 st/hr).

| Module ID | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 |
| PM2 | Roasting | $ 19,585,772 | $ - | $ 4,896,443 | $ 21,544,350 | $ 26,822,715 | $ 72,849,280 |
| PM3 | Leaching | $ 23,793,160 | $ - | $ 5,948,290 | $ 26,172,476 | $ 32,584,733 | $ 88,498,659 |
| PM4 | Al Precipitation | $ 5,801,484 | $ - | $ 1,450,371 | $ 6,381,633 | $ 7,945,133 | $ 21,578,621 |
| PM5 | REE Precipitation | $ 11,955,253 | $ - | $ 2,988,813 | $ 13,150,779 | $ 16,372,719 | $ 44,467,564 |
| PM6 | RE-Redissolution | $ 1,687,793 | $ - | $ 421,948 | $ 1,856,572 | $ 2,311,432 | $ 6,277,746 |
| PM7 | REE Oxalate Precipitation | $ 997,256 | $ - | $ 249,314 | $ 1,096,982 | $ 1,365,742 | $ 3,709,293 |
| PM8 | Sc SX Separation | $ 2,127,781 | $ 4,657,083 | $ 531,945 | $ 2,340,559 | $ 2,913,996 | $ 12,571,365 |
| PM11 | CM Recovery | $ 1,328,994 | $ - | $ 332,249 | $ 1,461,894 | $ 1,820,057 | $ 4,943,194 |
| | **TOTAL:** | **$ 69,741,153** | **$ 4,657,083** | **$ 17,435,288** | **$ 76,715,269** | **$ 95,510,510** | **$ 264,059,303** |

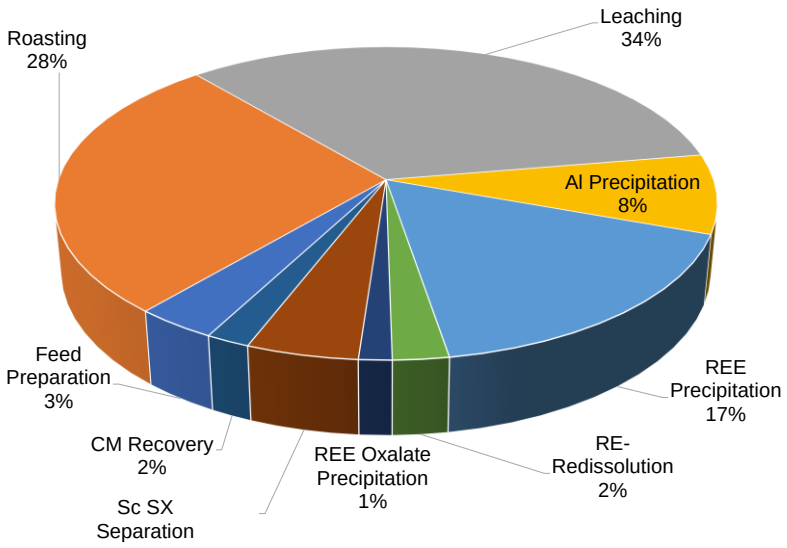



Figure 3.2.21. Capital cost breakdown for WK4 scenario

Final summaries of total operating cost by cost type and cost center are given in Table 3.2.22, while a graphical breakdown is shown in Figure 3.2.22. As shown, the total operating cost for this process was determined to be $2100/kg REE produced ($663/kg total product). An assessment of the cost breakdown shows that consumables are the single largest cost item, representing nearly 80% of the total costs. General overhead costs constituted the next largest item; however, this category was a much smaller fraction, only 13% of the total.

With respect to process module, the leaching operation was the costliest, representing over 50% of the total. Given the high relative significance of power and chemical costs, those expenses have been itemized separately, by process module, Table 3.2.23 and Table 3.2.24, respectively. This data shows that the total chemical cost is $1700/kg REE, with the primary contributors being leaching and to a lesser extent leachate precipitation. Likewise, power costs were much smaller in magnitude, only requiring $37/kg REE. Altogether, these results indicate that leaching acid is

the single highest cost item for process and likely, the item with the most potential for improvement.

Table 3.2.22. Total operating cost breakdown for commercial-scale plant, WK4 (500 st/hr.).

**Summary by Cost Type**

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 10,605.25 | $ 31,815.74 | $ 10,690,088 | $ 81 | $ 2.65 | I |
| Power | $ 4,873.98 | $ 14,621.94 | $ 4,912,972 | $ 37 | $ 1.22 | |
| Consumables | $ 222,931.09 | $ 668,793.27 | $ 224,714,537 | $ 1,702 | $ 55.73 | IIIIIIIIIIIIIIIIIIIIIIIII |
| Leasing/Tolling | $ - | $ - | $ - | $ - | $ - | |
| Waste Disposal | $ 698.70 | $ 2,096.10 | $ 704,291 | $ 5 | $ 0.17 | |
| General Overhead | $ 35,866.35 | $ 107,599.06 | $ 36,153,283 | $ 274 | $ 8.97 | IIII |
| TOTAL | $ 274,975 | $ 824,926 | $ 277,175,172 | $ 2,100 | $ 68.74 | |

**Summary by Process Module**

| Process Module | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Feed Preparation | $ 611 | $ 1,834 | $ 616,370 | $ 4.67 | $ 0.15 | |
| Roasting | $ 1,377 | $ 4,131 | $ 1,387,991 | $ 10.51 | $ 0.34 | |
| Leaching | $ 139,459 | $ 418,376 | $ 140,574,247 | $ 1,064.92 | $ 34.86 | IIIIIIIIIIIIIII |
| Al Precipitation | $ 12,526 | $ 37,578 | $ 12,626,100 | $ 95.65 | $ 3.13 | I |
| REE Precipitation | $ 5,887 | $ 17,662 | $ 5,934,525 | $ 44.96 | $ 1.47 | |
| RE-Redissolution | $ 17,326 | $ 51,979 | $ 17,464,957 | $ 132.31 | $ 4.33 | II |
| REE Oxalate Precipitation | $ 29,649 | $ 88,947 | $ 29,886,107 | $ 226.40 | $ 7.41 | III |
| Sc SX Separation | $ 3,049 | $ 9,146 | $ 3,072,923 | $ 23.28 | $ 0.76 | |
| CM Precipitation | $ - | $ - | $ - | $ - | $ - | |
| CM Redissolution | $ - | $ - | $ - | $ - | $ - | |
| CM Recovery | $ 17,921 | $ 53,763 | $ 18,064,289 | $ 136.85 | $ 4.48 | II |
| Biooxidation | $ - | $ - | $ - | $ - | $ - | |
| Reserved | $ - | $ - | $ - | $ - | $ - | |
| Reserved | $ - | $ - | $ - | $ - | $ - | |
| Reserved | $ - | $ - | $ - | $ - | $ - | |
| Unallocated | $ 47,170 | $ 141,511 | $ 47,547,663 | $ 360 | $ 11.79 | IIIII |
| TOTAL | $ 274,975 | $ 824,926 | $ 277,175,172 | $ 2,100 | $ 68.74 | |

**Summary by Cost Center**

| Cost Center | Code | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| Operating Labor | OL | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 23 | $ 0.76 | |
| Technical Labor | TL | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 9 | $ 0.29 | |
| Power/Utilities | PR | $ 4,873.98 | $ 14,621.94 | $ 4,912,972 | $ 37 | $ 1.22 | |
| Lease Agreements | LS | $ - | $ - | $ - | $ - | $ - | |
| Consumables/Reagents | CR | $ 222,931.09 | $ 668,793.27 | $ 224,714,537 | $ 1,702 | $ 55.73 | IIIIIIIIIIIIIIIIIIIIIIIII |
| Waste Disposal | WD | $ 698.70 | $ 2,096.10 | $ 704,291 | $ 5 | $ 0.17 | |
| Plant Overhead | OH | $ 38,790.85 | $ 116,372.55 | $ 39,101,175 | $ 296 | $ 9.70 | IIII |
| General Sales, and Administration | GSA | $ 3,461.45 | $ 10,384.35 | $ 3,489,142 | $ 26 | $ 0.87 | |
| Reserved | | | | | | | |
| Reserved | | | | | | | |
| TOTAL | | $ 274,975 | $ 824,926.11 | $ 277,175,172 | $ 2,100 | $ 68.74 | |

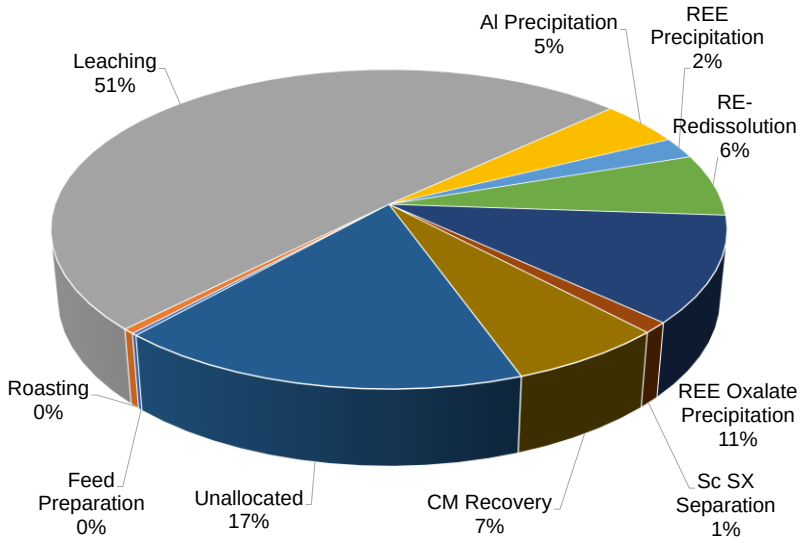



Figure 3.2.22. Operating cost breakdown for WK4 Scenario.

Table 3.2.23. Power cost summary by process module for commercial-scale plant, WK4 (500 st/hr).

| Module ID | Process Module | Total HP | Shift Cost | Daily Cost | Annual Cost | Cost Per Total Product kg | Cost Per Plant Feed Short-Ton |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | 1,065 | $ 510.68 | $ 1,532 | $ 514,764 | $ 3.90 | $ 0.13 |
| PM2 | Roasting | 198 | $ 94.99 | $ 285 | $ 95,747 | $ 0.73 | $ 0.02 |
| PM3 | Leaching | 5,058 | $ 2,424.73 | $ 7,274 | $ 2,444,130 | $ 18.52 | $ 0.61 |
| PM4 | Al Precipitation | 928 | $ 444.64 | $ 1,334 | $ 448,198 | $ 3.40 | $ 0.11 |
| PM5 | REE Precipitation | 984 | $ 471.49 | $ 1,414 | $ 475,265 | $ 3.60 | $ 0.12 |
| PM6 | RE-Redissolution | 683 | $ 327.52 | $ 983 | $ 330,138 | $ 2.50 | $ 0.08 |
| PM7 | REE Oxalate Precipitation | 283 | $ 135.83 | $ 407 | $ 136,913 | $ 1.04 | $ 0.03 |
| PM8 | Sc SX Separation | 499 | $ 239.42 | $ 718 | $ 241,331 | $ 1.83 | $ 0.06 |
| PM11 | CM Recovery | 469 | $ 224.69 | $ 674 | $ 226,486 | $ 1.72 | $ 0.06 |
| | **Total:** | **10,168** | **$ 4,873.98** | **$ 14,622** | **$ 4,912,972** | **$ 37.22** | **$ 1.22** |

Table 3.2.24. Chemical cost summary by process module for commercial-scale plant, WK4 (500 st/hr).

| Module ID | Process Module | Shift Cost | Daily Cost | Annual Cost | Cost Per Total Product kg | Cost Per Plant Feed Short-Ton |
|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 100.80 | $ 302 | $ 101,606 | $ 0.77 | $ 0.03 |
| PM2 | Roasting | $ 1,281.99 | $ 3,846 | $ 1,292,244 | $ 9.79 | $ 0.32 |
| PM3 | Leaching | $ 137,033.85 | $ 411,102 | $ 138,130,117 | $ 1,046.40 | $ 34.26 |
| PM4 | Al Precipitation | $ 12,081.25 | $ 36,244 | $ 12,177,902 | $ 92.25 | $ 3.02 |
| PM5 | REE Precipitation | $ 5,415.93 | $ 16,248 | $ 5,459,260 | $ 41.36 | $ 1.35 |
| PM6 | RE-Redissolution | $ 16,998.83 | $ 50,996 | $ 17,134,818 | $ 129.80 | $ 4.25 |
| PM7 | REE Oxalate Precipitation | $ 29,513.09 | $ 88,539 | $ 29,749,194 | $ 225.36 | $ 7.38 |
| PM8 | Sc SX Separation | $ 2,809.12 | $ 8,427 | $ 2,831,592 | $ 21.45 | $ 0.70 |
| PM11 | CM Recovery | $ 17,696.23 | $ 53,089 | $ 17,837,803 | $ 135.13 | $ 4.42 |
| | **Total:** | **$ 222,931.09** | **$ 668,793** | **$ 224,714,537** | **$ 1,702.32** | **$ 55.73** |

Using these cost values as well as the revenue from the sale of the mixed rare earth concentrate, a pro forma cash flow was developed and several economic indicators were derived. Table 3.2.25 provides the final economic summary for this scenario.

Table 3.2.25. Final Economic Summary for WK4

| Parameter | Unit | Value |
|---|---|---|
| ***Technical Results*** | | |
| Plant Feed Rate | t/hr | 453.6 |
| Plant Feed Grade | ppm | 329 |
| Overall REE Production | kg/hr | 16.4 |
| ***Economic Results*** | | |
| Overnight Capital Cost | $ | 264,059,303 |
| Constant Dollar OPEX | $ / yr | 277,175,172 |
| (unit conversion) | $/ st feed | 68.74 |
| (unit conversion) | $ /kg REE | 2,099.74 |
| Constant Dollar Revenue | $ / yr | 77,520,156 |
| Net Present Value (@10% discount) | $ | (1,598,005,780) |
| Internal Rate of Return | % | N/A |
| Payback Period | Operating Years | N/A |

While this scenario was not economically favorable on its own, it did provide a platform to conduct detailed sensitivity analyses to identify cost drivers as well as a potential pathway to economic viability. The first sensitivity analysis was evaluated to determine the influence of several technical parameters on total operating costs. Figure 3.2.23 shows this analysis, presented here as a tornado graph with the extent of each bar representing a +/- 30% change on the baseline input value. This data shows that CM recovery, leaching acid consumption, and leaching percent solids are the most sensitive parameters in the model, showing a range significantly greater than the remaining parameters. Power, labor, and other cost categories were not considered in this analysis due to their relatively low influence on cost when compared to the impact of chemical costs.

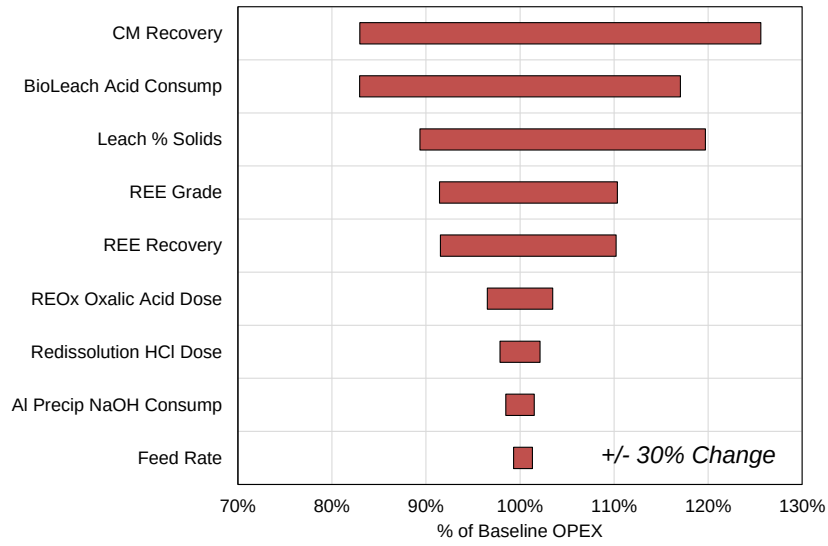



Figure 3.2.23. Sensitivity analysis showing the effect of key technical parameters on total operating cost for WK4. Limits of tornado graph indicate +/- 30% change in the input value.

Figure 3.2.24 shows the sensitivity of the overall plant capital cost as a function of plant feed. This curve follows a clear power law trend with an overall exponent of 0.57. The power law trend is anticipated given the capital cost estimation procedures; however, the exponent is lower than typical values for chemical processes that range from 0.6 to 0.7. This result is likely due to the high prevalence of unit operation with lower exponent values, including leach tanks, pumps, mixers, and filters (see Table 3.2.2).

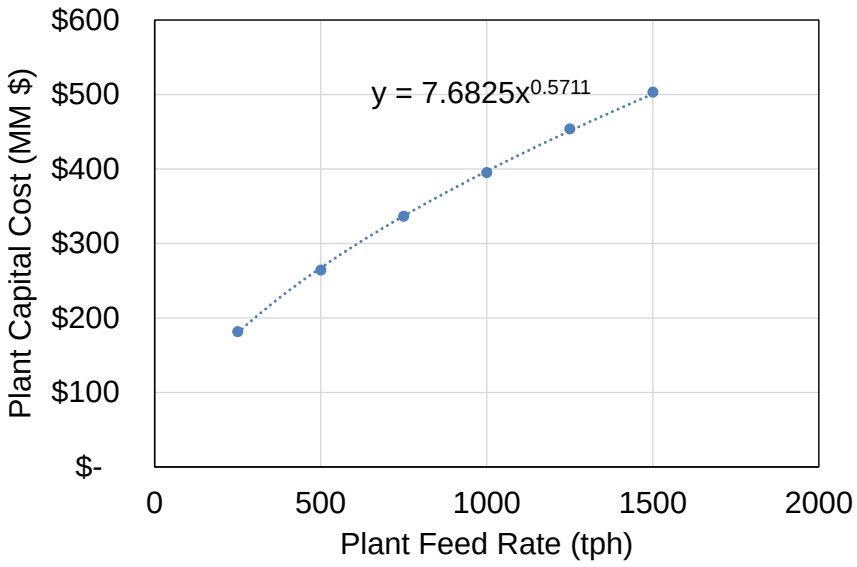



Figure 3.2.24. Sensitivity analysis showing the effect of plant feed rate on total plant capital costs for WK4.

Figure 3.2.25 shows a similar analysis depicting the plant operating costs ($/kg total product produced) as a function of plant feed. This graph illustrates the economy of scale associated with higher feed rates. Since a large portion of the operating costs are chemical costs, which scale linearly with feed rate, the cost savings due to economy of scale are very limited for this plant.

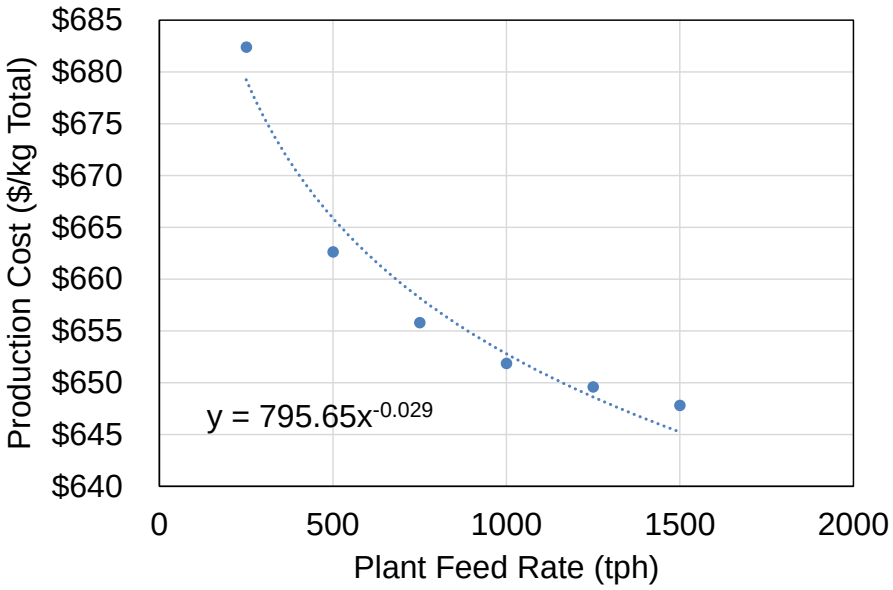



Figure 3.2.25. Sensitivity analysis showing the effect of plant feed rate on total plant operating costs for WK4 Scenario.

Since the one-variable-at-a-time tornado analysis showed that feed grade and acid cost were important variables with respect to processing cost, a two-factor analysis was conducted to evaluate the simultaneous influence of these two parameters. Figure 3.2.26 shows the results of this analysis, with feed grade varying between 200 and 1500 ppm, while the acid cost varies from a 50% reduction to a 50% increase relative to baseline values. This result shows that operating costs follow an exponential trend with respect to feed grade, with costs below $400/kg only occurring in cases where feed grade is greater than 1000 ppm. At low feed grades, the impact of acid reduction is more pronounced; however, as the feed grade increases, the effect that acid reduction has on total cost is more subtle.

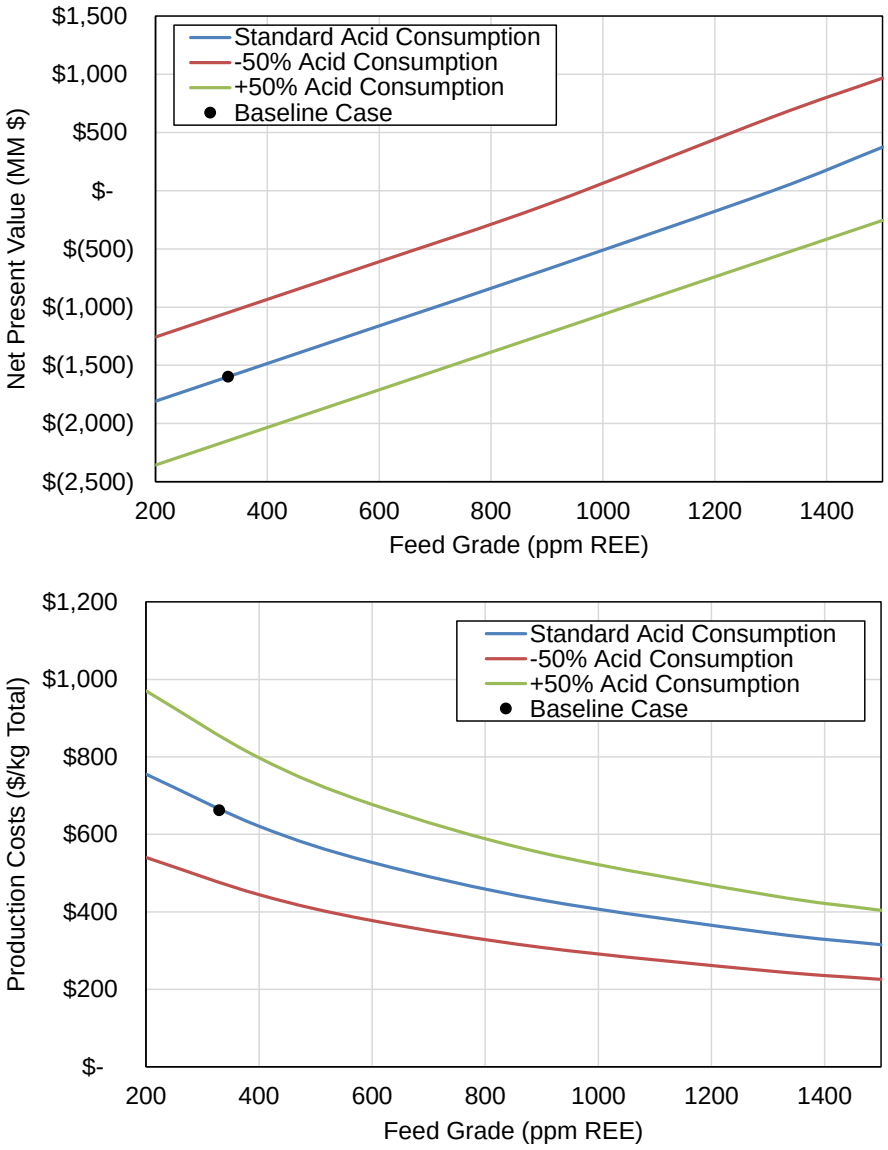



Figure 3.2.26. Sensitivity analysis showing the combined effect of plant feed grade and acid consumption/cost on net present value (top panel) and production cost (bottom panel), WK4 Scenario.

Lastly, Figure 3.2.27 shows the combined effect of scandium price and REE recovery on net present value. As shown, the high scandium price of $4,200/kg used in this analysis yields a breakeven REE recovery of approximately 45%. At lower scandium prices, the breakeven REE recovery increases, and at the lowest scandium price utilized in this analysis ($2,200/kg), the project fails to breakeven even at high REE recoveries. This plot clearly shows the high sensitivity of project economics to scandium price, which has been noted throughout the discussion.

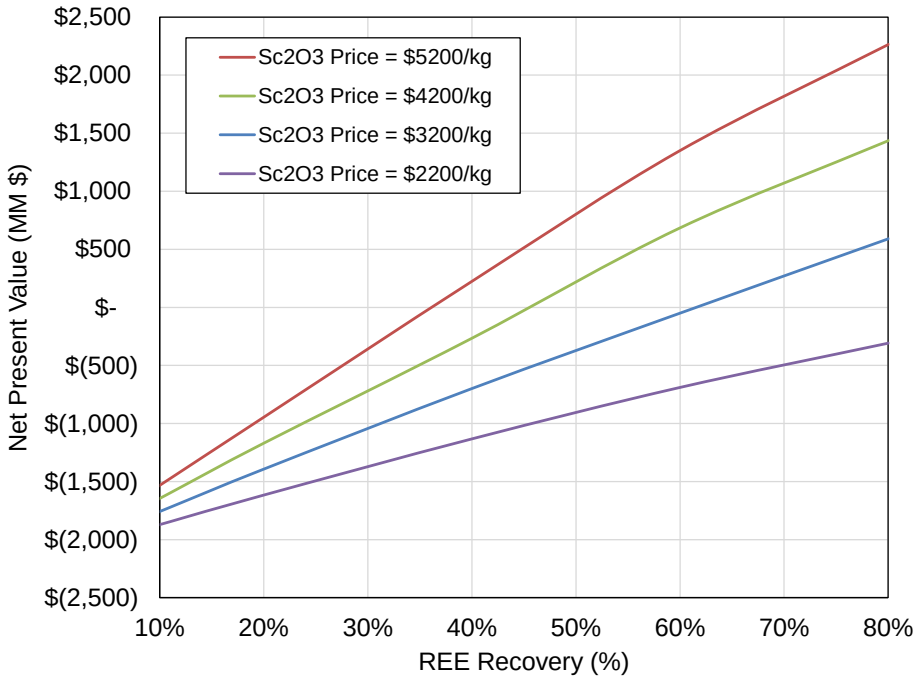



Figure 3.2.27. Sensitivity analysis showing the combined effect of REE recovery and Sc price on net present value, WK4 Scenario.

### *3.2.9 Detailed Analysis of SHXP-Alt-C4*

A second detailed analysis was conducted using the data from scenario SHXP-Alt-C4. The discussion in this section follows a similar pattern as that of the prior. Unlike WK4, SXHXP-Alt-C4 did produce an economically favorable outcome, and the results herein explain the causal factors prompting this response.

For this scenario, the nominal feed rate was fixed at 500 st/hr, with the REE grade and distribution shown in Table 3.2.26. Given these feed rate and grade values, the total mass flow of REEs into the process from the feed solids is calculated to be 567 kg/hr, while the total rate of non-REE CM was determined to be 781 kg/hr.

Table 3.2.26. Feedstock REE/CM Concentration used in SHXP-Alt-C4 Analysis

| Element | Feed Conc. (mg/kg) | Distribution (%) |
|---|---|---|
| Scandium | 34.80 | 2.78 |
| Yttrium | 236.02 | 18.88 |
| Lanthanum | 195.74 | 15.66 |
| Cerium | 377.56 | 30.20 |
| Praseodymium | 40.52 | 3.24 |
| Neodymium | 176.16 | 14.09 |
| Samarium | 37.15 | 2.97 |
| Europium | 8.51 | 0.68 |
| Gadolinium | 45.12 | 3.61 |
| Terbium | 6.15 | 0.49 |
| Dysprosium | 37.90 | 3.03 |
| Holmium | 7.37 | 0.59 |
| Erbium | 19.28 | 1.54 |
| Thulium | 4.23 | 0.34 |
| Ytterbium | 19.96 | 1.60 |
| Lutetium | 3.54 | 0.28 |
| Manganese | 505.39 | -- |
| Cobalt | 248.28 | -- |
| Nickel | 759.69 | -- |
| Zinc | 207.62 | -- |
| **Total REE** | **1,250.00** | **100.00** |
| **Total Non-REE CM** | **1,720.98** | **--** |

To determine the overall recovery of REE/CMs to the product stream, element-by-element recovery values from individual unit operations were combined in a simple multiplicative manner. Table 3.2.27 shows these recovery values, itemized by individual REE and unit operation. For this current scenario, these recovery values were determined from continuous pilot scale testing of the integrated process.

To support the economic analysis, these recovery values and other technical factors and reagent consumption rates were used to generate simplified material balances for each individual process module. These material balances not only show major solid mass and liquid flow, but they also show the balance of REEs throughout the circuit as well as the dosing rates for major consumables. Figure 3.2.28 through Figure 3.2.35 show these material balances for process modules PM1 through PM11, respectively (PM2, PM 9, and PM 10 are not utilized in this circuit configuration).

Table 3.2.27. Element-by-element recovery values by unit operation used in techno-economic analysis

| REE | Recovery in Various Unit Operations | | | | | | | Overall Recovery | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Leaching | Al/Fe Precipitation | REE Precipitation | RE Re-Dissolution | REOx Precipitation | Co Precipitation | Mn Precipitation | MREO Product | Co-Ni Product | Mn Product |
| Scandium | 87.5% | 51.7% | 56.8% | 78.4% | 3.1% | 13.8% | 51.1% | 0.0% | 0.0% | 0.1% |
| Yttrium | 98.0% | 92.5% | 74.6% | 83.5% | 89.6% | 80.5% | 95.6% | 50.6% | 4.7% | 1.1% |
| Lanthanum | 93.3% | 94.6% | 71.2% | 83.6% | 95.5% | 67.7% | 29.2% | 50.2% | 1.6% | 0.2% |
| Cerium | 95.8% | 89.9% | 94.8% | 83.1% | 98.9% | 72.3% | 100.0% | 67.0% | 0.6% | 0.2% |
| Praseodymium | 95.2% | 91.3% | 31.5% | 74.1% | 96.6% | 81.9% | 100.0% | 19.6% | 0.6% | 0.1% |
| Neodymium | 94.0% | 89.9% | 72.4% | 82.6% | 98.6% | 95.0% | 52.9% | 49.8% | 0.7% | 0.0% |
| Samarium | 96.1% | 87.6% | 79.3% | 83.2% | 98.3% | 80.0% | 100.0% | 54.6% | 0.8% | 0.2% |
| Europium | 96.2% | 81.7% | 73.1% | 83.4% | 99.3% | 65.9% | 100.0% | 47.6% | 0.2% | 0.1% |
| Gadolinium | 97.1% | 91.3% | 61.4% | 82.2% | 97.9% | 77.5% | 93.9% | 43.7% | 0.7% | 0.2% |
| Terbium | 91.2% | 79.5% | 54.5% | 91.3% | 99.4% | 52.1% | 98.3% | 35.8% | 0.1% | 0.1% |
| Dysprosium | 99.1% | 85.4% | 84.7% | 77.6% | 95.2% | 82.2% | 100.0% | 53.0% | 2.2% | 0.5% |
| Holmium | 98.4% | 92.4% | 64.6% | 82.2% | 91.0% | 49.5% | 21.4% | 44.0% | 2.2% | 0.5% |
| Erbium | 95.9% | 89.3% | 71.7% | 82.8% | 89.3% | 74.4% | 53.1% | 45.4% | 4.0% | 0.7% |
| Thulium | 96.8% | 87.7% | 62.9% | 87.2% | 56.6% | 18.0% | 29.0% | 26.4% | 3.6% | 4.8% |
| Ytterbium | 96.7% | 82.6% | 72.4% | 82.2% | 80.3% | 76.1% | 97.1% | 38.1% | 7.1% | 2.2% |
| Lutetium | 95.4% | 82.8% | 68.0% | 82.4% | 79.1% | 65.9% | 45.1% | 35.0% | 6.1% | 1.4% |
| Manganese | 98.5% | 96.6% | 88.9% | 77.1% | 0.6% | 79.3% | 96.3% | 0.4% | 51.3% | 12.9% |
| Cobalt | 92.3% | 88.7% | 99.5% | 79.5% | 1.8% | 98.4% | 96.2% | 1.1% | 62.7% | 1.0% |
| Nickel | 95.3% | 72.0% | 99.3% | 78.0% | 1.0% | 92.3% | 96.3% | 0.5% | 48.6% | 3.9% |
| Zinc | 93.3% | 88.7% | 99.7% | 79.9% | 2.5% | 99.6% | 100.0% | 1.6% | 64.1% | 0.2% |

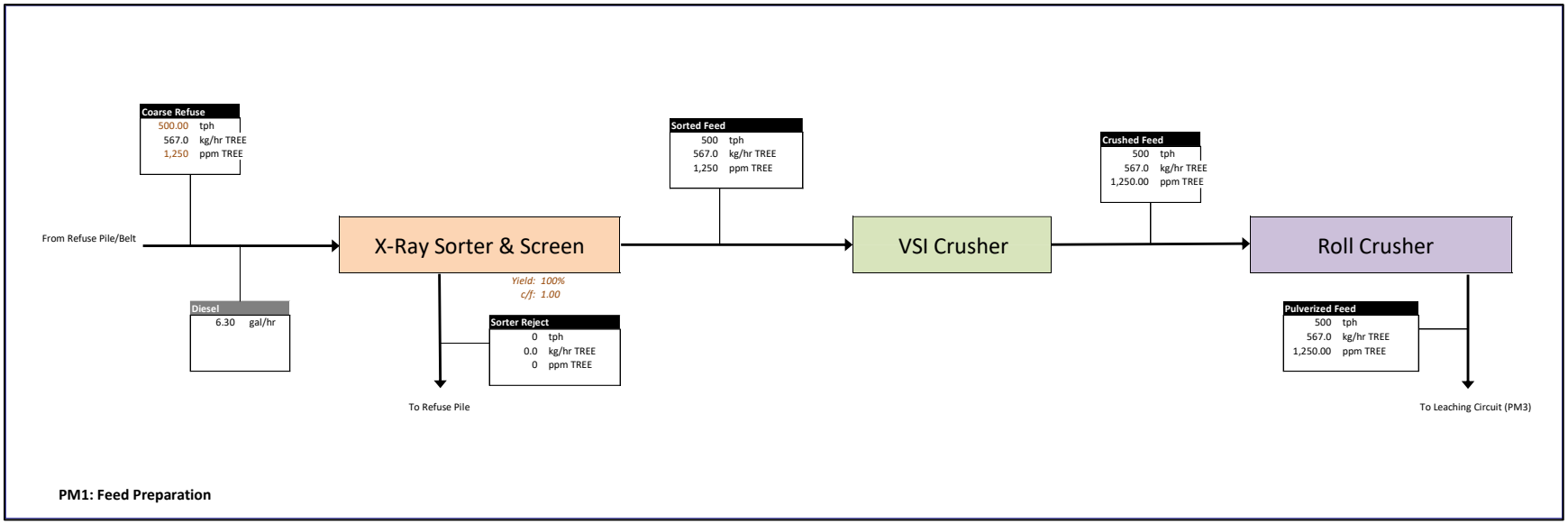



Figure 3.2.28. Block Flow Diagram for PM1: Feed Preparation (SHXP-Alt-C1).

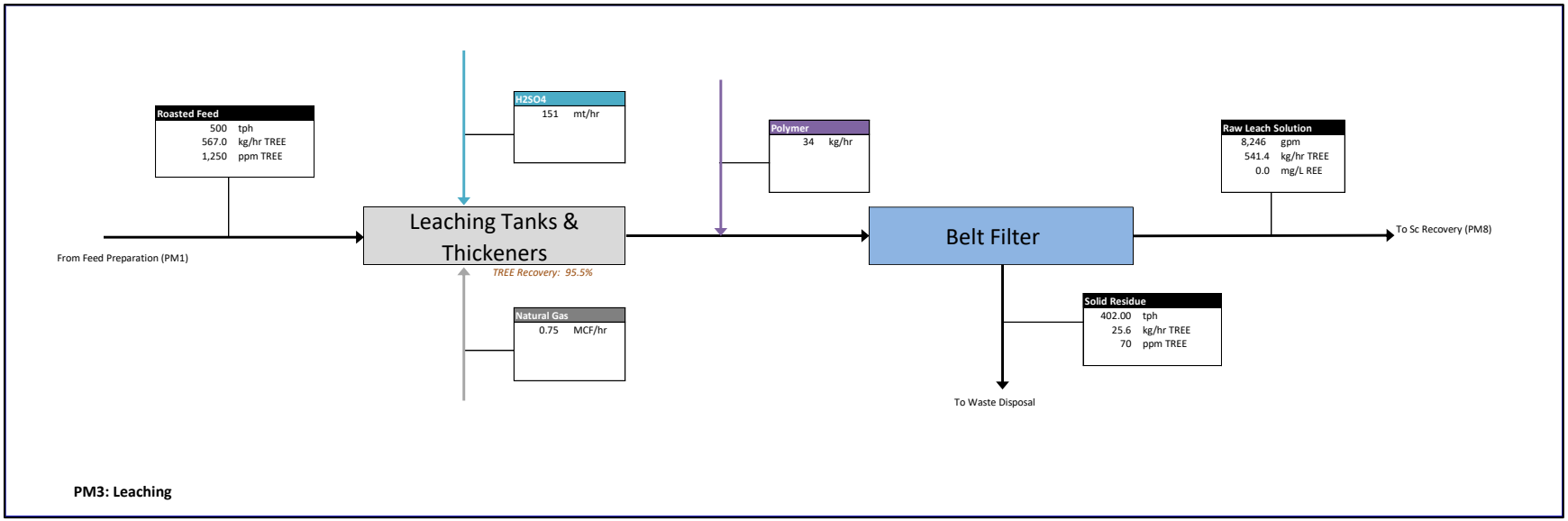



Figure 3.2.29. Block Flow Diagram for PM3: Leaching (SHXP-Alt-C1).

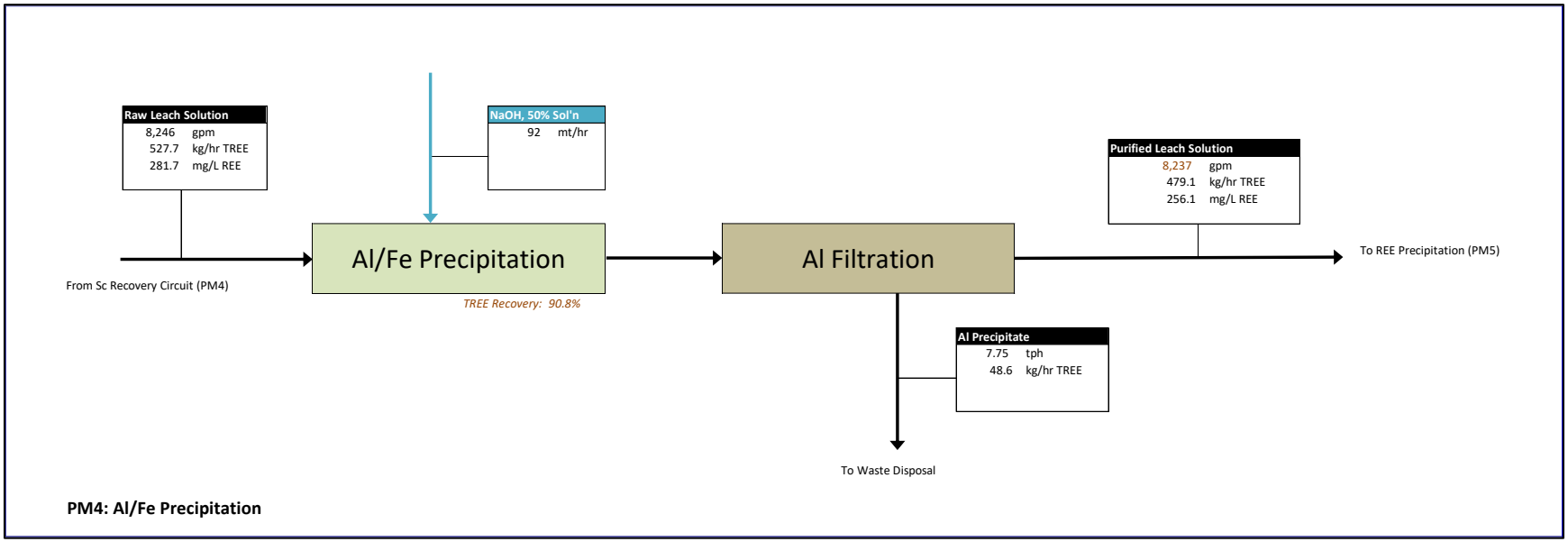



Figure 3.2.30. Block Flow Diagram for PM4 Al/Fe Precipitation (SHXP-Alt-C1).

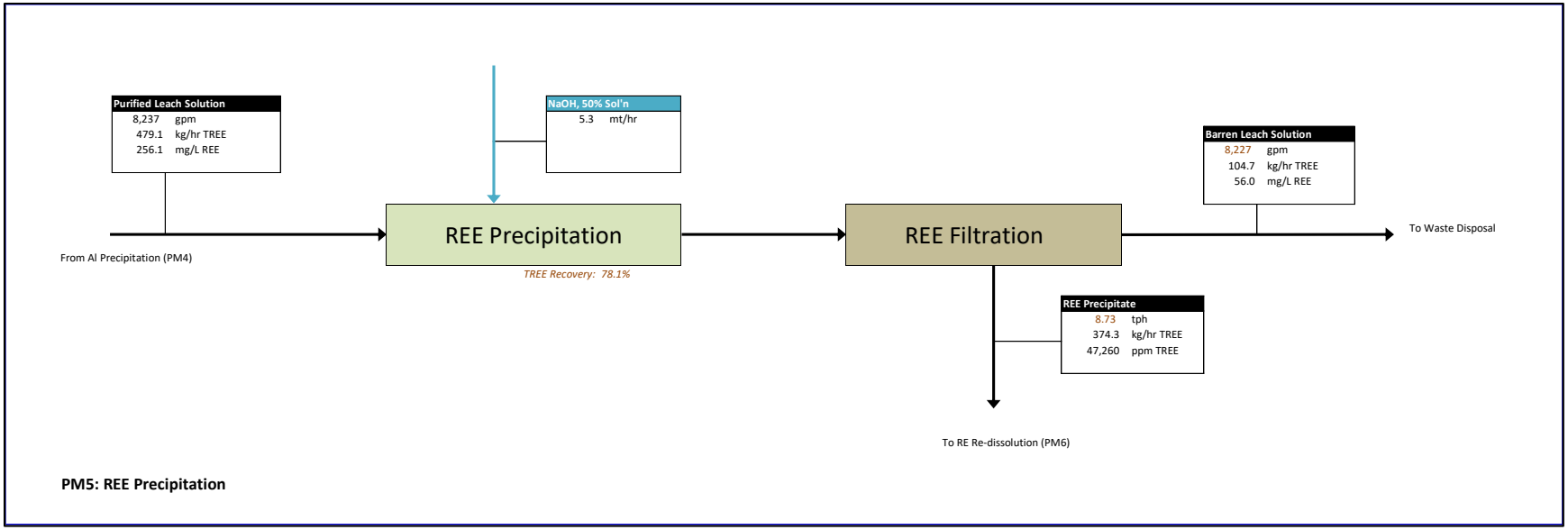



Figure 3.2.31. Block Flow Diagram for PM5: REE Precipitation (SHXP-Alt-C1).

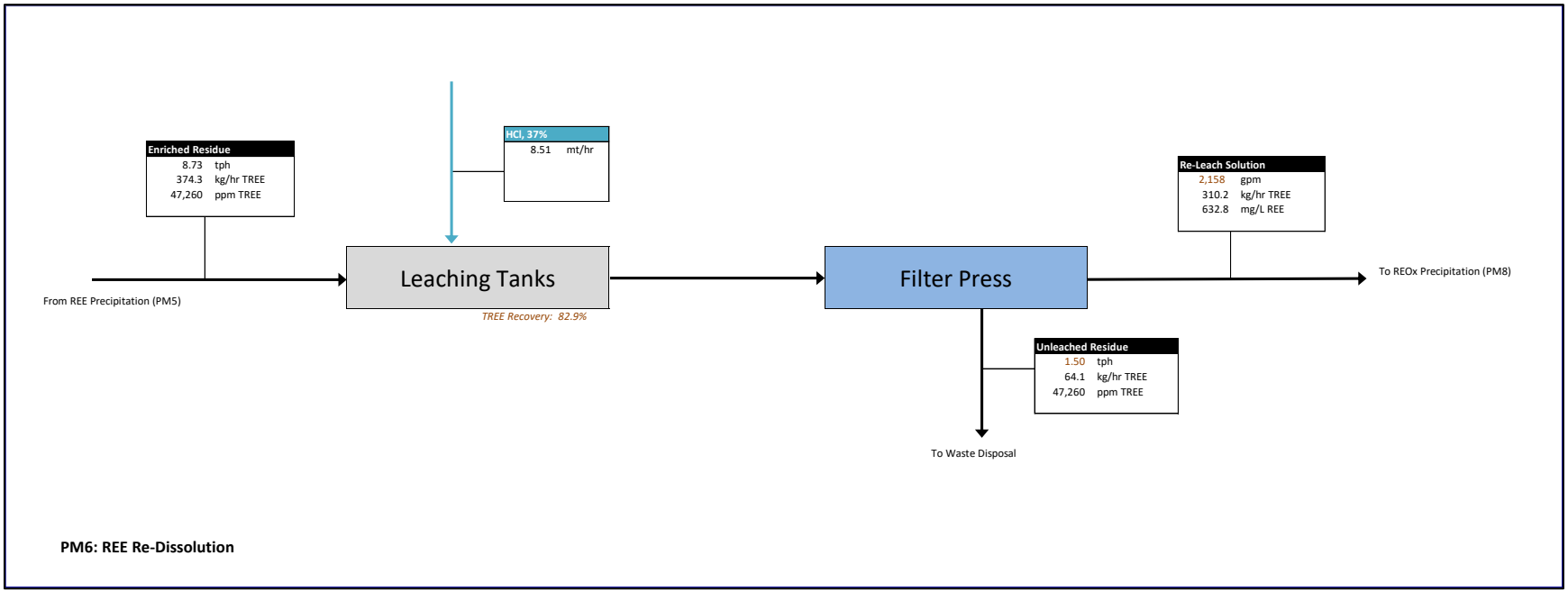



Figure 3.2.32. Block Flow Diagram for PM6: RE Re-dissolution (SHXP-Alt-C1).

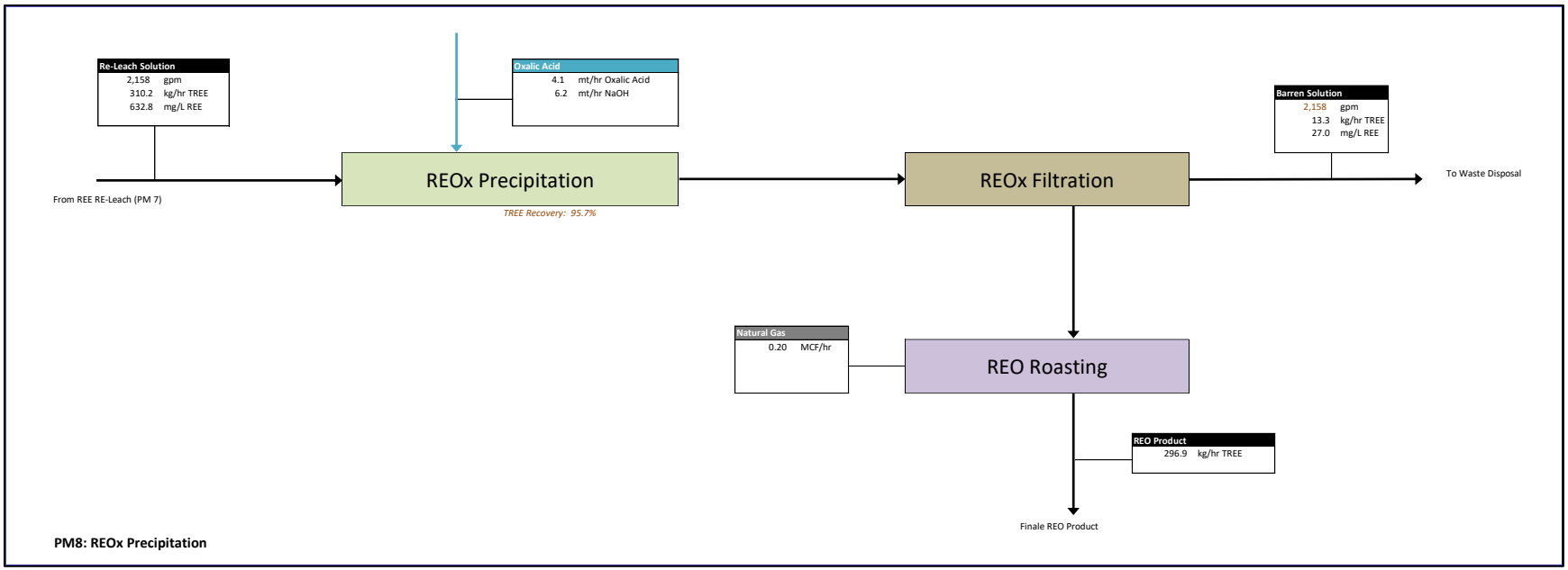



Figure 3.2.33. Block Flow Diagram for PM7: REOx Precipitation (SHXP-Alt-C1).

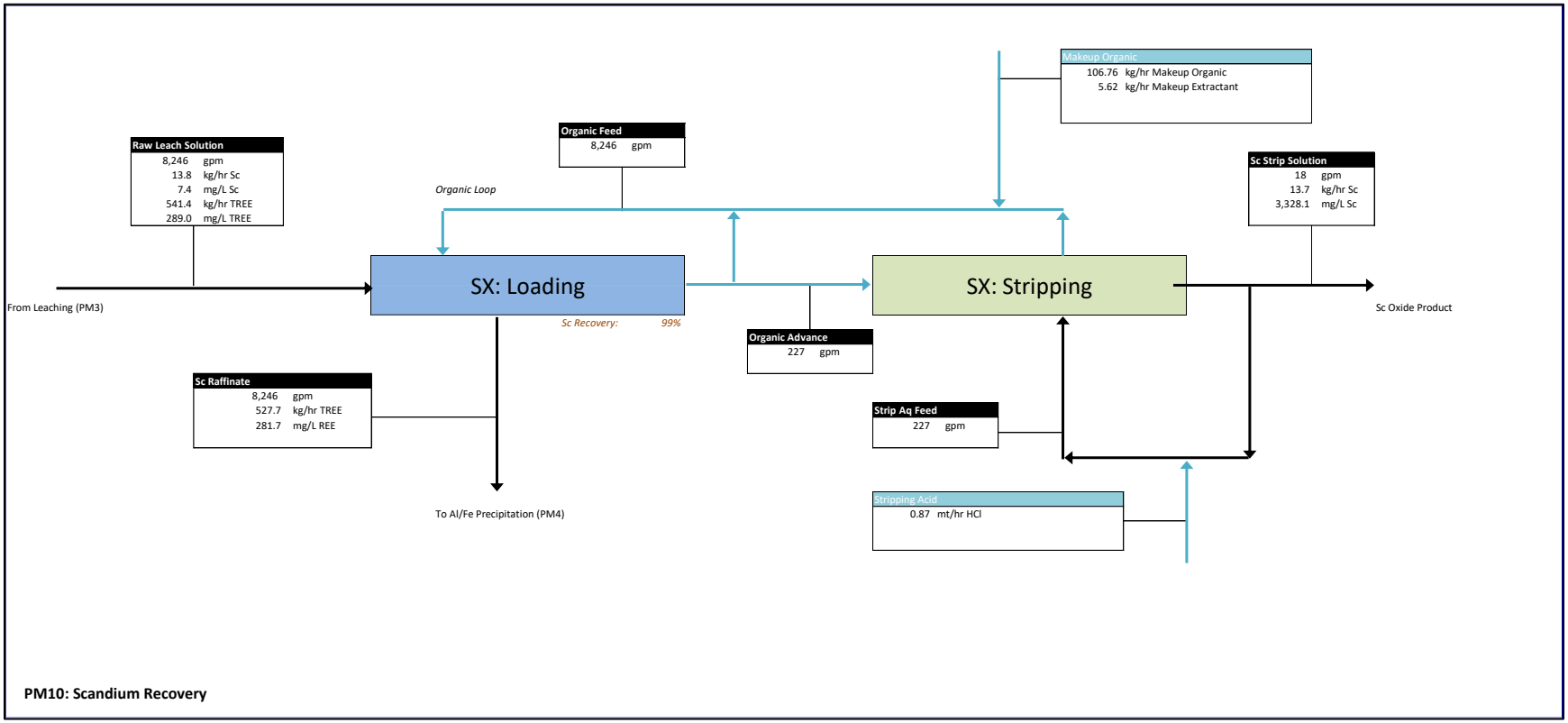



Figure 3.2.34. Block Flow Diagram for PM8: Sc Recovery (SHXP-Alt-C1).

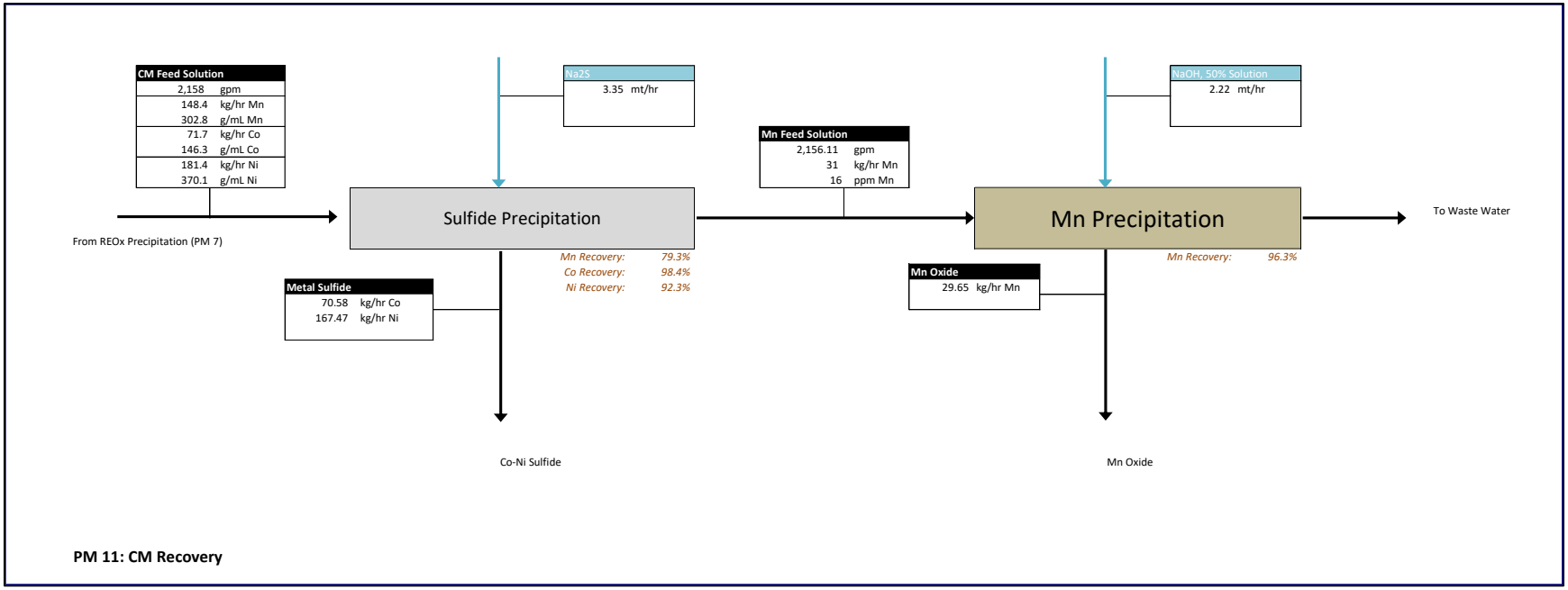



Figure 3.2.35. Block Flow Diagram for PM11 CM Recovery (SHXP-Alt-C1).

Using these material balances as well as the cost models descripted above, the total capital cost for the 500 st/hr plant was determined to be approximately $96 million. Table 3.2.28 shows this cost itemized by process module and the capital cost category, while Figure 3.2.36 shows a graphical breakdown of this same data. This analysis shows that the leaching circuit and REE precipitation circuit are the most significant capital costs, each contributing approximately 25% of the total.

Table 3.2.28. Module-level capital cost for commercial scale plant, SHXP-Alt-C4 (500 st/hr).

| Module ID | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 |
| PM3 | Leaching | $ 7,234,587 | $ - | $ 1,808,647 | $ 7,958,046 | $ 9,907,767 | $ 26,909,047 |
| PM4 | Al Precipitation | $ 3,448,822 | $ - | $ 862,206 | $ 3,793,705 | $ 4,723,162 | $ 12,827,895 |
| PM5 | REE Precipitation | $ 5,795,457 | $ - | $ 1,448,864 | $ 6,375,003 | $ 7,936,879 | $ 21,556,204 |
| PM6 | RE-Redissolution | $ 1,274,195 | $ - | $ 318,549 | $ 1,401,615 | $ 1,745,010 | $ 4,739,369 |
| PM7 | REE Oxalate Precipitation | $ 1,019,128 | $ - | $ 254,782 | $ 1,121,040 | $ 1,395,695 | $ 3,790,645 |
| PM8 | Sc SX Separation | $ 2,162,479 | $ 4,803,650 | $ 540,620 | $ 2,378,727 | $ 2,961,514 | $ 12,846,989 |
| PM11 | CM Recovery | $ 1,035,762 | $ - | $ 258,940 | $ 1,139,338 | $ 1,418,476 | $ 3,852,515 |
| | **TOTAL:** | **$ 24,434,089** | **$ 4,803,650** | **$ 6,108,522** | **$ 26,877,498** | **$ 33,462,486** | **$ 95,686,245** |

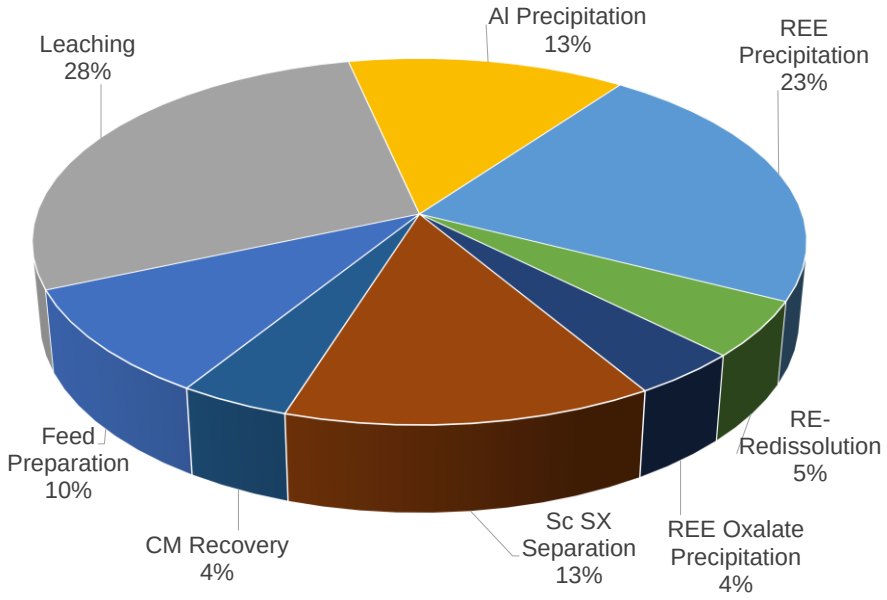



Figure 3.2.36. Capital cost breakdown for SHXP-Alt-C4 scenario.

Final summaries of total operating cost by cost type and cost center are given in Table 3.2.29, while a graphical breakdown is shown in Figure 3.2.37. The total operating cost for this process was determined to be $194/kg REE produced ($81.3/kg total product). An assessment of the cost breakdown shows that consumables are the single largest cost item, representing nearly 84% of the total costs. General overhead costs constituted the next largest item; however, this category was a much smaller fraction, only 13% of the total.

With respect to process module, the leaching operation was the costliest, representing over 50% of the total. Given the high relative significance of power and chemical costs, those expenses have been itemized separately, by process module, Table 3.2.30 and Table 3.2.31, respectively. This data shows that the total chemical cost is $163/kg REE, with the primary contributors being

leaching and to a lesser extent leachate precipitation. Likewise, power costs were much smaller in magnitude, only requiring $1.01/kg REE. Altogether, these results indicate that leaching acid is the single highest cost item for process and likely, the item with the most potential for improvement.

Additional itemized line-by-line costing details for all process modules and all cost components can be found in the *Appendix* to this report.

Table 3.2.29. Total operating cost breakdown for commercial-scale plant, SHXP-Alt-C4 (500 st/hr.).

**Summary by Cost Type**

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 10,759.75 | $ 32,279.26 | $ 10,845,830 | $ 4 | $ 2.69 | |
| Power | $ 2,617.50 | $ 7,852.49 | $ 2,638,437 | $ 1 | $ 0.65 | |
| Consumables | $ 422,038.20 | $ 1,266,114.61 | $ 425,414,508 | $ 163 | $ 105.51 | IIIIIIIIIIIIIIIIIIIIIIII |
| Leasing/Tolling | $ - | $ - | $ - | $ - | $ - | |
| Waste Disposal | $ 414.60 | $ 1,243.81 | $ 417,921 | $ 0 | $ 0.10 | |
| General Overhead | $ 65,374.51 | $ 196,123.52 | $ 65,897,504 | $ 25 | $ 16.34 | IIII |
| TOTAL | $ 501,205 | $ 1,503,614 | $ 505,214,199 | $ 194 | $ 125.30 | |

**Summary by Process Module**

| Process Module | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Feed Preparation | $ 611 | $ 1,834 | $ 616,370 | $ 0.24 | $ 0.15 | |
| Roasting | $ - | $ - | $ - | $ - | $ - | |
| Leaching | $ 266,475 | $ 799,426 | $ 268,607,095 | $ 103.15 | $ 66.62 | IIIIIIIIIIIIII |
| Al Precipitation | $ 92,195 | $ 276,584 | $ 92,932,080 | $ 35.69 | $ 23.05 | IIIII |
| REE Precipitation | $ 5,677 | $ 17,030 | $ 5,722,151 | $ 2.20 | $ 1.42 | |
| RE-Redissolution | $ 18,592 | $ 55,777 | $ 18,741,007 | $ 7.20 | $ 4.65 | I |
| REE Oxalate Precipitation | $ 25,941 | $ 77,822 | $ 26,148,229 | $ 10.04 | $ 6.49 | I |
| Sc SX Separation | $ 3,144 | $ 9,433 | $ 3,169,634 | $ 1.22 | $ 0.79 | |
| CM Precipitation | $ - | $ - | $ - | $ - | $ - | |
| CM Redissolution | $ - | $ - | $ - | $ - | $ - | |
| CM Recovery | $ 12,020 | $ 36,061 | $ 12,116,378 | $ 4.65 | $ 3.01 | |
| Biooxidation | $ - | $ - | $ - | $ - | $ - | |
| Reserved | $ - | $ - | $ - | $ - | $ - | |
| Reserved | $ - | $ - | $ - | $ - | $ - | |
| Reserved | $ - | $ - | $ - | $ - | $ - | |
| Unallocated | $ 76,549 | $ 229,647 | $ 77,161,255 | $ 30 | $ 19.14 | IIII |
| TOTAL | $ 501,205 | $ 1,503,614 | $ 505,214,199 | $ 194 | $ 125.30 | |

**Summary by Cost Center**

| Cost Center | Code | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| Operating Labor | OL | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 1 | $ 0.76 | |
| Technical Labor | TL | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 0 | $ 0.29 | |
| Power/Utilities | PR | $ 2,617.50 | $ 7,852.49 | $ 2,638,437 | $ 1 | $ 0.65 | |
| Lease Agreements | LS | $ - | $ - | $ - | $ - | $ - | |
| Consumables/Reagents | CR | $ 422,038.20 | $ 1,266,114.61 | $ 425,414,508 | $ 163 | $ 105.51 | IIIIIIIIIIIIIIIIIIIIIIII |
| Waste Disposal | WD | $ 414.60 | $ 1,243.81 | $ 417,921 | $ 0 | $ 0.10 | |
| Plant Overhead | OH | $ 66,628.64 | $ 199,885.91 | $ 67,161,666 | $ 26 | $ 16.66 | IIII |
| General Sales, and Administration | GSA | $ 5,286.32 | $ 15,858.97 | $ 5,328,614 | $ 2 | $ 1.32 | |
| Reserved | | | | | | | |
| TOTAL | | $ 501,205 | $ 1,503,613.69 | $ 505,214,199 | $ 194 | $ 125.30 | |

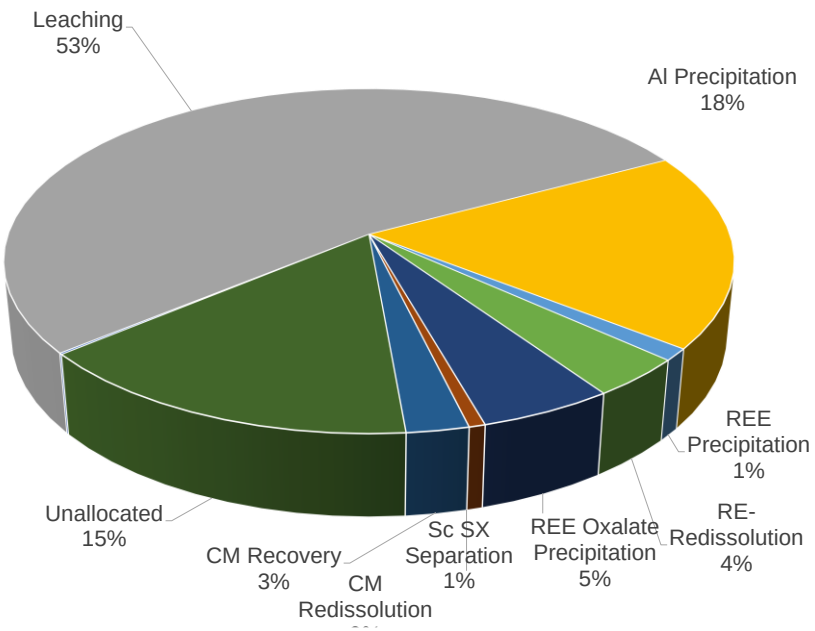



Figure 3.2.37. Operating cost breakdown for SHXP-Alt-C4 Scenario.

Table 3.2.30. Power cost summary by process module for commercial-scale plant, SHXP-Alt-C4

| Module ID | Process Module | Total HP | Shift Cost | Daily Cost | Annual Cost | Cost Per Total Product kg | Cost Per Plant Feed Short-Ton |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | 1,065 | $ 510.68 | $ 1,532 | $ 514,764 | $ 0.20 | $ 0.13 |
| PM3 | Leaching | 1,296 | $ 621.02 | $ 1,863 | $ 625,992 | $ 0.24 | $ 0.16 |
| PM4 | Al Precipitation | 815 | $ 390.67 | $ 1,172 | $ 393,800 | $ 0.15 | $ 0.10 |
| PM5 | REE Precipitation | 839 | $ 402.40 | $ 1,207 | $ 405,616 | $ 0.16 | $ 0.10 |
| PM6 | RE-Redissolution | 442 | $ 212.07 | $ 636 | $ 213,765 | $ 0.08 | $ 0.05 |
| PM7 | REE Oxalate Precipitation | 184 | $ 88.05 | $ 264 | $ 88,753 | $ 0.03 | $ 0.02 |
| PM8 | Sc SX Separation | 515 | $ 246.95 | $ 741 | $ 248,927 | $ 0.10 | $ 0.06 |
| PM11 | CM Recovery | 304 | $ 145.65 | $ 437 | $ 146,820 | $ 0.06 | $ 0.04 |
| | **Total:** | **5,460** | **$ 2,617.50** | **$ 7,852** | **$ 2,638,437** | **$ 1.01** | **$ 0.65** |

Table 3.2.31. Chemical cost summary by process module for commercial-scale plant, SHXP-Alt-C4

| Module ID | Process Module | Shift Cost | Daily Cost | Annual Cost | Cost Per Total Product kg | Cost Per Plant Feed Short-Ton |
|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 100.80 | $ 302 | $ 101,606 | $ 0.04 | $ 0.03 |
| PM3 | Leaching | $ 265,854.27 | $ 797,563 | $ 267,981,103 | $ 102.91 | $ 66.46 |
| PM4 | Al Precipitation | $ 91,803.85 | $ 275,412 | $ 92,538,281 | $ 35.54 | $ 22.95 |
| PM5 | REE Precipitation | $ 5,274.34 | $ 15,823 | $ 5,316,534 | $ 2.04 | $ 1.32 |
| PM6 | RE-Redissolution | $ 18,380.20 | $ 55,141 | $ 18,527,242 | $ 7.11 | $ 4.60 |
| PM7 | REE Oxalate Precipitation | $ 25,852.65 | $ 77,558 | $ 26,059,476 | $ 10.01 | $ 6.46 |
| PM8 | Sc SX Separation | $ 2,897.53 | $ 8,693 | $ 2,920,707 | $ 1.12 | $ 0.72 |
| PM11 | CM Recovery | $ 11,874.56 | $ 35,624 | $ 11,969,557 | $ 4.60 | $ 2.97 |
| | **Total:** | **$ 422,038.20** | **$ 1,266,115** | **$ 425,414,508** | **$ 163.37** | **$ 105.51** |

Using these cost values as well as the revenue from the sale of the mixed rare earth concentrate, a pro forma cash flow was developed, and several economic indicators were derived. Table 3.2.32 provides the final economic summary for this scenario.

Table 3.2.32. Final Economic Summary for SHXP-Alt-C4

| Parameter | Unit | Value |
|---|---|---|
| ***Technical Results*** | | |
| Plant Feed Rate | t/hr | 453.6 |
| Plant Feed Grade | ppm | 1,250 |
| Overall REE Production | kg/hr | 322.9 |
| ***Economic Results*** | | |
| Overnight Capital Cost | $ | 95,686,245 |
| Constant Dollar OPEX | $ / yr | 505,214,199 |
| (unit conversion) | $/ st feed | 125.30 |
| (unit conversion) | $ /kg REE | 194.02 |
| Constant Dollar Revenue | $ / yr | 782,160,698 |
| Net Present Value (@10% discount) | $ | 2,129,048,297 |
| Internal Rate of Return | % | 180.2% |
| Payback Period | Operating Years | 1 |

Additional sensitivity analyses were conducted to assess the robustness of these economic outcomes. The first sensitivity analysis was evaluated to determine the influence of several technical parameters on total operating costs. Figure 3.2.38 shows this analysis, presented here as a tornado graph with the extent of each bar representing a +/- 30% change on the baseline input value. This data shows that leaching acid consumption, leaching percent solids, and REE grade are the most sensitive input parameters in the model, showing a range significantly greater than the remaining parameters. Power, labor, and other cost categories were not considered in this analysis due to their relatively low influence on cost when compared to the impact of chemical costs.

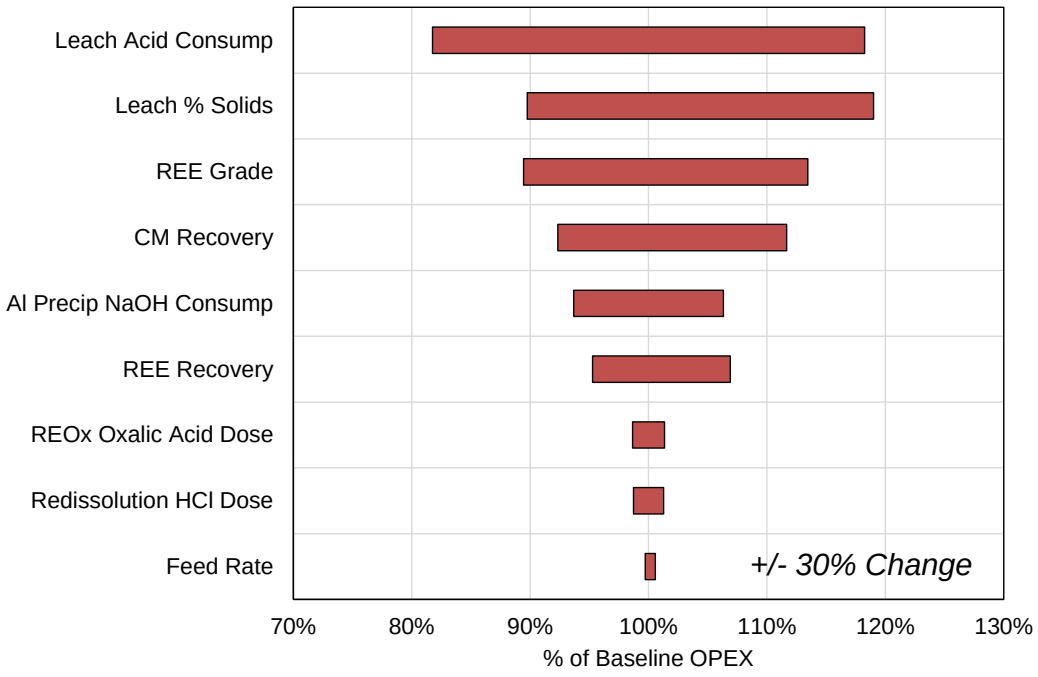



Figure 3.2.38. Sensitivity analysis showing the effect of key technical parameters on total operating cost for SHXP-Alt-C4. Limits of tornado graph indicate +/- 30% change in the input value.

Figure 3.2.39 shows the sensitivity of the overall plant capital cost as a function of plant feed. This curve follows a clear power law trend with an overall exponent of 0.59. The power law trend is anticipated given the capital cost estimation procedures; however, the exponent is commensurate with typical values for chemical processes that range from 0.6 to 0.7.

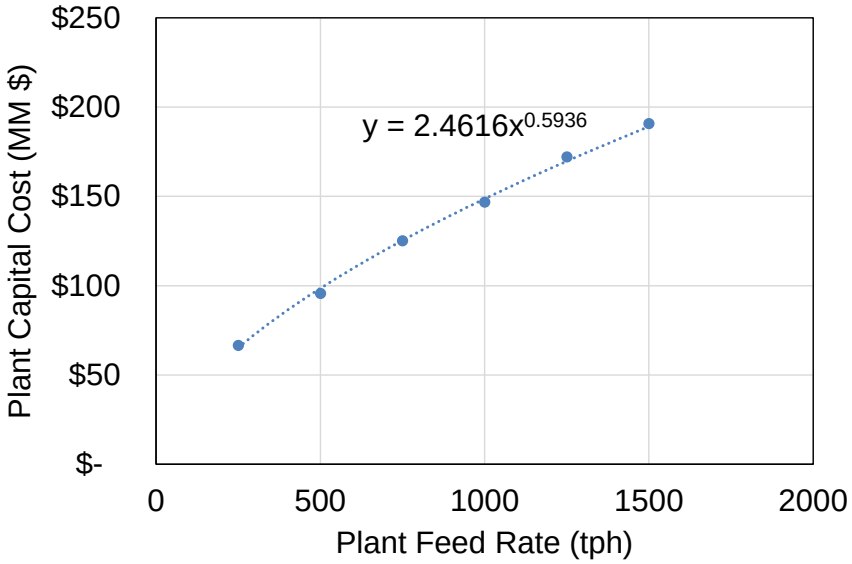



Figure 3.2.39. Sensitivity analysis showing the effect of plant feed rate on total plant capital costs for SHXP-Alt-C4.

Figure 3.2.40 shows a similar analysis, depicting the plant operating costs ($/kg total product produced) as a function of plant feed. This graph illustrates the economy of scale associated with higher feed rates. Since a large portion of the operating costs are chemical costs, which scale linearly with feed rate, the cost savings due to economy of scale are very limited for this plant.

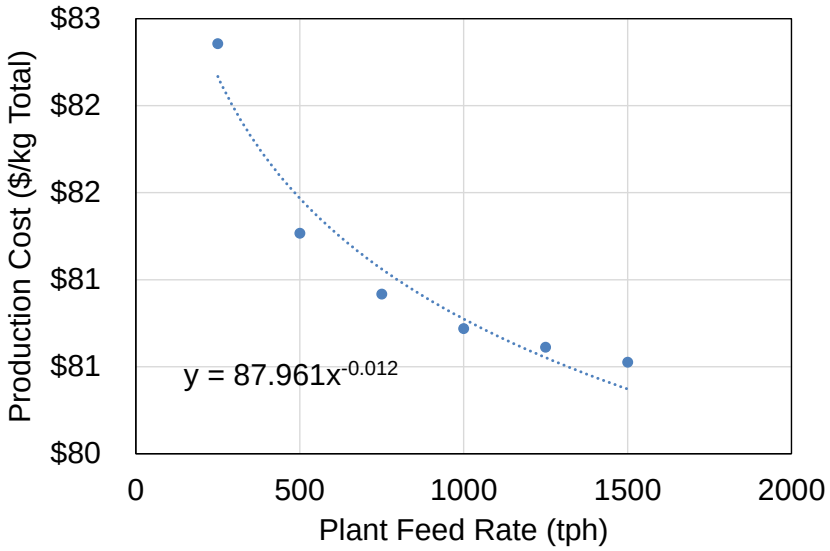



Figure 3.2.40. Sensitivity analysis showing the effect of plant feed rate on total plant operating costs for SHXP-Alt-C4.

Since the one-variable-at-a-time tornado analysis showed that feed grade and acid cost were important variables with respect to processing cost, a two-factor analysis was conducted to evaluate the simultaneous influence of these two parameters. Figure 3.2.41 shows the results of this analysis, with feed grade varying between 800 and 2000 ppm, while the acid cost varies from a 50% reduction to a 50% increase relative to baseline values. This result shows that operating costs follow a slightly exponential trend with respect to feed grade, with costs below $100/kg achievable at increased acid concentrations and grades greater than 1400 ppm. At low feed grades, the impact of acid reduction is more pronounced; however, as the feed grade increases, the effect that acid reduction has on total cost is more subtle.

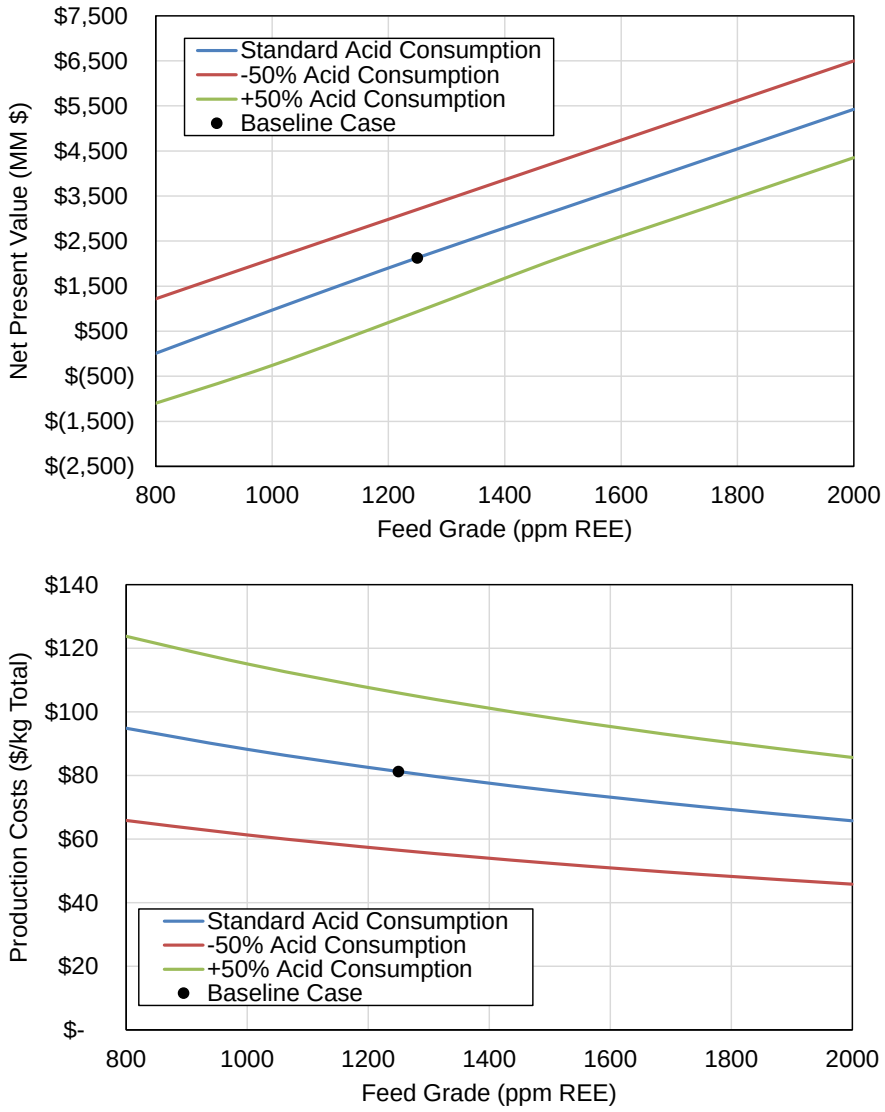



Figure 3.2.41. Sensitivity analysis showing the combined effect of plant feed grade and acid consumption/cost on net present value (top panel) and production cost (bottom panel), SHXP-Alt-C4 Scenario.

Lastly, Figure 3.2.42 shows the combined effect of scandium price and REE recovery on net present value. As shown, the high scandium price of $4,200/kg used in this analysis yields a breakeven REE recovery of approximately 30%. At lower scandium prices, the breakeven REE recovery increases, and at the lowest scandium price utilized in this analysis ($2,200/kg), the project fails to breakeven except at greater than 80% REE recoveries.

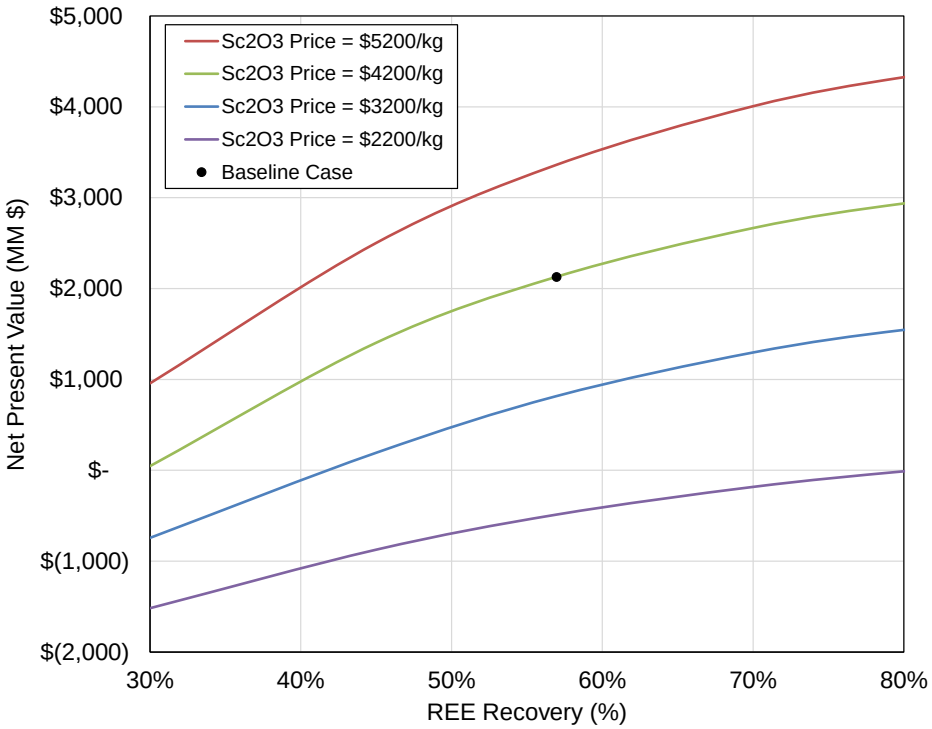



Figure 3.2.42. Sensitivity analysis showing the combined effect of REE recovery and Sc price on net present value, SHXP-Alt-C4 Scenario.

### 3.3 PROVIDE SPLIT SAMPLES (TASK 4)

Various circuit arrangements with different process set points using multiple feedstocks were tested in the rare earth pilot facility as part of the continuous improvement. Since the operating conditions changed between some tests, it also impacted the grade of rare earth and critical material oxide products, which was an anticipated outcome. As such, the samples selected for shipment to the funding agency were collected from the tests resulting in the highest product purity. A complete list of the sample codes shipped to NETL along with the description is shown in Table 3.3.1.

Table 3.3.1 Product sample list with the description of each product [All samples roasted at 750□C for 2hrs]

| Test Code- Sample Type-Container | Description |
|---|---|
| PLS3-REO-B | Rare earth oxide product produced from the treatment of a pregnant leach solution (PLS) of a coal coarse refuse heap. [Test Code Reading Instruction: (3: 3rd test, REO: Rare Earth Oxides, B: Container B)] |
| WK1-REO-A | Rare earth oxide product generated through the treatment of pregnant leachate solution (PLS) of a 600°C roasted coarse refuse coal sample. [Test Code Reading Instruction: (1: 1st test, REO: Rare Earth Oxides, A: Container A)] |
| Scandium Product | Enriched scandium oxide product produced from the treatment of a loaded organic. |
| WK1-CMO | Cobalt, Nickel, and Zinc product generated through the treatment of pregnant leachate solution (PLS) of a 600°C roasted coarse refuse coal sample. [Test Code Reading Instruction: (1: 1st test, REO: Rare Earth Oxides, A: Container A)] |
| PLS3-CMO | Cobalt, Nickel, and Zinc product generated through the treatment of pregnant leachate solution (PLS) of a coal coarse refuse heap. [Test Code Reading Instruction: (3: 3rd test, REO: Rare Earth Oxides, B: Container B)] |
| CP Roast | Lab generated cobalt product |
| NP Roast | Lab generated nickel product |

## 3.4 OPTIMIZATION OF PHYSICAL SEPARATIONS (TASK 6)

Innovative physical processing processes and circuits were evaluated and optimized to prepare material prior to the REE extraction and recovery operations. Specific objectives of this pre-processing stage were to: (1) increase the concentration of REEs in the feed to downstream unit operations; (2) remove and isolate acid consuming minerals such as calcite, dolomite, and other carbonates; and (3) remove and isolate acid producing minerals such as pyrite. For the first sub-objective, the Recipient evaluated the feasibility and suitability of novel sorting technologies to achieve dry-based separations as a means of producing a higher grade feedstock for the leaching process. For the second and third sub-objectives, density set points were established that selectively separated calcite and pyrite from the REE-bearing alumino-silicates present in coal refuse. Feed material for testing was collected for the pilot-plant feed material collected from the industrial host site. All tests were conducted using laboratory-scale equipment in this task. Parametric studies were performed to identify the test conditions that optimally achieved the aforementioned process objectives.

### *3.4.1 X-ray Sorting System*

**X-ray Sorting System Upgradation**: The DriJet uses a dual scan x-ray transmission system to separate particles according to specific gravity, which in turn has been shown to promote moderate enrichment of REEs in the product stream. The mobile unit was upgraded with an improved materials handling system that includes upgraded feed and takeaway conveyors, belt scales on all feed and product streams, an improved feeding system, and upgrades to the electrical connections. A mobile screening unit having the capability of processing 20 tph of material was installed at the x-ray sorter location to screen out the plus 2-inch and minus ½-inch material as shown in Figure 3.4.1. The unit was modified to include support structure for conveyor belts to transport material to an adjacent storage pile. In addition, conveyors with weigh belt scales were installed on the feed, ejected and pass through process streams of the dual x-ray transmission sorter. As a result, continuous monitoring of the feed rate and mass flows to each product stream was achievable. The new setup is shown in Figure 3.4.2. Altogether, these improvements promoted higher throughput with fewer labor requirements.



Figure 3.4.1. Mobile screen unit and associated conveyor used to remove the material coarser than 2-inch and finer than ½-inch from the x-ray sorter feed.



Figure 3.4.2. Dual x-ray sorter and conveyor system providing instantaneous feed rate and mass yield information.

**X-ray sorter calibration**: Coarse reject material collected at a preparation plant treating coal from the Fire Clay seam was density fractionated, color coded and scanned to characterize the material and provided calibration information for the continuous unit (Figure 3.4.3a and Figure 3.4.3b). The material was processed through the sorter at a rate of approximately 5 tph and samples collected from the high- and low-density product streams. The particles in Figure 3.4.3b that appear as a mix of orange and green are mixed-phased particles of coal and mineral matter, which typically contain elevated concentrations of REEs. As such, these particles were targeted for recovery in the sorter tests. The x-ray images of the sorter products in Figure 3.4.3c and Figure 3.4.3d indicate relatively effective separations with minor misplacement. Particularly, most of the mixed-phased particles were recovered into the low-density product stream as desired.

Results obtained from two tests are provided in Table 3.4.1. The objective of Test No. 1 was a separation density of around 1.8 SG, which provided a significant reduction in the ash content of the material from 85.80% to 37.35% while recovering only 5.0% of the feed weight to the low-density product stream. Although the TREE content on an ash basis in the product stream was relatively high (667 ppm), the content on a whole sample basis was reduced to a value below the feed content due to organic dilution. In the second test, separation cut point was changed to approximately 2.2 SG. The results demonstrated more favorable REE upgradation relative to the first test. The ash-based REE content in the low-density product was 850 ppm while the whole sample content was 496 ppm, indicating significant enrichment. As such, 2.2 SG setpoint was selected as the optimal cut point for the sorter operation.

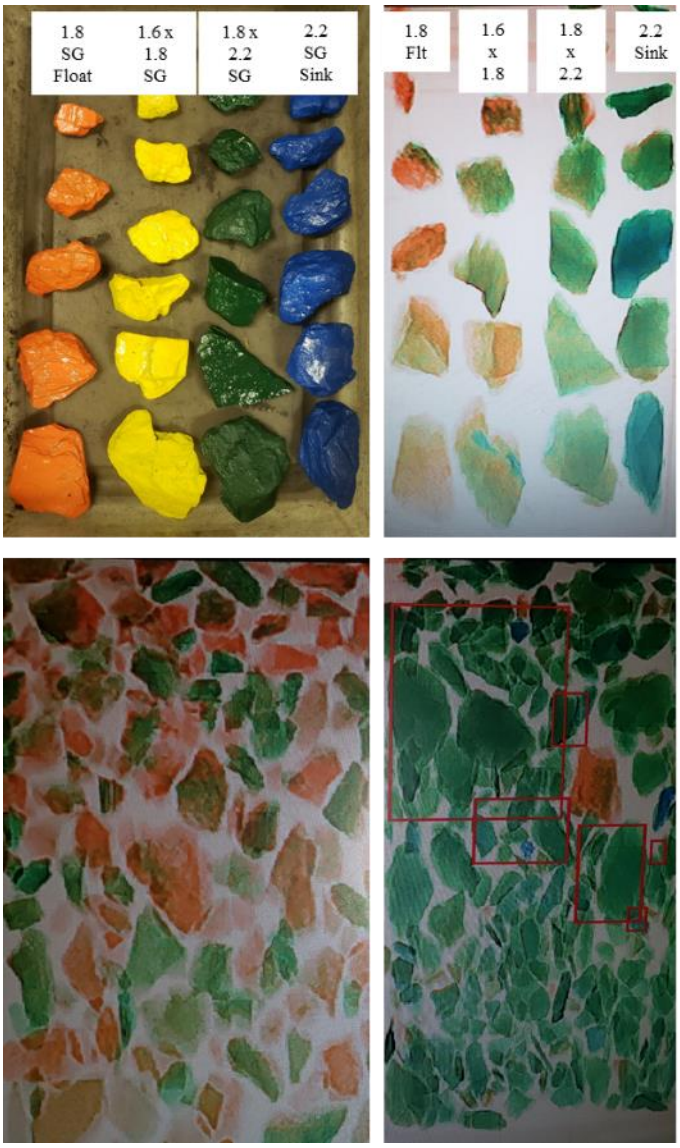



Figure 3.4.3. Particle characterization of Fire Clay coarse refuse for x-ray transmission sorting: a) color coded particles recovered from a density fractionation study; b) x-ray transmission image of the color-coded particles; c) low density product from a DXRT production unit and d) corresponding high density product.

Table 3.4.1.REE enrichment of Fire Clay coarse refuse material using a DXRT sorter: TREE ppm, a = dry, total mixed REE content on an ash-basis; and TREE ppm, w = dry whole sample basis content).

| Parameter | Feed | Test No. 1 | | Test No. 2 | |
|---|---|---|---|---|---|
| | | Product | Tailings | Product | Tailings |
| Ash (%) | 85.80 | 37.25 | 88.42 | 58.37 | 91.22 |
| TREE (ppm,a) | 366 | 667 | 350 | 850 | 301 |
| TREE (ppm,w) | 314 | 248 | 309 | 496 | 275 |
| Mass Yield (%) | 100.0 | 5.0 | 95.0 | 16.5 | 83.5 |

### *3.4.2 Pyrite and Calcite Concentration*

**Pyrite Concentration from Coal Preparation Plant**: Coal-based pyrite was concentrated from an operating preparation plant as a feedstock for an acid generation process by bio-oxidation. Various coal streams within the preparation plant were sampled (Figure 3.4.4) for testing the viability of pyrite concentration using a mineral spiral including:

1. High frequency screen overflow;
2. Clean coal sieve underflow;
3. Stack sizer tails;
4. Column flotation tails; and
5. Bottom of column flotation.

Figure 3.4.4 shows the flowsheet section including the circuits A and B in the plant in sampling points 1, 2, and 3 were located. Circuit C uses flotation columns instead of the StackSizer units that are utilized for ultrafine coal recovery in Circuits A and B. Sampling points 4 and 5 were located within the column flotation system of Circuit C.

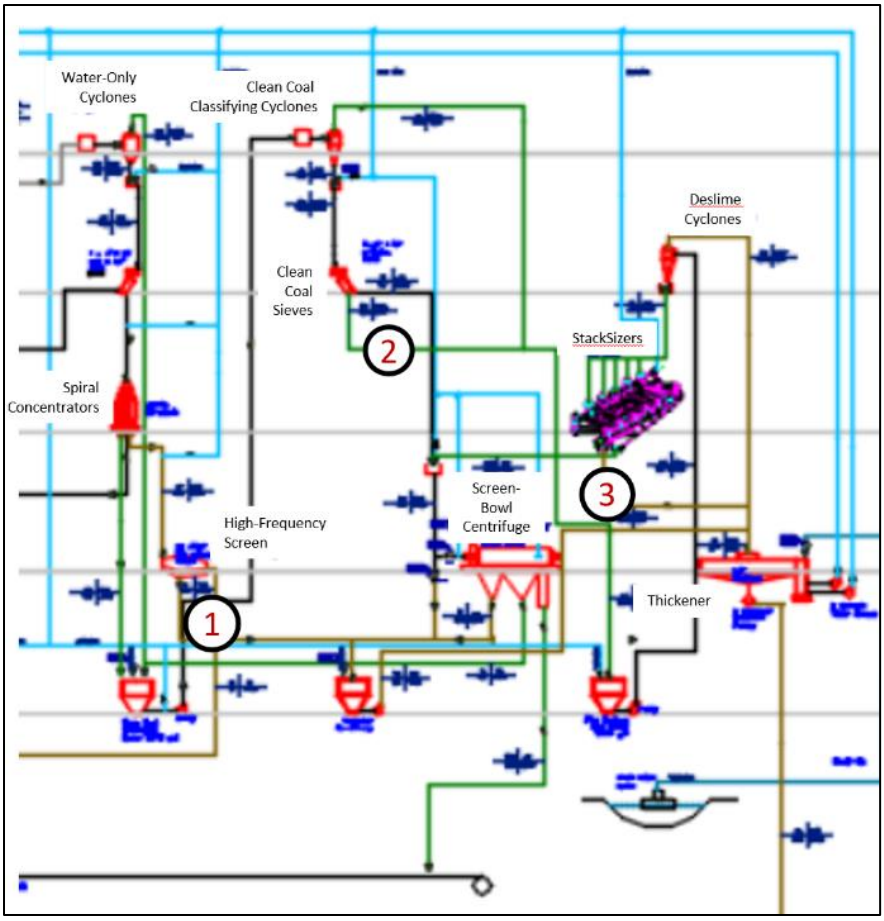



Figure 3.4.4 Flowsheet section showing the sampling points for testing the feasibility of pyrite concentration.

To assess the viability of density separation for pyrite concentration, a dense media separation of the samples was conducted using an undiluted solution of lithium meta tungstate (LMT) at a specific gravity of 2.95. The solids mass yield values for the 2.95 S.G. sink was highest for the clean coal sieve underflow with 21.1% material reporting to the sink as shown in Table 3.4.2 and Figure 3.4.5.

Table 3.4.2. Sink-float analysis for different samples at a density cut-point of 2.95 S.G.

| Sample | Float Mass (g) | Sink Mass (g) | Total Mass (g) | Float (%) | Sink (%) |
|---|---|---|---|---|---|
| High frequency screen overflow | 40.6 | 5.2 | 45.8 | 88.7 | 11.3 |
| Stack sizer tails | 20.8 | 2.7 | 23.5 | 88.4 | 11.6 |
| Flotation tails | 22.0 | 2.4 | 24.4 | 90.1 | 9.9 |
| Clean Coal sieve underflow | 20.4 | 5.4 | 25.8 | 78.9 | 21.1 |
| Bottom of Column Flotation | 19.5 | 3.3 | 22.8 | 85.5 | 14.5 |

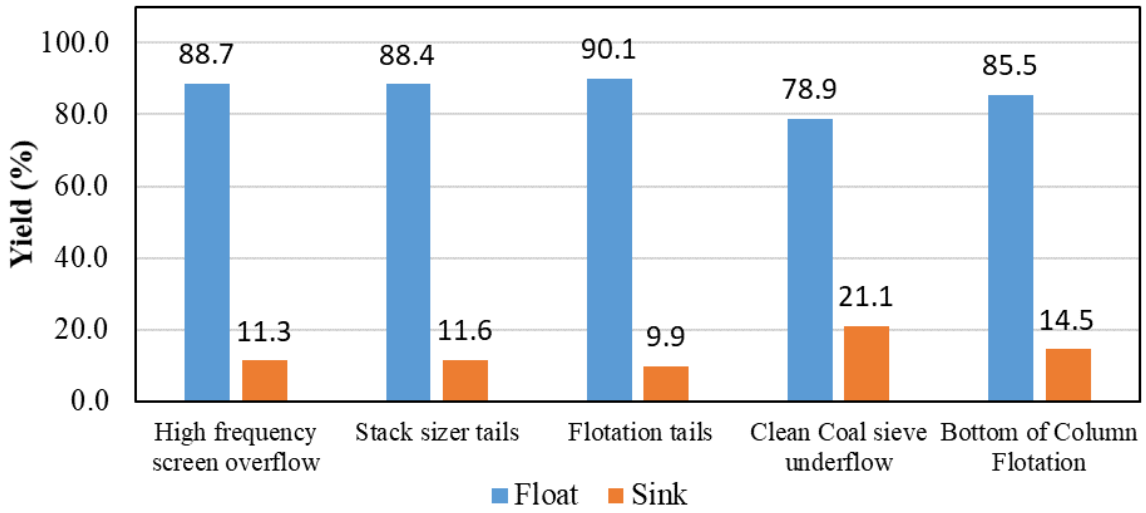



Figure 3.4.5. Sink-float analysis for different samples at a density cut-point of 2.95 S.G.

The proximate analysis was performed on all samples by LECO TGA 701 equipment. The results of the proximate analysis showed that all the solid samples were dry with very low moisture content (Table 3.4.3). The high frequency screen overflow, flotation column tails, and the sample collected from the bottom of the flotation column contained relatively high ash content. The high frequency screen overflow and flotation column tails contained over 85% ash whereas the sample collected from the bottom of the flotation column contained approximately 75% ash content. The high frequency screen also contained relatively high sulfur content, which was close to 8%. The flotation tails contained significantly lower sulfur content with 3.25% sulfur on a dry basis.

Table 3.4.3. Proximate analysis of the samples from different coal streams for pyrite concentration.

| Sample ID | Moisture (%) | Ash (%), db | Sulfur (%), db |
|---|---|---|---|
| High-frequency screen overflow | 2.35 | 85.67 | 7.97 |
| Stack sizer tails | 1.71 | 45.07 | 4.86 |
| Flotation tails | 0.42 | 85.79 | 3.25 |
| Clean Coal sieve underflow | 0.00 | 48.17 | 5.47 |
| Bottom of Column Flotation | 1.34 | 74.38 | 6.81 |

The samples were also analyzed by the x-ray fluorescence (XRF) to measure the concentration of major elements (Table 3.4.4). For all samples, SiO2 and Al2O3 comprised the majority of the total weight, i.e., > 65% by weight. The other two elements which dominated the composition were Fe2O3 and CaO, which reflects the relatively high pyrite and calcite contents. A high pyrite content is desirable for acid generation process by bio-oxidation whereas calcite content is largely detrimental for acid generation although a small portion is beneficial for microbe population growth.

Table 3.4.4. XRF analysis for samples from different coal streams for pyrite concentration.

| Sample ID | $SiO_2$ (%) | $Al_2O_3$ (%) | $Fe_2O_3$ (%) | CaO (%) | MgO (%) | MnO (%) | $K_2O$ (%) |
|---|---|---|---|---|---|---|---|
| High frequency screen overflow | 47.31 | 16.74 | 14.18 | 12.75 | 1.48 | 0.07 | 3.19 |
| Stack sizer tails | 52.67 | 15.71 | 13.64 | 9.47 | 1.36 | 0.07 | 2.74 |
| Flotation tails | 62.52 | 17.08 | 8.98 | 7.28 | 1.46 | 0.06 | 2.98 |
| Clean Coal sieve underflow | 51.08 | 14.59 | 14.69 | 11.26 | 1.29 | 0.07 | 2.51 |
| Bottom of Column Flotation | 52.40 | 13.01 | 13.41 | 13.64 | 1.17 | 0.09 | 2.31 |

| Sample ID | $Na_2O$ (%) | $P_2O_5$ (%) | $TiO_2$ (%) | BaO (%) | SrO (%) | $SO_3$ (%) | Total (%) |
|---|---|---|---|---|---|---|---|
| High frequency screen overflow | 0.29 | 0.79 | 0.73 | 0.23 | 0.02 | 1.84 | 99.63 |
| Stack sizer tails | 0.35 | 0.44 | 0.79 | 0.15 | 0.02 | 2.44 | 99.84 |
| Flotation tails | 0.32 | 0.37 | 0.85 | 0.10 | 0.02 | 1.58 | 103.60 |
| Clean Coal sieve underflow | 0.29 | 0.44 | 0.73 | 0.15 | 0.02 | 2.47 | 99.61 |
| Bottom of Column Flotation | 0.32 | 0.54 | 0.66 | 0.20 | 0.02 | 2.26 | 100.04 |

Based on the cumulative analytical findings, it was concluded that the StackSizer tailings stream was tthe most suitable for pyrite concentration due to the high pyrite concentration as well as low calcite concentration. Additionally, the smaller particle size of the stack sizer tails stream aids in the acid generation by the bio-oxidation process.

The elemental contents including REEs were measured by ICP-OES analysis. The solid samples were digested using ASTM procedure D6357-11. The standard solid sample supplied by the National Institute of Standards (NIST) was used as a quality control standard for each batch of solids analyzed. The ICP-OES analysis was carried out by KGS. The REE concentrations in the samples taken from different streams analyzed were very similar (Figure 3.4.6). The light REE concentration was the major component of the total REE concentration in the samples. The elements cerium, lanthanum, neodymium, and scandium formed the majority of the TREE concentration. Yttrium had the highest concentration of all the heavy REEs in the samples. The detailed elemental analyses are shown in Table 3.4.5.

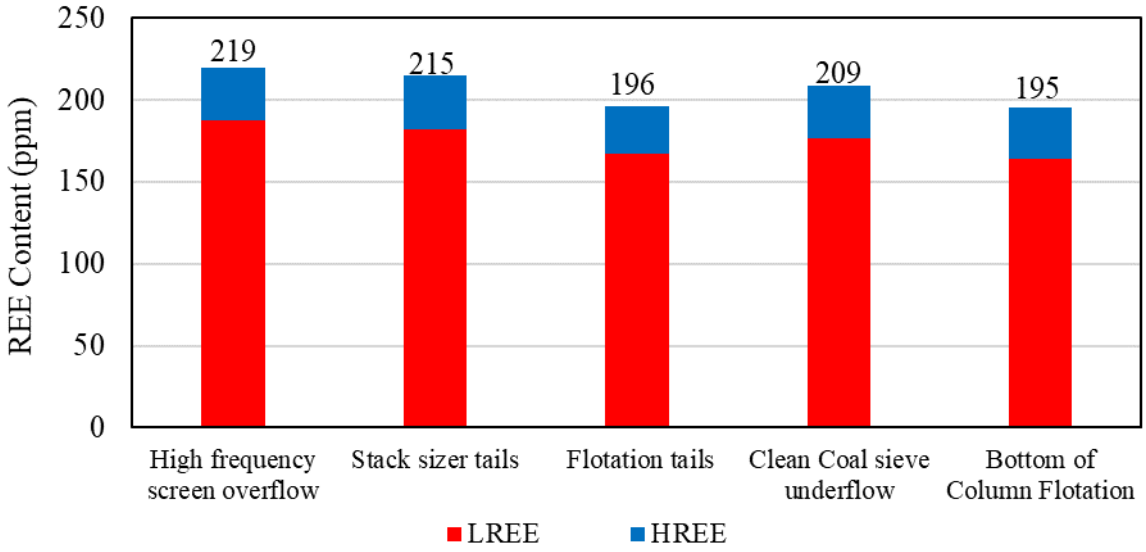



Figure 3.4.6. REE concentration (ppm) of samples from different coal streams for pyrite concentration.

Table 3.4.5. REE analysis of samples from different coal streams for pyrite concentration.

| Sample ID | Sc (ppm) | La (ppm) | Ce (ppm) | Pr (ppm) | Nd (ppm) | Sm (ppm) | Eu (ppm) | Gd (ppm) | LREE (ppm) |
|---|---|---|---|---|---|---|---|---|---|
| High frequency screen overflow | 34 | 33 | 52 | 10 | 35 | 7 | 1 | 15 | 187 |
| Stack sizer tails | 28 | 32 | 53 | 11 | 34 | 8 | 2 | 13 | 182 |
| Flotation tails | 25 | 31 | 50 | 9 | 32 | 7 | 1 | 11 | 167 |
| Clean Coal sieve underflow | 32 | 31 | 46 | 11 | 33 | 7 | 2 | 15 | 177 |
| Bottom of Column Flotation | 35 | 28 | 37 | 10 | 31 | 7 | 1 | 16 | 164 |
| Sample ID | Tb (ppm) | Dy (ppm) | Ho (ppm) | Er (ppm) | Tm (ppm) | Yb (ppm) | Lu (ppm) | Y (ppm) | HREE (ppm) |
| High frequency screen overflow | 2 | 3 | 0 | 3 | 1 | 3 | 2 | 19 | 32 |
| Stack sizer tails | 2 | 3 | 0 | 3 | 1 | 3 | 2 | 19 | 33 |
| Flotation tails | 1 | 3 | 0 | 3 | 1 | 3 | 1 | 17 | 29 |
| Clean Coal sieve underflow | 2 | 2 | 0 | 3 | 1 | 4 | 2 | 19 | 32 |
| Bottom of Column Flotation | 2 | 2 | 0 | 2 | 1 | 3 | 2 | 18 | 31 |

SEM analysis of different samples confirmed that the pyrite in the coal occurred either in crystalline form or framboidal form. The framboidal pyrite, which has higher surface area, is desirable for the bio-oxidation process aiding in acid generation (Figure 3.4.7).

(a) (b)

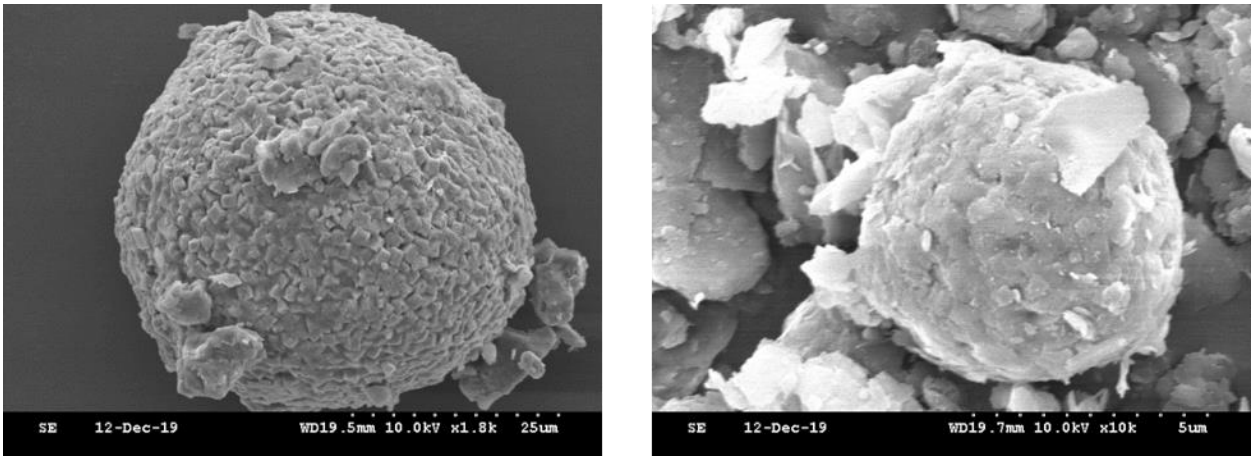



Figure 3.4.7. SEM images of the coal samples from (a) column flotation bottom and (b) stack sizer underflow showing evidence of the presence of framboidal pyrite.

A rougher-cleaner spiral circuit was installed in the coal preparation plant to recover and concentrate pyrite from the StackSizer underflow stream (Figure 3.4.8) for pyrite concentration. The rougher spiral was a coal spiral (Figure 3.4.9 a) while the cleaner spiral was a single start mineral spiral (Figure 3.4.9 b) MG-12 supplied by Mineral Technologies USA. The high-density product stream of the rougher spiral was fed to the cleaner spiral. The cleaner spiral produced the final high-density concentrate containing primarily pyrite as shown in Figure 3.4.10. This circuit arrangement was used to provide concentrated pyrite throughout the project. The pyrite concentrate was further concentrated by a circuit in the pilot plant that included a 100 micron screen, low-intensity magnetic separator, and a classifying cyclone.

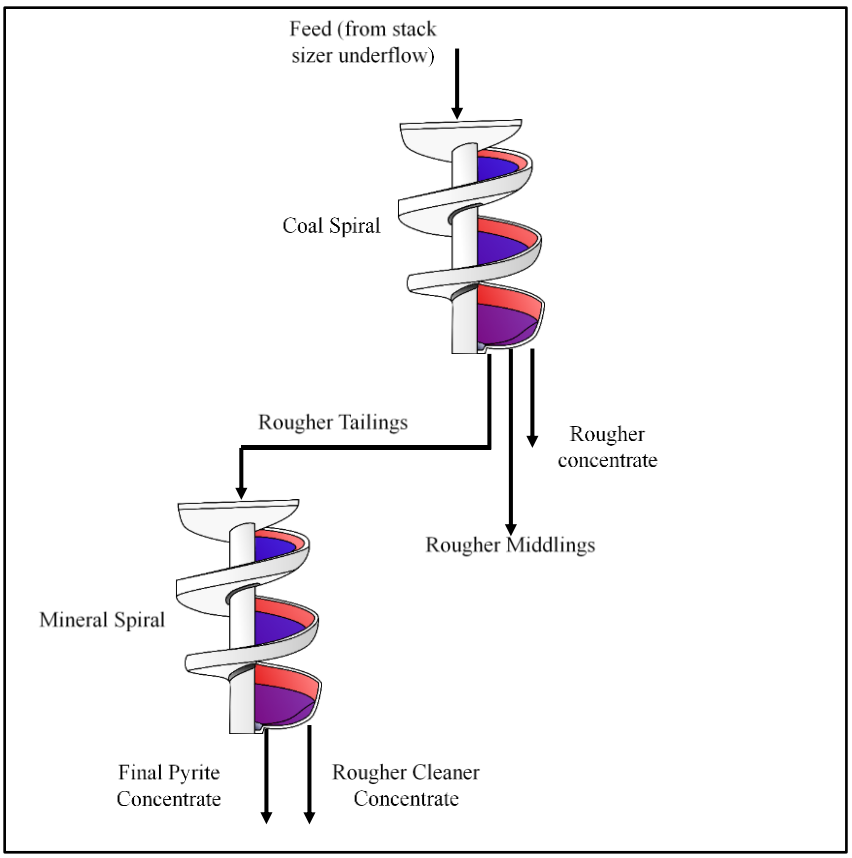



Figure 3.4.8. Schematic for pyrite concentration from stack sizer underflow using 2 spirals in series




Figure 3.4.9. (a) Mineral spiral and (b) coal spiral for pyrite concentration.

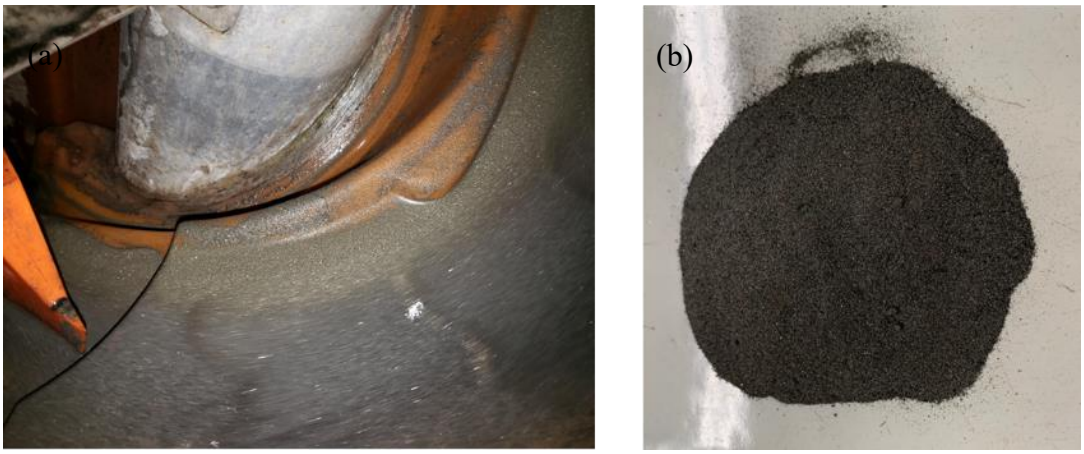



Figure 3.4.10. (a) Pyrite concentration in the mineral spiral (b) pyrite concentrate.

The final pyrite concentrate was subjected to proximate analysis and XRF analysis to determine the pyrite content. The results of proximate analysis (Table 3.4.6) showed that the total sulfur content of the concentrate was 31.56%. The XRF analysis results (Table 3.4.7) showed the total Fe concentration to be 30.37%. This value converted to FeS2 content was 65.02% based on stoichiometry. The XRF analysis also revealed that the CaO% was 8.6%, which equates to 15.37% calcite. The conversions based on stoichiometry assume that the entire concentration of iron and calcium in the sample was due to pyrite and calcite presence, respectively.

Table 3.4.6. Proximate analysis of the pyrite concentrate.

| Sample | Moisture (%) | Ash (%), db | Volatiles (%), db | Sulfur (%), db |
|---|---|---|---|---|
| Pyrite Concentrate | 0.10 | 73.09 | 24.15 | 31.56 |

Table 3.4.7. XRF analysis of the pyrite concentrate.

| Sample | Total Fe (%) | $SiO_2$ (%) | $Al_2O_3$ (%) | CaO (%) | MgO (%) | MnO (%) | $Na_2O$ (%) |
|---|---|---|---|---|---|---|---|
| Pyrite Concentrate | 30.37 | 11.45 | 2.50 | 8.61 | 0.36 | 0.09 | 0.00 |
| | Total S (%) | $K_2O$ (%) | $P_2O_5$ (%) | $TiO_2$ (%) | BaO (%) | SrO (%) | Total (%) |
| | 31.56 | 0.35 | 0.47 | 0.29 | 1.88 | 0.02 | 87.95 |

**Calcite and pyrite separation circuit:** Twenty 55-gallon drums of the rougher-cleaner tailings material were collected from the rougher-cleaner spiral pyrite and calcite concentration circuit in Alliance Coal's River View preparation plant to treat the underflow of the StackSizer stream. Representative samples were collected from each stream and analyzed for mineralogical content by XRD as well as ash and total sulfur contents using ASTM procedures. As shown in Table 3.4.8, the pyrite was concentrated into the rougher-cleaner tailings stream as indicated by the XRD pyrite analysis and the high total sulfur content. Calcite was distributed somewhat evenly in the various process streams. Silica was the dominant mineral component in each of the process streams.

Table 3.4.8. Mineral content as determined by XRD in the rougher-cleaner spiral circuit process streams as well as ash and total sulfur contents on a dry basis.

| Process Stream | Ash (%) | T. Sulfur (%) | Percentage | | | |
|---|---|---|---|---|---|---|
| | | | Pyrite | Silica | Calcite | Kaolinite |
| Rougher Feed | 39.79 | 4.82 | 8.4 | 47.1 | 19.5 | 23.8 |
| Rougher Concentrate | 29.00 | 4.3 | 11.2 | 44.8 | 17.5 | 6.3 |
| Rougher Middlings | 36.68 | 4.62 | 10.2 | 47.9 | 7.7 | 29.0 |
| Rougher Cleaner Concentrate | 74.04 | 8.97 | 14.2 | 57.6 | 17.7 | 9.2 |
| Rougher Cleaner Tailings | 77.55 | 30.54 | 38.1 | 36.4 | 17.7 | 4.7 |

The mineral composition of the rougher-spiral tailings material (also referred to as the pyrite concentrate) was analyzed by particle size using XRD. As shown in Table 3.4.9, the weight split was nearly equal for material coarser and finer than 75 microns (200 mesh). The material coarser than 75 microns was found to be mostly calcite (63.7%) which indicated that the fraction was a promising source for base material to increase the pH of the process solutions when needed. The material finer than 75 microns had a pyrite content of 34.2%. Calcite and silica contents in this fraction are relatively high.

Table 3.4.9. Mineral composition by particle size in the rougher-cleaner tailings stream (pyrite concentrate) as determined by XRD.

| Size Fraction | Weight (%) | Percentage | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | | Pyrite | Silica | Calcite | Kaolinite | Barite | Montmoril. | Bohmite |
| +100 mesh | 8.84 | 6.3 | 6.4 | 68.2 | 4.9 | --- | --- | 14.2 |
| 100 mesh x 200 mesh | 43.72 | 20.0 | 14.1 | 62.8 | 3.1 | 0.0 | --- | --- |
| 200 mesh x 325 mesh | 14.88 | 43.3 | 12.2 | 35.4 | 1.1 | 0.6 | 7.4 | --- |
| -325 mesh | 32.56 | 30.1 | 44.8 | 21.0 | 3.7 | 0.3 | --- | --- |
| Total | 100.00 | 25.5 | 23.1 | 45.6 | 3.2 | 0.2 | 1.1 | 1.3 |

Based on the analytical findings, efforts to recover and concentrate both calcite and pyrite involved screening of the rougher-cleaner tailings material with the oversize material being the final calcite product to be used for increasing solution pH values for process streams being treated for precipitation purposes. Given the significant density differences between pyrite (SG=4.8) and calcite/silica (SG≈2.7) as well as the high silica content in the fraction smaller than 45 microns, the underflow of the screening process was processed using a classifying cyclone at an elevated feed pressure (e.g., 25 psi) from which the underflow was an enriched pyrite product. Further enrichment may be possible using a fluidized-bed gravity separator and/or a low-intensity wet drum magnetic separator. Despite the XRD analysis results not showing a significant presence of material that would be ferromagnetic, a preliminary assessment using a hand magnet revealed the presence of magnetic material in the finer than 75-micron fraction.

Since calcite is a good base material and hinders acid production, a physical separation circuit was constructed utilizing a wet vibrating screen, low intensity magnetic drum separator to remove magnetite and a classifying cyclone to remove silica and calcite from the -75 micron pyrite material as shown in Figure 3.4.11. The 20 barrels of pyrite pre-concentrate was processed through the circuit. Since a significant amount of pyrite still existed in the material coarser fractions, a MG-6 spiral was purchased and used to enrich the calcite and produce a relatively rich pyrite fraction.



Figure 3.4.11. Physical separation circuit used to enrich the calcite and pyrite in the pyrite pre-concentrate material. Circuit includes a wet vibrating screen, low intensity magnetic separator and classifying cyclone.

### *3.4.3 Pyrite Concentration Pilot Plant Circuit*

To enrich the pyrite concentrate collected from the operating coal plant, the circuit described in section 1.1.2.2 was used to create a purer pyrite concentrate while separating out the calcite that was mostly a negative contributor to acid production. A fluidized-bed system using Lamella plates was also evaluated. The solids concentrations of the underflow and overflow of Lamella plate separator were measured to determine their effectiveness as a substitute for a filter press within the pyrite circuit.

**Methods:** Approximately 30.4 kg of pyrite circuit feed was mixed into 38 gallons of water to prepare a feed slurry of approximately 20% solids by mass. The feed was added by a peristaltic pump into a vibratory screener with a mesh of 45 microns. This separated the larger particles, such as calcite and silica, and left smaller particles to flow through the screen, such as the desired pyrite particles. Afterward, the screen underflow was fed through a low-intensity magnetic (LIM) drum separator to remove magnetite and other magnetic material. The clearance between the rotary magnet and the bottom drum was around 0.17-inch to provide enhanced magnetite removal.

From the LIM separator, the slurry flowed to a conical mixing tank where it was pumped through a centrifugal pump into a cyclone. The underflow of the cyclone produced pyrite concentrate, which was sent back to the mixing tank, while the cyclone overflow, consisting of mostly clay solids and water, was pumped into the Lamella plates to concentrate the solids. Clarified water from the Lamella plates was recycled back to the system as spray water for the LIM separator and as make-up water in the feed mix-sump.

For a controlled test of the circuit, two batches of 20% solid slurry were prepared and separately ran through the pyrite circuit. As the liquid level in the conical mixing tank used to feed the classifying cyclones lowered, clean tap water was added to the tank to maintain a level needed for

mixing. A total of 230 gallons of clean water were added to the system during the process. Overall, 57 kg of pyrite concentrate were produced.

**Results:** Pyrite concentrate samples produced during this process were analyzed to determine any improvement on the quality of the pyrite percentage within the cleaned pyrite concentrate. Solids concentration was measured as a function of time for the overflow and underflow streams of the Lamella plate separator to assess their effectiveness.

Three different concentrated pyrite samples were taken from the final product and analyzed with an ICP-OES to quantify the iron content. These concentrations were used to calculate enrichment ratio to determine if a higher percentage of Fe was concentrated through the circuit. Enrichment ratio is defined as the concentration of Fe in the final concentrate over the concentration of Fe in the feed material. Sample 1 had an enrichment ratio of 1.25 and Sample 2 had an enrichment ratio of 1.09. Enrichment ratio of Sample 3 could not be calculated due to the concentration of Fe being unreadable by the ICP (exceeding the units limits).

Solid concentrations of Lamella overflow and underflow were measured by recovering the solids by filtration, drying and weighing the dry solids. The dry solids weight was divided by the sample slurry weight to determine the solid concentration. The solid concentration in the Lamella underflow increased during the process while the amount in the overflow decreased. Changes in the percent solids of Lamella overflow and underflow throughout the process indicates the effectiveness of the Lamella plates as a clarifier (Figure 3.4.12). The recycled water from the Lamella plate separator had sufficiently low solids concentration to be used as make-up water.

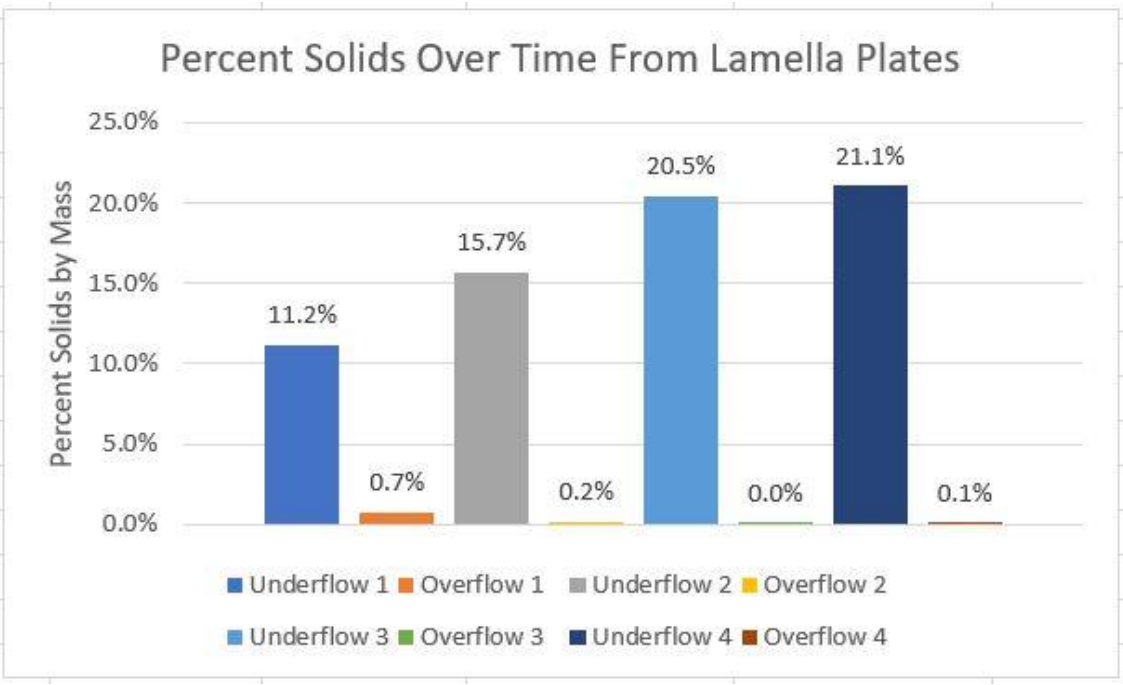



Figure 3.4.12. Percent solids by mass of Lamella plate overflow and underflow.

### *3.4.4 REE Characterization*

To evaluate the REE mode of occurrence in the density fractionated West Kentucky No. 13 and Fire Clay course refuse samples, a detailed sequential extraction study was conducted. The

sequential extraction tests utilized a four-step extraction sequence to identify the ion exchangeable, carbonate, metal oxide, acid soluble, and residual REE phases (Figure 3.4.13). The test program utilized both as-received material as well as material that was calcined for two hours at 600°C in a laboratory muffle furnace to investigate the phase changes imparted by the calcining/roasting process. All tests were repeated in triplicate to ensure experimental reproducibility. Figure 3.4.14 summarizes the test matrix.

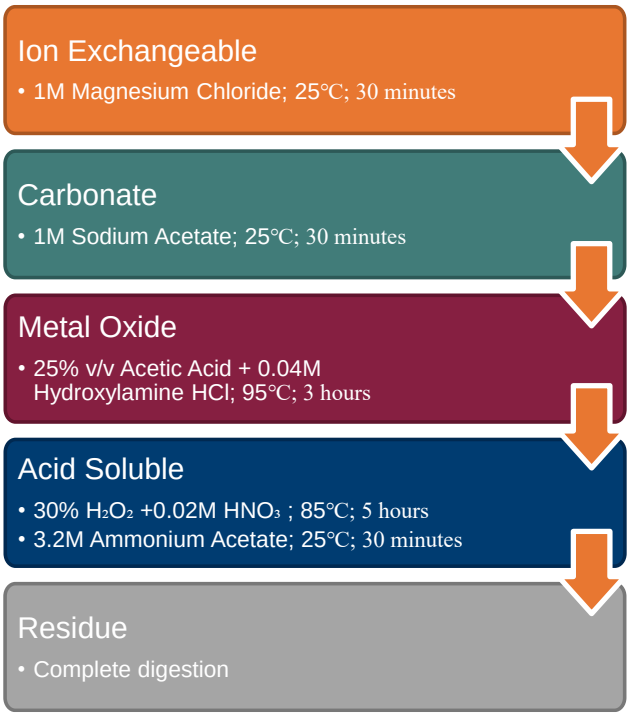



Figure 3.4.13 Test Procedure for 4-step sequential chemical extraction.

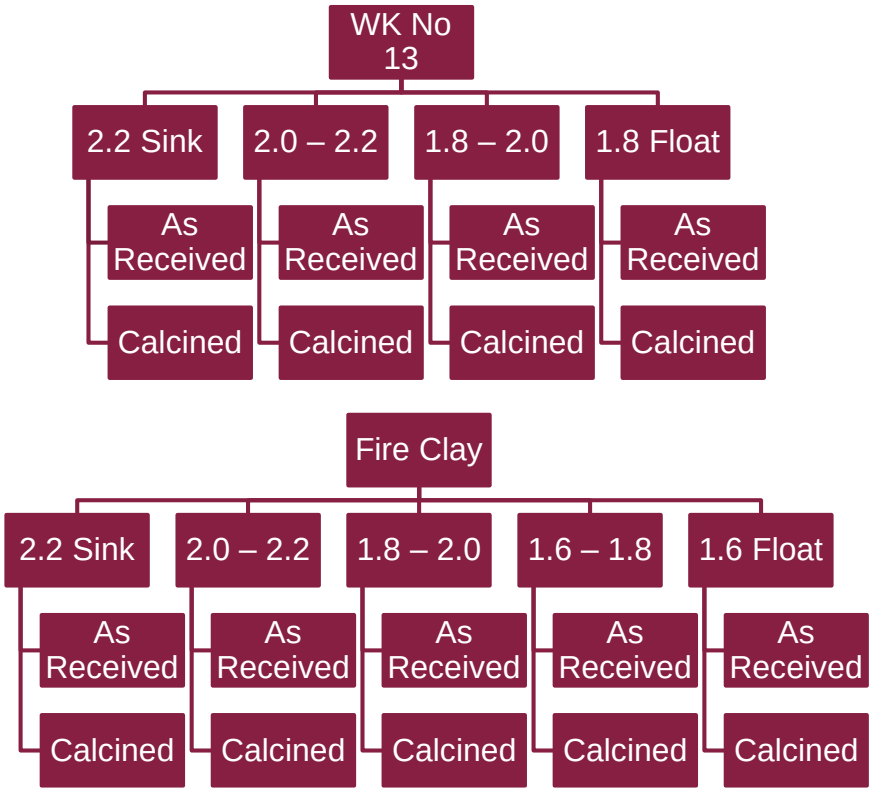



Figure 3.4.14. Sequential chemical extraction test matrix.

XRD results obtained from the 2.0 x 2.2 SG material of the West Kentucky No. 13 and Fire Clay samples are shown in Figure 3.4.15 and Figure 3.4.16, respectively. While the samples show some distinct characteristics, both contain similar major minerals including quartz, kaolinite, and pyrite. In both cases, calcining at 600°C led to the dehydroxylation of the kaolinite. The data also appears to indicate some conversion of pyrite to hematite; however, the hematite signatures in the calcined material are not very strong.

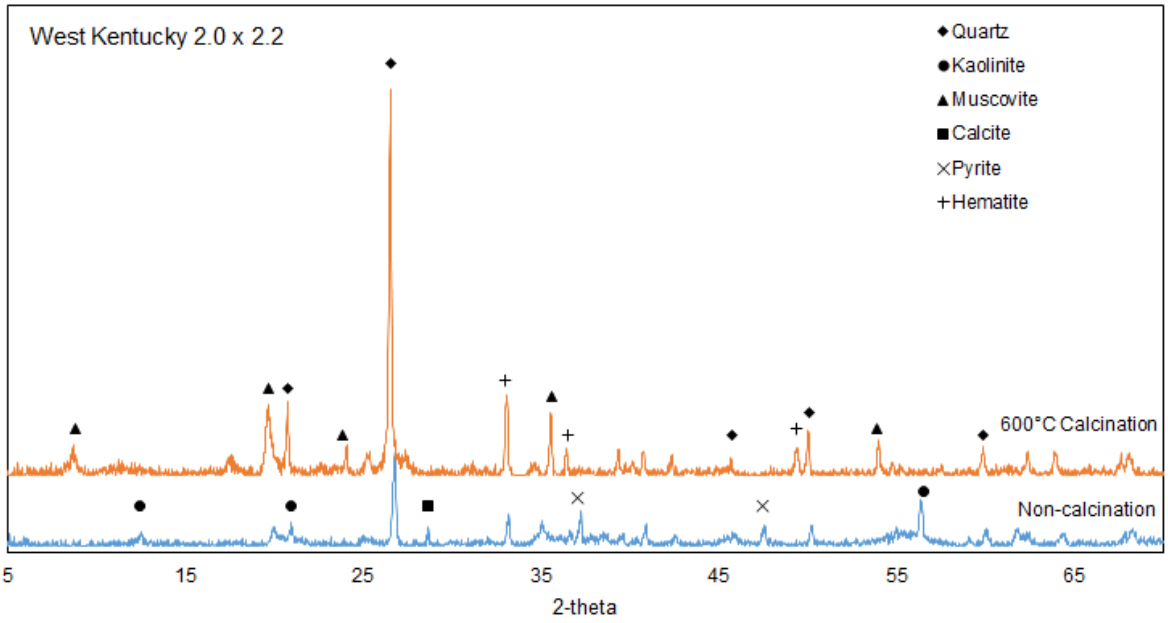



Figure 3.4.15. XRD Data for West KY No. 13 2.0 x 2.2 Material.

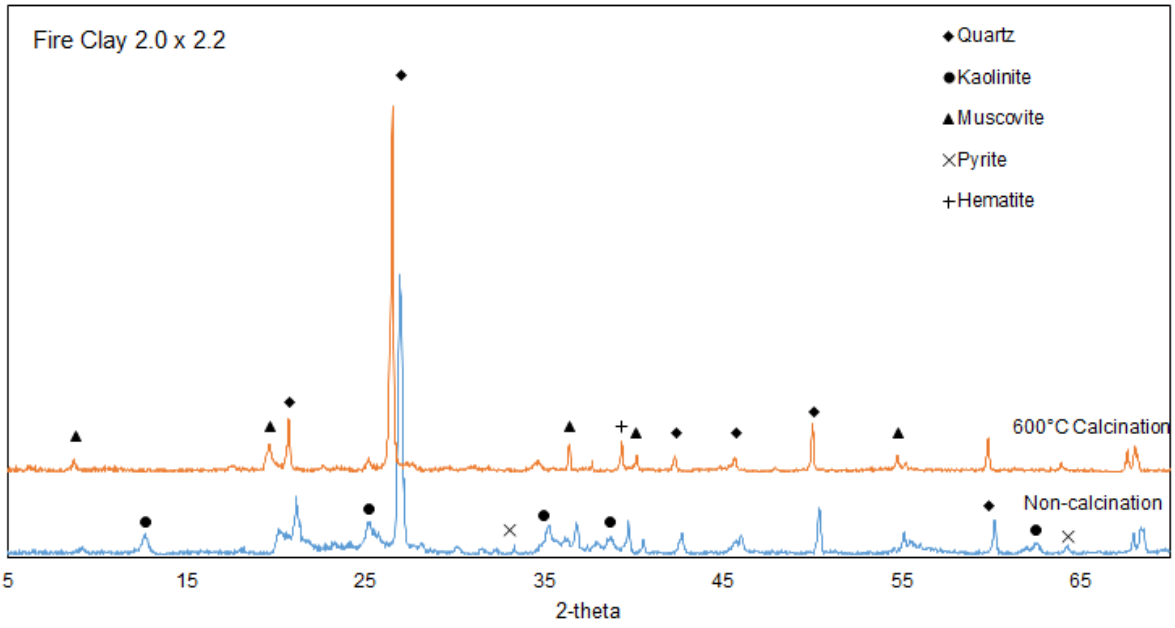



Figure 3.4.16. XRD Data for Fire Clay 2.0 x 2.2 Material

Figure 3.4.17 through Figure 3.4.20 show the raw sequential extraction data for the otal REEs, LREE, HREE, and Scandium. In all cases, the as-received material tended to have poor leachability with greater than 90% of REEs residing in the residue phase. This trend was consistent for both samples and across all density classes with only slight variations distinguishing the materials. After calcination, the materials tended to respond better to leaching, as the residual fraction was reduced to 40% to 80% in most cases. Moreover, most of the residual phases were

converted to metal oxides as this fraction often saw the most substantial gains. Carbonates and acid soluble phases were also increased after leaching, albeit to a lower degree. The ion exchangeable phase was negligible in all cases.

Figure 3.4.21 through Figure 3.4.23 show the same data plotted as the absolute change in fractionation (with and without calcination) as a function of density class. These data provide a means of quantifying the impact of calcination in transforming the mode of REE occurrence in the material – as a function of density class. Figure 3.4.21 shows the TREE data (which closely matches the LREEs), while Figure 3.4.22 shows the data for Dy, which is indicative of most HREEs. The trends in Figure 3.4.21 show that the West Kentucky No. 13 tended to respond better to roasting across all SG classes. The middling density classes (1.8 x 2.0 SG and 2.0 x 2.2 SG) showed the highest degree of conversation. This conversion was signified by a slight increase in the carbonate and acid soluble fractions (<5%) and a significant increase in the metal oxide fraction (40% to 50%). The Fire Clay material showed similar trends, albeit to a reduced degree. Interestingly, the trend of conversation rate as a function of specific gravity (SG) was reversed for Dy and the HREEs when compared to the LREEs. For HREEs, greater conversion was seen for the lower SG fractions, i.e., <1.7 SG. Conversion of HREEs was generally less than that of LREEs; however, most of increases were still observed to be in the metal oxide fraction. Scandium follows a similar trend to that of Dy; however, the difference between the two samples is less pronounced (Figure 3.4.23).

Overall, this data shows that roasting is effective at converting material from a residual phase to a metal oxide phase; however, the degree of this conversion varies with sample type and with SG fraction. LREEs (and TREEs by proxy given the high fraction of LREEs) tend to show superior conversion for middle to high density fractions, while HREEs tend to show better conversion for lower density fractions. This result confirms that the host minerals are quite distinct for HREE and LREEs.

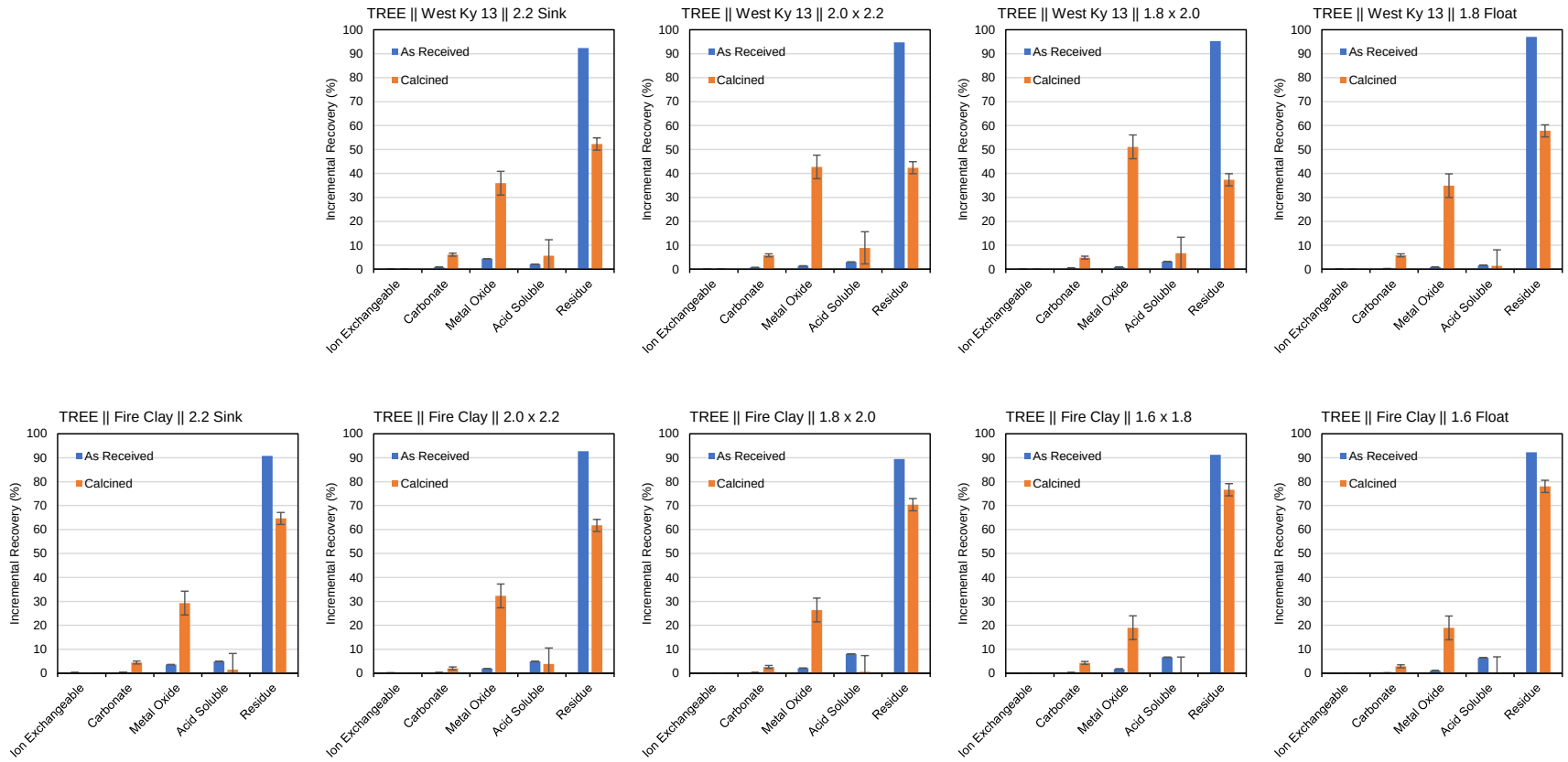



Figure 3.4.17. Sequential chemical extraction data for total REE.

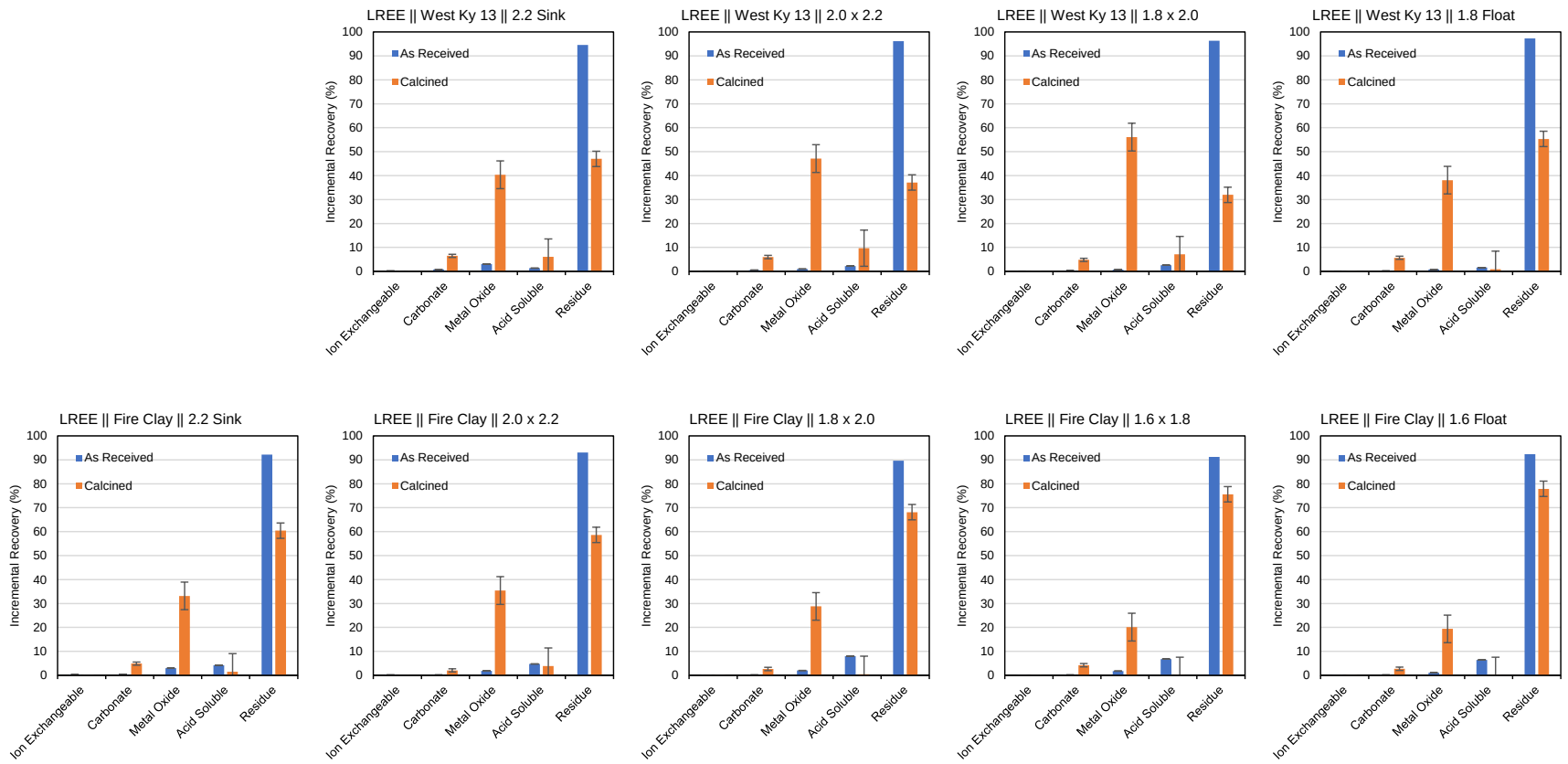



Figure 3.4.18. Sequential chemical extraction data for LREE.

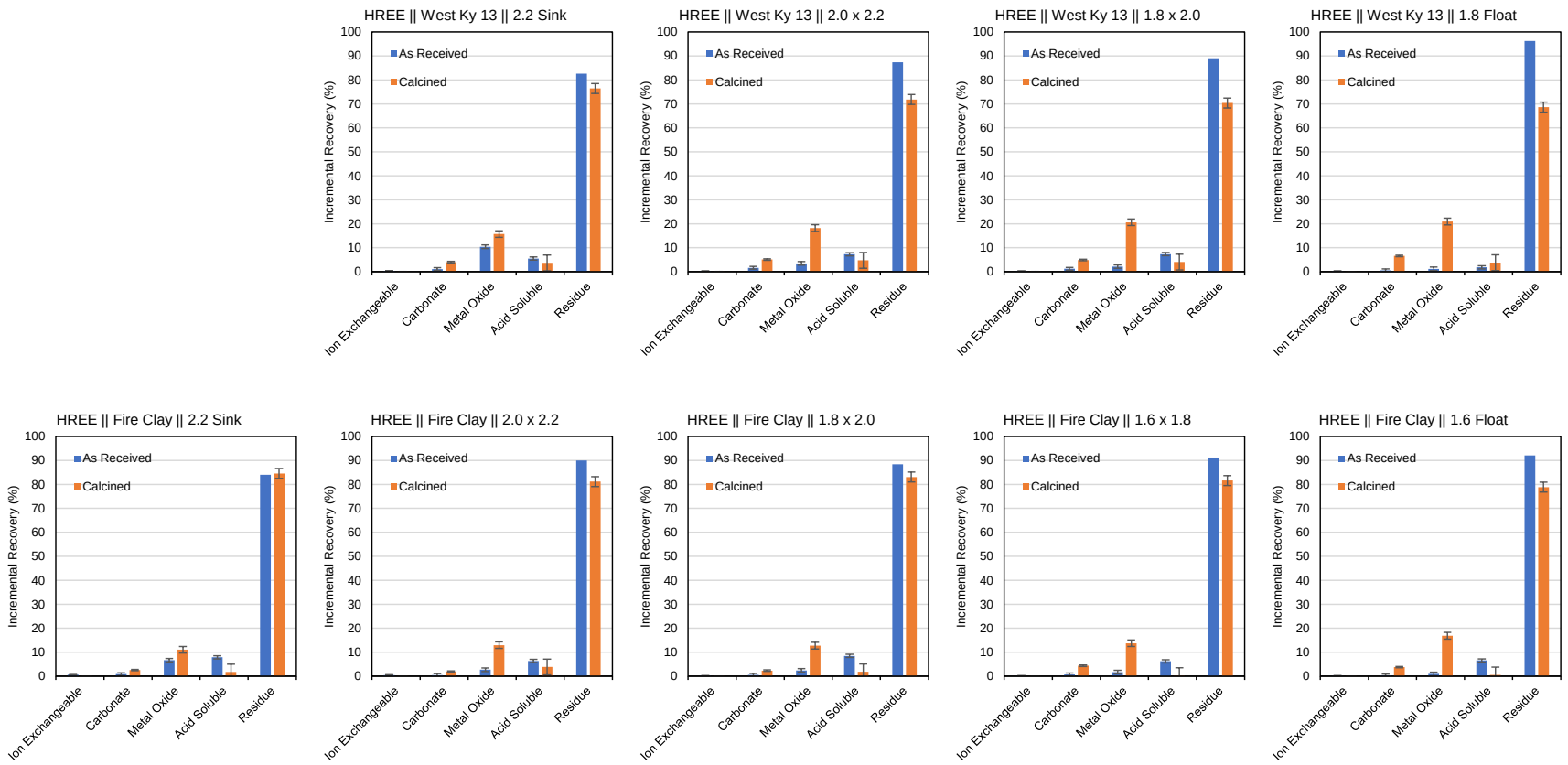



Figure 3.4.19. Sequential chemical extraction data for HREE.

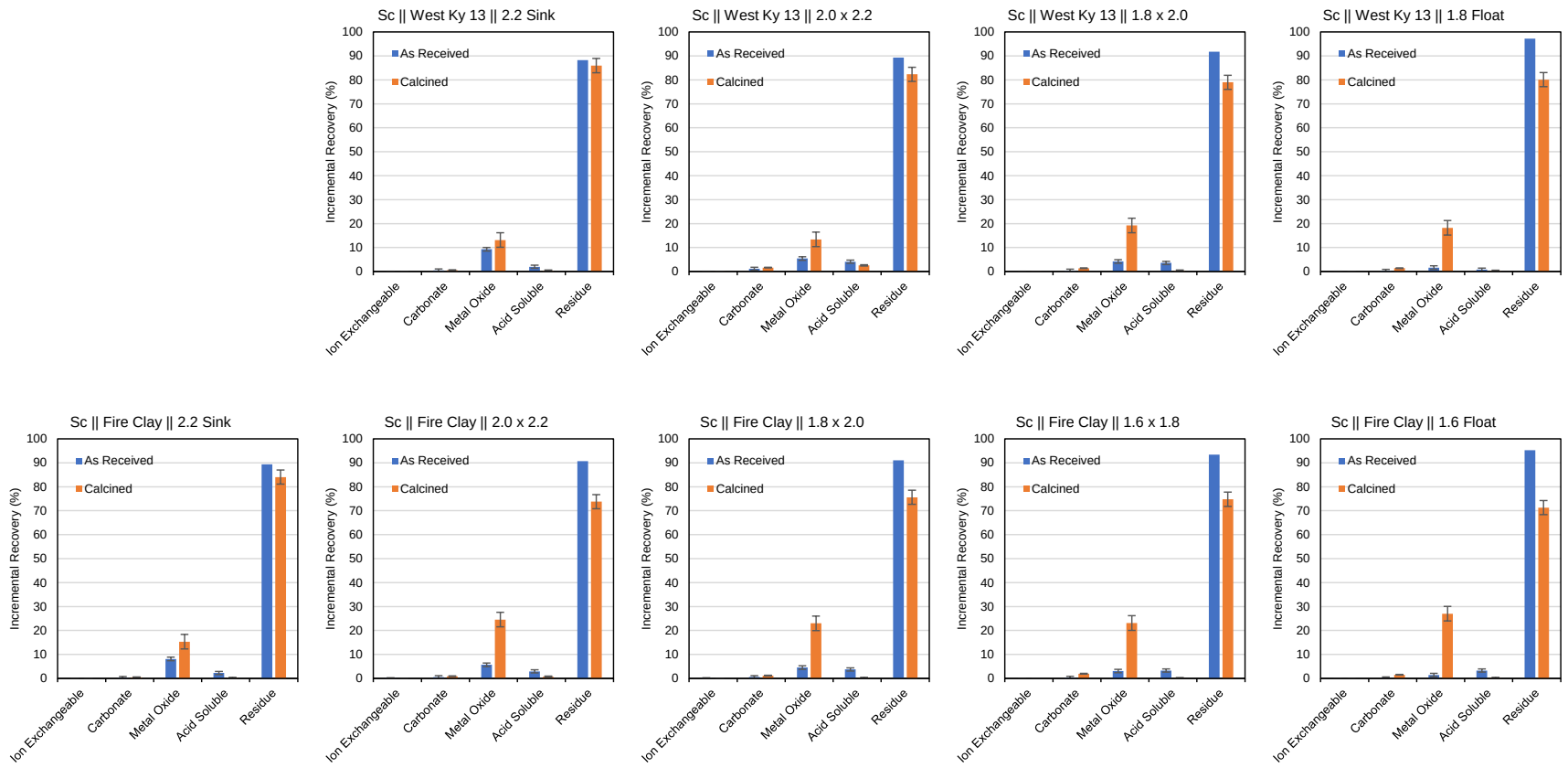



Figure 3.4.20. Sequential chemical extraction data for Sc.

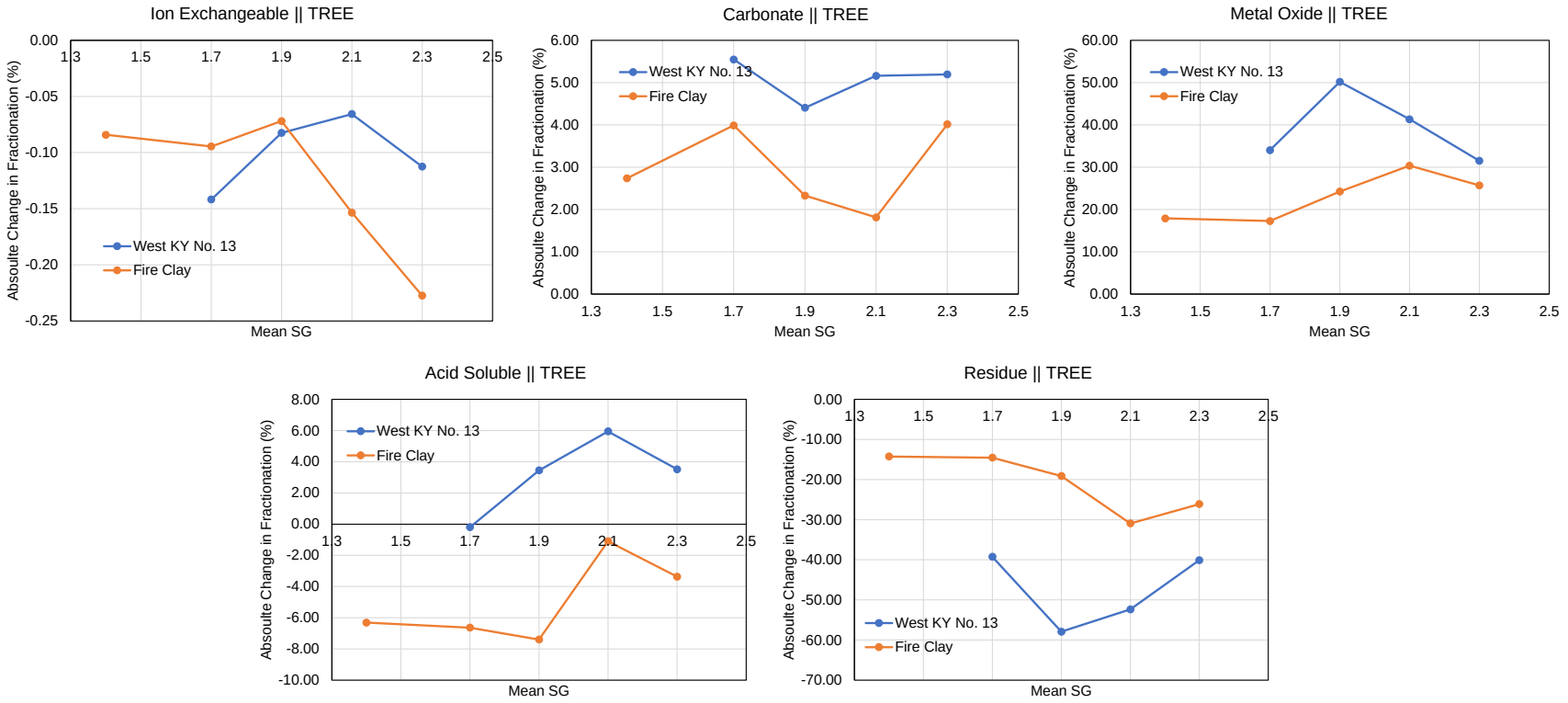



Figure 3.4.21. Change in fractionation observed for the total rare earths between as received and calcined density fraction of the West Kentucky No. 13 and Fire Clay coarse refuse samples. A positive number indicates that the fraction in the respective phase was increased after calcining at 600° C for two hours.

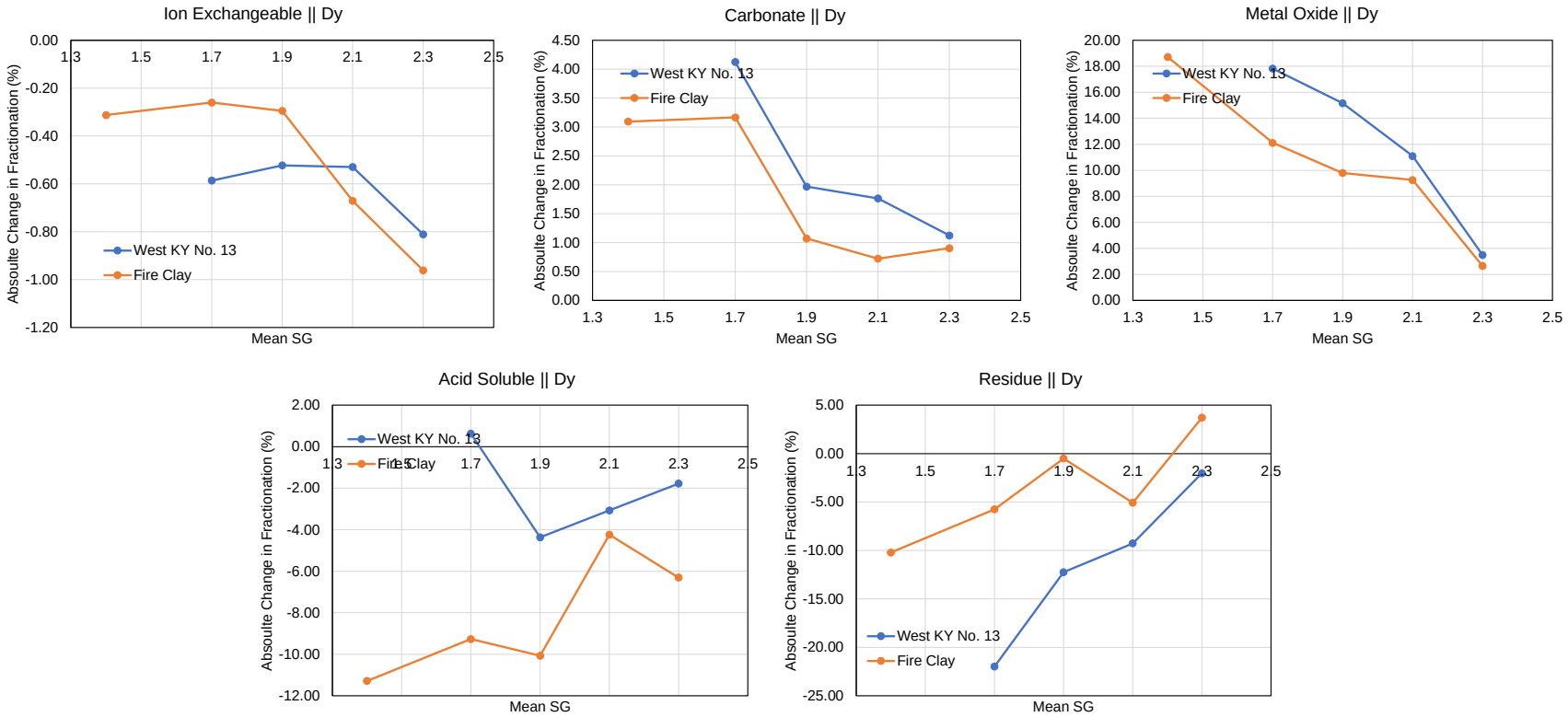



Figure 3.4.22. Change in fractionation observed for dysprosium (Dy) between as received and calcined density fraction of the West Kentucky No. 13 and Fire Clay coarse refuse samples. A positive number indicates that the fraction in the respective phase was increased after calcining at 600° C for two hours.

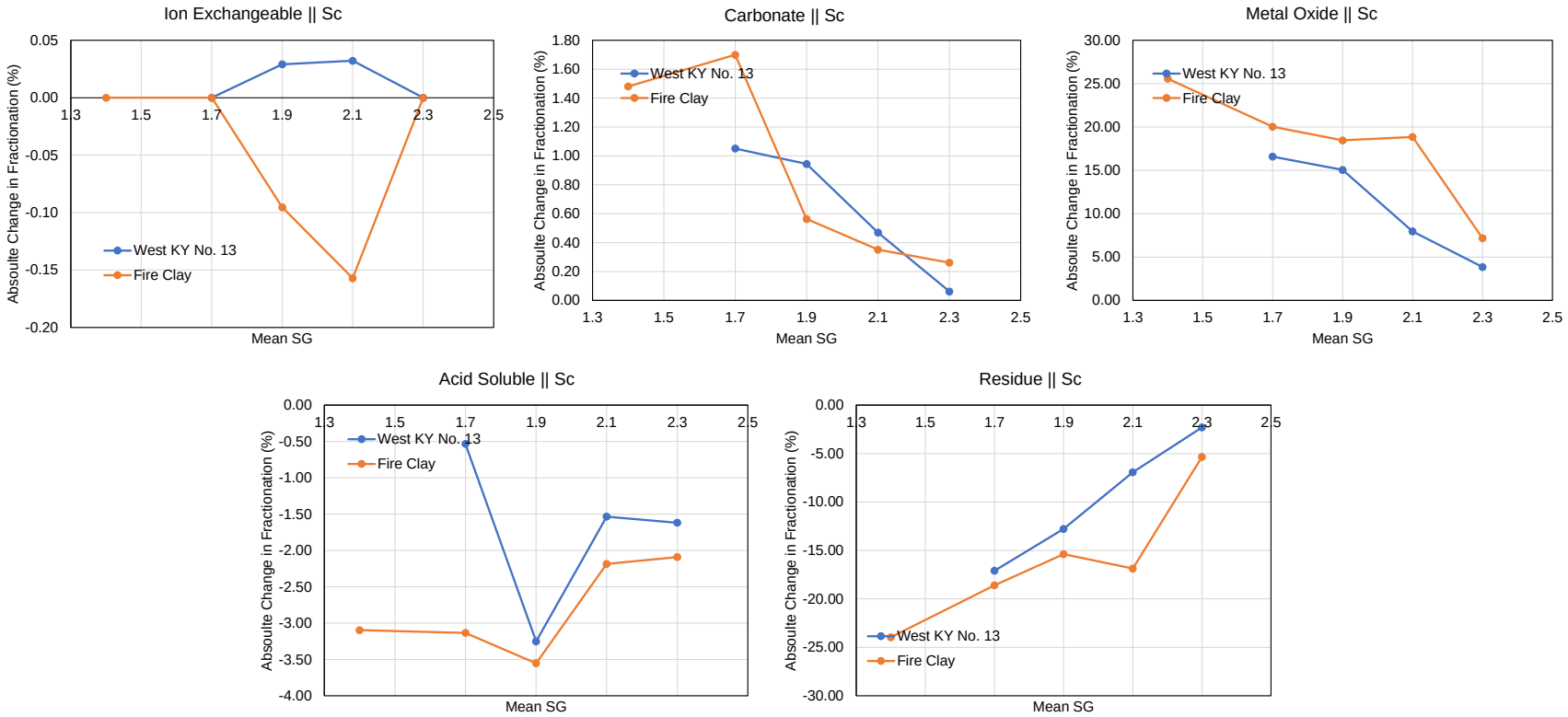



Figure 3.4.23. Change in fractionation observed for scandium (Sc) between as received and calcined density fraction of the West Kentucky No. 13 and Fire Clay coarse refuse samples. A positive number indicates that the fraction in the respective phase was increased after calcining at 600° C for two hours.

## 3.5 OPTIMIZATION OF BIO-OXIDATION AND ACID GENERATION (TASK 7)

The primary objective of the study was to study and develop a bio-oxidation system that is capable of producing sulfuric acid from naturally occurring coal pyrite to be used as the lixiviant for extracting rare earths and critical materials from bituminous coal refuse. Active bacteria cultures of *Acidithiobacilus ferrooxidans* and related organisms were acquired from previously used cultures at the University of Utah and supplemented with industrial sources with active biooxidation related leaching operations and biological supply companies. The bacterial cultures were grown in a combination of leaching solution that was supplemented with traditional medium, containing essential salts with potassium, nitrogen, magnesium, and phosphorous. The cultures were sparged with air to provide both oxygen and carbon dioxide, which are necessary for metabolism and growth. Redox potentials were monitored to check bioactivity and periodic ferrous oxidation tests were performed to quantify the bacterial population using a predetermined quantity of ferrous ion addition and the relative rate of oxidation as determined by electrochemical testing.

Process parameters effects were evaluated to quantify their influence on biooxidation performance using 1.5 and 3.0 liter bioreactors. The parameters included stirring speed, particle size, solids feed rate, and gas flow rate on biooxidation performance. The effects on biooxidation performance were measured by electrochemical monitoring of the ferrous oxidation rates and oxygen consumption. A design of experiments was used to optimize the processing parameters. Optimization was based on acid production performance measured by a combination of product flow rate and pH. The optimimum parameter values were used to initiate the bioreactors at the pilot scale facilities starting with a 100-gallon unit and staged to 250 gallon tanks and 3000 gallon reactors.

### *3.5.1 Bioreactor Background and Parametric Effects*

In this project, key operating parameters that impact key performance indicators for biooxidation, which are acid production and biooxidation rate, were evaluated. The key parameters that were evaluated are residence time, solids concentration, air flow rate, stirring speed, temperature, and nutrient availability. The evaluations and assessments were performed using bioreactors such as shown in Figure 3.5.1.

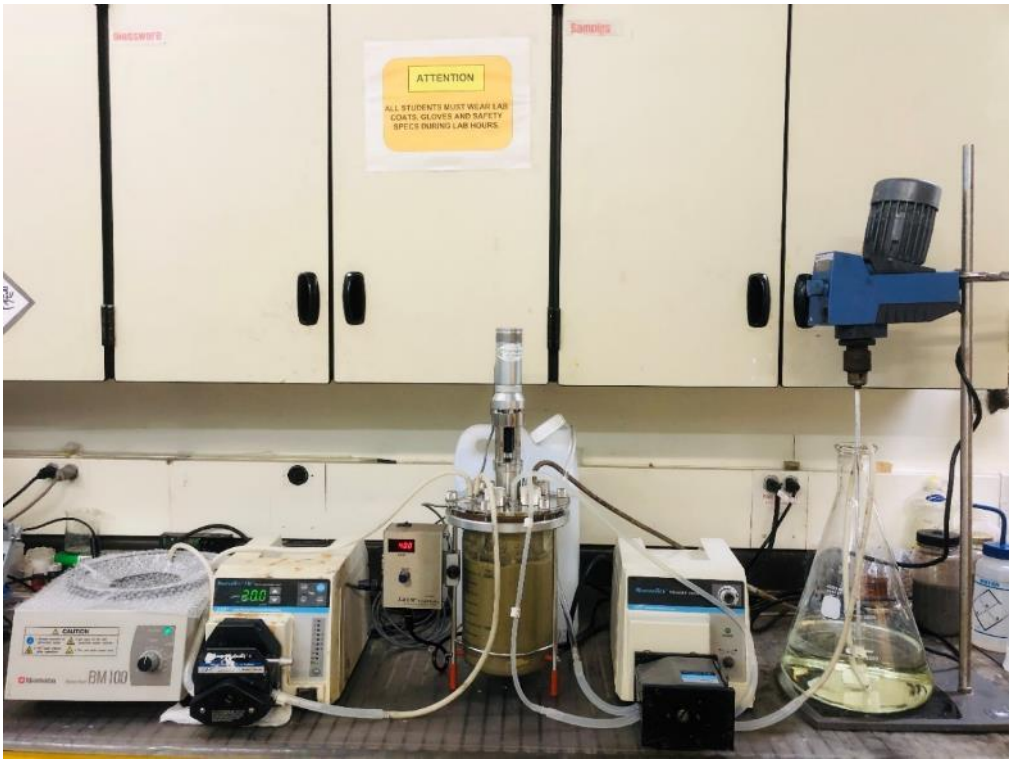



Figure 3.5.1. View of bioreactor (center) and related support equipment and tubing.

As general background information, the bacteria catalyze reactions that can produce acid from sulfide minerals. A simplified set of reactions that describes what happens generally is given as:

$2Fe^{2+} + 0.5O_2 + 2H^+ \leftrightarrow 2Fe^{3+} + H_2O$ (Biotic – ferrous ion oxidation)

$FeS_2 + 2Fe^{3+} \leftrightarrow 3Fe^{2+} + 2S_o$ (Abiotic - indirect bioleaching)

$2S_o + 3O_2 + 2H_2O \leftrightarrow 2SO_4^{2-} + 4H^+$ (Biotic - direct bioleaching)

Or

$15Fe^{2+} + 3.75O_2 + 15H^+ \leftrightarrow 15Fe^{3+} + 7.5H_2O$ (Biotic – ferrous ion oxidation)

$FeS_2 + 15Fe^{3+} + 8H_2O \leftrightarrow 15Fe^{2+} + 15H^+ + HSO_4^- + FeSO_4^+$ (Abiotic - indirect)

Correspondingly, the bacteria can cause direct or indirect leaching, and they can produce ferric ions from ferrous ions using oxygen and acid, then produce acid from pyrite. Testing with pure pyrite leaching in ferric sulfate media indicated that the ratio of $Fe^{3+}$ consumed per $FeS_2$ is 14.8. This indicated the primary chemical oxidation of pyrite results primarily in sulfate production rather than elemental sulfur. Chemical analyses confirmed negligible amounts of elemental sulfur are produced on particles. This also indicates that the demand for ferric ions in pyrite oxidation is high. A schematic diagram of biooxidation of pyrite is shown in Figure 3.5.2.

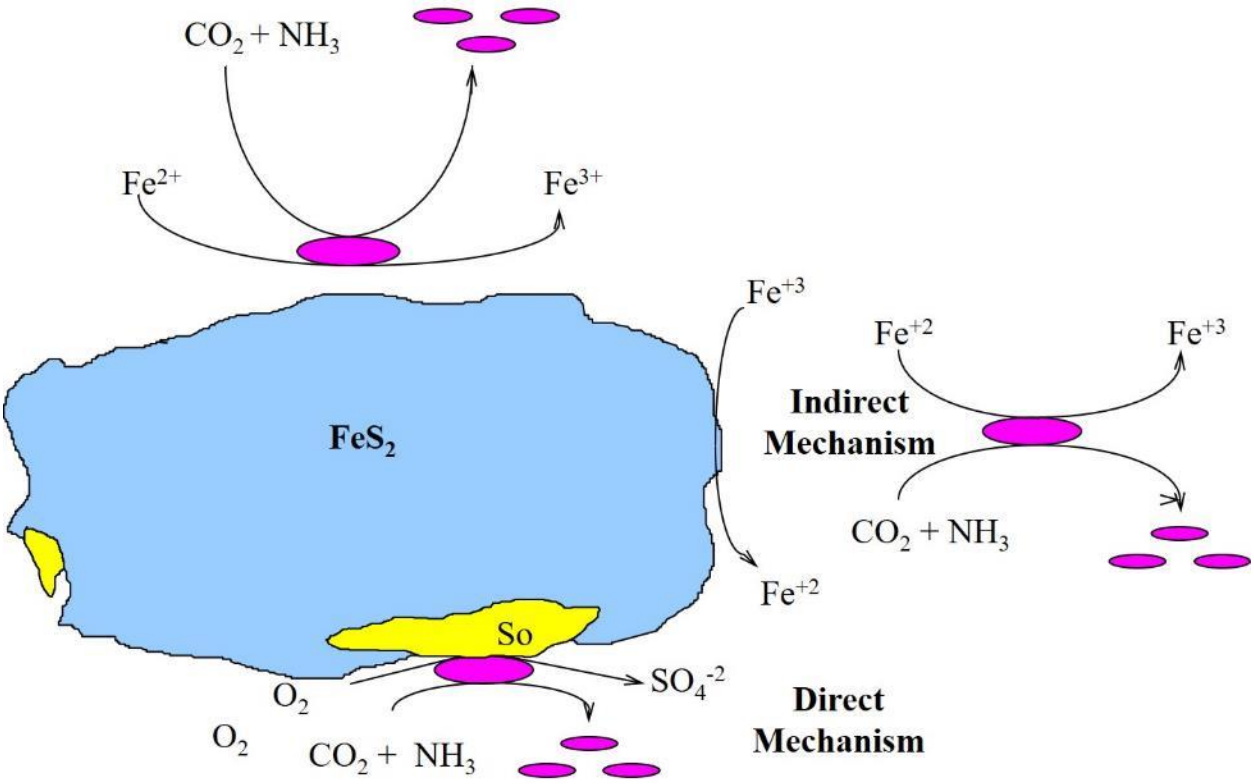



Figure 3.5.2: Schematic diagram of biooxidation of pyrite (after M. L. Free, Hydrometallurgy, 2022)

The rate of biooxidation was found to be as high as 4 grams of ferrous ions oxidized per liter per hour based on testing with additions of ferrous sulfate, followed by corresponding electrochemical potential measurements and conversion to ferrous ion oxidation rate, and example of which is presented in Figure 3.5.3.

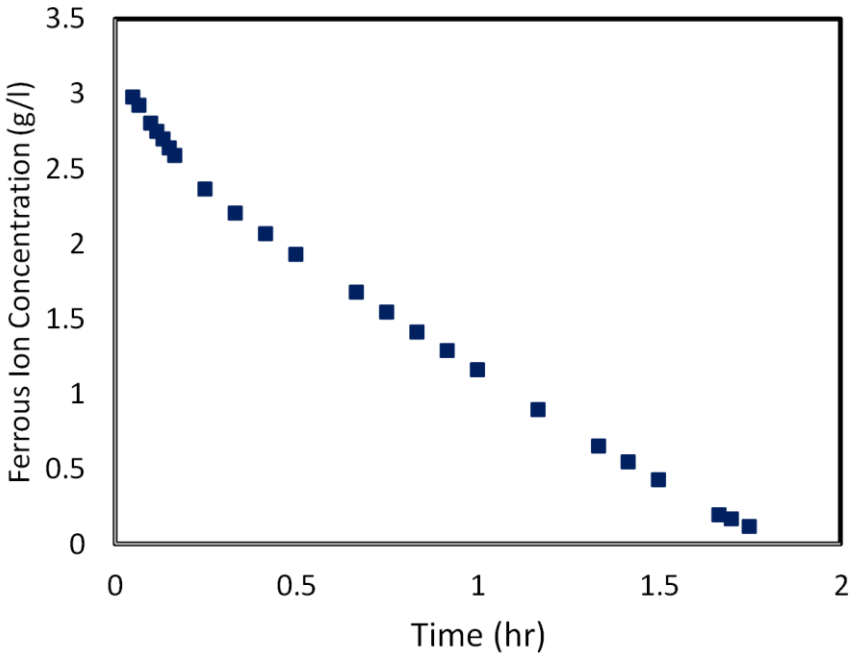



Figure 3.5.3: Comparison of ferrous ion concentration versus time for typical biooxidation testing in which ferrous ions were added to a solution and subsequently oxidized by bacteria.

The effects of changing residence time, solids concentration, air flow, stirring speed, temperature, and nutrient availability on biooxidation performance were assessed. Figure 3.5.4 shows the effect of solids concentration on biooxidation rate and Figure 3.5.5 shows the related acid production rate. These figures show that solids concentration had a significant influence on performance. Although 9 wt % solids had a slight improvement in acid production rate, there was also a decrease in pyrite utilization efficiency and in the biooxidation rate. Previous calculations also suggested that 6 wt % pyrite concentrate in the feed was an appropriate level for the solids concentration that matched the ability of the bacteria to utilize the available pyrite in a typical 4-day residence time.

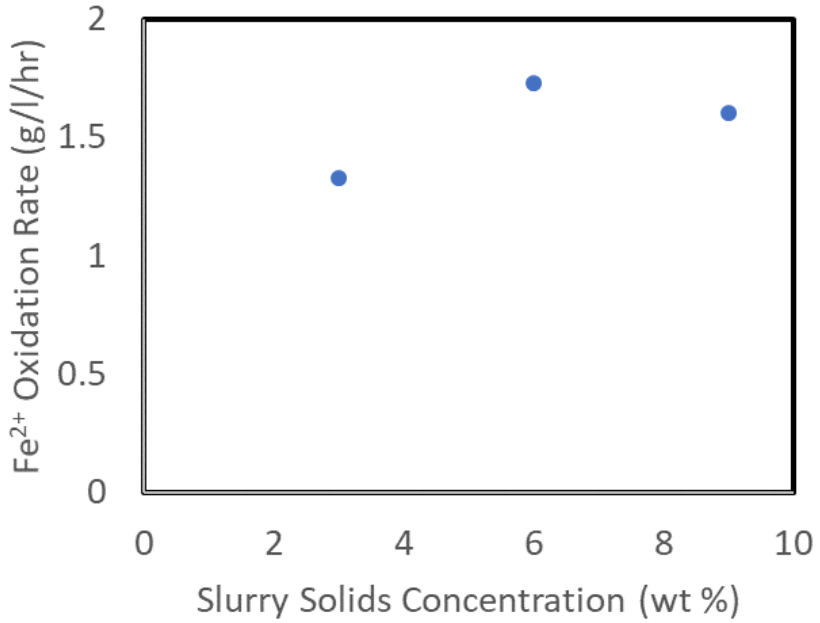



Figure 3.5.4. Comparison of slurry solids concentration on ferrous ion oxidation rate. The baseline conditions for this comparison are 500 RPM stirring speed, 35ºC, 6 wt % solids, 1.0 l/m air flow rate, 9 K medium without iron, and 6-day residence time.

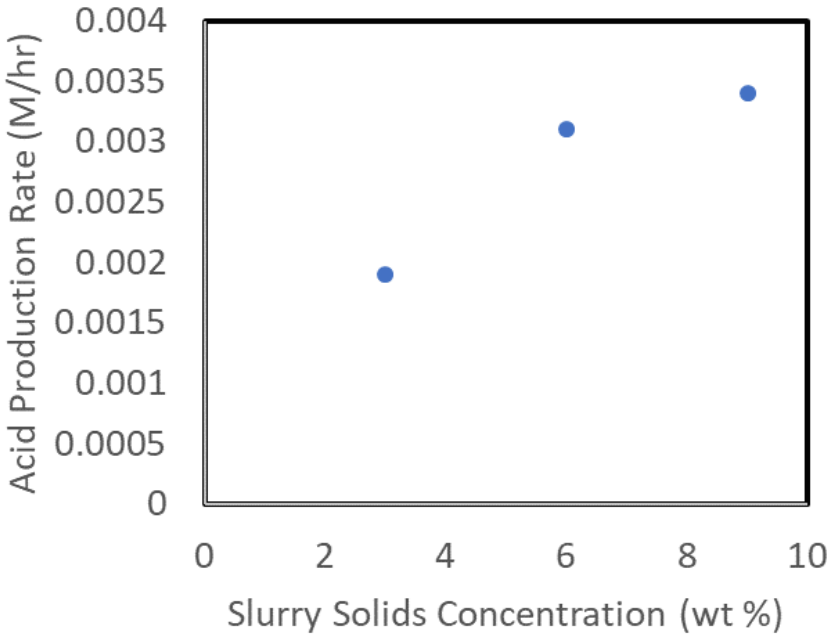



Figure 3.5.5. Comparison of slurry solids concentration on acid production rate. The baseline conditions for this comparison are 500 RPM stirring speed, 35ºC, 6 wt % solids, 1.0 l/m air flow rate, 9 K medium without iron, and 6-day residence time.

The effects of air flow rate on performance are shown in Figure 3.5.6. Figure 3.5.7 shows the related biooxidation rate and acid production rate performance levels, respectively. These result showed that a flow rate of 1 l/minute was best for the associate rotational speed of 500 rpm.

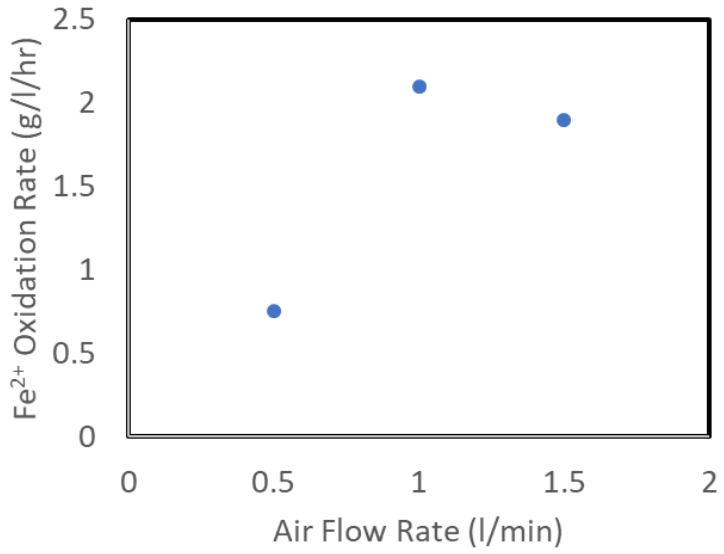



Figure 3.5.6. Comparison of air flow rate and ferrous ion oxidation rate. The baseline conditions for this comparison are 500 RPM stirring speed, 35°C, 6 wt % solids, 1.0 l/m air flow rate, 9 K medium without iron, and 6-day residence time.

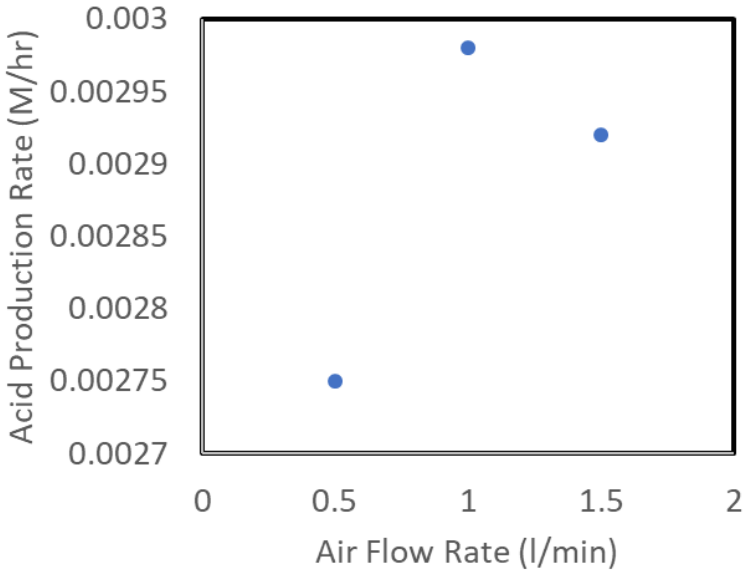



Figure 3.5.7. Comparison of air flow rate and acid production rate. The baseline conditions for this comparison are 500 RPM stirring speed, 35°C, 6 wt % solids, 1.0 l/m air flow rate, 9 K medium without iron, and 6-day residence time.

The effects of changing nutrient concentrations are illustrated in Figure 3.5.8. Note that the general nutrient formulation for 9 K medium, which was the general nutrient addition standard.

9 K Medium (without ferrous sulfate heptahydrate)

$(NH_4)_2SO_4$: 6.0 g/l

KCl: 0.2 g/l

$K_2HPO_4$ : 1 g/l

$MgSO_4.7H_2O$: 1 g/l

$Ca(NO_3)_2$ : 0.02 g/l

Note that because pyrite is $FeS_2$ there was more than enough iron in the feed solids, so the iron was missing from the 9K medium in this application.

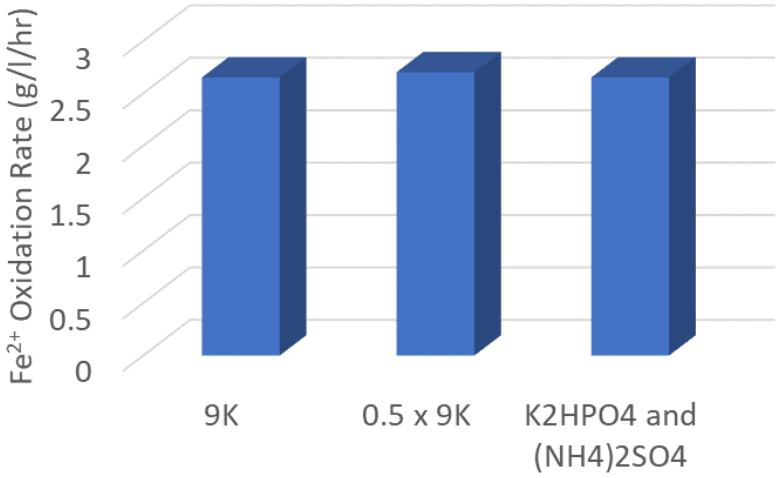



Figure 3.5.8. Comparison of nutrient additions on ferrous ion oxidation rate. The baseline conditions for this comparison are 500 RPM stirring speed, 35ºC, 6 wt % solids, 1.0 l/m air flow rate, 9 K medium without iron, and 6-day residence time.

The results of the nutrient evaluation indicated the nutrient addition could be reduced substantially to include only 0.8 grams of $K_2HPO_4$ per liter and 1.6 grams of $(NH_4)_2SO_4$ per liter. The reduction in the use of nutrients provided a significant cost savings relative to initial projections. The effect of temperature on biooxidation performance is shown in Figure 3.5.9. The data showed that increasing temperature was beneficial up to 35ºC. However, the data also showed that increasing the temperature to 40ºC was very detrimental to performance. Thus, temperature was a very critical parameter to control well.

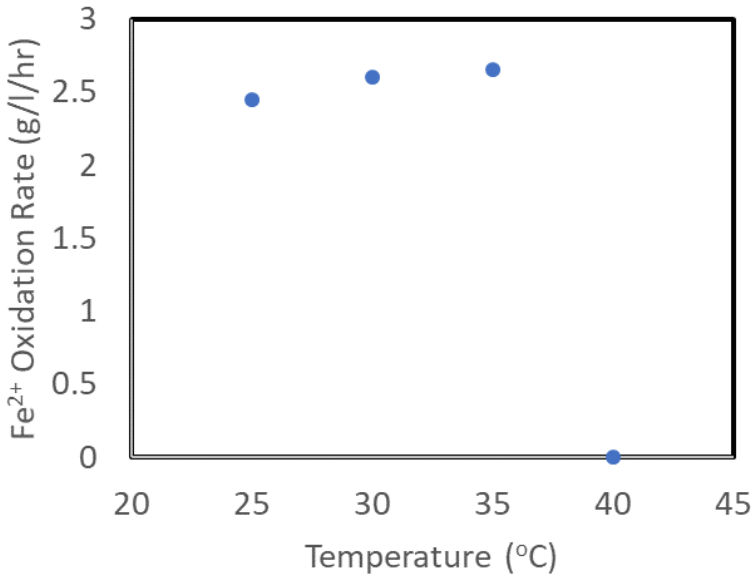



Figure 3.5.9. Effect of temperature on ferrous ion production rate.

The effects of residence time on biooxidation performance are presented in Figure 3.5.10 and Figure 3.5.11. These data show that a residence time of 4 days resulted in the best performance. The lower performance for 2.5 days of residence was attributed to the fast rate of effluent removal of bacteria relative to the time required for the bacteria to reproduce. The short residence time may also have reduced the overall utilization of pyrite, thereby decreasing performance. The longer residence times above 4 days could result in depletion of resources as pyrite was completely dissolved at longer residence times, making the available feed in limited supply as residence times became long, thereby reducing performance.

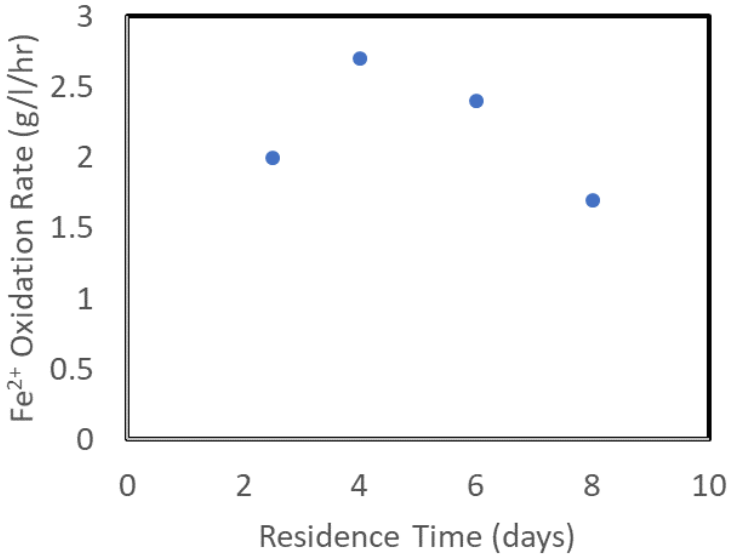



Figure 3.5.10. Comparison of residence time and ferrous ion oxidation rate.

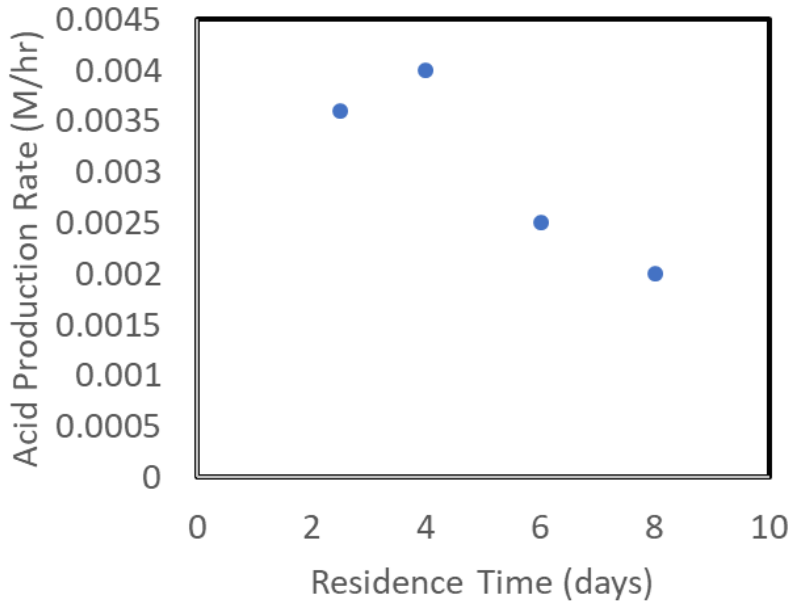



Figure 3.5.11. Comparison of residence time and acid production rate.

The effect of stirring speed on biooxidation performance is presented in Figure 3.5.12. These data indicate that performance increased as stirring speed increases.

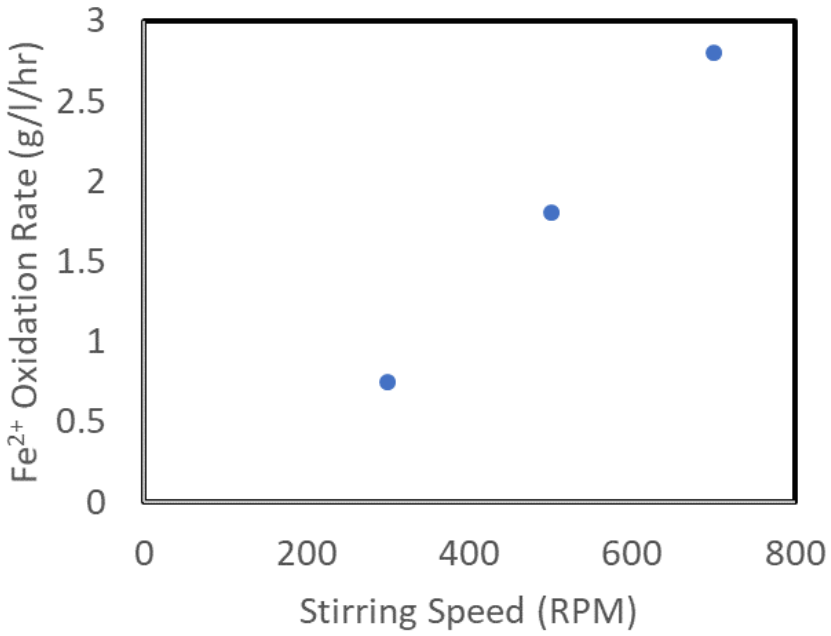



Figure 3.5.12. Initial comparison of stirring speed and ferrous ion oxidation rate.

The data from these tests indicated there were some clear optimum levels of performance for parameters such as air flow rate and residence time. The data also showed potential for increasing

performance by increasing stirring rate. Therefore, additional testing was performed to determine optimum levels near observed performance peaks presented in previous plots. The optimization testing focused on stirring speed and residence time. The corresponding results are presented in Figure 3.5.13 and Figure 3.5.14. Note that the optimum performance is achieved at a stirring speed of 650 RPM and 5 days of residence time.

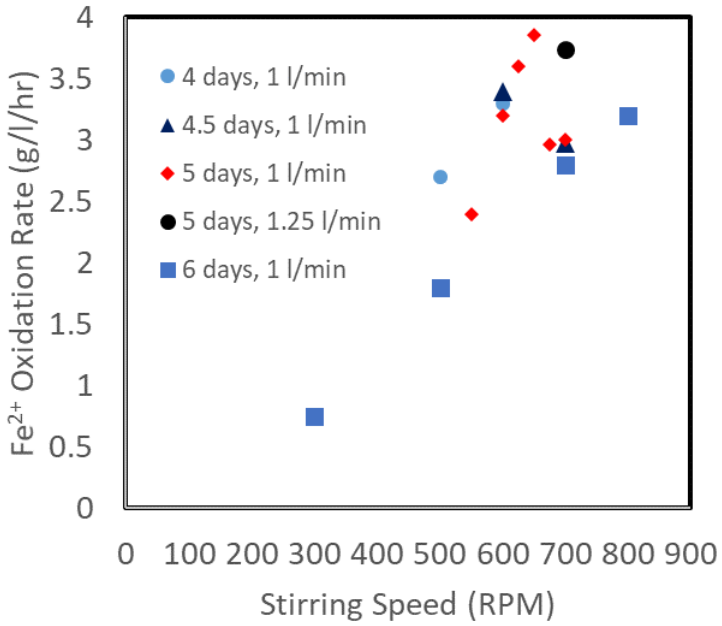



Figure 3.5.13. Detailed, more comprehensive comparison of stirring speed and ferrous ion oxidation rate.

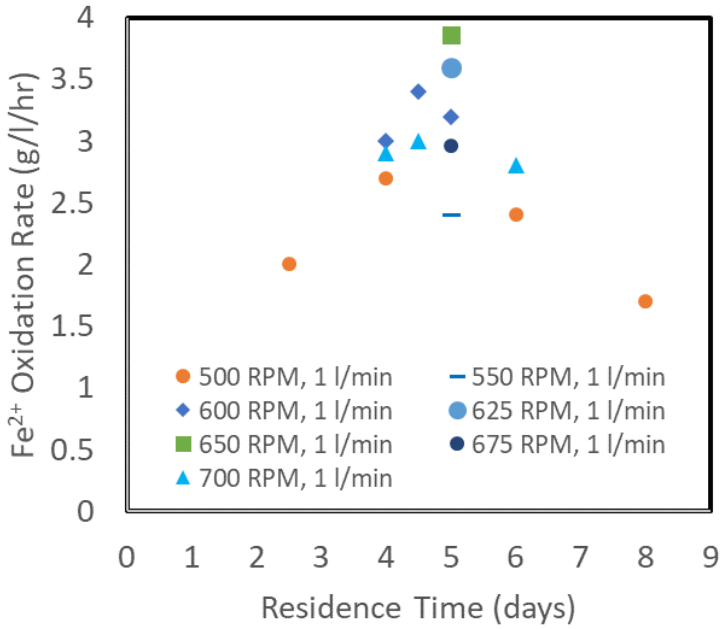



Figure 3.5.14. Detailed, more comprehensive comparison of residence time and ferrous ion oxidation rate.

Representative plots of pH data along with operator notes for one of the bioreactors during most of the last year of the project are presented in Figure 3.5.15 and Figure 3.5.16.

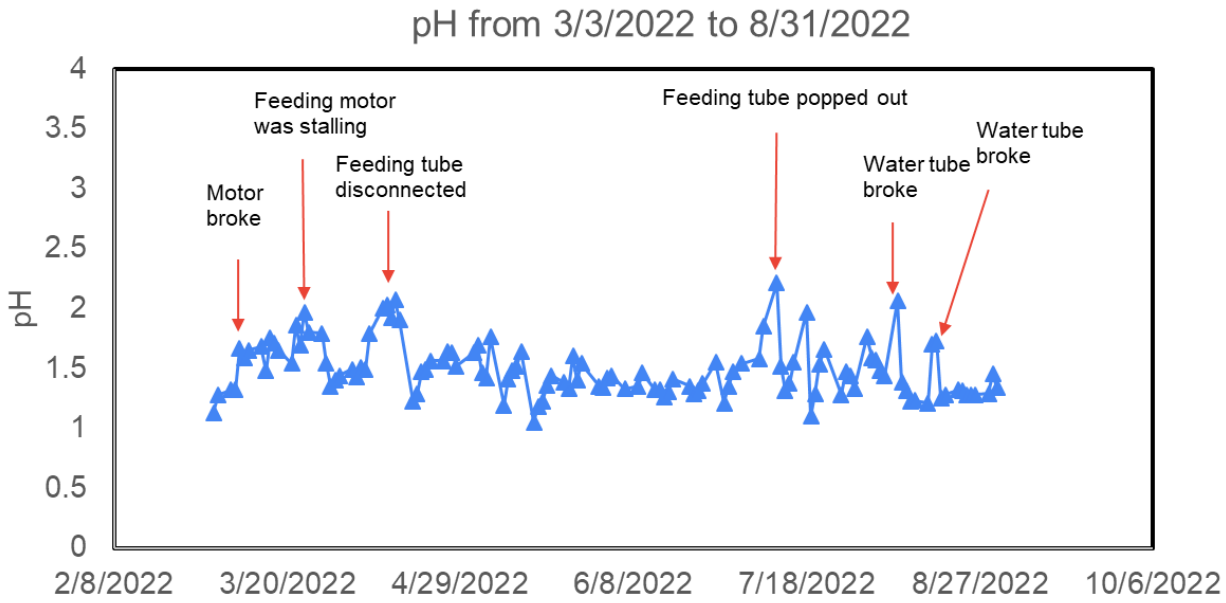



Figure 3.5.15. Comparison of pH data in the WBB 419 bioreactor from 3/3/2022 to 8/31/2022.

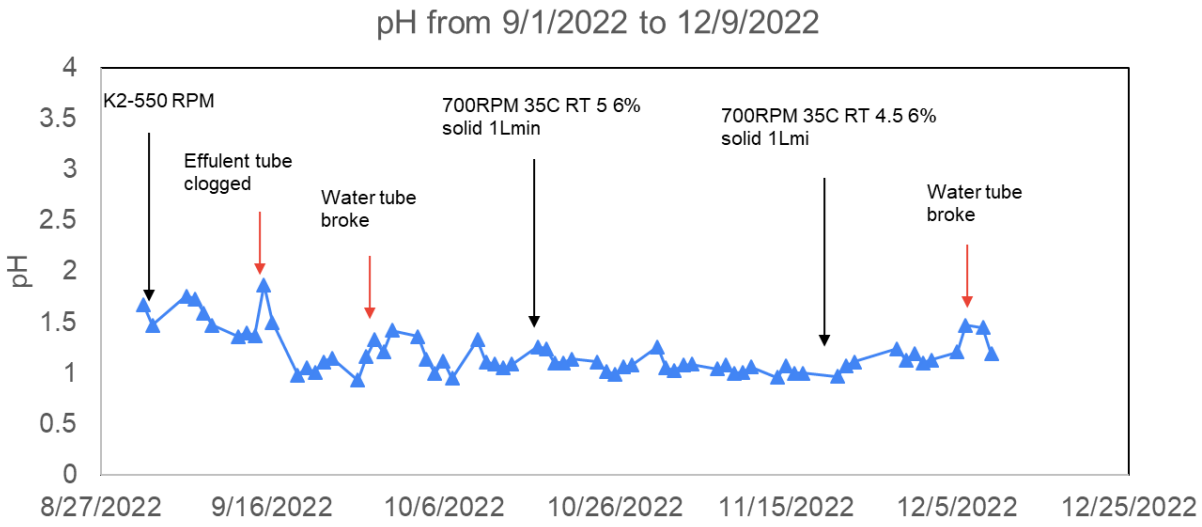



Figure 3.5.16. Comparison of pH data in the WBB 419 bioreactor from 9/1/2022 to 12/09/2022.

Representative plots of Eh data for one of the bioreactors during most of the last year of the project are presented in Figure 3.5.17 and Figure 3.5.18.

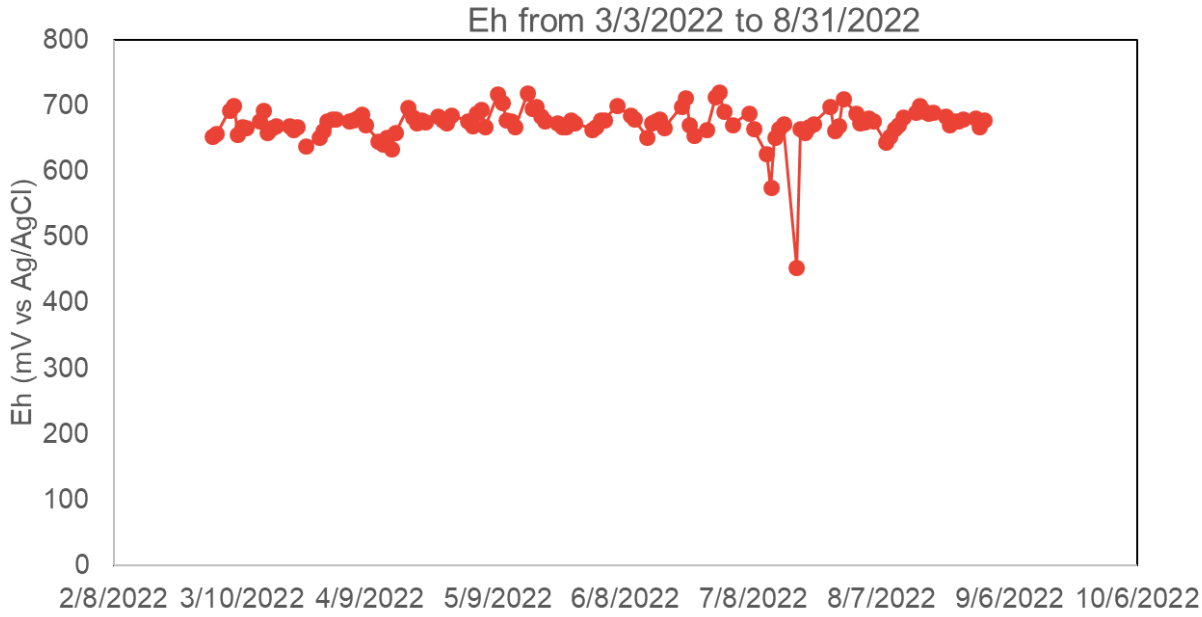



Figure 3.5.17. Comparison of Eh data in the WBB 419 bioreactor from 3/3/2022 to 8/31/2022.

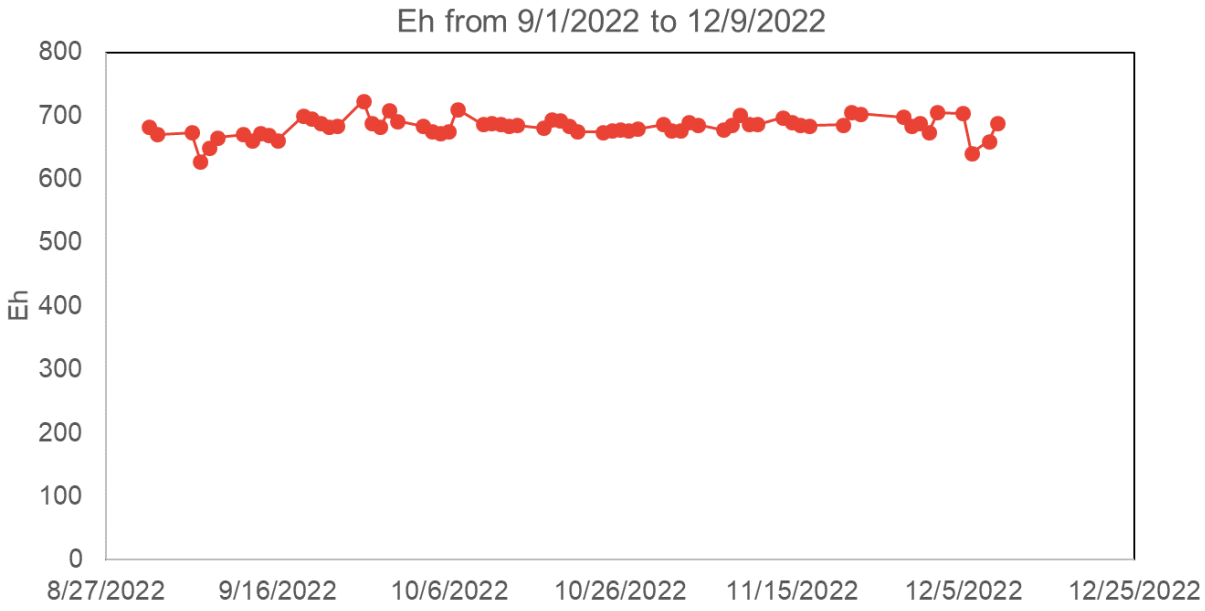



Figure 3.5.18. Comparison of Eh data in the WBB 419 bioreactor from 9/1/2022 to 12/09/2022.

Representative plots of ferrous ion oxidation data for one of the bioreactors during most of the last year of the project are presented in Figure 3.5.19 and Figure 3.5.20.

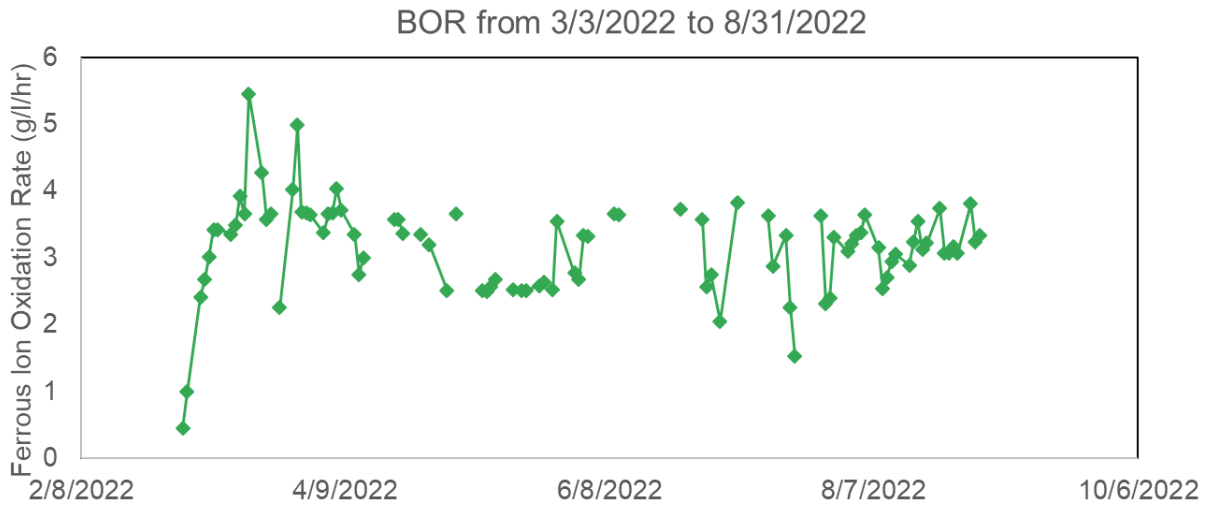



Figure 3.5.19. Comparison of ferrous ion oxidation rate (biooxidation rate or BOR) data in the WBB 419 bioreactor from 3/3/2022 to 8/31/2022.

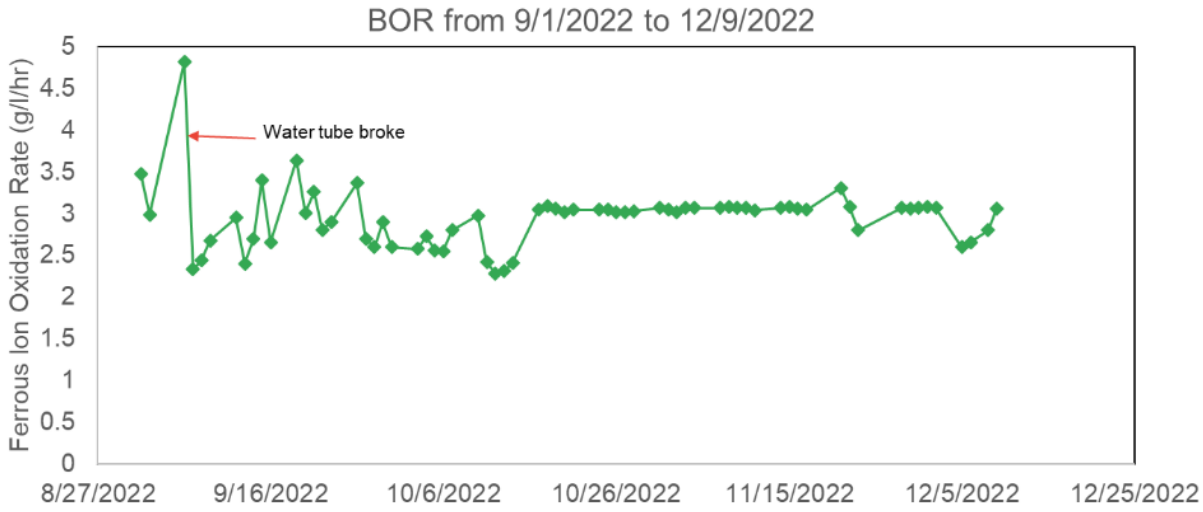



Figure 3.5.20. Comparison of ferrous ion oxidation rate (biooxidation rate or BOR) data in the WBB 419 bioreactor from 8/27/2022 to 12/9/2022.

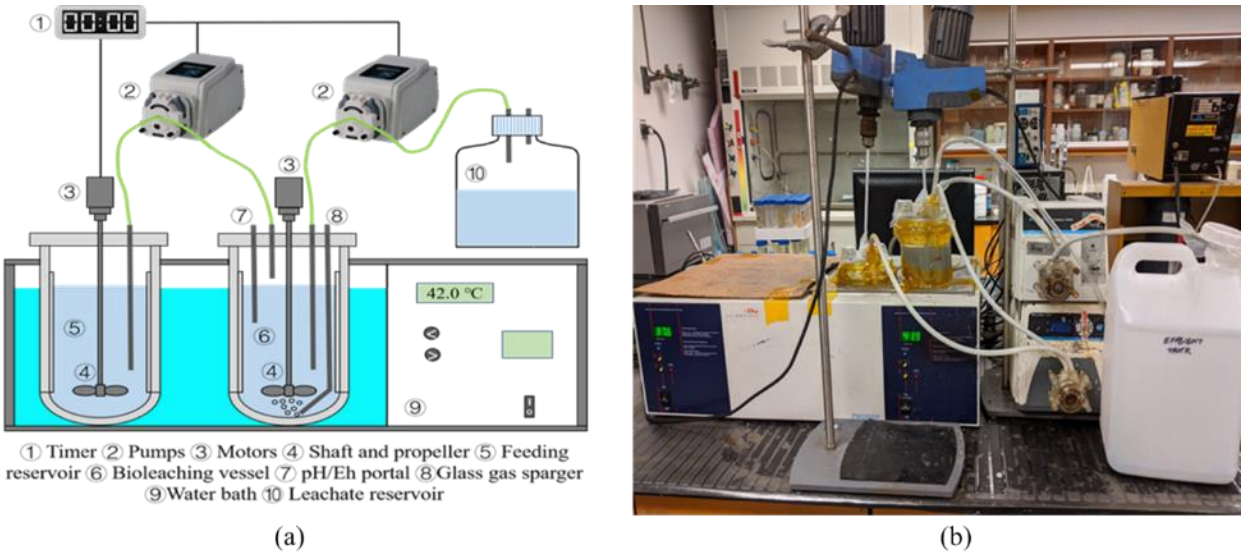
These bioreactor data plots illustrated the variations associated with parameter changes as well as laboratory related issues that impacted result while running these reactors 24 hours per day, seven days per week. The data also showed that steady-state conditions could be achieved for sustained periods of time, provided that there were no mechanical or laboratory facility related issues that impact the operation of the bioreactors.

### *3.5.2 Bioreactor Assembly and Automation*

**1.5 L Bioreactor:** In order to generate acid from pyrite using biooxidation method, initial tests were performed using primary experimental setup that have facility of air supply, temperature control, and provision for automated supply of nutrients (9k media). After validation, a more professional bioreactor setup was purchased and experiments were conducted. These tests were continued for some time but were stopped due to University restrictions caused by COVID-19. After a mandatory lockdown due to COVID-19 situation, the PIs and team successfully restarted our bioreactors on May 05, 2020, after establishing appropriate safety protocols and receiving permission from the University of Utah. The previously preserved bacterial cultures were used to provide the inoculum for bioleaching experiments in two different buildings in order to avoid discontinuity in case of future emergency shutdowns.

The team performed necessary procedures to restart the bioreactors, which included cleaning steps, installation of all parts, re-filling of new solutions, maintenance of temperature (inside the reactor ~35°C) , pH control ( ~1.4 using $H_2SO_4$), and providing the supply of 9K medium (See Figure 3.5.21(a)). The bioreactor was filled with 300ml of stored bioleaching solution containing bacterial culture along with 1 L 9K medium that was already prepared to make the total volume to 1.3 L in the bioleaching reactor. Overhead stirrer speed was set up at 680 rpm, and gas flow was set up at 0.5 L/min, which was changed to 0.2 L/min later. After that, when Eh reached to 600 mV, pyrite feeding was started. The pyrite feeding slurry was made with 5 g of pyrite concentrate (approximately 60% pyrite) and 100 ml of 9K medium and was added into the feeding vessel. At

that point, the automatic feeding process was started. A timer was adjusted to feed 100 ml feeding slurry in the reactor and to take out 100 ml leaching solution from the reactor per day. Eh and pH changes were also tracked. Figure 3.5.21(b) shows the digital image of the bioreactor after the restart.




① Timer ② Pumps ③ Motors ④ Shaft and propeller ⑤ Feeding reservoir ⑥ Bioleaching vessel ⑦ pH/Eh portal ⑧ Glass gas sparger ⑨ Water bath ⑩ Leachate reservoir

(a) (b)

Figure 3.5.21 (a) Schematic diagram of bioreactor system; (b) Digital image of bio-reactor

**3L Bioreactor:** A 3 L automatic bioreactor purchased from Chemglass Life Sciences has been assembled in the laboratory and setup after ongoing COVID-19 situation, as shown in Figure 3.5.22.



Figure 3.5.22. Chemglass life sciences 3 L automatic bioreactor assembly.

The pictures of the 3 L bioreactor after feeding 9K medium is shown in Figure 3.5.23, and other accessories, such as the circulating water heating, air injection, and the automatic feeding system, that were connected to the new reactor (see Figure 3.5.24). Similar bioreactor startup procedures were followed as explained previously. The bacteria population was increased gradually, and the Eh (690 mV, recorded on 6/28/2020) and pH vs. time pattern indicates the new 3L bioreactor was performing well and the bacteria in it were reproducing.




Figure 3.5.23. Pictures of the chemglass life science 3 L bioreactor.

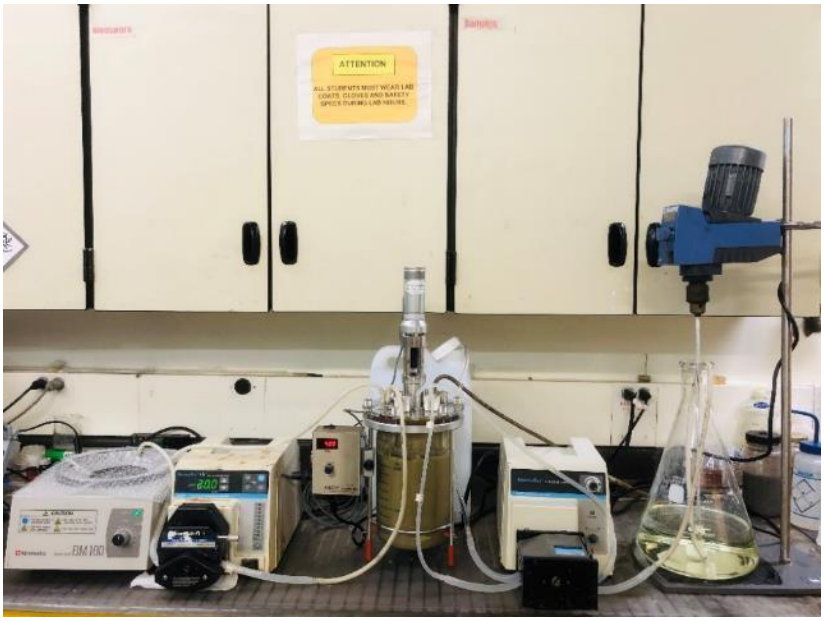



Figure 3.5.24. New bioreactor Set up.

### *3.5.3 Bacteria Adaption to the Bioleaching Solution with Pyrite Concentrate*

**pH, Eh, and Ferrous Oxidation Rate of the 1.5L Bioreactor:** On May 19, 2020, only 9K medium (with $FeSO_4$) was fed to the bioleaching vessel, and Eh and pH were tracked regularly. When Eh reached 600 mV and pH was 1.03 on May 27, 2020, then automatic feeding was started with 5 g of 60% pure pyrite with 100 ml 9K medium (without $FeSO_4$) per day and kept feeding this amount automatically until June 15, 2020. when the Eh reached 650 mV and pH was varying between 1.25 to1.30, then the feeding amount was increased to 10 g of pyrite concentrate with 200 ml of 9K medium per day. In this period, the Eh had its highest value, which was 671 mV. However, there were no significant change in pH. Thus, the automatic feeding amount was increased to 300 ml of 9K medium with 15 g of pyrite concentrate per day on June 19, 2020. However, a small decrease in Eh value was observed after feeding with this new composition. There were no remarkable changes in pH. Figure 3.5.25 show the results for Eh and pH values vs. time in this period of time.

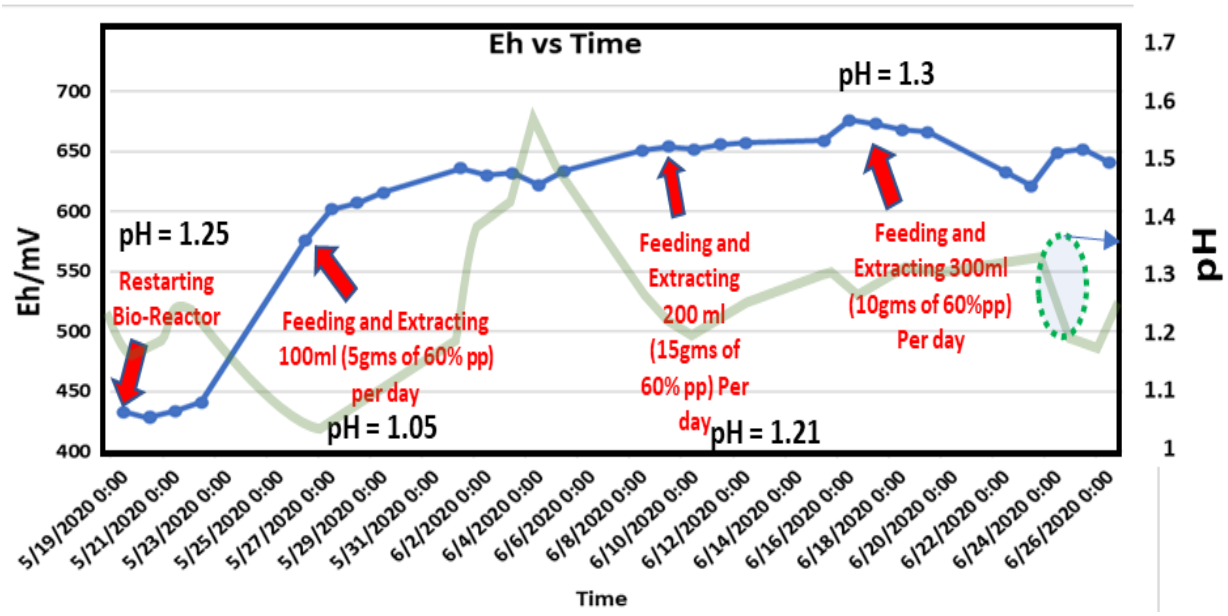



Figure 3.5.25. Changes in Eh and pH values vs. time for different amount of Feeding solution

The bacterial activity has been examined by performing ferrous oxidation tests. These tests were conducted on different days and utilized different amounts of $FeSO_4 \cdot 7H_2O$. Ferrous sulfate heptahydrate was added to the 1.5 L bacterial solution in the reactor while Eh and pH were tracked to see the changes. The Eh vs. time is measured and recorded for every test. By following the Eh data, the ferrous concentration vs. time and the ferrous ion consumption rate vs. time, the rate of bacterial activity can be dertermined. An overall summary (such as ferrous ion consumption rate for different Eh valuse and after addition of ferrous sulfate) is shown in Figure 3.5.26 In these tests the rotation speed was set at 680 RPM, and the gas flow rate of 0.2 L/min was maintained. The data in Figure 3b show that the rate of oxidation increased from 0.7 to 1.6 g of ferrous oxidized per liter of solution per hour over the course of growing the bacteria for about 20 days. The rate of oxidation prior to the COVID-19 lab closure was about 2.5-3.0 g of ferrous ion oxidation per liter per day, so the reactors should be matching the pre closure levels by the end of July.

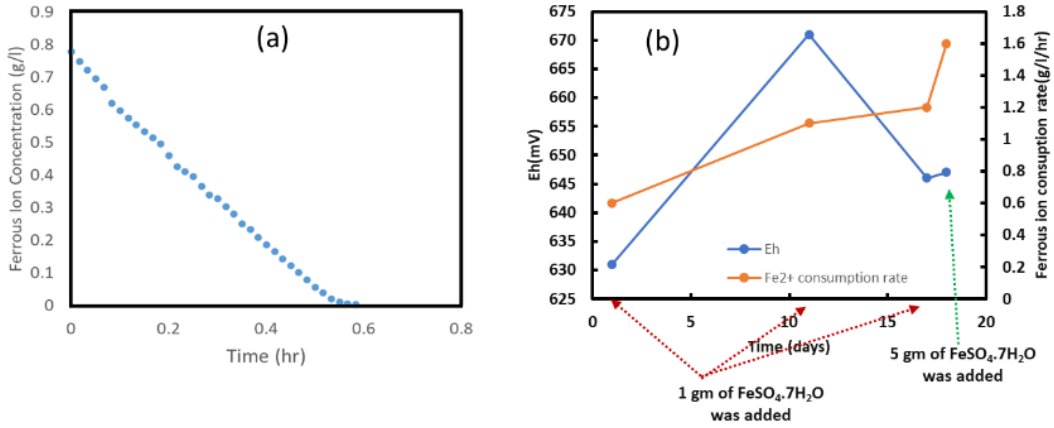



Figure 3.5.26. (a) Bio-oxidation rate assessment after addition of 5 g $FeSO_4 \cdot 7H_2O$ on 06/23/2020 at Eh 647 mV; (b) overall comparison of bio-oxidation rate for different days and at different Eh values.

From 7.1.2020 to 7.15.2020 the bioreactor was operated with one time feeding in a day with 100 ml of feeding slurry for each feeding. The solid percentage in the feeding composition of the feeding solution was 8% and 100 ml of 9K medium was used as well. On July 16, 2020 instead of using 100 ml 9k medium; only 50 ml of 9K medium and 50 ml of DI water was added with 8% of solids to see the stability of the bio-oxidation performance. Then the feeding rate was increased by raising feeding two times/day. On 8/13/2020 60% pure pyrite feeding (size: smaller than 100 mesh) has been started with the composition of pyrite-8g, 9k-50ml, DI water-50ml with two times of feeding in a day. Residence time - 7.5 days on 8.27.2020 air flow rate was increased to 0.35 L/min from 0.2 L/min after doing the central composite design analysis. On 9/16/2020 feeding was changed to three times/ day with a residence time of 5 days and it was continued for the next eight days and no significant changes were observed during this time. From 9.25.2020, four times feeding in a day has been started. The residence time was ~ 3.75 days. The gas flow rate was increased to 0.3 L/min from 0.2 L/min on 8/27/2020 and continued until this report was made. The rotation speed was 680 RPM. The Eh and pH data during this period of time is shown in Figure 3.5.27. From this figure, it is clear that the bioreactor was running at a relatively stabilized condition with a pH of ~ 1.1 to 1.3 and Eh was ~ 630 mV to 660 mV.

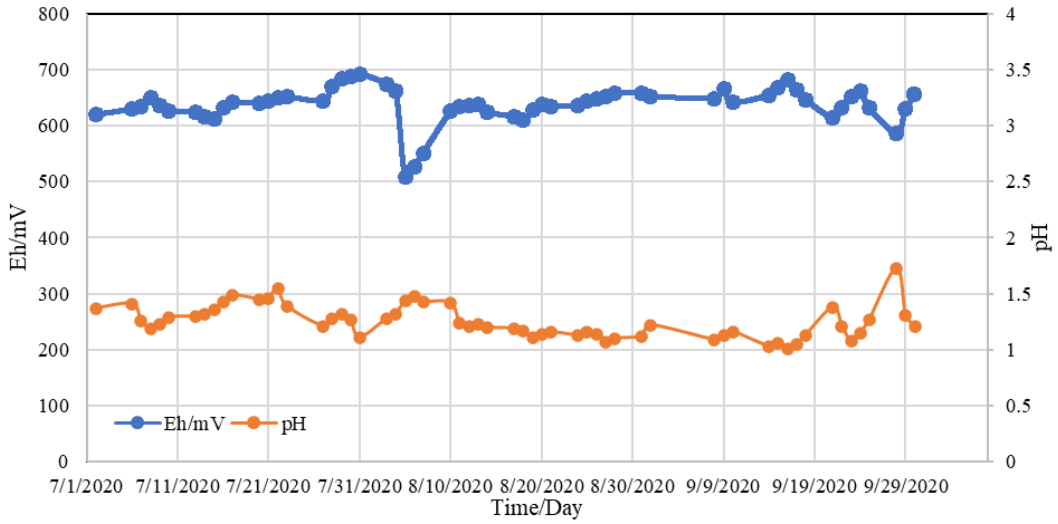



Figure 3.5.27. Eh and pH data of the 1.5 L bioreactor from 7/1/2020 to 9/30/2020.

By using a ferrous oxidation test bacterial activity has been evaluated by putting 1 g of $FeSO_4 \cdot 7H_2O$ into the 1.5 L bacterial solution in the reactor while Eh and pH measured without adding any feeding solution before starting of oxidation test. This test was conducted at different times of the bioleaching process by the same amount of $FeSO_4 \cdot 7H_2O$ to see the changes in bacterial activity. During this oxidation test, the same operating parameters were maintained. The propeller rotation speed 680 RPM, gas flow rate 0.3 L/min The Eh vs. time was measured and recorded for every test. By following the Eh data, the initial Eh vs. time and the ferrous ion consumption rate vs. time inclinations are portrayed in Figure 3.5.28. As can be seen, the initial Eh values were approximately the same and the ferrous oxidation rate has been increased a lot compared to its initial state.

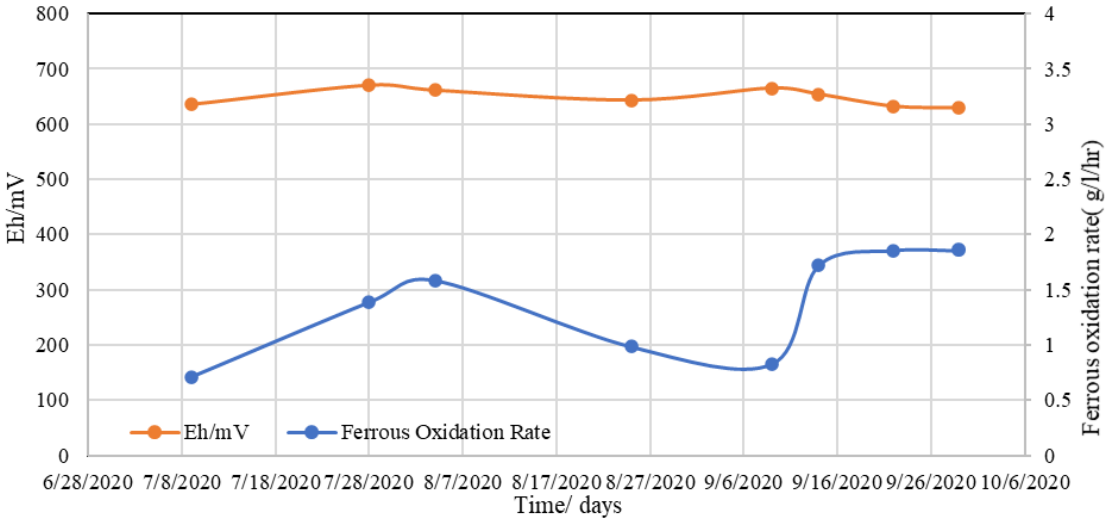



Figure 3.5.28 Changes in Ferrous oxidation rate over the time.

The 1.5 L Bioreactor was operated with a substantial percentage in the feeding material of 8%. The feeding rate was four times/ day with 100 ml of feeding slurry to the Bioreactor and 100 ml

effluent slurry out of the reactor per feeding from October 1 to October 22. The gas flow rate was 0.3 L/min, and the rotation speed of the propeller was 680 RPM. This period with the residence time of 3.75, Eh varied between 621mV to 655mV, and pH stayed in a range of 1.21 to 1.33. On October 23, magnetite-free and screened less than 100mesh pyrite powder was fed into the 1.5L bioreactor with the same feed percentage. Same operating parameters were maintained, and feeding was two times/ day. During the feeding of magnetite-free pyrite, Eh and pH were not stable at all. Information can be found in Figure 3.5.29. Later magnetite-free pyrite feeding was stopped on November 2, and regular pyrite feeding was started again by maintaining the same compositions and parameters and Continued the parameters until 9th November with two times and four times feeding per day. Eh, and pH was unstable During this period because there was a crack was found in the bioreactor vessel, and all the solution was wasted after two days. From November 10, 2020, to November 22, 2020, the entire operation was paused due to the COVID 19 case in department. On November 23, Bioreactor was found empty, and it was restarted again. Bioreactor was restarted on November 23, 2020, by maintaining the following parameters. On December 4, when Bioreactor was found in a stable position. Pyrite feeding was started with two times/ day feeding with the same composition but with the new fine pyrite. From 12/11/2020 to 12/15/2020, feeding was stopped due to less Bacterial activity.

Later on, 15th December, frequent feeding was started. The feeding amount was then increased on December 17 and continued until this report was made. Below information's about frequent feeding is shown.

- Feeding composition- 8g fine pyrite
- 9K medium- 50ml
- DI water-50ml
- Frequent feeding started: 12/15/2020
- Residence time: 10 days
- Feeding 6 times a day
- In every feeding we provided 25ml feeding solution
- Feeding solution contains
- 12.5ml of DI water (Total--75 ml in a day)
- 12.5ml of 9k medium (Total- 75 ml in a day)
- 2g of fine pyrite (Total -12g in a day)
- Residence time- 10 days
- Feeding amount increased: 12/17/2020
- 50ml of feeding solution with 8% of solids
- 6 times a day
- Residence time- 5 days

During this time, Eh stayed in a range of 603mV to 612 mv. pH was increasing steadily and when it reached to 1.49 then pH was adjusted by $H_2SO_4$ and continued. Then pH was found pretty stable. All the information's about Eh and pH change from October 2020 to December 2020 has been showed in Figure 3.5.29.

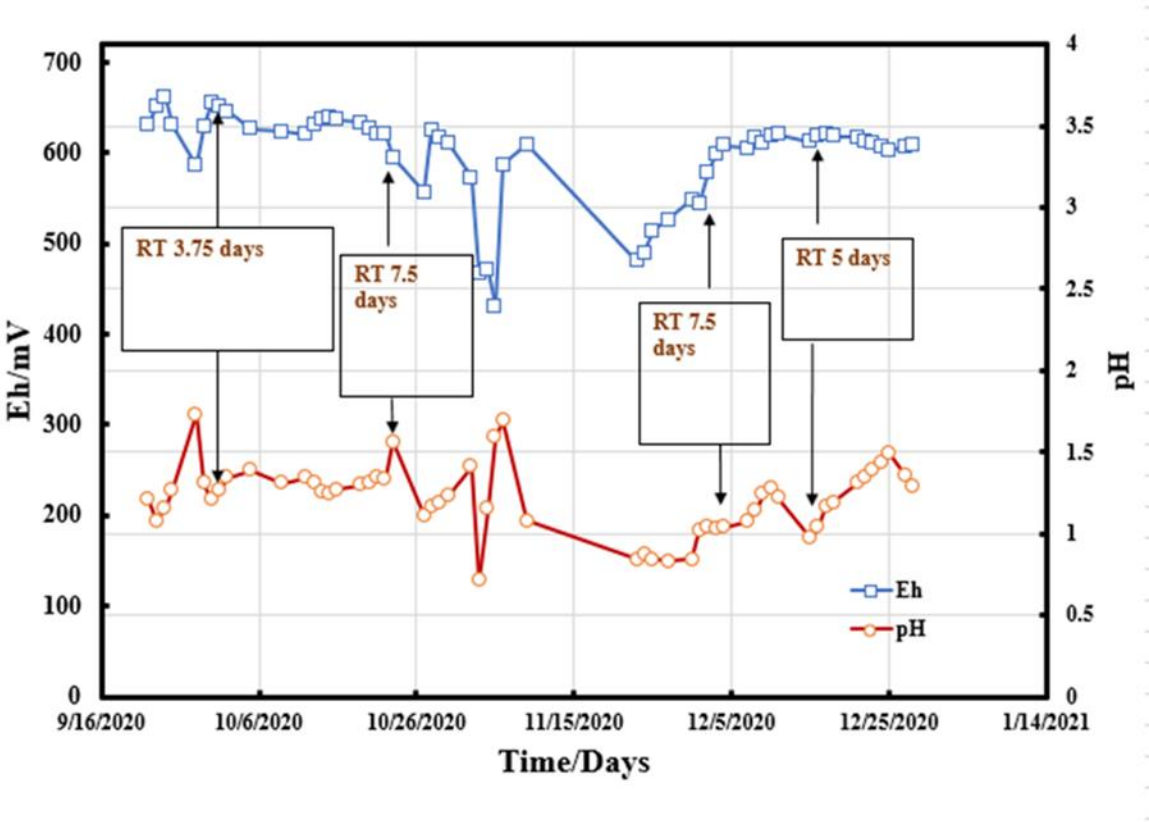



Figure 3.5.29 Eh and pH data of 1.5L Bio Reactor

The bioactivity of the bacteria was monitored regularly with the ferrous oxidation test. The initial Eh before the ferrous oxidation test and the ferrous oxidation rate vs. time are shown in Figure 3.5.30. During 60% pure pyrite feeding Bio oxidation rate varied in between 1.789 g/l/hr to 2.008 g/l/hr . After starting magnetite free screened leess than 100 mesh particles biooxidation decreased to 1.153. At the starting of using the Fine pyrite B.O.R was found 0.441 g/l/hr and later it was increased to 2.04g/l/hr and after starting the frequent feeding; B.O.R. is in a range of 1.306 g/l/hr to 1.905 g/l/hr. During all these experiments only 1g $FeSO_4.7H_2O$ has beed used to conduct the assay. The operating parameters were kept unchanged.

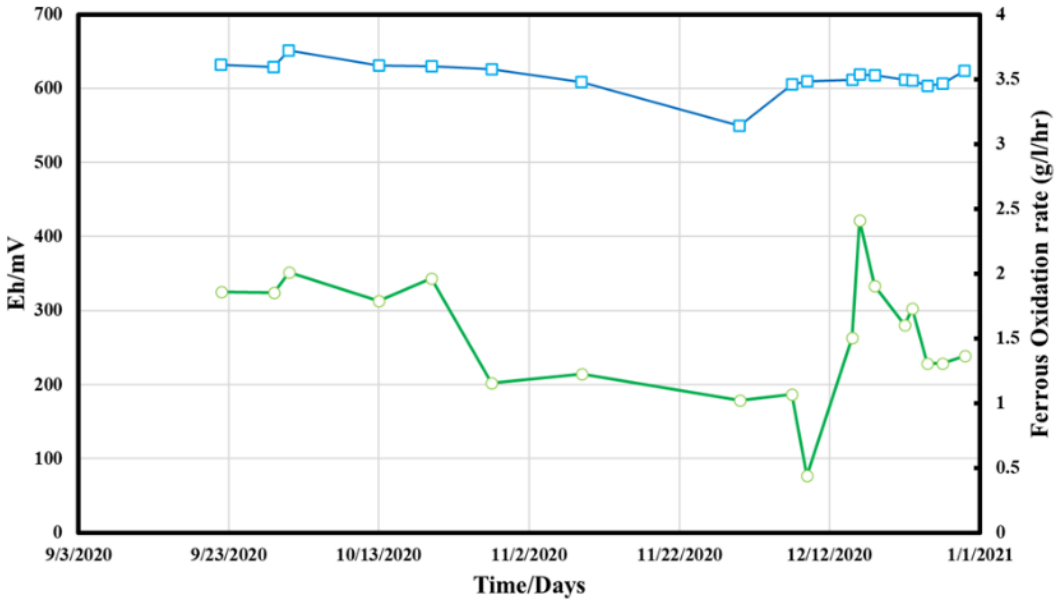



Figure 3.5.30 Changes in the bio-oxidation rate vs time

**pH, Eh, and ferrous oxidation rate of the 3 L bioreactor:** The Eh and pH changes vs. time in the begining of the test is shown in Figure 3.5.31. Ferrous oxidation tests were carried out on a regular base to determine the bioactivity of the bacteria. In each test, 2 g of $FeSO_4 \cdot 7H_2O$ were added to the 3 L bioreactor with 2.2-2.4 L of bioleaching solution. The Eh was tracked after adding ferrous sulfate. From the Eh change, the ferrous ion consumption rate can be estimated, which is in a positive correlation with the population and bioactivity of the bacteria. The measured ferrous ion consumption rate as well as the initial Eh when the test was conducted are shown in Figure 3.5.32. It shows the ferrous ion consumption rate is increasing gradually and indicate the bacteria in the new bioreactor are also in a good state and growing.

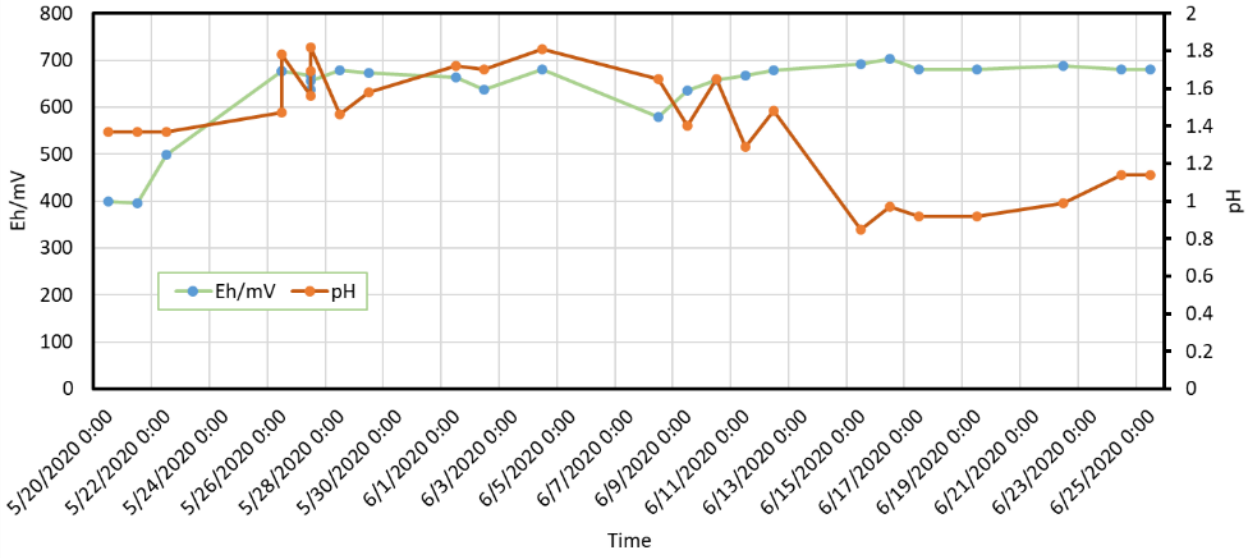



Figure 3.5.31. Changes in Eh and pH values vs. time in the new 3L bioreactor.

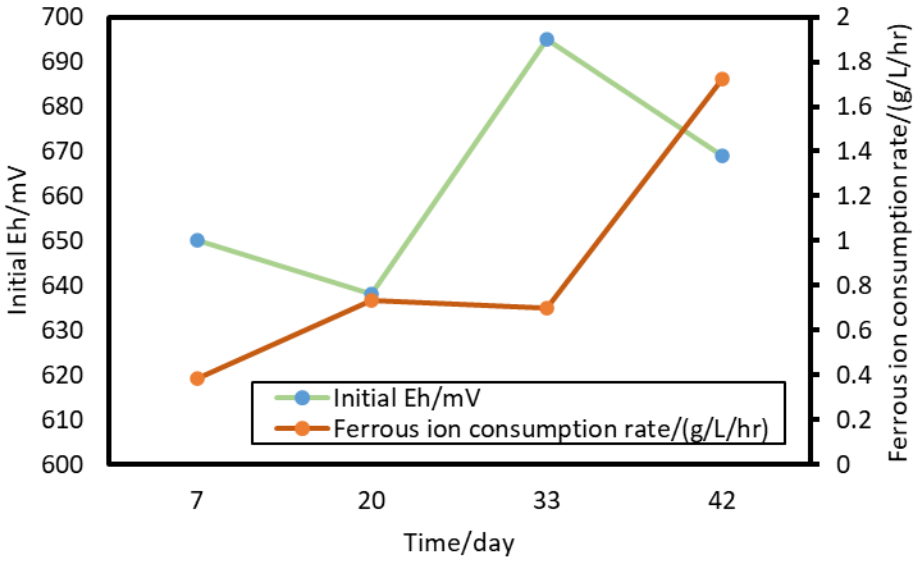



Figure 3.5.32. Overall comparison of bio-oxidation rate for different days and at different Eh values in the new 3L bioreactor.

The bioreactor was operated with a set of previously optimized operating parameters for a period of two weeks to check the stability of the biooxidation performance. The solid percentage in the feeding material was 8% and the feeding rate is twice a day with 100 ml of feeding slurry to the bioreactor and 100 ml effluent slurry out of the reactor per feeding. The gas flow rate was 1.5 L/min and rotation speed of the propeller was 550 RPM. The Eh and pH data during this period of time is shown in Figure 3.5.33. From this figure, it is clear that the bioreactor was running at a relatively stabilized condition with the pH around 1.3 and Eh round 670 mV.

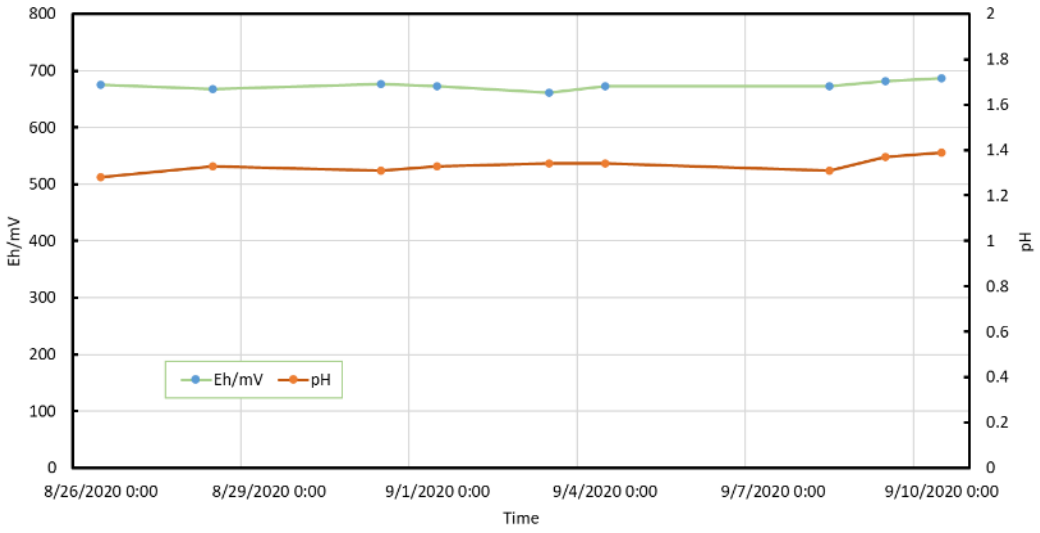



Figure 3.5.33. Eh and pH data of the 3 L bioreactor.

The bioactivity of the bacteria was measured with the ferrous oxidation test. The initial Eh before the measurement and the average ferrous oxidation rate are shown in Figure 3.5.34. As can be

seen, the initial Eh values are approximately the same and the ferrous oxidation rate has been increased from less than 1 g/L/hr in July to more than 2.5 g/L/hr in September, which indicates the bioactivity has been increased dramatically during this period of time.

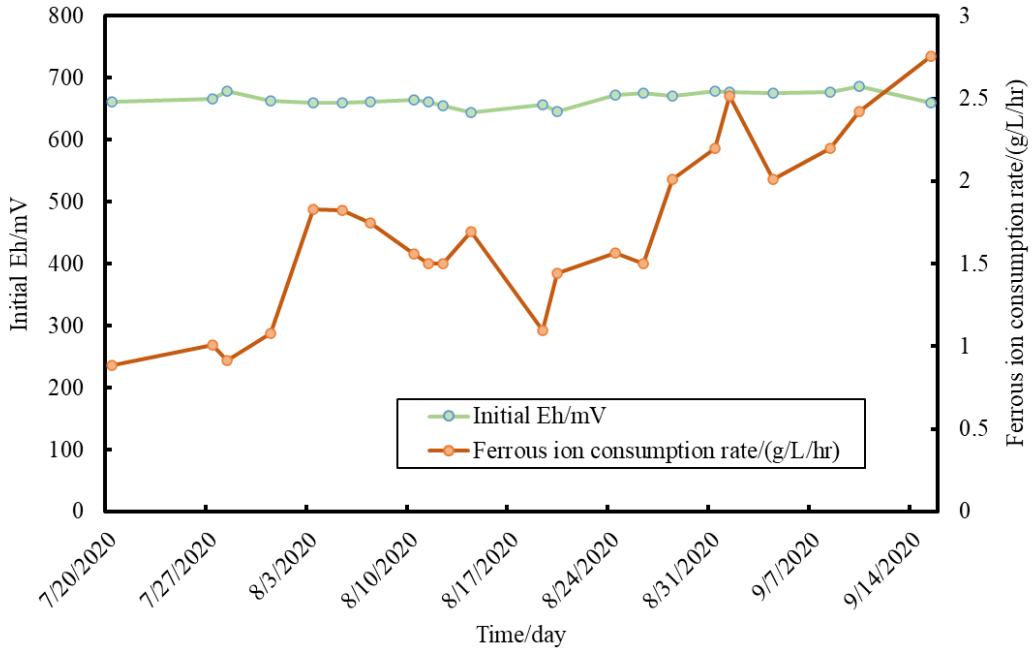



Figure 3.5.34. The initial Eh before the measurement and the average ferrous oxidation rate vs. time.

The pH and Eh changes over this quarter are shown in Figure 3.5.35. In early October, the effect of feeding rate was studied, detailed results are shown in Figure 3.5.36. To increase the bioreactor throughput and the study the effect of feeding rate (residence time) on the performance of the bioreactor. Different feeding rates were applied in the bioreactor. The solid percentage in the feeding material was 8% and the effect of feeding rate (residence time) on the pH and Eh of the bioleaching solution was studied and shown in Figure 1. In each feeding, 100 ml of feeding slurry was fed to the bioreactor and 100 ml effluent slurry was taken out of the reactor. When the feeding rate changes, a new steady state of pH and Eh can be built in several days. With 2 time per day feeding, the pH can be stabilized at around 1.25; with 3 times per day feeding, the pH is 1.35; with 4 times per day feeding, the pH is 1.45.

In November, magnetite-free pyrite concentrate was fed to the bioreactor in order to reduce the acid consumption by the oxide material. However, it turned out that the magnetite-free material contains more calcium oxide which broken the balance in the bioreactor. Then material was switched back to the mixed pyrite. Feeding rate was changed to 4 times a day after stabilized. pH can go up to 1.6 under 4 times a day feeding. After that, more frequent feeding of 8 times a day 50 ml each time was applied to provide more stable $CO_2$ supply to the bacteria.

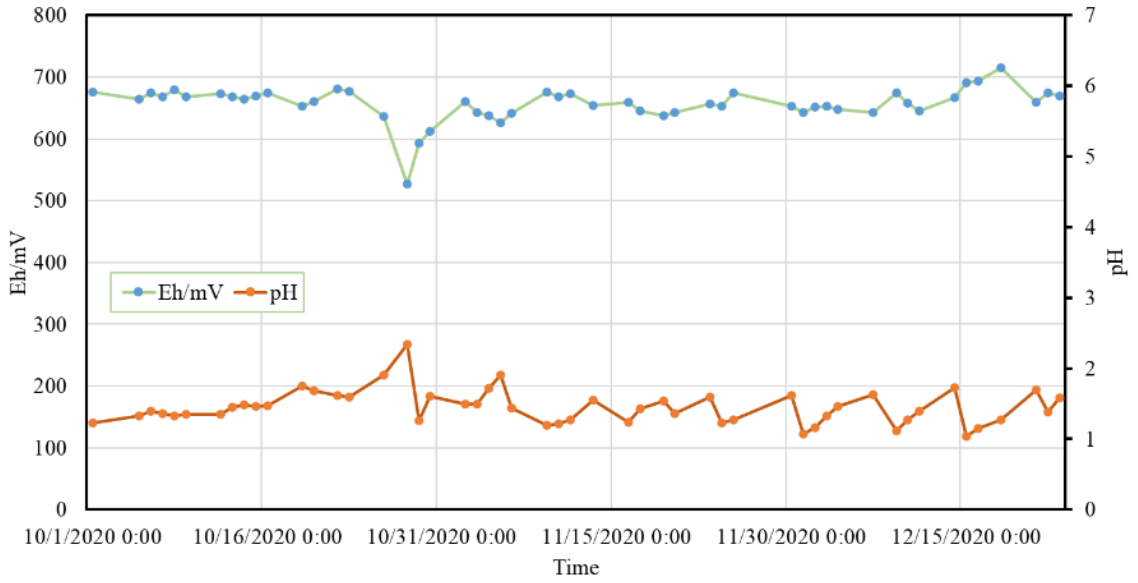



Figure 3.5.35: Eh and pH data of the 3L bioreactor

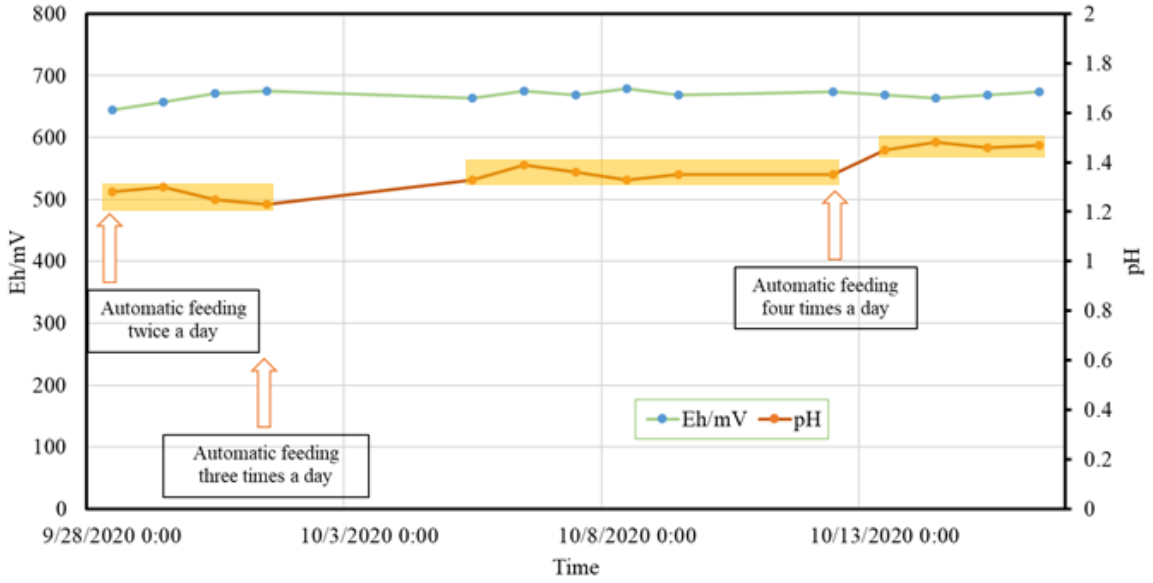



Figure 3.5.36 Eh and pH under different feeding rates (residence times)

The bioactivity of the bacteria was measured with the ferrous oxidation test. The initial Eh before the measurement and the average ferrous oxidation rate are shown in Figure 3.5.37. The initial Eh and the ferrous oxidation rate fluctuated with time due to the change of operating conditions for various studies. The highest ferrous oxidation rate can be higher than 2.5 g/L/hr but generally stay in the range of 1.5 g/L/hr to 2.5 g/L/hr. As can be seen, in early October, after the stabilizing period, the Eh reached a relatively steady state while the ferrous oxidation rate exhibited increasing trend and finally reached a plateau. Comparing the pH and the ferrous oxidation rate, the pH increase caused by the increasing feeding rate is not because of the reduction of the ferrous oxidation rate. The ferrous oxidation rate is still in a high level even the pH is increasing, which indicates the

increase of feeding rate did not wash out the bacteria in the bioreactor. When the feeding rate is switched to 4 times a day, the ferrous oxidation rate is increasing as pyrite is more easily available under 4 times a day feeding, which means the bacteria population is building up. To study the effect of 9K medium composition then reduce the cost of reagents, 9K medium with different compositions were used and ferrous oxidation rate was tracked during the test. When the 9K medium with only 1.5 g/L $(NH_4)_2SO_4$ was used, the average ferrous oxidation rate decreased to about 1.5 g/L/hr. Then 0.6 g/L $K_2HPO_4$ was added to the 9K medium, which increased the ferrous oxidation rate.

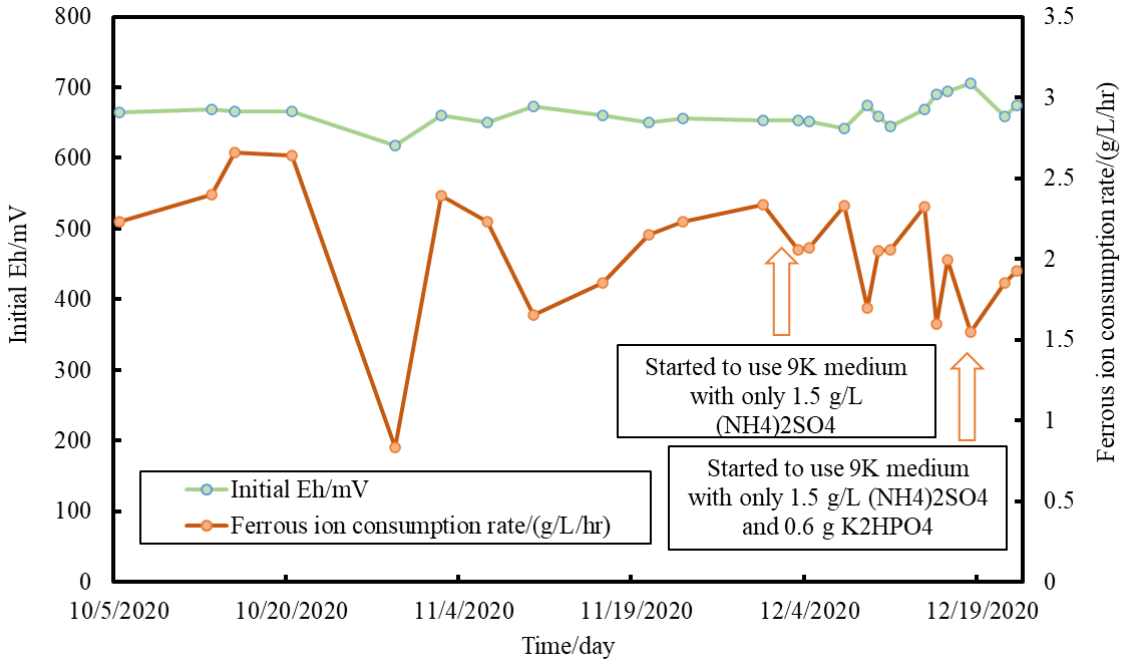



Figure 3.5.37. The initial Eh before the measurement and the average ferrous oxidation rate vs. time

### *3.5.4 Characterization of the Feeding and Effluent Solid*

**SEM-EDS Studies:** Pyrite ($FeS_2$) powder was used as received from University of Kentucky. Three different kinds of pyrite (Concentrated pyrite, 20% pure pyrite and 60 % pure pyrite) has been used. 20 % pure and 60% pure pyrite was received as slurry. But they were used as powder after drying in 145℃ in a low temperature furnace. Eh and pH of the leaching solution is being tracked regularly. All different kinds of pyrite were used. Figure 3.5.38 shows three different kinds of pyrite powder.




Figure 3.5.38. Pictures of different kinds of Pyrite powders.

Figure 3.5.39 shows the SEM images of all kinds coal waste pyrite samples with same magnification. Each pyrite sample had their own unique micromorphological structure as shown. SEM image of the 60% pure pyrite had more particles showing a sharp crystal structure in some particles as it has concentrated pyrite. 20% pure pyrite particles clearly showing different structure. Figure 3.5.40 shows the SEM images of concentrated pyrite with different magnification.

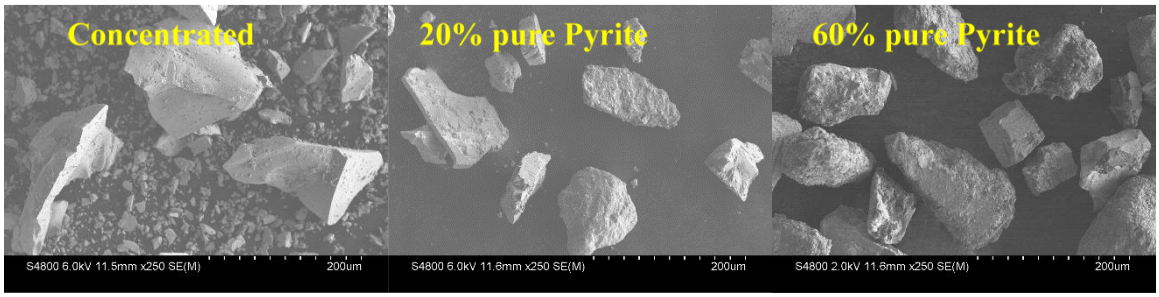



Figure 3.5.39. SEM images of pyrite samples at same magnifications.

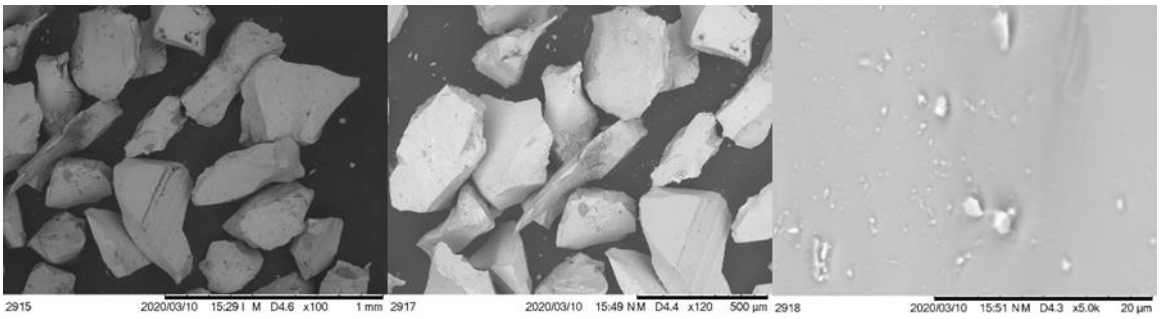



Figure 3.5.40. SEM images of concentrated pyrite with different magnification.

Figure 3.5.41 and Figure 3.5.42 show the resulting EDS mapping image and spectrum of 20% pure pyrite by SEM-EDS, respectively.

Table 3.5.1 shows the point analysis on the weight percentage of each element in Figure 3.5.42 in the 20% pure pyrite. Table 3.5.2 shows the concentration percentages of elements on the entire scanning surface. This data shows that Fe and S are present in the sample with a concentration of 10.1% and 5.7%, respectively.

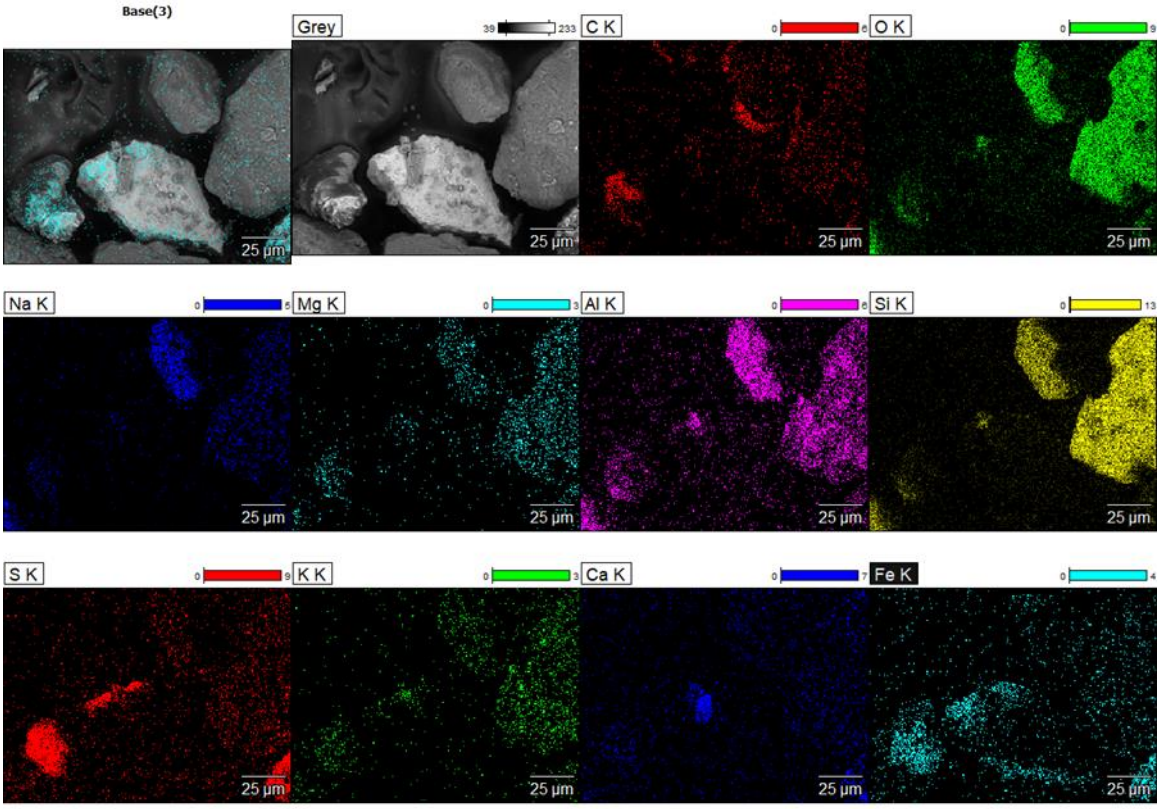



Figure 3.5.41. EDS mapping image of the 20% pure pyrite.

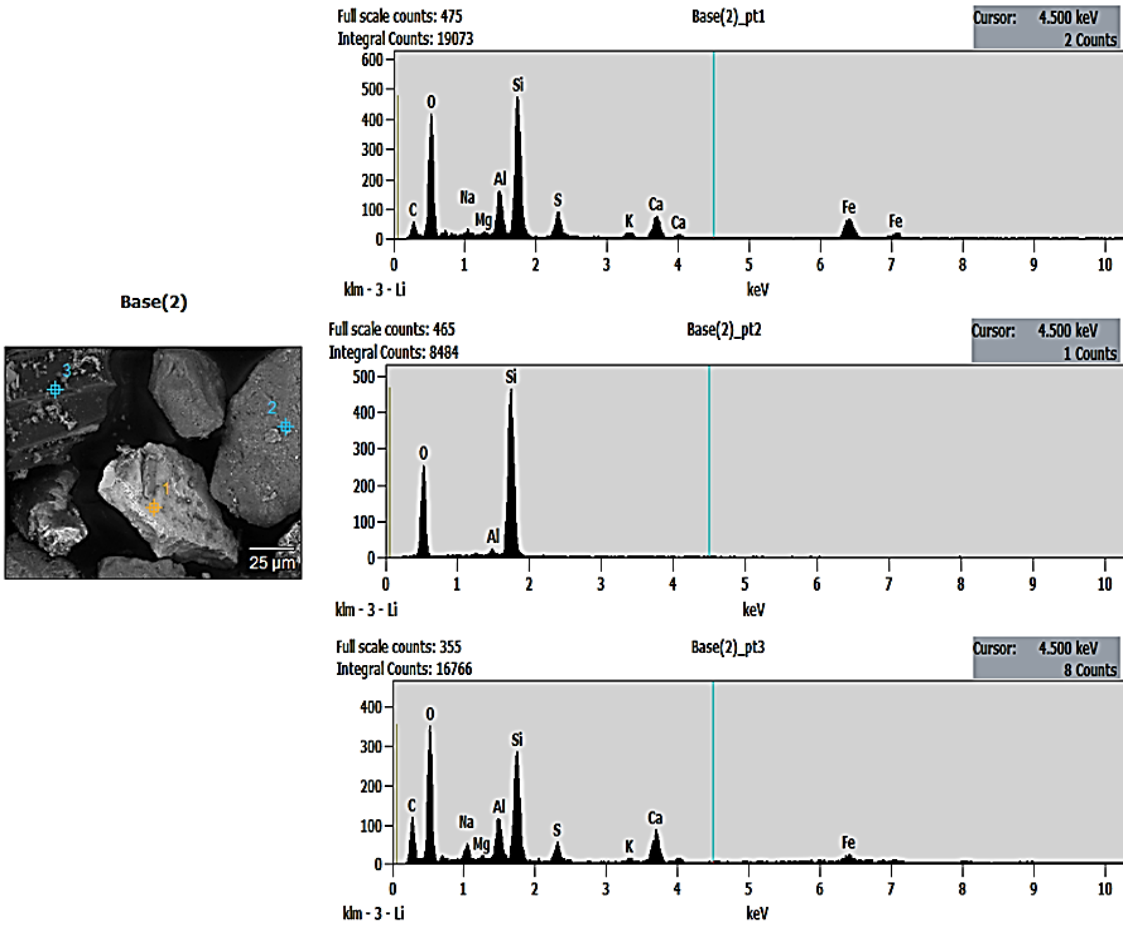



Figure 3.5.42. EDS spectrum of 20% pure pyrite on three different particles.

Table 3.5.1 Weight percentage of each element from the point analysis in Figure 3.5.42.

| Weight % | C | O | Na | Mg | Al | Si | S | K | Ca | Fe |
|---|---|---|---|---|---|---|---|---|---|---|
| Base(2)_pt1 | 7.5 | 38.2 | 1.2 | 0.5 | 6.2 | 19.7 | 3.7 | 1.2 | 5.3 | 16.5 |
| Base(2)_pt2 | | 52.7 | | | 1.4 | 45.9 | | | | |
| Base(2)_pt3 | 16.4 | 44.9 | 3.0 | 0.4 | 5.6 | 14.4 | 2.5 | 0.6 | 6.6 | 5.6 |

| % Error (+/- 1 Sigma) | C | O | Na | Mg | Al | Si | S | K | Ca | Fe |
|---|---|---|---|---|---|---|---|---|---|---|
| Base(2)_pt1 | ±0.4 | ±0.6 | ±0.2 | ±0.1 | ±0.3 | ±0.4 | ±0.1 | ±0.2 | ±0.2 | ±1.1 |
| Base(2)_pt2 | | ±1.1 | | | ±0.2 | ±0.7 | | | | |
| Base(2)_pt3 | ±0.5 | ±0.7 | ±0.2 | ±0.2 | ±0.3 | ±0.4 | ±0.1 | ±0.1 | ±0.2 | ±0.7 |

Table 3.5.2 The concentration percentages of elements on the entire scanning surface.

| Element | Weight % | Weight % Error | Atom % | Atom % Error |
|---|---|---|---|---|
| C | 7.7 | ±0.3 | 13.7 | ±0.5 |
| O | 39.5 | ±0.5 | 52.6 | ±0.7 |
| Mg | 0.3 | ±0.1 | 0.3 | ±0.1 |
| Al | 3.7 | ±0.2 | 2.9 | ±0.1 |
| Si | 24.1 | ±0.3 | 18.3 | ±0.2 |
| Si | --- | --- | --- | --- |
| S | 5.7 | ±0.2 | 3.8 | ±0.1 |
| S | --- | --- | --- | --- |
| K | 1.1 | ±0.1 | 0.6 | ±0.1 |
| K | --- | --- | --- | --- |
| Ca | 7.7 | ±0.3 | 4.1 | ±0.2 |
| Ca | --- | --- | --- | --- |
| Fe | 10.1 | ±0.5 | 3.8 | ±0.2 |
| Fe | --- | --- | --- | --- |
| Total | 100.0 | | 100.0 | |

Particle size distribution analysis has been conducted to analyze the information about the size and range of particles representative of 60% pure pyrite. Two sets of experiment were conducted. One experiment was done by using 60% pure pyrite powder (10 g) from RC#2 bucket and another test was done by analyzing 100 g of mixed powder from four different buckets. As it can be seen weight ratio is almost same for individual and mixed bucket pyrite and most of the powders size range is in between 80 to 250 microns. Table 3.5.3 shows the results of Particle size distribution analysis.

Table 3.5.3 Particle size distribution of the pyrite concentrate.

| Mesh Size | 60% pp -10g (RC#2) | 60% pp -100g (mixed) |
|---|---|---|
| ≤30 | 0.011g | 0.14g |
| ≤65 | 0.543g | 5.30g |
| ≤100 | 1.40g | 14.61g |
| ≤200 | 6.94g | 70.98g |
| ≤250 | 0.238g | 6.67g |
| ≤325 | 0.032g | 1.74g |
| Total | 9.16g | 99.4g |

From 7/1/2020 to 8/12/2020 regular size 60% pure pyrite was used and from 8/13/2020 less than 100 mesh of pyrite concentrate powder was fed to the bioreactor till the date report was made. Figure 3.5.43 shows the digital image and SEM images of regular size, larger than 100 mesh, and smaller than 100 mesh of pyrite concentrate particle. It can be easily seen from the image that each pyrite sample has its own unique micromorphological structure. Smaller particles show more sharper crystal structure than the larger particles.

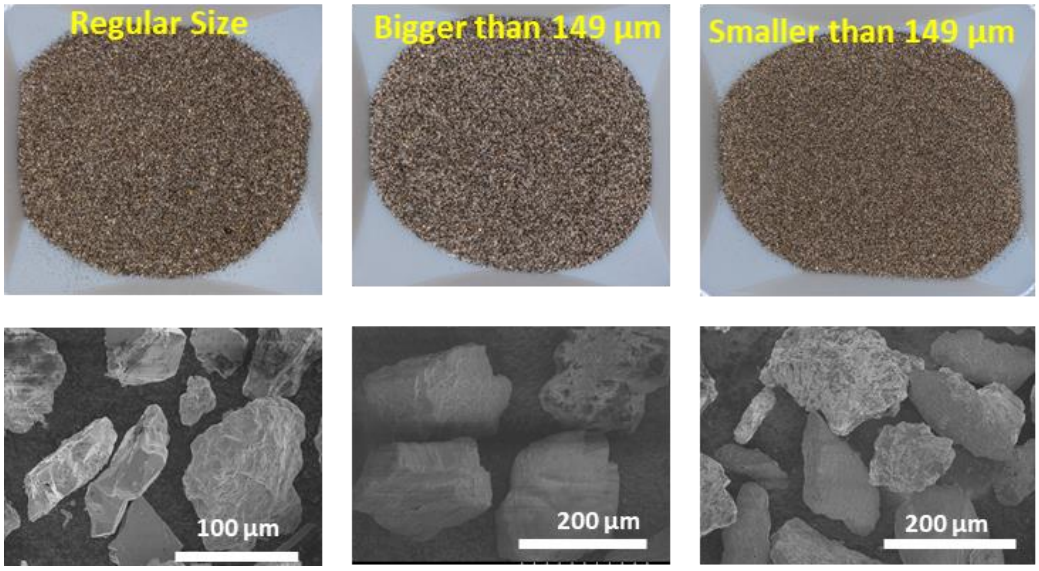



Figure 3.5.43. Digital and SEM images of regular size, larger than 100mesh, and smaller than 100mesh pyrite concentrate particle.

Two samples of effluent solid were collected and prepared on different dates from the 3L bioreactor. The pictures and SEM images of these two samples are shown in Figure 3.5.44. As can be seen, the two solid samples look similar and the SEM images both show solid particles with similar crystal structure.

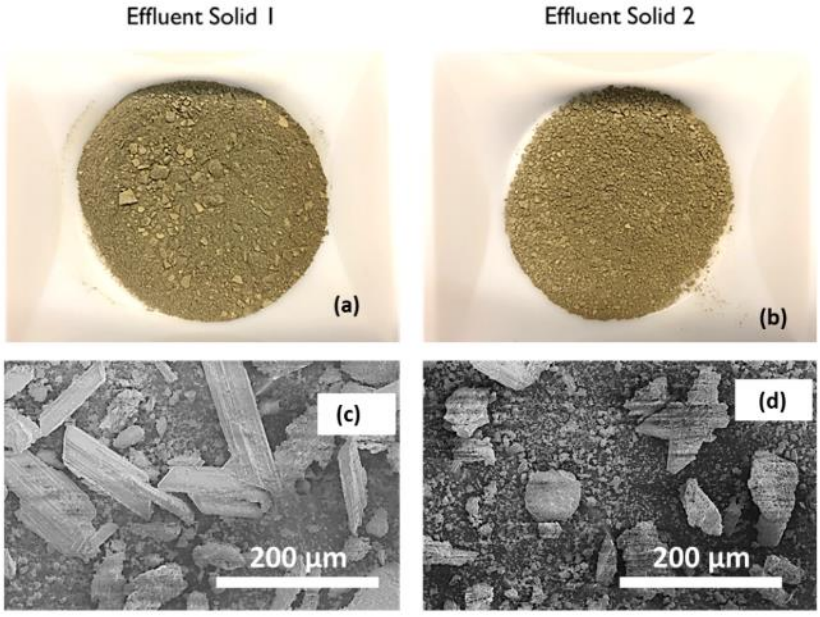



Figure 3.5.44. Digital (a-b) and SEM (c-d) images of the effluent solids.

Figure 3.5.45 shows the resulting EDS mapping image and spectrum of less than 100 mesh of pyrite concentrate and Figure 3.5.46 shows the results of the solids from effluent from the 1.5 L

bioreactor by SEM-EDS. Figure 3.5.47 shows the point analysis on the weight percentage of regular size particles, larger than 100mesh particles, and smaller than 100mesh particles of the pyrite concentrate. This data shows that Fe and S percentage in smaller size particles is higher than the larger particles.

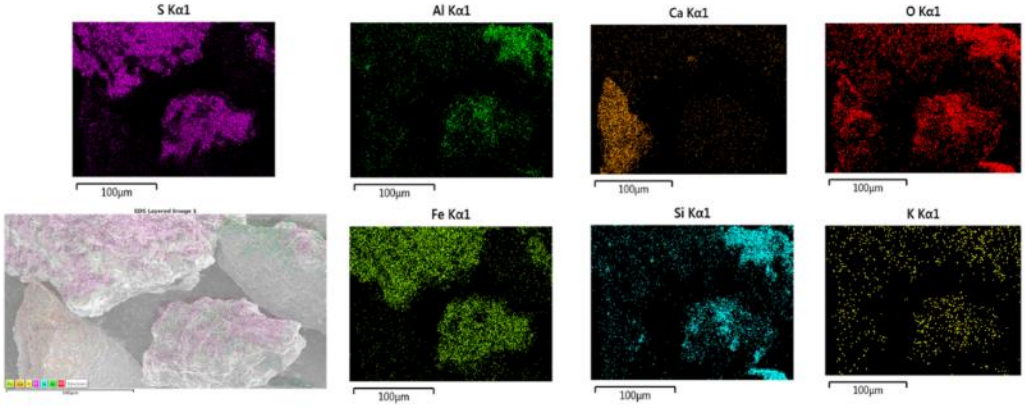



Figure 3.5.45. EDS mapping image of less than 100 mesh particles of pyrite concentrate.

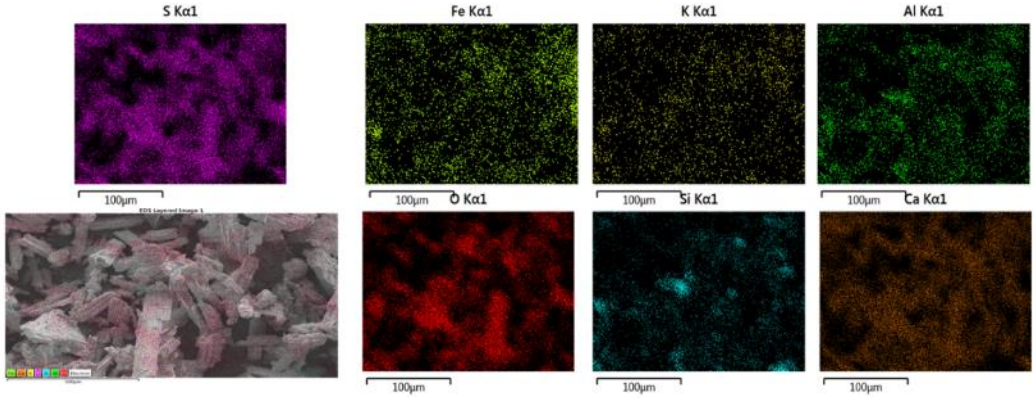



Figure 3.5.46. EDS mapping of solids from effluent.

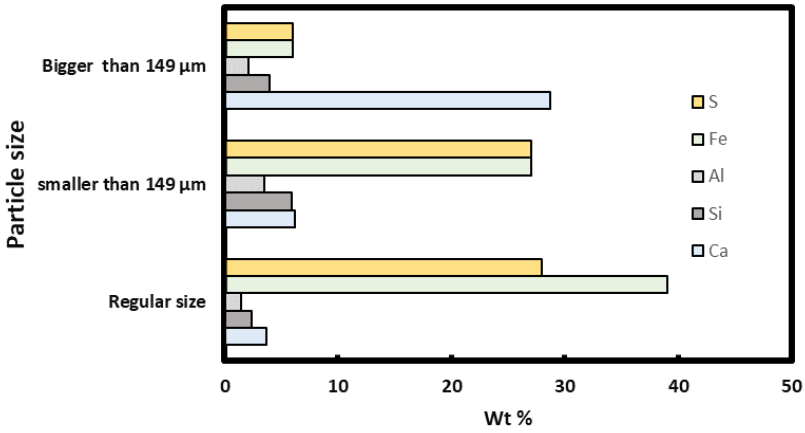



Figure 3.5.47. Weight percentage of regular size particles, larger than 100mesh particles, and smaller than 100mesh particles of pyrite concentrate (based on map sum spectrum).

EDS mapping were conducted on the effluent solid samples from the 3L bioreactor, as shown in Figure 3.5.48 and Figure 3.5.49. The main elements in the samples is S, Fe, Si, Ca, O, Al, and K. Based on the distribution of each elements in the particles, the possible compositions of each particle can be inferred. Most of the particles might be $CaSO_4$, and some small particles may be associated with $SiO_2$ and $Al_2O_3$. Very fine particles of K-jarosite may also exist in the solid.

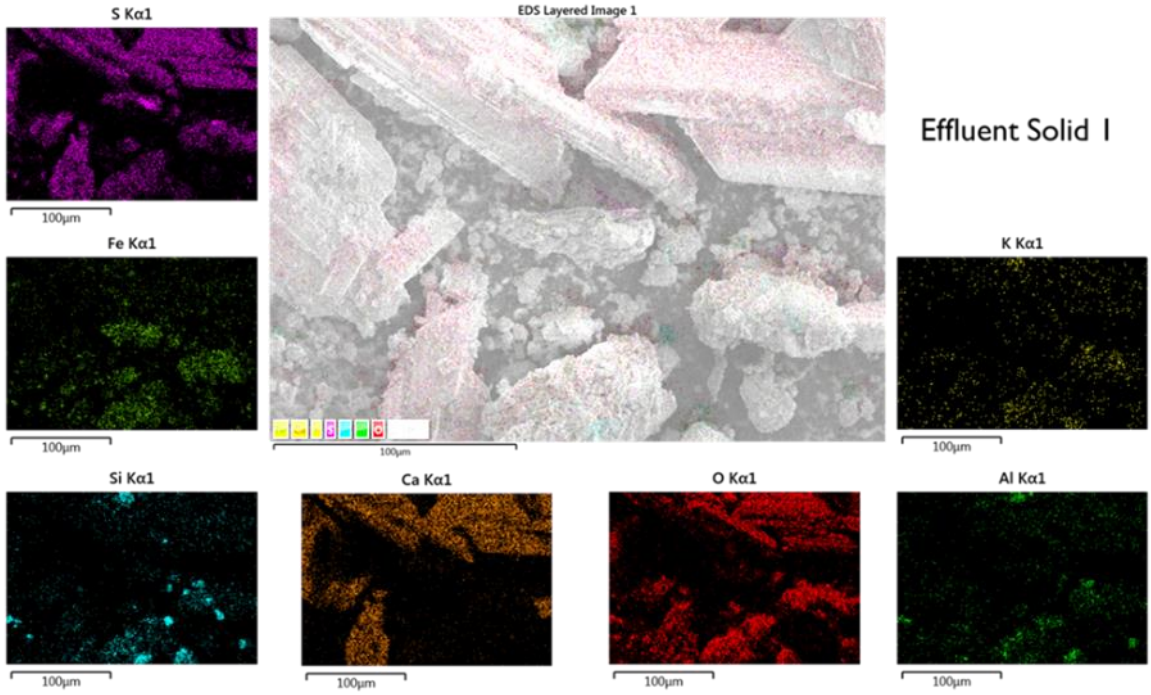



Figure 3.5.48. EDS mapping of the effluent solid 1.

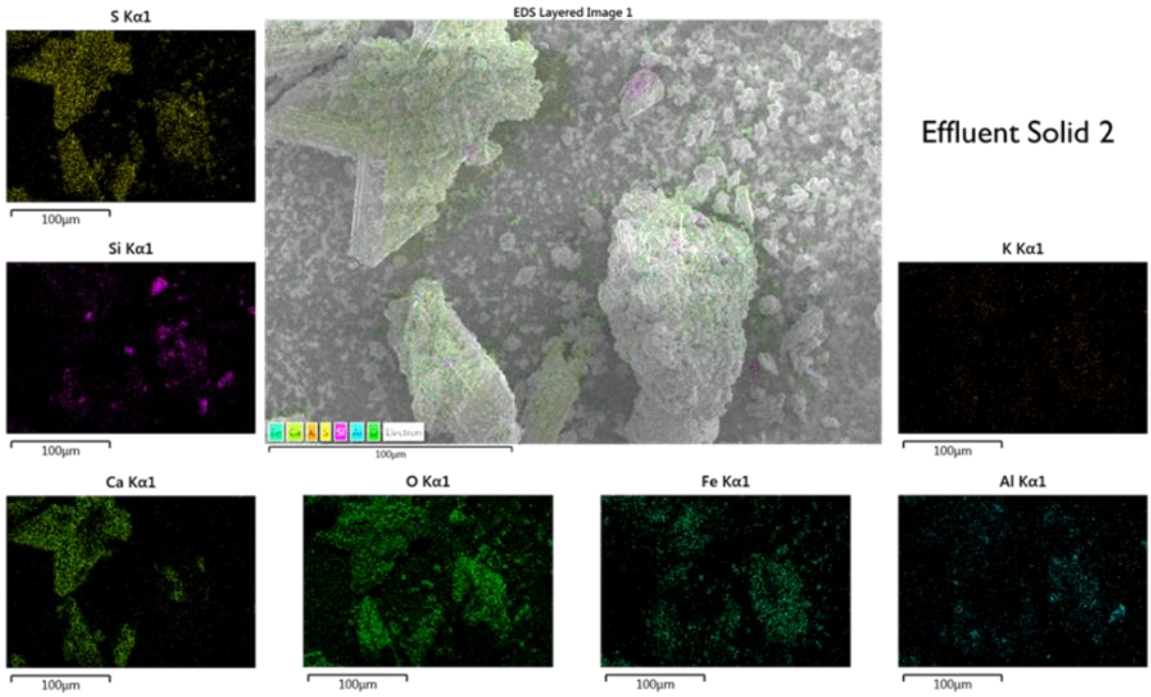



Figure 3.5.49. EDS mapping of the effluent solid 2.

To further understand the composition of the feeding and effluent material, the cross-section of the pyrite concentrates and the solid from the effluent slurry were examined with SEM and EDS. Pyrite concentrate and effluent solid particles were mounted in the epoxy with the hardener, then samples were polished to show the cross-section of the particles, as shown in Figure 3.5.50.



Figure 3.5.50. Mounted samples of the pyrite concentrate and the effluent solid.

To increase the conductivity of the sample, gold sputtering was conducted on the samples to coat an 8 nm gold conducting layer. Then the SEM and EDS were conducted on these samples. The SEM images of the pyrite concentrate cross section are shown in Figure 3.5.51. As can be seen, the cross sections of the pyrite concentrate particles had been exposed by polishing. EDS mapping analysis on the elemental composition of these particles are shown in Figure 5 and Figure 6. Based on the elemental composition of each particle, the possible chemical compositions of the particles were labeled in the SEM image. In Figure 3.5.52, lots of $FeS_2$ particles were observed with some $SiO_2$ and $CaCO_3$ particles. However, in Figure 3.5.53, more $SiO_2$ and $CaCO_3$ particles are observed than the $FeS_2$ particles. it's worth noting that a $ZnS_2$ particle was present in the sample, confirmed the existence of metal sulfide other than iron.

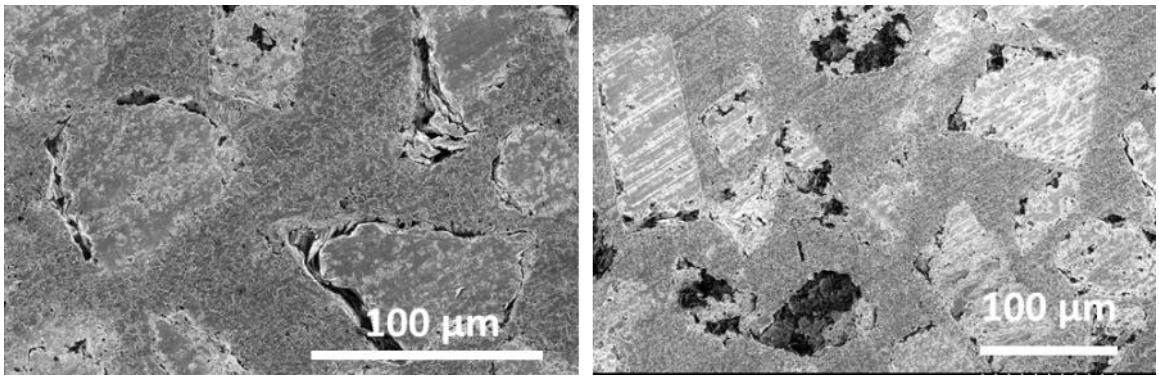



Figure 3.5.51. SEM images on the cross section of the pyrite concentrate particles.

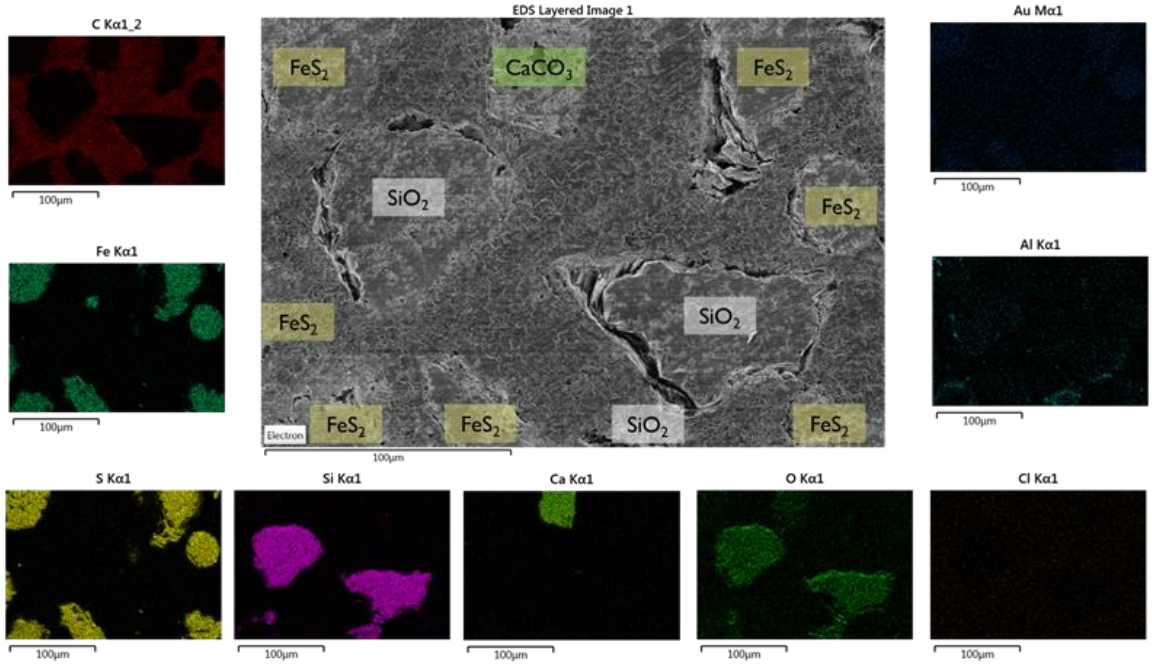



Figure 3.5.52. EDS mapping on the cross section of the pyrite concentrate particles spot 1.

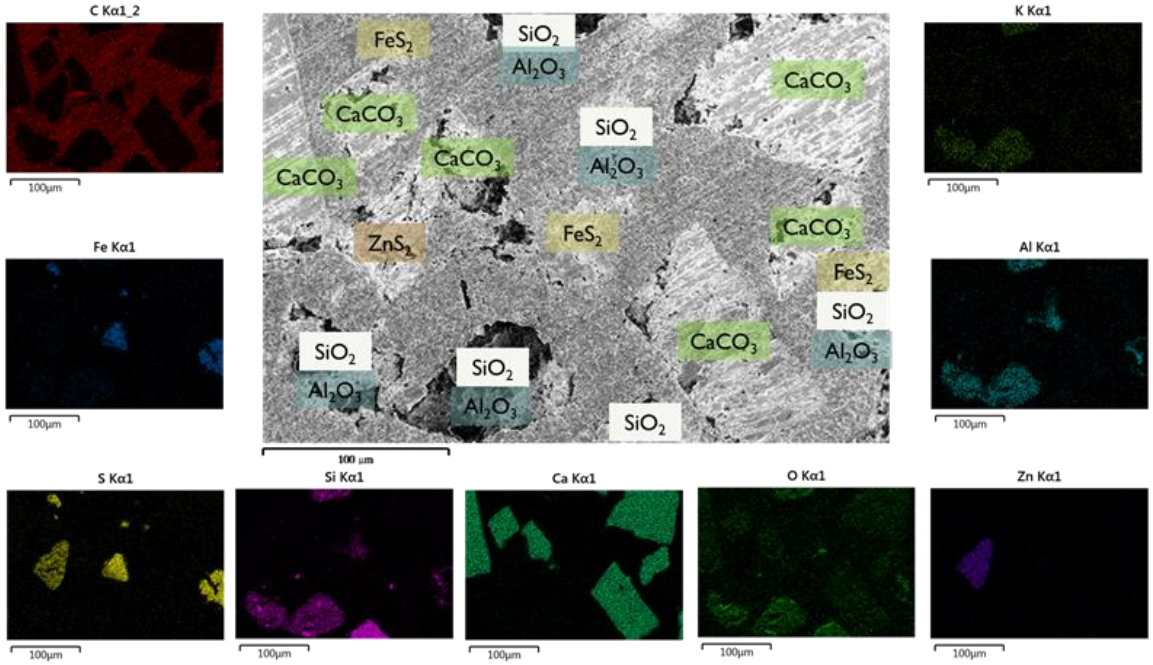



Figure 3.5.53. EDS mapping on the cross section of the pyrite concentrate particles spot 2.

Similarly, the SEM images of the effluent solid from the 3 L bioreactor are shown in Figure 3.5.54. Compared with the pyrite concentrate feeding material, the shape and size of the particles were different, indicating that the composition of these particles may be different.

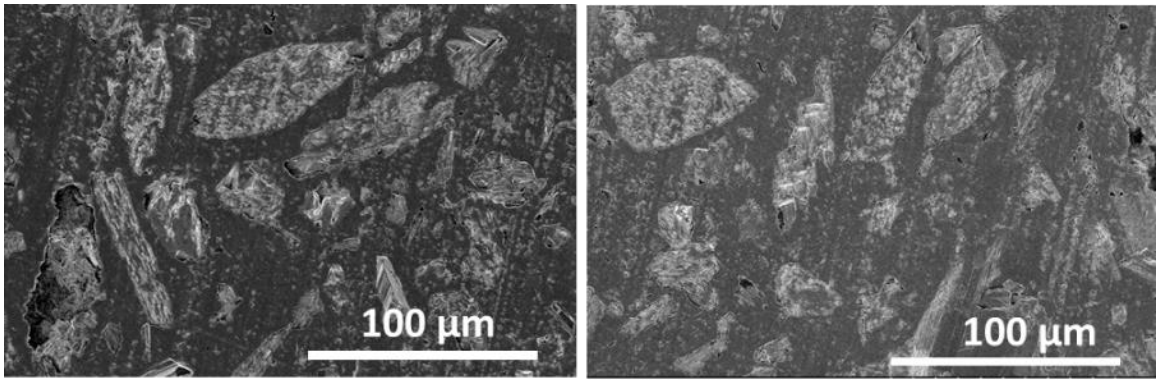



Figure 3.5.54. SEM images on the cross section of the effluent solid from the 3L bioreactor.

EDS mappings of these two spots in Figure 3.5.54 are shown in Figure 3.5.55 and Figure 3.5.56. It is clear that most of the particles in the effluent solid is $CaSO_4$, which was the product of the reaction between $CaCO_3$ in the pyrite concentrate and the $H_2SO_4$ in the bioleaching solution. This indicated that a large amount of the acid in the bioleaching solution was consumed by the $CaCO_3$ impurity in the pyrite concentrate. To increase the acid generation in the bioreactor, pyrite with low content of $CaCO_3$ should be used. Other chemicals observed in the effluent solid were $SiO_2$, $Al_2O_3$, and very little unreacted $FeS_2$.

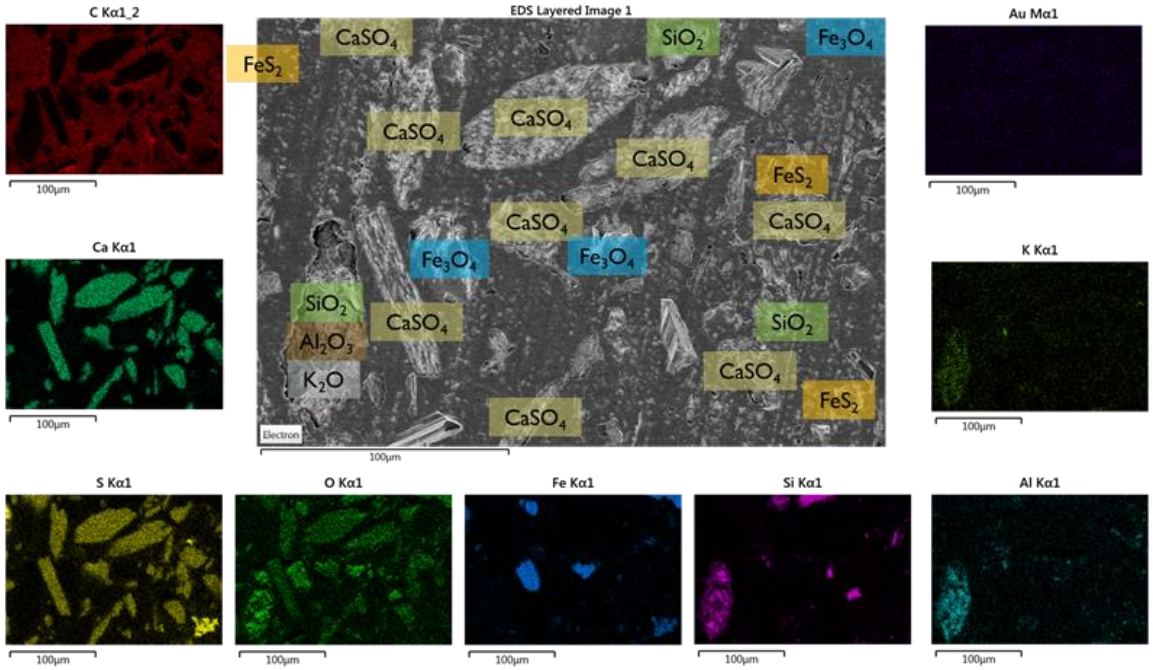



Figure 3.5.55. EDS mapping on the cross section of the effluent solid particles spot 1.

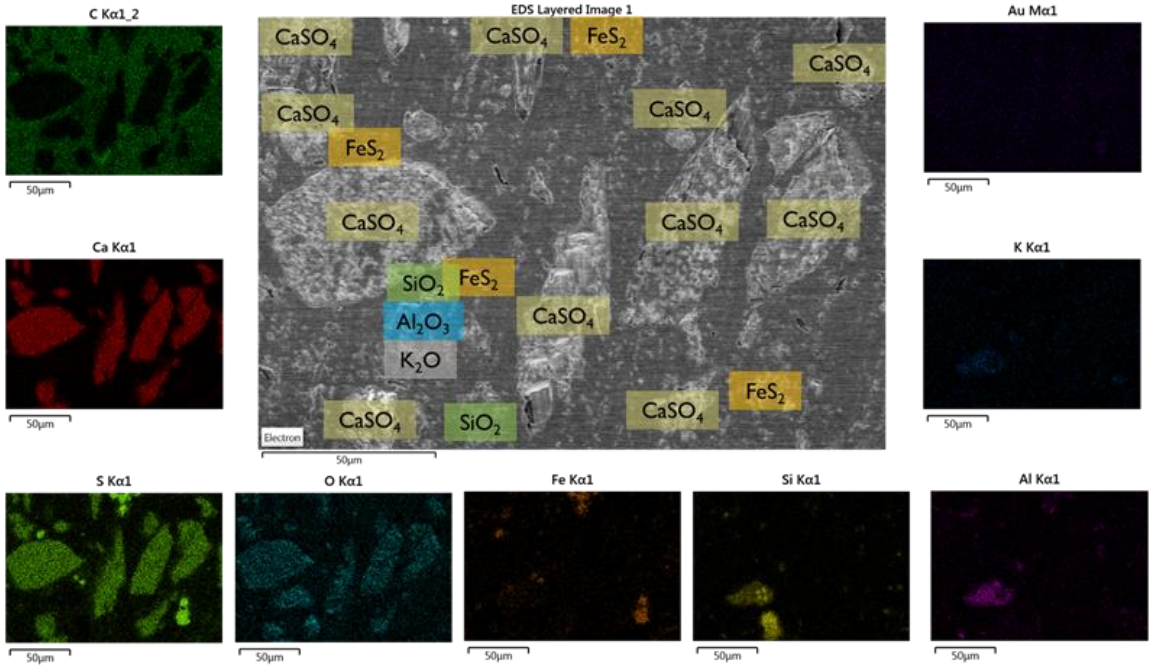



Figure 3.5.56. EDS mapping on the cross section of the effluent solid particles spot 2.

Figure 3.5.57 shows the SEM image and EDS mapping image of the effluent solids from 1.5L bioreactor. Similar to the 3L reactor, this data shows that there were some unreacted $FeS_2$ were still in the effluent solids and also some trace of Silica and alumina also there.

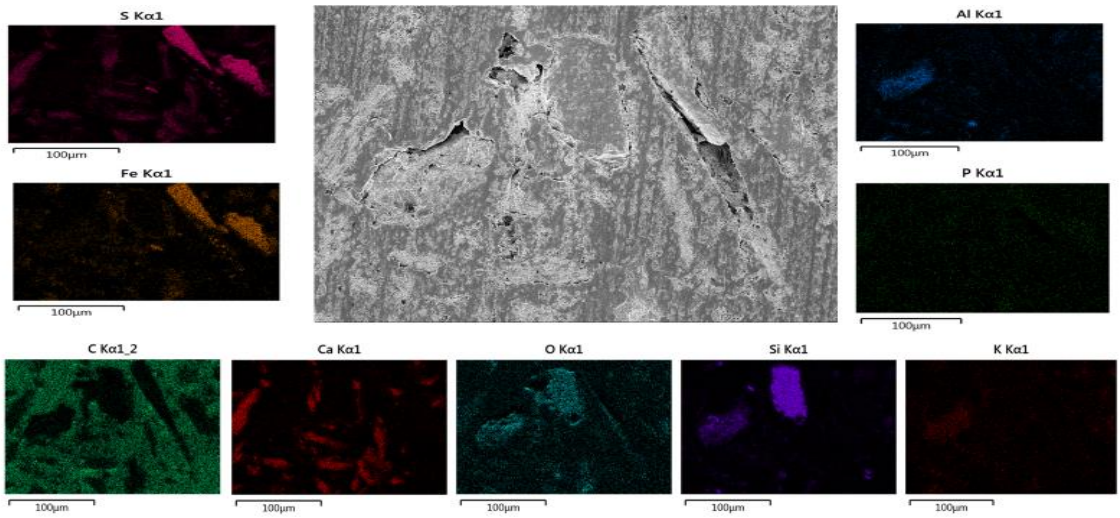



Figure 3.5.57. SEM image and EDS mapping results of cross-section of effluent solids from 1.5 L reactor.

**XRD Analysis**: Figure 3.5.58 and Figure 3.5.59 show the XRD patterns of the pyrite concentrate and the effluent solid from the 3L bioreactor. As can be seen, the main composites in the pyrite concentrate were pyrite, calcite and silica. While in the effluent solid, the main composites were calcium sulfate, jarosite, and silica. There were no peaks associated with pyrite, which indicated that most of the jarosite has been consumed in the bioreactor. However, pyrite residues might be covered with the jarosite or calcium sulfate precipitate formed in the bioreactor. To confirm this, the solid needed to be grinded to remove possible precipitate shells on the residue pyrite. Figure 3.5.60 shows the XRD results for the pyrite concentrate particles of size less than 100 mesh after grinding with an agate mortar. Here some intense peaks of the iron sulfide can be seen. Figure 3.5.61 and Figure 3.5.62 shows the XRD results of the effluent solids from the 3L and the 1.5L bioreactors after grinding with an agate mortar. From these two images, it was noted that there were few peaks of iron sulfide alongside with jarosite. Few additional peaks that correspond to calcite, were also noted in these patterns.

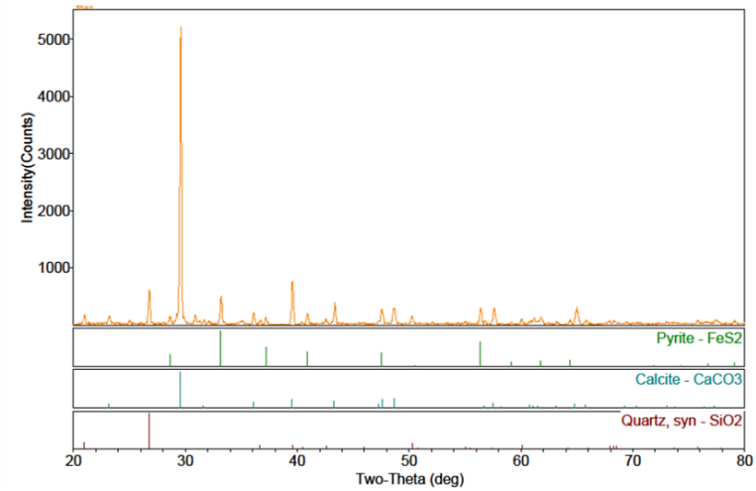



Figure 3.5.58. XRD results of the less than 100 mesh pyrite concentrate particles.

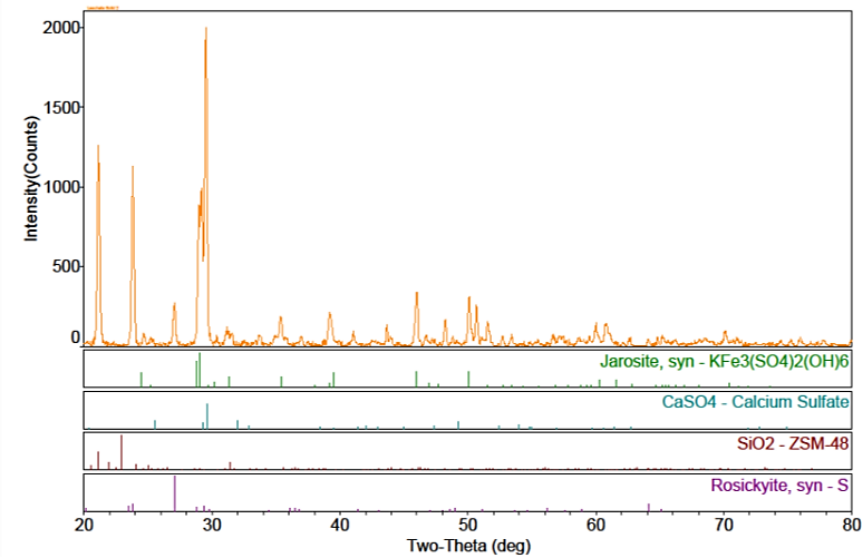



Figure 3.5.59. XRD results of the effluent solids from the 3L bioreactor.

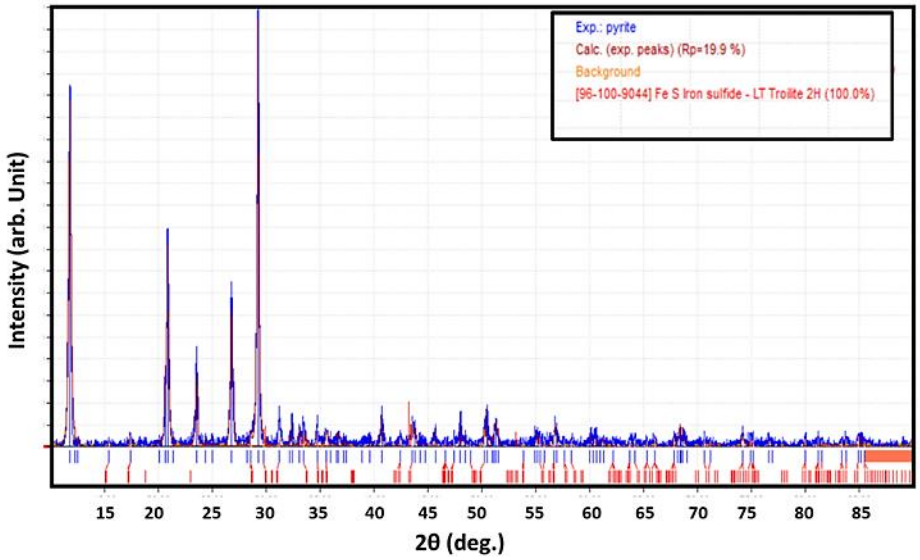



Figure 3.5.60. XRD results for the less than 100mesh pyrite concentrate particles.

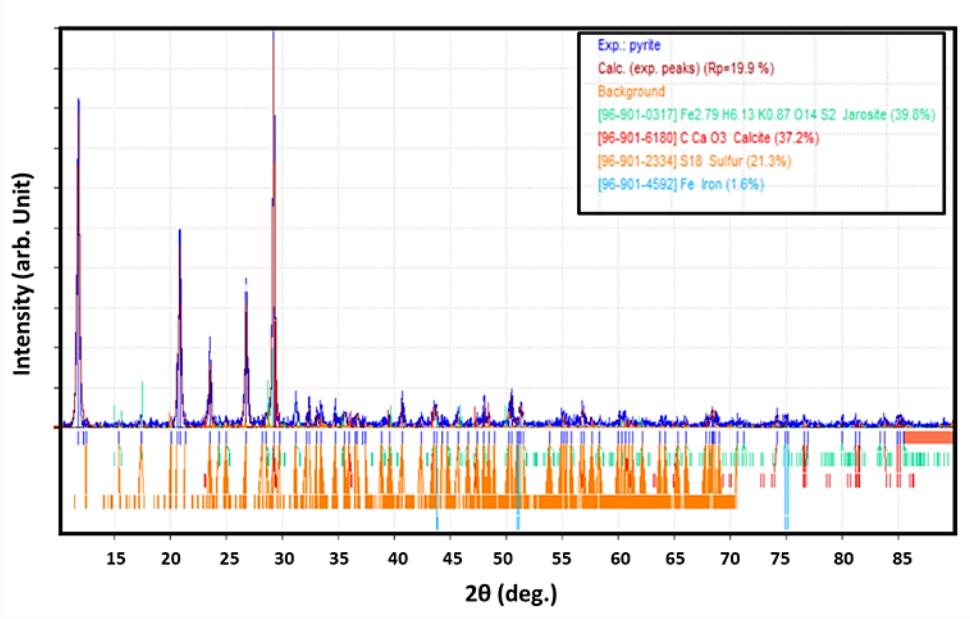



Figure 3.5.61. XRD results of the effluent solids from the 3L bioreactor.

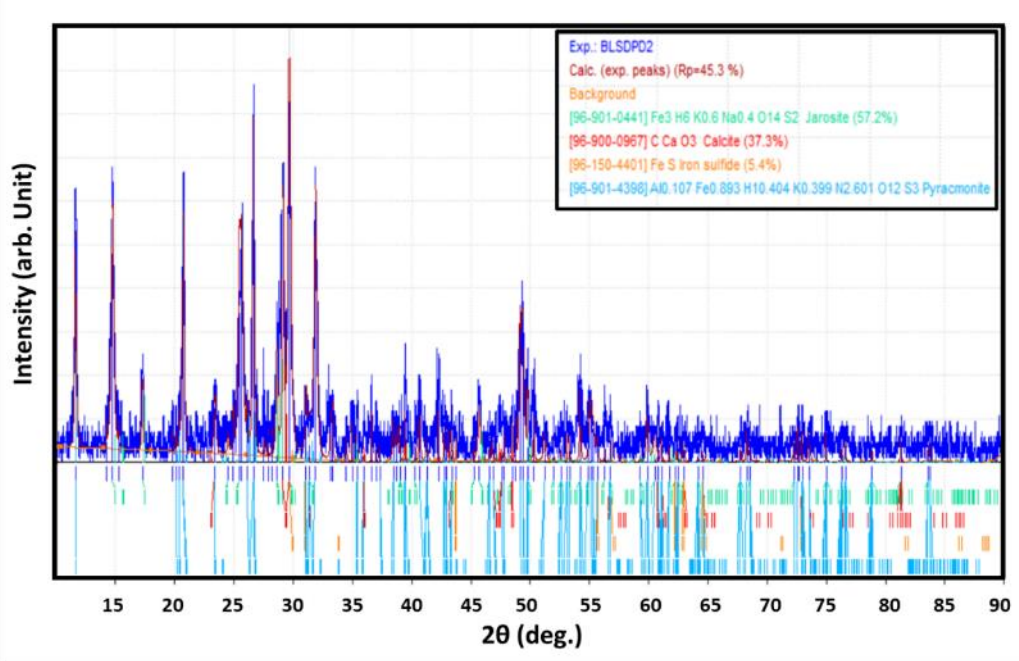



Figure 3.5.62. XRD results of the effluent solids from the 1.5L bioreactor.

**ICP-MS Analysis on the Digestion Solution:** 0.1 g of pyrite concentrate feeding material and 0.1 effluent solid from the 3L bioreactor were dissolved in 10 ml of aqua regia, respectively, then the digestion solution was diluted to 1000 ml. ICP samples were prepared for the two digestion solutions and the bioleaching solutions collected during the 8.5-day mass balance test. The analytical ICP results of the pyrite concentrate digestion solution are shown in Table 3.5.4 and Figure 3.5.63. As it shows, the main elements in the pyrite concentrates are Fe and Ca. Other

elements with minor concentrations are Mg, Zn, Al, Na, Cu, and Co. No Si, K, and P were detected in the digestion solution. Concentrations of REEs are also included in this table, which was used to track the mass balance of the REEs in the bioleaching process.

Table 3.5.4 ICP results of the pyrite concentrate digestion solution.

| Element | Measured Concentration/ppb | RSD | Dilution Factor | Concentration/(mg/L) | STD | Mass/mg | Percentage/% |
|---|---|---|---|---|---|---|---|
| Fe | 282.763 | 1.8 | 100 | 28.2763 | 0.5089734 | 28.2763 | 28.2763 |
| Ca | 1501.598 | 3.1 | 10 | 15.01598 | 0.46549538 | 15.01598 | 15.01598 |
| Mg | 199.051 | 1.9 | 1 | 0.199051 | 0.00378197 | 0.199051 | 0.199051 |
| Zn | 148.179 | 1.3 | 1 | 0.148179 | 0.00192633 | 0.148179 | 0.148179 |
| Al | 138.268 | 1.7 | 1 | 0.138268 | 0.00235056 | 0.138268 | 0.138268 |
| Na | 40.215 | 2.4 | 1 | 0.040215 | 0.00096516 | 0.040215 | 0.040215 |
| Cu | 4.901 | 1.3 | 1 | 0.004901 | 6.3713E-05 | 0.004901 | 0.004901 |
| Co | 3.286 | 2.2 | 1 | 0.003286 | 7.2292E-05 | 0.003286 | 0.003286 |
| Si | 0 | - | - | 0 | 0 | 0 | 0 |
| K | 0 | - | - | 0 | 0 | 0 | 0 |
| P | 0 | - | - | 0 | 0 | 0 | 0 |
| Sc | 0.386 | 10.8 | 1 | 0.000386 | 4.1688E-05 | 0.000386 | 0.000386 |
| Y | 0.731 | 0.2 | 1 | 0.000731 | 1.462E-06 | 0.000731 | 0.000731 |
| La | 0.859 | 3.2 | 1 | 0.000859 | 2.7488E-05 | 0.000859 | 0.000859 |
| Ce | 2.267 | 3.8 | 1 | 0.002267 | 8.6146E-05 | 0.002267 | 0.002267 |
| Pr | 0.279 | 11.8 | 1 | 0.000279 | 3.2922E-05 | 0.000279 | 0.000279 |
| Nd | 1.207 | 5.6 | 1 | 0.001207 | 6.7592E-05 | 0.001207 | 0.001207 |
| Sm | 0.374 | 8 | 1 | 0.000374 | 0.00002992 | 0.000374 | 0.000374 |
| Eu | 0.062 | 28.3 | 1 | 0.000062 | 1.7546E-05 | 0.000062 | 0.000062 |
| Gd | 0.665 | 4.1 | 1 | 0.000665 | 2.7265E-05 | 0.000665 | 0.000665 |
| Tb | 0.031 | 12.3 | 1 | 0.000031 | 3.813E-06 | 0.000031 | 0.000031 |
| Dy | 0.191 | 2.3 | 1 | 0.000191 | 4.393E-06 | 0.000191 | 0.000191 |
| Ho | 0.03 | 9.4 | 1 | 0.00003 | 0.00000282 | 0.00003 | 0.00003 |
| Er | 0.098 | 7.2 | 1 | 0.000098 | 7.056E-06 | 0.000098 | 0.000098 |
| Tm | 0.009 | 3.5 | 1 | 0.000009 | 3.15E-07 | 0.000009 | 0.000009 |
| Yb | 0.194 | 1.5 | 1 | 0.000194 | 0.00000291 | 0.000194 | 0.000194 |
| Lu | 0.014 | 33.9 | 1 | 0.000014 | 4.746E-06 | 0.000014 | 0.000014 |

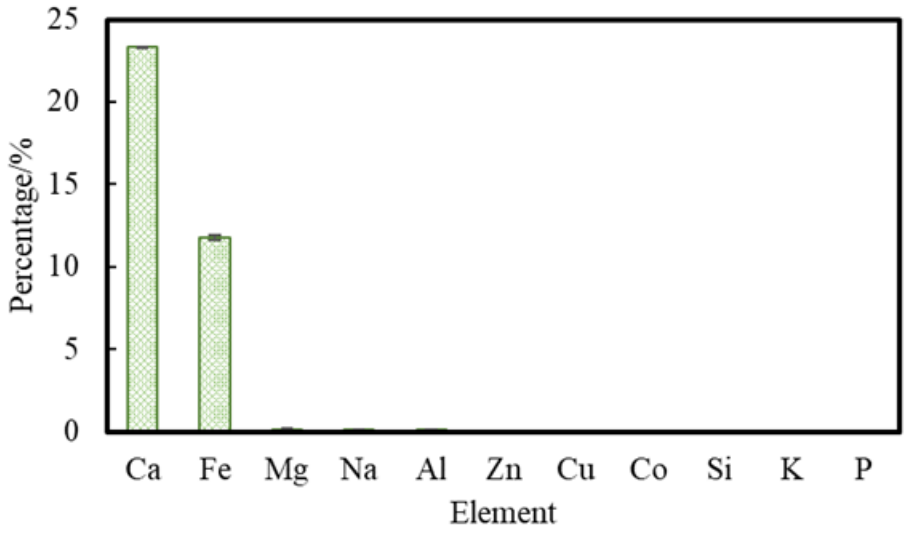



Figure 3.5.63. ICP results of the pyrite concentrate digestion solution

The analytical ICP results of the effluent solid digestion solution are shown in Table 3.5.5 and Figure 3.5.64. As it shows, the main elements in the pyrite concentrates were still Fe and Ca. However, the elements with highest percentage becomes Ca as most of the Fe was dissolved into the bioleaching solution while Ca was precipitated as $CaSO_4$.

Table 3.5.5 ICP results of the effluent solid digestion solution in the 3L bioreactor.

| Element | Measured Concentration/ppb | RSD | Dilution Factor | Concentration/(mg/L) | SD | Mass/mg | Percentage/% |
|---|---|---|---|---|---|---|---|
| Ca | 2332.996 | 2 | 10 | 23.32996 | 0.4665992 | 23.32996 | 23.32996 |
| Fe | 117.718 | 2.2 | 100 | 11.7718 | 0.2589796 | 11.7718 | 11.7718 |
| Mg | 13.767 | 2.5 | 10 | 0.13767 | 0.00344175 | 0.13767 | 0.13767 |
| Na | 13.076 | 2.1 | 10 | 0.13076 | 0.00274596 | 0.13076 | 0.13076 |
| Al | 117.932 | 0.3 | 1 | 0.117932 | 0.0003538 | 0.117932 | 0.117932 |
| Zn | 61.939 | 1.6 | 1 | 0.061939 | 0.00099102 | 0.061939 | 0.061939 |
| Cu | 7.873 | 2.4 | 1 | 0.007873 | 0.00018895 | 0.007873 | 0.007873 |
| Co | 0.785 | 2.9 | 1 | 0.000785 | 2.2765E-05 | 0.000785 | 0.000785 |
| Si | 0 | - | - | 0 | 0 | 0 | 0 |
| K | 0 | - | - | 0 | 0 | 0 | 0 |
| P | 0 | - | - | 0 | 0 | 0 | 0 |
| Sc | 0.202 | 9 | 1 | 0.000202 | 0.00001818 | 0.000202 | 0.000202 |
| Y | 0.339 | 37.9 | 1 | 0.000339 | 0.00012848 | 0.000339 | 0.000339 |
| La | 0.991 | 4 | 1 | 0.000991 | 0.00003964 | 0.000991 | 0.000991 |
| Ce | 2.619 | 3.5 | 1 | 0.002619 | 9.1665E-05 | 0.002619 | 0.002619 |
| Pr | 0.326 | 7 | 1 | 0.000326 | 0.00002282 | 0.000326 | 0.000326 |
| Nd | 1.334 | 2.6 | 1 | 0.001334 | 3.4684E-05 | 0.001334 | 0.001334 |
| Sm | 0.398 | 2.4 | 1 | 0.000398 | 9.552E-06 | 0.000398 | 0.000398 |
| Eu | 0.064 | 6.6 | 1 | 0.000064 | 4.224E-06 | 0.000064 | 0.000064 |
| Gd | 0.527 | 3.7 | 1 | 0.000527 | 1.9499E-05 | 0.000527 | 0.000527 |
| Tb | 0.019 | 5.3 | 1 | 0.000019 | 1.007E-06 | 0.000019 | 0.000019 |
| Dy | 0.112 | 11.5 | 1 | 0.000112 | 0.00001288 | 0.000112 | 0.000112 |
| Ho | 0.013 | 2.7 | 1 | 0.000013 | 3.51E-07 | 0.000013 | 0.000013 |
| Er | 0.043 | 14.2 | 1 | 0.000043 | 6.106E-06 | 0.000043 | 0.000043 |
| Tm | 0.004 | 4.9 | 1 | 0.000004 | 1.96E-07 | 0.000004 | 0.000004 |
| Yb | 0.135 | 11.3 | 1 | 0.000135 | 1.5255E-05 | 0.000135 | 0.000135 |
| Lu | 0.007 | 7.3 | 1 | 0.000007 | 5.11E-07 | 0.000007 | 0.000007 |

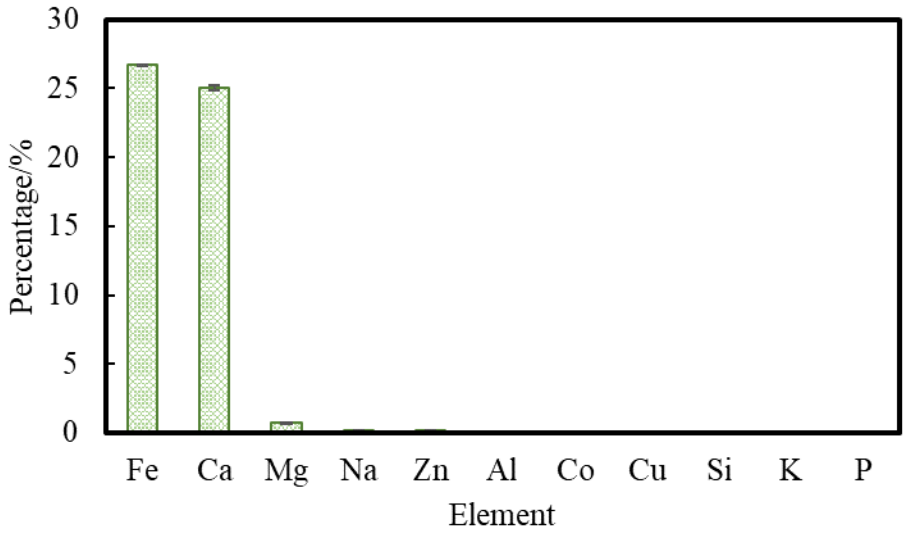



Figure 3.5.64. ICP results of the effluent solid digestion solution in the 3L bioreactor.

The ICP results of the bioleaching solution collected during the 8.5-day mass balance test is shown in **Error! Reference source not found.** and Figure 3.5.65. It is clear that in the bioleaching solution, Fe was the element with the highest concentration, which is 24.79 g/L. While the Ca concentration was only 0.93 g/L because most of the Ca weny to the effluent solid in the form of $CaSO_4$ precipitate. Other elements with minor concentrations were the same as it was in the pyrite concentrate. REE concentrations are included in this table.

Table 3.5.6 ICP results of the bioleaching solution collected during the mass balance test.

| Element | Measured Concentration/ppb | RSD | Dilution Factor | Concentration/(g/L) | SD |
|---|---|---|---|---|---|
| Fe | 247.893 | 4.6 | 100000 | 24.7893 | 1.1403078 |
| Ca | 929.128 | 0.3 | 1000 | 0.929128 | 0.00278738 |
| Zn | 108.98 | 0.5 | 1000 | 0.10898 | 0.0005449 |
| Mg | 632.178 | 0.3 | 100 | 0.0632178 | 0.00018965 |
| Al | 236.895 | 0.7 | 100 | 0.0236895 | 0.00016583 |
| P | 40.368 | 3.3 | 100 | 0.0040368 | 0.00013321 |
| Co | 2.506 | 1.5 | 1000 | 0.002506 | 0.00003759 |
| Na | 16.09 | 2 | 100 | 0.001609 | 0.00003218 |
| Cu | 2.307 | 1.6 | 100 | 0.0002307 | 3.6912E-06 |
| Si | 0 | - | - | 0 | 0 |
| K | 0 | - | - | 0 | 0 |
| Sc | 1.865 | 5.4 | 100 | 0.0001865 | 1.0071E-05 |
| Y | 6.381 | 1.1 | 100 | 0.0006381 | 7.0191E-06 |
| La | 3.05 | 0.8 | 100 | 0.000305 | 0.00000244 |
| Ce | 6.709 | 1.9 | 100 | 0.0006709 | 1.2747E-05 |
| Pr | 0.821 | 2 | 100 | 0.0000821 | 1.642E-06 |
| Nd | 3.645 | 1.9 | 100 | 0.0003645 | 6.9255E-06 |
| Sm | 0.936 | 4.8 | 100 | 0.0000936 | 4.4928E-06 |
| Eu | 0.242 | 2.9 | 100 | 0.0000242 | 7.018E-07 |
| Gd | 1.352 | 1.7 | 100 | 0.0001352 | 2.2984E-06 |
| Tb | 0.209 | 2.2 | 100 | 0.0000209 | 4.598E-07 |
| Dy | 1.231 | 5.1 | 100 | 0.0001231 | 6.2781E-06 |
| Ho | 0.248 | 3.7 | 100 | 0.0000248 | 9.176E-07 |
| Er | 0.66 | 0.3 | 100 | 0.000066 | 1.98E-07 |
| Tm | 0.087 | 2.8 | 100 | 0.0000087 | 2.436E-07 |
| Yb | 0.497 | 1.6 | 100 | 0.0000497 | 7.952E-07 |
| Lu | 0.068 | 3.9 | 100 | 0.0000068 | 2.652E-07 |

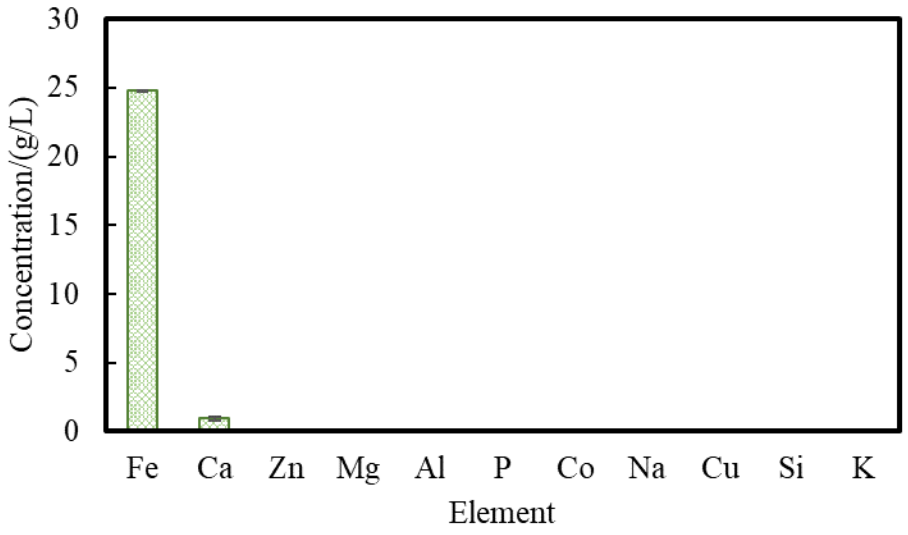



Figure 3.5.65. ICP results of the bioleaching solution collected during the mass balance test.

The amount of REEs fed in to the bioreactor in the form of pyrite concentrate and came out of the bioreactor in the effluent solid and in the bioleaching solution were calculated based on the mass or volume of the solid and liquid as well as the measured concentrations of each element in these three streams. The calculated input and output mass of each elements and the errors between them are shown in Table 3.5.7 and Figure 3.5.66. As can be seen, the errors of most of the REE are within 20%.

Table 3.5.7 Mass balance of the REE during the 8.5-day mass balance test.

| Elements | Input/mg | Output/mg | Error/% |
|---|---|---|---|
| 45 Sc | 0.65041 | 0.568386 | -12.6112 |
| 89 Y | 1.231735 | 1.490311 | 20.99279 |
| 139 La | 1.447415 | 1.782036 | 23.11855 |
| 140 Ce | 3.819895 | 4.486543 | 17.45199 |
| 141 Pr | 0.470115 | 0.556135 | 18.29774 |
| 146 Nd | 2.033795 | 2.322819 | 14.21105 |
| 147 Sm | 0.63019 | 0.668019 | 6.002825 |
| 153 Eu | 0.10447 | 0.122516 | 17.27348 |
| 157 Gd | 1.120694 | 0.902876 | -19.436 |
| 159 Tb | 0.052235 | 0.059003 | 12.95606 |
| 163 Dy | 0.321835 | 0.34764 | 8.018022 |
| 165 Ho | 0.05055 | 0.057695 | 14.13492 |
| 166 Er | 0.16513 | 0.164387 | -0.44983 |
| 169 Tm | 0.015165 | 0.019517 | 28.69502 |
| 172 Yb | 0.32689 | 0.256209 | -21.6223 |
| 175 Lu | 0.02359 | 0.020253 | -14.1467 |

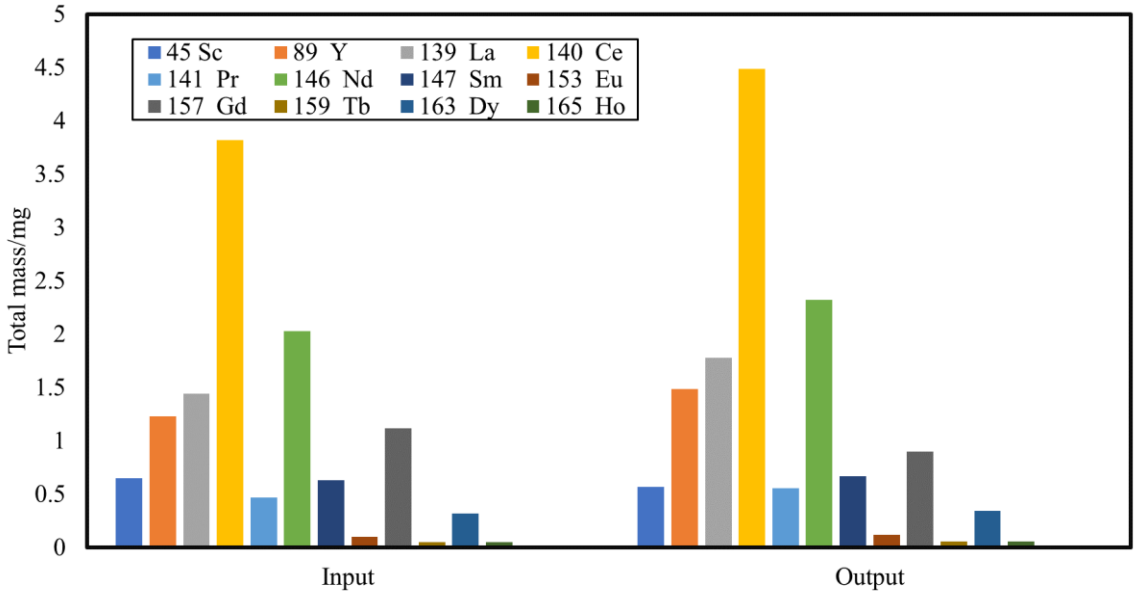



Figure 3.5.66. Total mass of the REE input and output during the 8.5-day mass balance test.

During the bioleaching process, part the REEs in the pyrite concentrate was leached into the bioleaching solution. The recovery (percentages dissolved to the bioleaching solution) of REEs in the bioleaching solution was calculated and plotted in Figure 3.5.67. The recovery in the bioleaching solution for almost half of the REEs was higher than 50%. The overall recovery of REEs in the bioleaching solution was 33.4%.

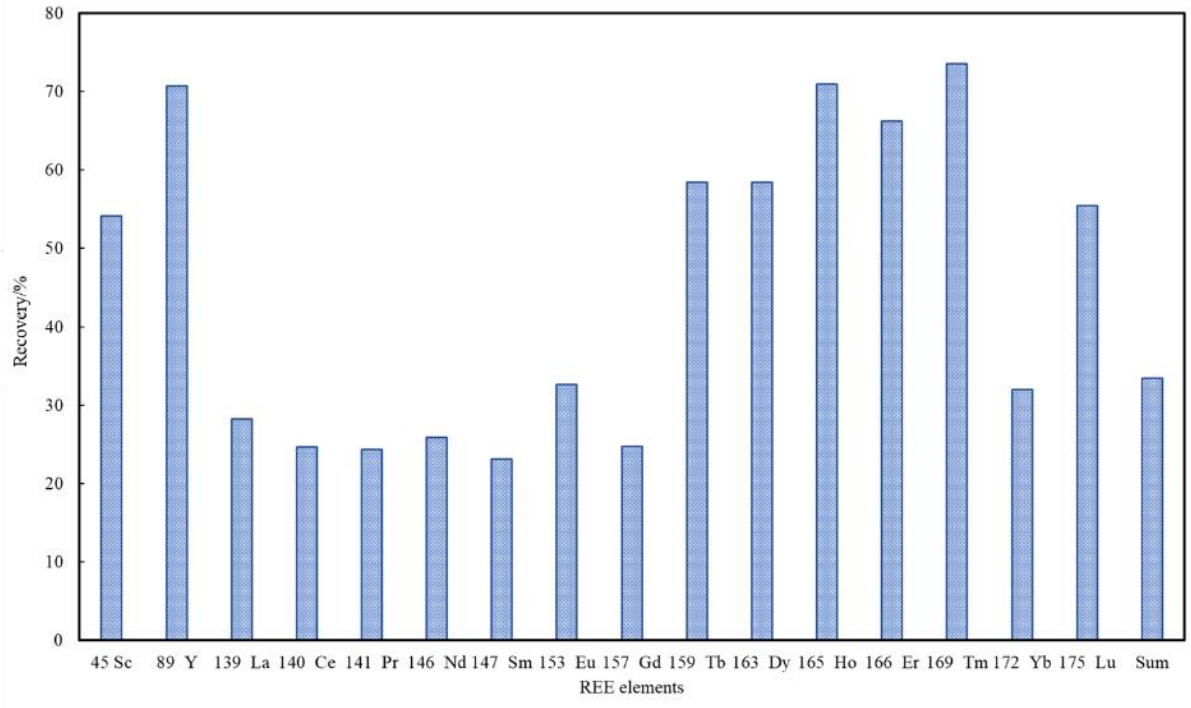



Figure 3.5.67. The recoveries of REEs in the bioleaching solution.

Similar mass balance calculations were made for the main elements such as Fe and Ca and the results are shown in Table 3.5.8. Similar to REEs, the errors of most of the main elements were within 20%.

Table 3.5.8 Mass balance of the main elements during the 8.5-day mass balance test.

| Elements | Input/g | Output/g | Error/% |
|---|---|---|---|
| Fe | 47.64557 | 56.09268 | 17.72906 |
| Ca | 25.30193 | 31.63804 | 25.04203 |
| Mg | 0.335401 | 0.281959 | -15.9338 |
| Zn | 0.249682 | 0.259743 | 4.02972 |
| Al | 0.232982 | 0.191267 | -17.9046 |
| Cu | 0.008258 | 0.01054 | 27.63064 |
| Co | 0.005537 | 0.005148 | -7.02641 |

The recoveries of these elements in the bioleaching solution were calculated and shown in Figure 3.5.68. For Fe, Zn, and Co, the recovery is higher than 70%. For other elements such as Ca, the recovery was quite low because most of it is went to the effluent solid in the form of precipitate.

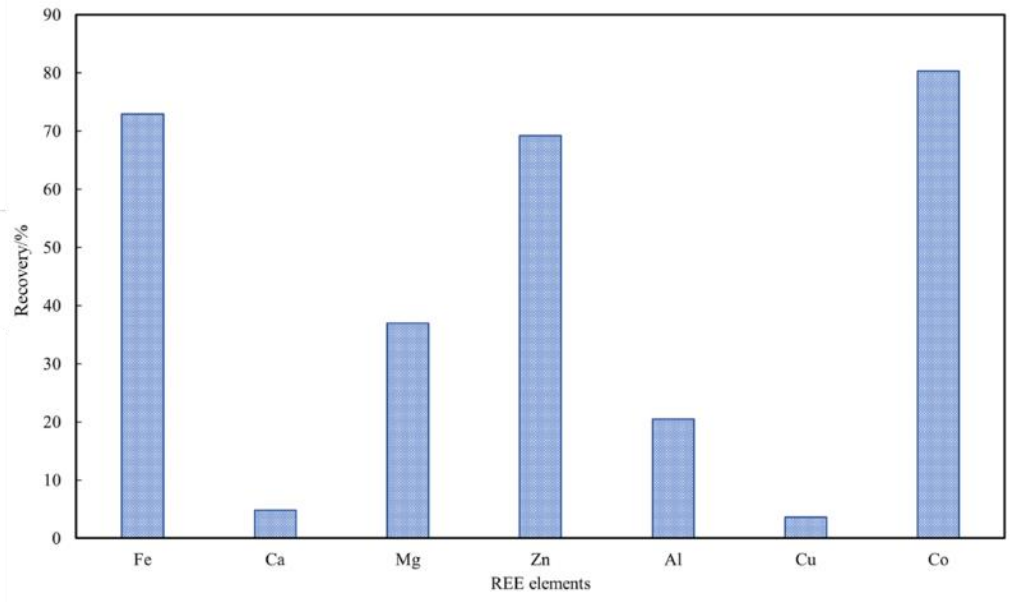



Figure 3.5.68. The yields of the main elements in the bioleaching solution.

Magnetite-free pyrite concentrate was prepared from the RC tail #2 pyrite concentrate with screening, magnetic separation, and splitting, as the pictures shown in Figure 3.5.69. The separation procedures are as follows:

1) Sample is set to dry
2) Dried sample is screened with 100 mesh
3) Magnet is used to remove magnetite
4) Step #3 is repeated multiple until no further magnetite is present in the sample
5) The final sample is splited to a homogenous composition using a splitter.

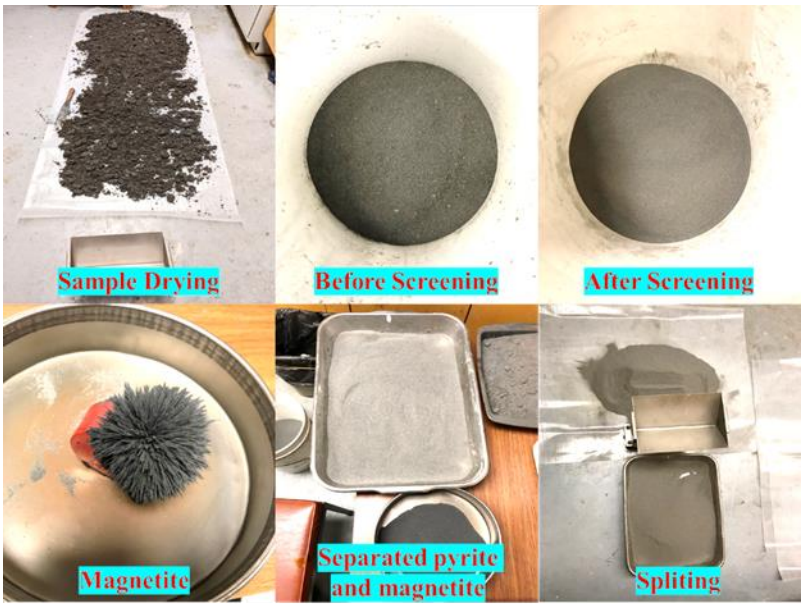



Figure 3.5.69 Magnetite-free pyrite separation procedure pictures.

The solid samples collected during the magnetic separation process including RC Tail # 2 < 100 mesh no magnetite, RC Tail # 2 > 100 mesh, RC Tail # 2 magnetite, and the samples from RC Tail #1, RC Tail # 4, and the mixture of 3 buckets were digested with aqua regia (1 g solid to 100 ml of aqua regia) and then diluted for ICP-MS analysis. Concentrations of multiple elements such as Fe and Ca were measured and plotted in Figure 3.5.70. Samples from RC Tail #1 and #4 buckets contains more Fe than Ca, while samples from RC Tail #2 and the mixture of 3 buckets have more Ca than Fe. Samples with particle size > 100 mesh have more Ca than Fe, which needed to be removed before sending to the bioreactor. Magnetite separation samples contains mostly Fe, which indicatedthe separation process was successful. Therefore, pyrite concentrate from bucket #1 and #4 may be used as the favorable feeding material after removing magnetite.

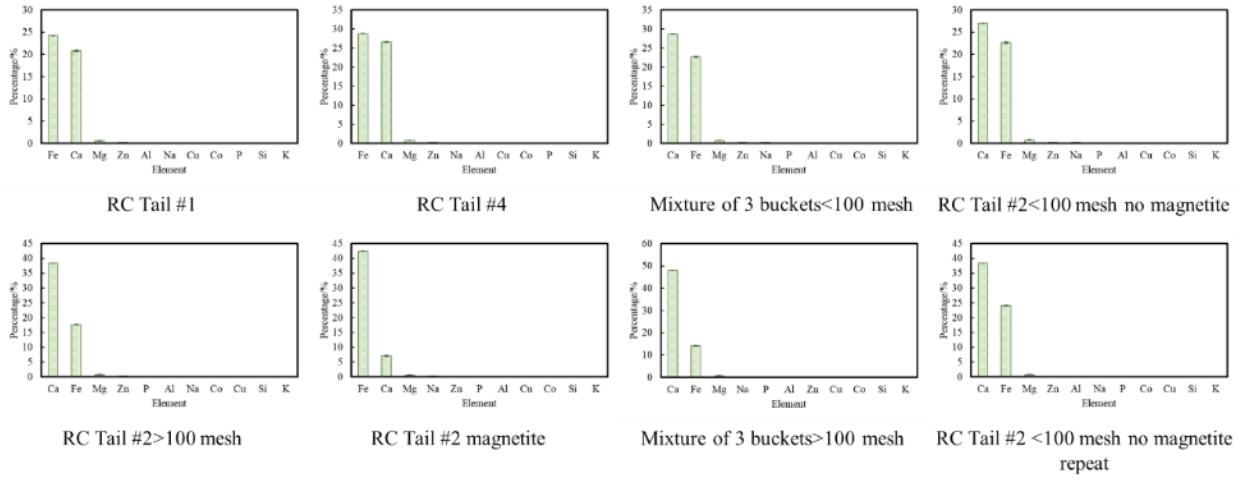



Figure 3.5.70. Element percentages in the pyrite concentrate samples measured with ICP-MS.

One bucket of coarse pyrite and one bucket of fine pyrite was received on November 24; 2020. Both of the pyrite was in slurry form. Then they were dried in the room in the room temperature. Figure 3.5.71 shows the digital image of dry coarse and fine pyrite. Also Particle size distribution analysis was been conducted to analyze the information about the size and range of particles representative of new pyrite. The experiment was done by using both of the new pyrite powder (10 g). It can be seen most of the coarse pyrite size range was in between 75 to 240 microns and fine pyrite size range is in between 40 to 75 microns. Table 3.5.9 shows the results of Particle size distribution analysis.

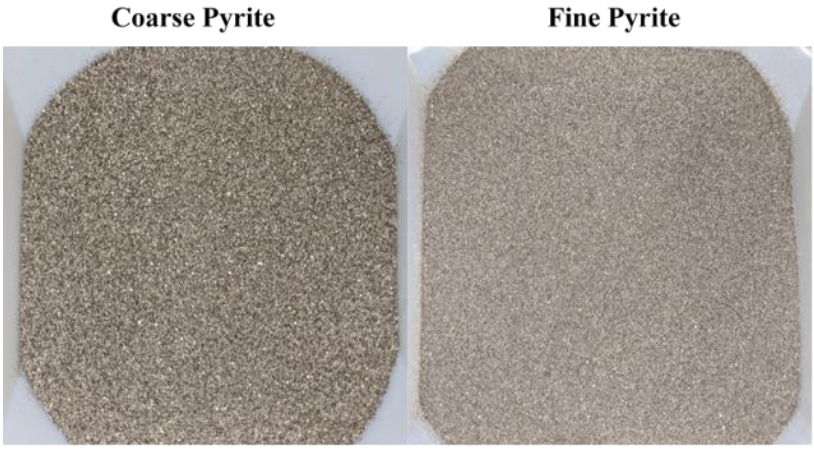



Figure 3.5.71 Digital image of dry coarse and fine pyrite.

Table 3.5.9 Particle size distribution of coarse and fine pyrite.

| Mesh Size | Coarse Pyrite(g) | Fine Pyrite(g) |
|---|---|---|
| >30 | 00 | 00 |
| >60;<30 | 0.07 | 0.026 |
| >65;<60 | 0.16 | 0.054 |
| >100;<65 | 2.88 | 0.374 |
| >200;<100 | 7.22 | 7.90 |
| >325;<200 | 0.33 | 1.18 |
| <325 | 0.08 | 0.366 |
| Total | 10g | 10g |

To determine the carbonate concentration in the mixed pyrite used before, coarse pyrite, and fine pyrite, carbonate titration was conducted with a back titration method. The procedures are as follows:

- Prepare 25 ml 0.5 M HCl
- Weigh 1.0 g pyrite concentrate
- Mix the pyrite concentrate with HCl in an Erlenmeyer flask and shake well
- Leave the mixture for 24 hrs

- Filter the mixture and collect all the solution, rinse the solid and flask with DI water, collect all the rinse solution
- Use 0.5 M NaOH solution to titrate the solution collected in last step to pH 7.0
- Based on the volume of NaOH consumed, calculate the amount of carbonate in the solid

The volume of NaOH consumed to do the titration as well as the calculated $CaCO_3$ percentage (assume all the carbonate is $CaCO_3$) in the three kinds of pyrite concentrates are shown in Table 3.5.10. It is clear that the coarse pyrite had more carbonate in it and was not suitable for the bioreactor. However, the fine pyrite contained only 12.51% of carbonate, and consumed less acid in the bioleaching process while providing sufficient $CO_2$, which made it a suitable material for bioreactor operation.

Table 3.5.10 The volume of NaOH consumed to do the titration and the calculated $CaCO_3$ percentage (assume all the carbonate is $CaCO_3$)

| | Mixed pyrite | Coarse Pyrite | Fine pyrite |
|---|---|---|---|
| Volume of NaOH/ml | 14 | 10 | 20 |
| CaCO3 percentage/% | 27.52 | 37.53 | 12.51 |

Figure 3.5.72 shows the SEM image of coarse and fine pyrite at the same magnification. From the images it can easily defined that each individual had their own micromorphological structure and size difference.

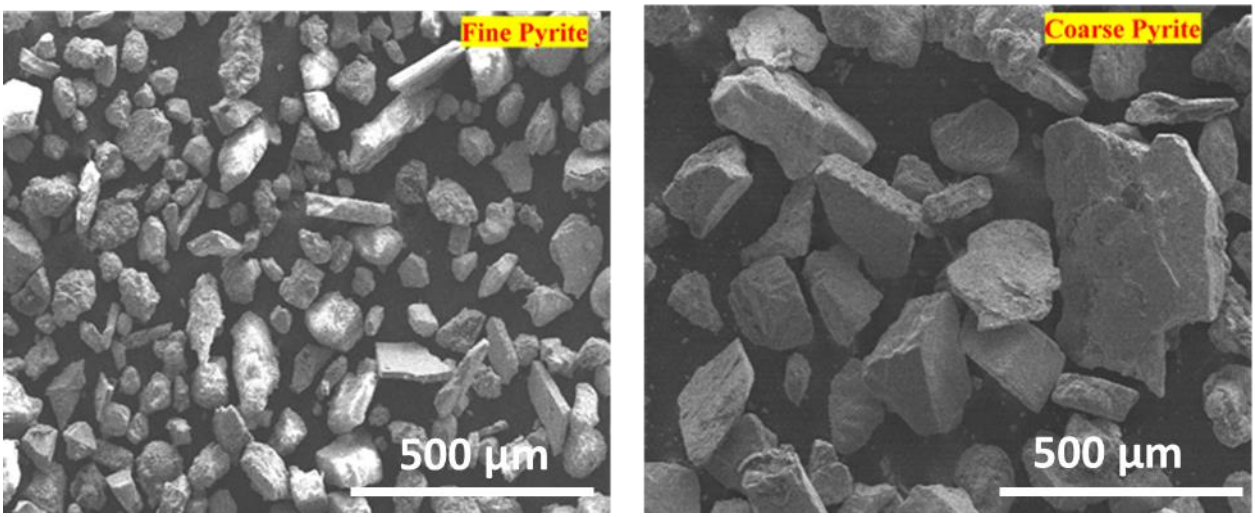



Figure 3.5.72 SEM image of Fine and Coarse pyrite at same magnification.

Figure 3.5.73 shows the resulting EDS mapping image and spectrum of fine pyrite and Figure 3.5.74 of EDS mapping results of Coarse pyrite. It was determined that fine pyrite had higher percentage of Fe and S at the same points than coarse pyrite. Trace of Al, Si , Ca and C can be easily seen in the images.

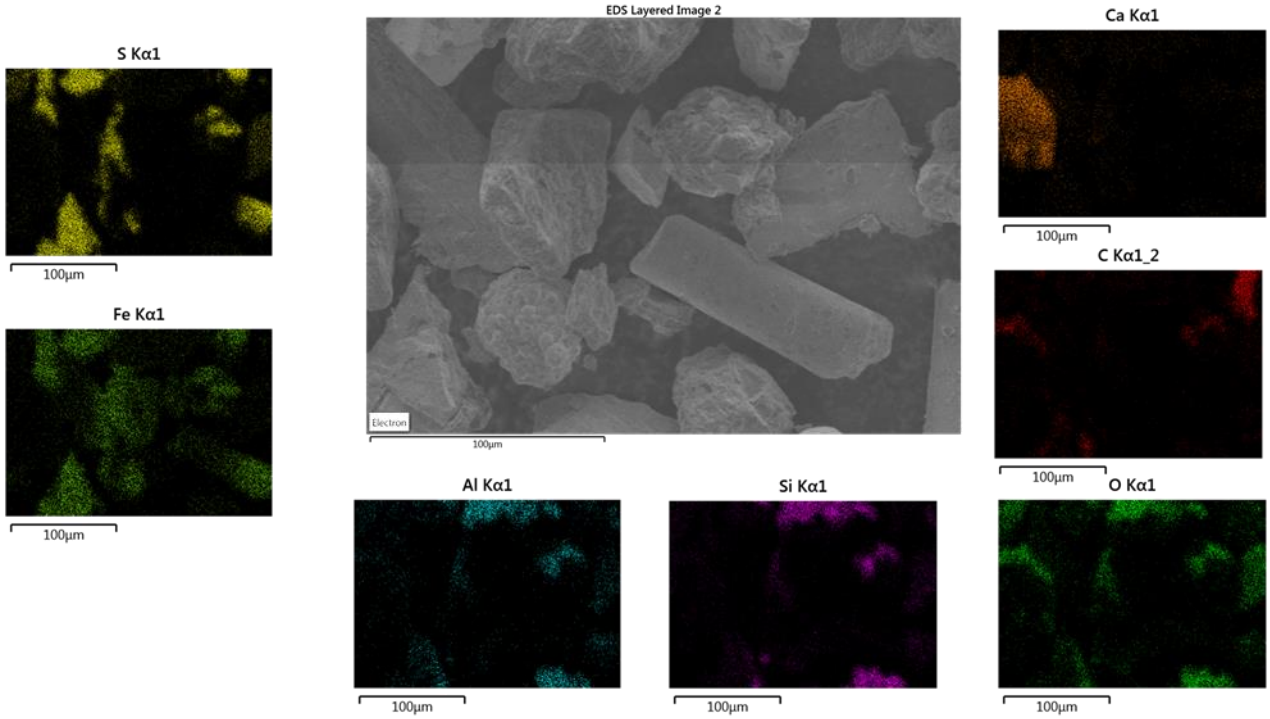



Figure 3.5.73 EDS mapping image of fine pyrite

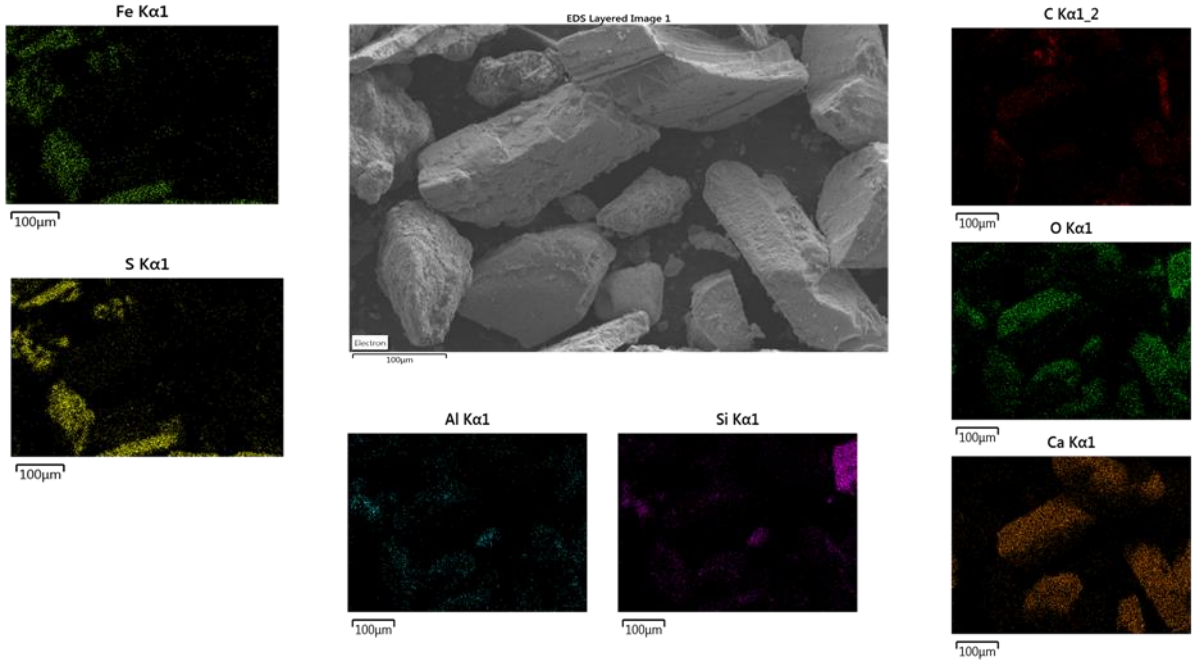



Figure 3.5.74 EDS mapping image of coarse pyrite

All the samples were digested with aqua regia (1 g solid to 100 ml of aqua regia) and then diluted for ICP-MS analysis. Concentrations of multiple elements such as Fe and Ca were measured and plotted in Figure 3.5.75, Figure 3.5.76 and Figure 3.5.77. In Figure 3.5.76 ICP digestion results of

60% pure pyrite from three different buckets has been showed for both screened and unscreened materials. All the samples contain more Fe than Ca. Also, it was observed that pyrite from bucket RC 4 had highest percentage of Fe and bucket RC 2 had the lowest percentage of Fe. Figure 3.5.76 shows the result of ICP-MS digestion result for screened and unscreened fine pyrite and coarse pyrite. It can be easily seen that screened materials had more Fe percentage compared to unscreened materials and fine pyrite had higher percentage of Fe than coarse pyrite. The other elements with fewer concentrations are Mg, Al, Zn, and Co. No Na, Si, K, Cu was detected in Fine pyrite, coarse pyrite and 60% pure pyrite. In Figure 3.5.77 20% pure pyrite ICP result has been showed. It had more Ca than Fe and also a higher percentage of alumina and silica. Figure 3.5.78 shows estimation of calcite parentage for all 12 samples.

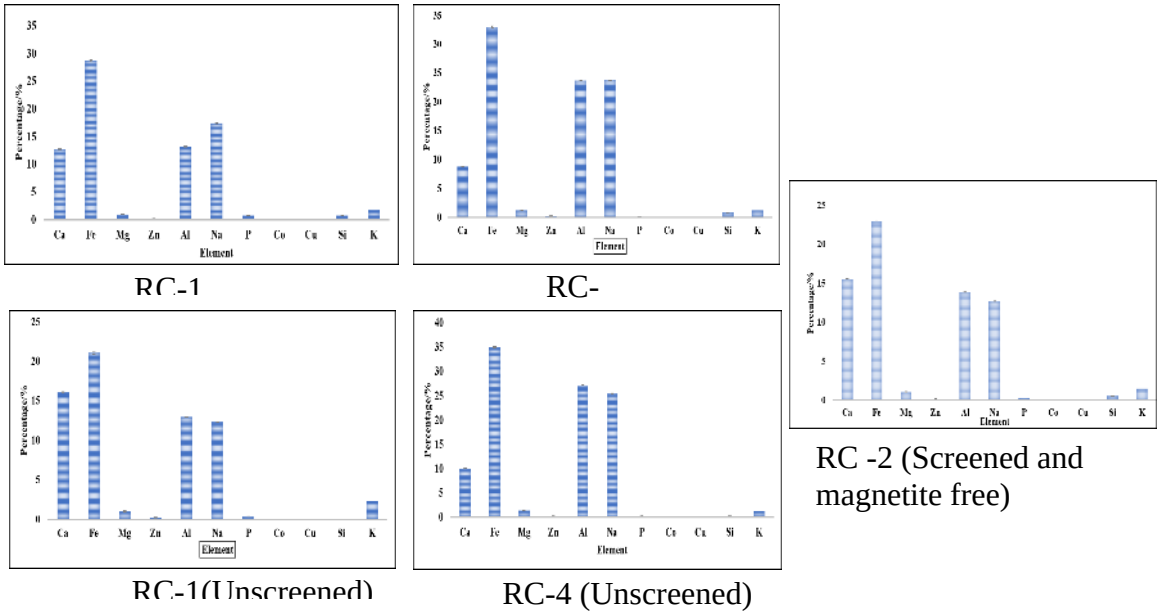



Figure 3.5.75 ICP-Ms result of 60% pure pyrite from three different buckets

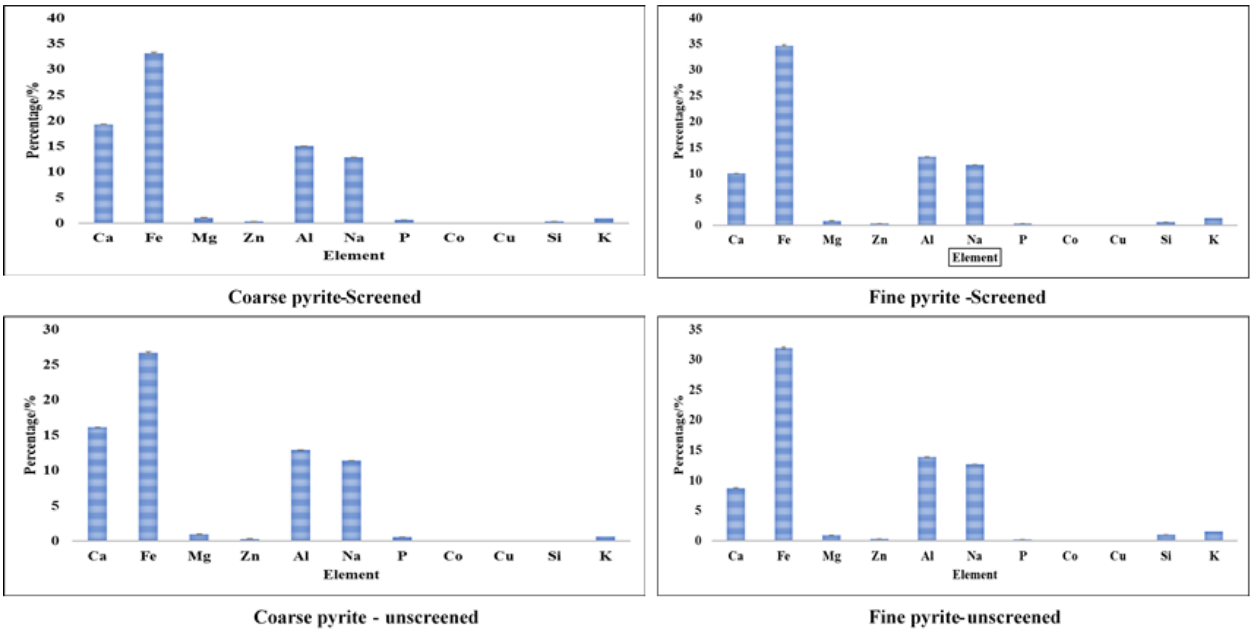



Figure 3.5.76. ICP-Ms result of coarse pyrite and fine pyrite

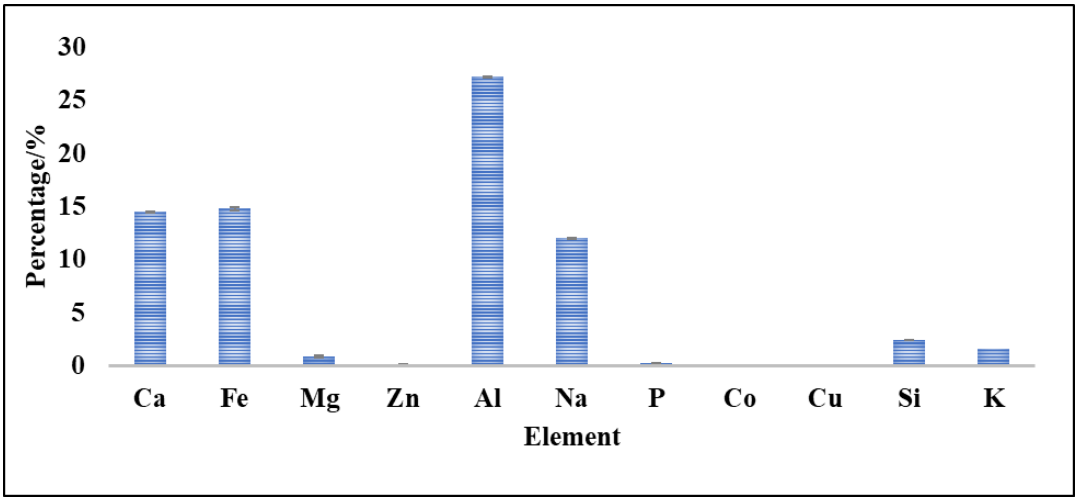



Figure 3.5.77. ICP-Ms result of unscreened 20% pure pyrite

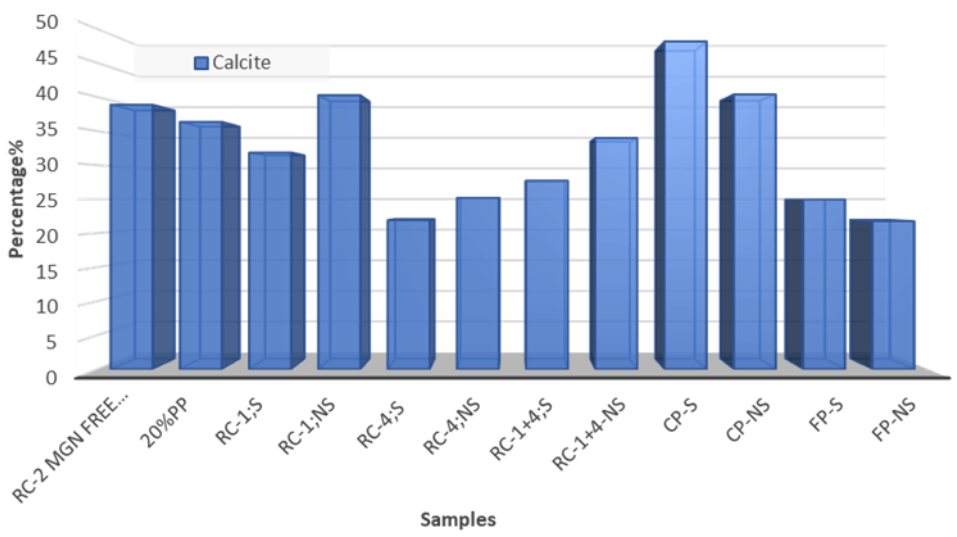



Figure 3.5.78. Comparison in Pyrite and calcite parentage for all 12 samples

**Elemental Sulfur Analysis:** Elemental sulfur analysis wasconducted to see the trace any kind of unreacted sulphur in the effluent solids. During the experiment 1 g of effluent solid was taken and then this solid was rinsed twice in pH 1.5 water and dried in 108 ℃ to do heat treatment of solid. Heat treatment helped sulphur to dissolve in carbon di sulfide solution. After the heat treatment solid was mixed with the 10ml carbon di sulfide solution and stabilize the solution for 30mins. After stabilizing solution was decanted, centrifuged and filtered. Then the solution was evaporated, and weight of the beaker was gathered before and after the evaporation. The procedures and process pictures of the elemental sulfur test are shown in Figure 3.5.79. As calculated, only 0.23% of the effluent solids is elemental sulfur, which indicates that most of the sulfur in the pyrite had been transformed into sulfate ions in the solution rather than elemental sulfur.

- Amount of effluent solid - 1 g
- Amount of carbon disulfide - 10ml
- Wight of the beaker before evaporation - 27.9570 g
- Weight of the beaker after evaporation - 27.9547 g
- Amount of residue - 0.0023 grams
- Percentage of elemental sulfur - 0.23%

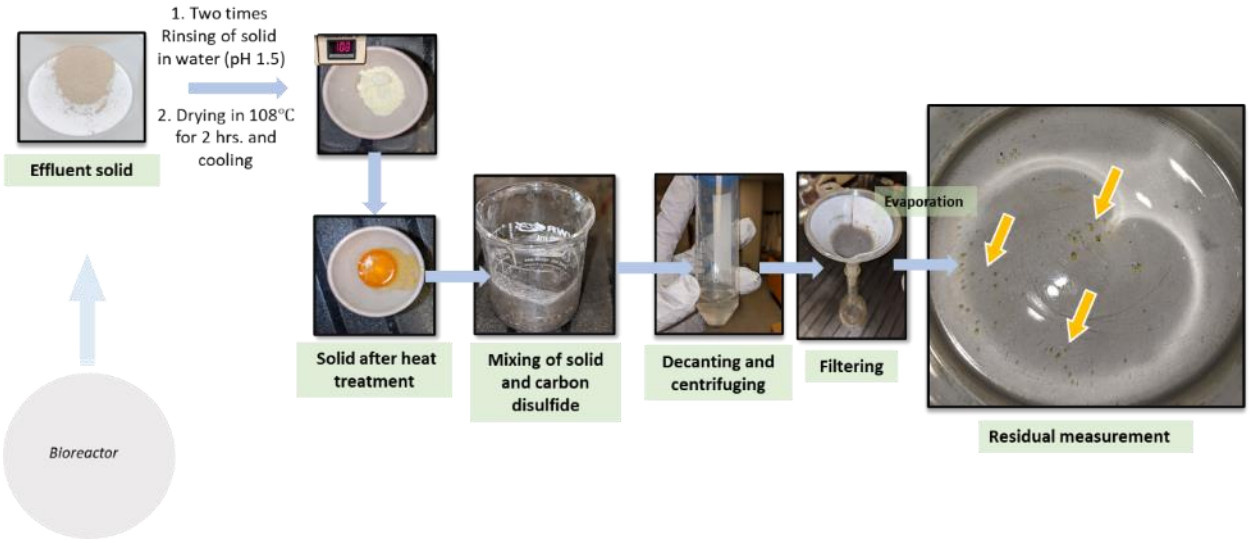



Figure 3.5.79. The procedures and process pictures of the elemental sulfur test.

### *3.5.5 The Effect of Processing Parameters and Optimization*

**Oxygen Consumption Rate Measurement**: The oxygen concentration in the input and output gas of the bioreactor was measured with a Neutronics Model OA-1 Oxygen Analyzer when the air flow rate was fixed at 1.5 L/min. The effective oxygen concentration in the air decreased with the elevation. At 5000 feet (approximately the altitude of the lab where the bioreactor was located), the effective oxygen centration is 17.3%. Therefore, the ambient oxygen percentage was set to 17.3%.

The sensor used in the OA-1 PPM Oxygen Analyzer was made of the solid-state oxygen ion conducting (electrolyte) zirconium oxide. Due to oxygen vacancies in the ceramic lattice, at temperature over 700 degrees Centigrade, oxygen ions were mobile in the solid material. When two gases of differing oxygen concentrations were on opposite sides of a zirconium oxide membrane, each side with a conductive (platinum) electrode material, a DC voltage was generated that was a function of the difference in oxygen concentration. In practice, one side of the membrane was known reference gas, typically air, and the opposite side was the unknown sample gas to be measured. The measurement was done with a zirconium oxide senser and calculated based on Eq. (1). The oxygen consumption rate, $FeS_2$ reaction rate, and the time to consume 8 g 60% pyrite were calculated with Eq. (2) – Eq. (4).

$$E = \frac{RT}{nF} \ln \frac{P1}{P2} \qquad (1)$$

Where:

$E$ = is the measured voltage.

$R$ = universal gas constant 8.314 joule/(mole· K).

$T$ = sensor temperature in K.

$n$ = number of electrons transferred (4).

$F$ = Faraday's constant 96,500 coul./mole.

$P1$ = Oxygen partial pressure in air reference.

$P2$ = Oxygen partial pressure in sample gas.

$$R_{O_2} = \frac{flow\,rate(L/min)\times(O_2\%_{inlet}\text{-}O_2\%_{outlet})}{22.4(L/mol)}\times\frac{60min}{hr} \quad (2)$$

$$R_{FeS_2} = \frac{4}{15}R_{O_2} \quad (3)$$

$$t = \frac{8\times 60\%}{120\times R_{FeS_2}} \quad (4)$$

The oxygen partial pressure P1 in this equation was set to the partial pressure when the effective oxygen concentration is 17.3%. As can be seen in Figure 3.5.80, the oxygen concentration in the input gas is 18.0%, while the oxygen concentration dropped to 16.8% in the output gas, which means 1.2% of the oxygen in the air is consumed. This measurement was done just after automatic feeding occurred.




(a) (b)

Figure 3.5.80. Measured oxygen concentration in the (a) input gas and (b) output gas.

Based on these data, the oxygen consumption rate can be calculated:

$$R_{O_2} = \frac{1.5L/min\times(18.0\%\text{-}16.8\%)}{22.4L/mol}\times\frac{60min}{hr} = 0.0482mol/hr$$

The consumption of every 4 mol of $FeS_2$ needs 15 mol of oxygen, therefore, the $FeS_2$ consumption rate can be estimated:

$$R_{FeS_2} = \frac{4}{15} R_{O_2} = \frac{4}{15} \times 0.0482 mol/hr = 0.0128 mol/hr$$

Therefore, if the oxygen consumption rate was consistent in the bioreactor, the $FeS_2$ in 8g of 60% pyrite should be consumed in 3.125 hr. However, the actual situation was more complex considering the variation of the oxygen concentration and the varied reaction rate due to the shrinking of the solid particles.

**Effect of Operating Parameters on Ferrous Oxidation Rate:** The effect of the stirrer rotation speed and the air flow rate on the biooxidation performance was studied by measuring the ferrous oxidation rate. First, the effect of the rotation speed was investigated by measuring the ferrous oxidation rate under different RPM. The results of this test are shown in Figure 3.5.81. It is clear that when the RPM increased from 200 to 400, the ferrous oxidation rate increased dramatically. However, as the RPM increased from 400 to 700, the rate did not change a lot. Therefore, with the gas flow rate of 1.5 L/min, the optimized rotation speed should be 400-500 rpm.

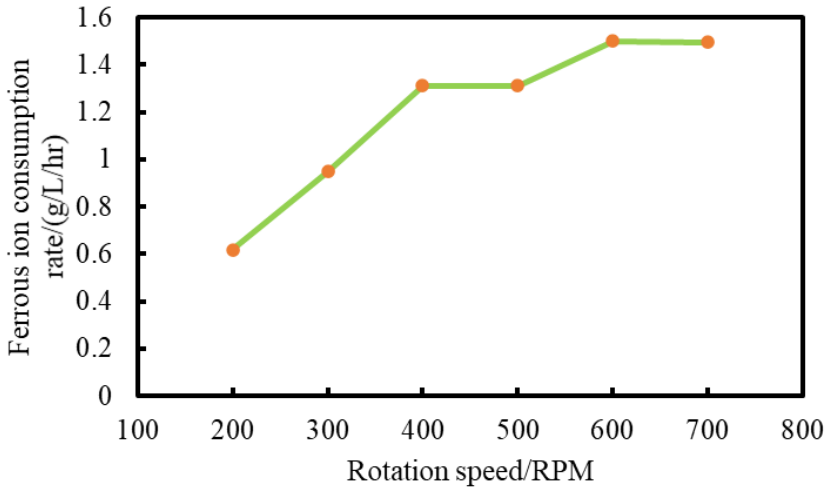



Figure 3.5.81. Ferrous ion consumption rate vs. rotation speed.

The other detailed study was conducted with the response surface design method to find the best combination of the gas flow rate and the rotation speed. The experimental design generated with Minitab is shown in table below.

Table 3.5.11 Response Surface Design-2 factors-uncoded.

| StdOrder | RunOrder | PtType | Blocks | Stirring Speed/RPM | Gas Flow Rate/(L/min) |
|---|---|---|---|---|---|
| 12 | 1 | 0 | 1 | 500 | 1.5 |
| 10 | 2 | 0 | 1 | 500 | 1.5 |
| 7 | 3 | -1 | 1 | 500 | 0.79 |
| 3 | 4 | 1 | 1 | 400 | 2 |
| 1 | 5 | 1 | 1 | 400 | 1 |
| 5 | 6 | -1 | 1 | 358.58 | 1.5 |
| 9 | 7 | 0 | 1 | 500 | 1.5 |
| 6 | 8 | -1 | 1 | 641.42 | 1.5 |
| 11 | 9 | 0 | 1 | 500 | 1.5 |
| 8 | 10 | -1 | 1 | 500 | 2.21 |
| 13 | 11 | 0 | 1 | 500 | 1.5 |
| 2 | 12 | 1 | 1 | 600 | 1 |
| 4 | 13 | 1 | 1 | 600 | 2 |

The contour plot, surface plot, the main effect plots are shown in Figure 3.5.82. With the two parameters studied, the optimized gas flow rate and stirring speed were 1.5 L/min and 550 rpm, respectively.

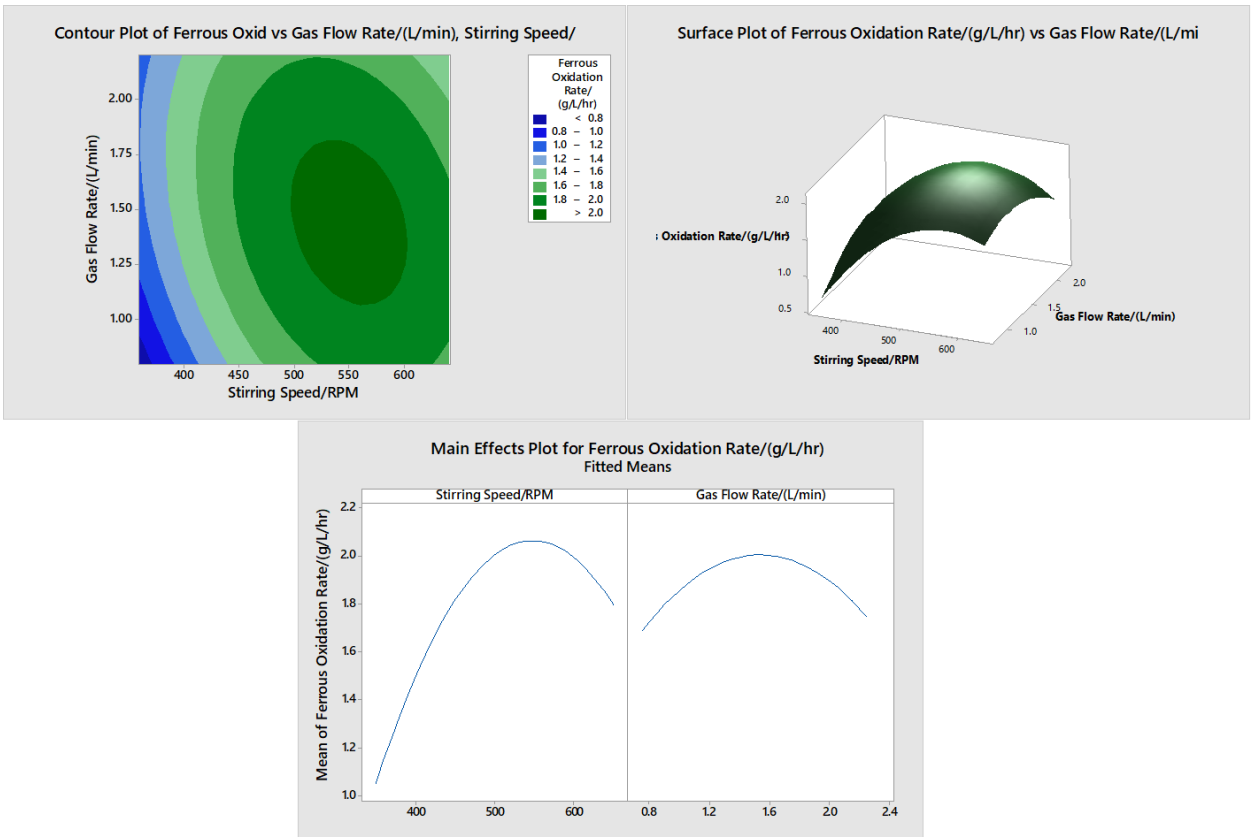



Figure 3.5.82. Effects of the gas flow rate and stirring speed on the ferrous oxidation rate.

To observe the changes in the bioleaching solution temperature due to the diverse air flow rate this experiment was conducted in the 1.5 L reactor. During experiment, six different air flow rates was used and the bioleaching solution temperature was measured by the laboratory thermometer. From the Figure 3.5.83, it can be seen that the solution temperature decreased with the increase of the air flow rate. Therefore, to give the bacteria the best condition (i.e. 35℃), the temperature of the water bath had to be adjusted accordingly.

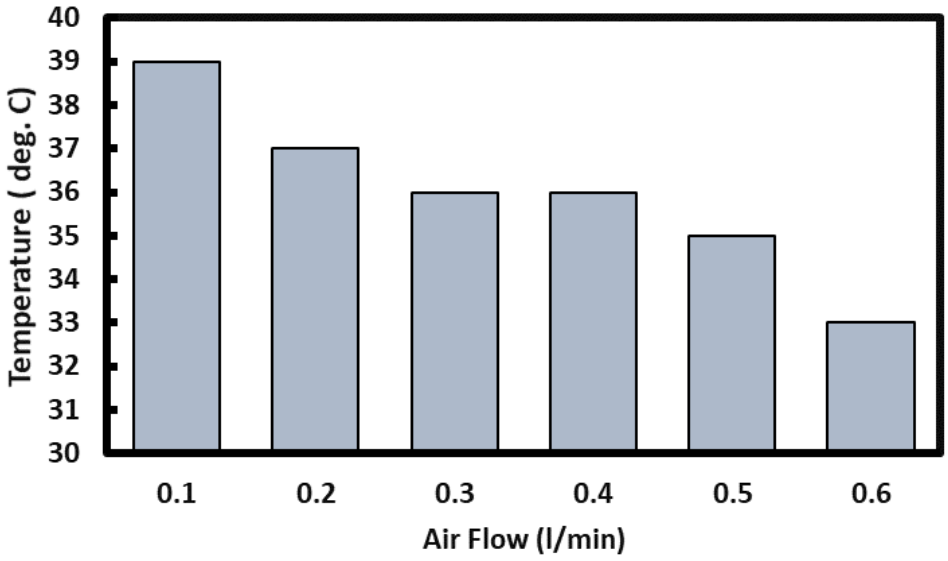



Figure 3.5.83. Effect of Air flow rate on the bioleaching solution temperature.

Response surface design is a way of presenting the data primarily visualizing the research findings. For instance, if there are two independent variables rotation speed and airflow rate and, in such case, to find the effect of that on biooxidation rate, the associated visual representation is called response surface methodology. The response surface was analyzed in Minitab. The primary reason to do the response surface was to see not just one number of rotation speed and or air flow rate was the most suitable instead the authors want to evaluate a range of regions where rotation speed or air flow rate would be more suitable. Figure 3.5.84 shows the contour plot and surface plot for the design for the 1.5 L reactor. It can be seen from the figure that that rotation speed from 650 to 750 and airflow rate from 0.3 L/min to 0.4 L/min looked optimum so that the rotation speed could be set in between 650 rpm to 750 rpm and airflow rate in between 0.3 L/min to 0.4 L/min and there will be no significant change in the comfort level. Such information suggested that 700rpm was significantly better than the other rotation speed and 0.35 L/min was significantly better than other air flow rate in present experimental settings.

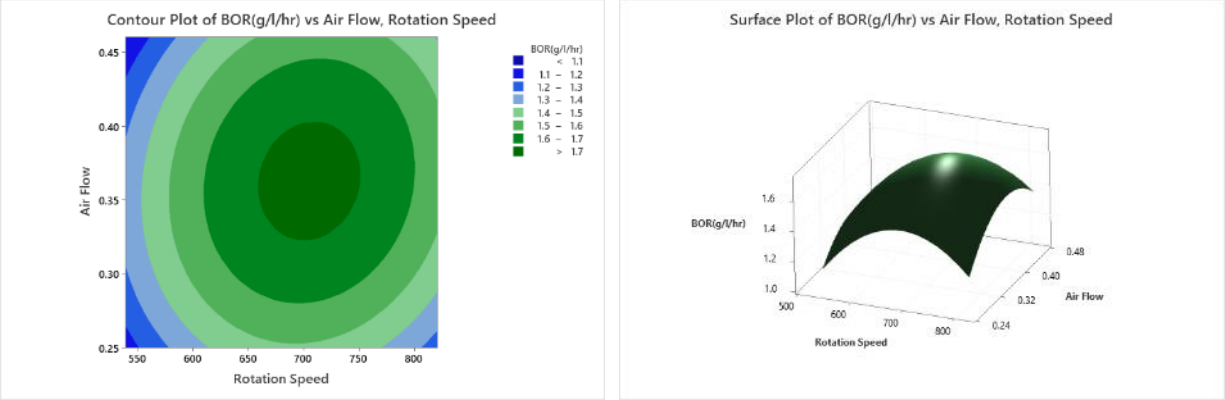



Figure 3.5.84. Contour plot and Surface plot of central composite design.

**Effect of Operating Parameters on the Oxygen Consumption Rate:** The effect of gas flow rate on the oxygen consumption were studied by measuring the oxygen concentration in the inlet and outlet gas with the Cerion OA-1 oxygen analyzer. The oxygen concentration in the inlet gas was measured as 20.3%. The oxygen concentrations measured and the oxygen consumption rate, $FeS_2$ reaction rate, and the time to consume 8 g 60% pyrite under different gas flow rate is shown in Table 3.5.12.

Table 3.5.12 The oxygen concentration in the outlet gas and the calculated consumption rate.

| Gas flow rate/(L/min) | Oxygen content in outlet gas/% | Oxygen consumption rate/(mol/hr) | $FeS_2$ consumption rate/(mol/hr) | Time to consume 8 g pyrite concentrate/hr |
|---|---|---|---|---|
| 0.8 | 18.5 | 0.0386 | 0.0103 | 3.89 |
| 1.5 | 18.8 | 0.0603 | 0.0161 | 2.49 |
| 2 | 19.3 | 0.0536 | 0.0143 | 2.80 |
| 2.5 | 19.8 | 0.0335 | 0.0089 | 4.48 |
| 3 | 20 | 0.0241 | 0.0064 | 6.22 |

The oxygen concentrations in the inlet and outlet gas are shown in Figure 3.5.85. The oxygen consumption rate and the time to consume 8 g 60% pyrite under different gas flow rate is shown in Figure 3.5.86. As it shows, the oxygen concentration in the outlet gas increased with the gas flow rate, while the oxygen consumption rate did not follow a monotonic increasing or decreasing trend. An optimized value of the oxygen consumption rate is shown when the gas flow rate was 1.5. In this case, only the effect of gas flow rate was studied. To get an optimized value for both the gas flow rate and the stirring speed, a response surface design experiment was conducted with the Minteq software.

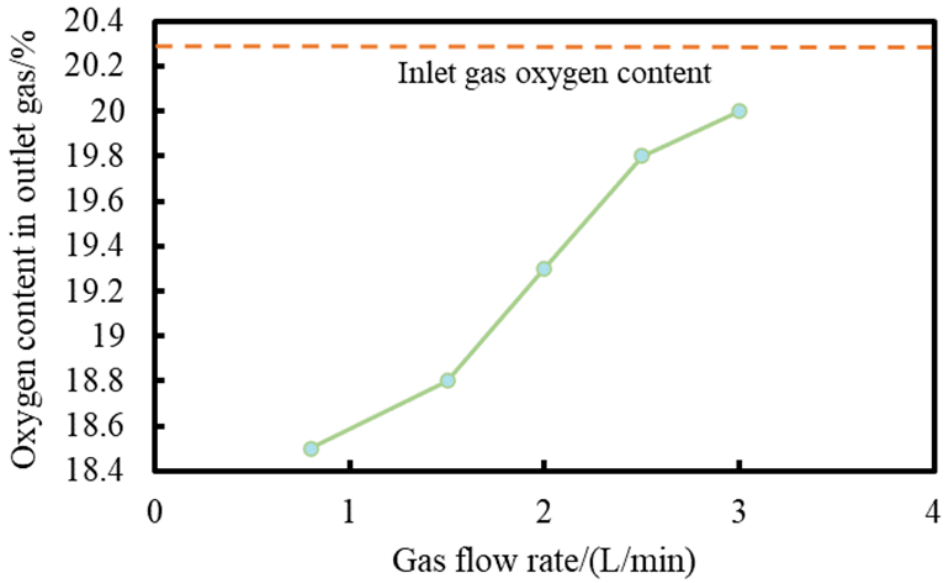



Figure 3.5.85. The oxygen concentrations in the inlet and outlet gas under various gas flow rate.

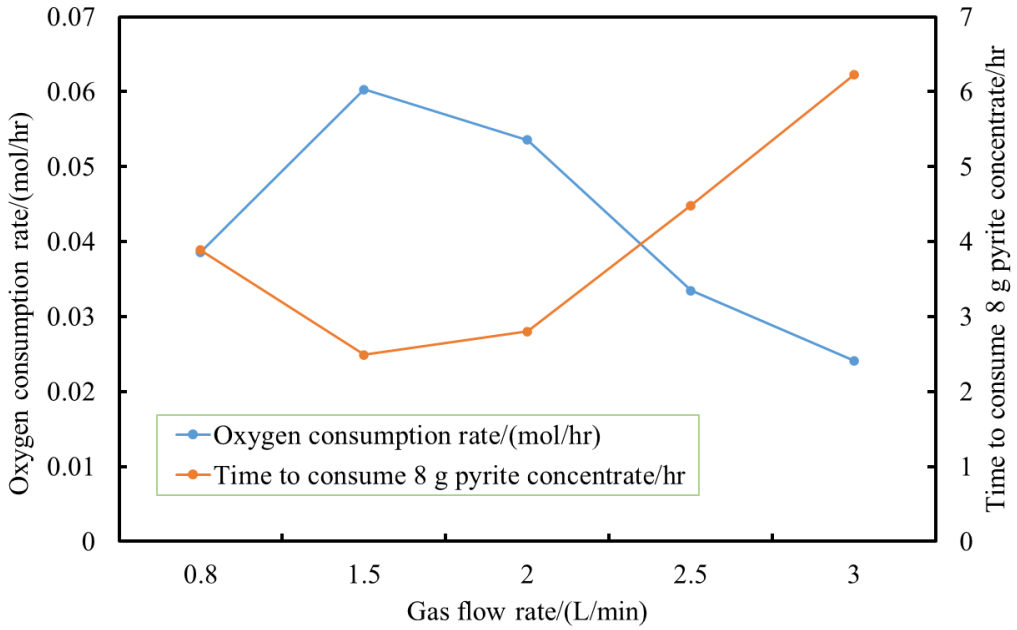



Figure 3.5.86. The oxygen consumption rate and the time to consume 8 g 60% pyrite under different gas flow rate.

The designed experiments are shown in table below. Total 13 tests were designed and conducted under different combinations of the gas flow rate and stirring speed.

Table 3.5.13 Response surface design experiments parameters on the oxygen consumption rate.

| StdOrder | RunOrder | PtType | Blocks | Stirring Speed/RPM | Gas Flow Rate/(L/min) | Oxygen consumption rate/(mol/hr) |
|---|---|---|---|---|---|---|
| 12 | 1 | 0 | 1 | 500 | 1.5 | 0.048214 |
| 10 | 2 | 0 | 1 | 500 | 1.5 | 0.044196 |
| 7 | 3 | -1 | 1 | 500 | 0.792893 | 0.019117 |
| 3 | 4 | 1 | 1 | 400 | 2 | 0.026786 |
| 1 | 5 | 1 | 1 | 400 | 1 | 0.008036 |
| 5 | 6 | -1 | 1 | 358.5786 | 1.5 | 0.008036 |
| 9 | 7 | 0 | 1 | 500 | 1.5 | 0.008036 |
| 6 | 8 | -1 | 1 | 641.4214 | 1.5 | 0.008036 |
| 11 | 9 | 0 | 1 | 500 | 1.5 | 0.004018 |
| 8 | 10 | -1 | 1 | 500 | 2.207107 | 0.005912 |
| 13 | 11 | 0 | 1 | 500 | 1.5 | 0.004018 |
| 2 | 12 | 1 | 1 | 600 | 1 | 0.002679 |
| 4 | 13 | 1 | 1 | 600 | 2 | 0.005357 |

The contour plot and the surface plot of the optimization test are shown in Figure 3.5.87. It is clear that there was one optimized oxygen consumption rate under one certain condition. This test showed the direction of optimization on the operating parameters.

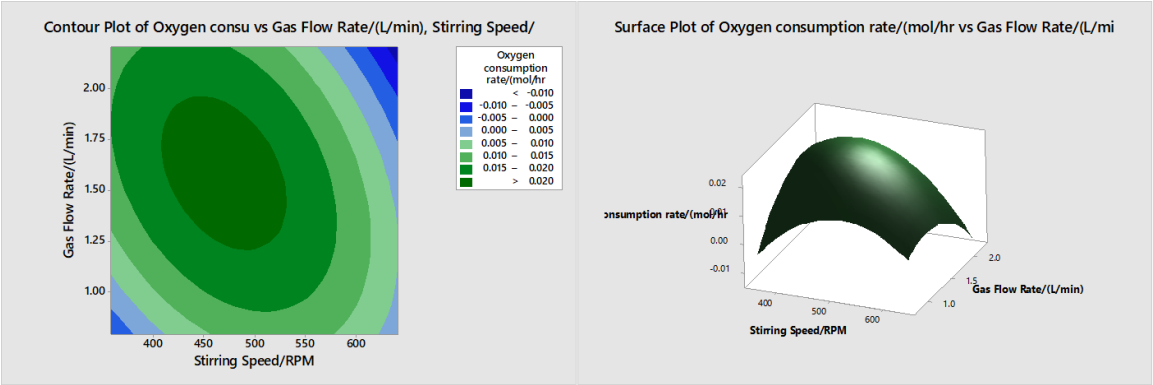



Figure 3.5.87. The contour plot and the surface plot of the optimization test.

The interaction plot for oxygen consumption rate and the main effects plot for oxygen consumption rate are shown in Figure 3.5.88. These plots show how the oxygen consumption rate changed with one operating parameter when the other one parameter was fixed. Based on this study, the optimized value of the stirring speed was around 500 RPM and the optimized value of the gas flow rate was around 1.5 when the maximized parameter was oxygen consumption rate.

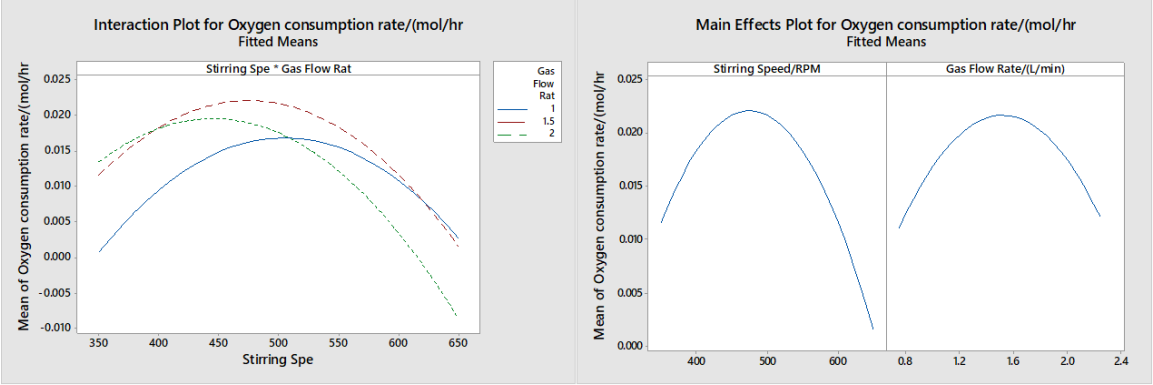



Figure 3.5.88. The interaction plot and the main effects plot for oxygen consumption rate.

### *3.5.6 Estimation of the Acid Generation Rate*

**Titration Test on the Bioleaching Solution:** Titration test was conducted through out the test to determine the acid concentration in the leaching solution. The test procedures are listed below.

(1) Prepare 0.1 M NaOH solution by dissolving NaOH pellet into DI water.
(2) Take 20 ml of the leaching solution from the bioreactor and filter it to remove solid particles.
(3) Prepare a burette and fill it with the 0.1 M NaOH solution, record the initial reading on the burette.
(4) Put 10 mL of the filtered leaching solution to an Erlenmeyer flask and put it under the burette.
(5) Open the valve of the burette and start titration, track the pH of the solution with a pH meter till the pH reading reach 2.5, close the valve.
(6) Record the final reading on the burette.

Test 1

With this procedure, one titration test was conducted. The acid concentration was determined with the calculations shown below.

$C_{NaOH}$= 0.1 M

$V_{NaOH}$= 32.3 ml

$V_{Acid}$= 10 ml

We know,

$C_{NaOH} \cdot V_{NaOH} = C_{Acid} \cdot V_{Acid}$

$0.1M \times 32.3mL = C_{Acid} \times 10mL$

$C_{Acid} = 0.323M$

With the activity coefficient of H+ calculated in the previous section ( $\gamma_{H+}$=0.295), the initial pH of the solution can be back calculated:

$pH = -\log(\gamma_{H+} \cdot C_{Acid}) = -\log(0.295 \times 0.323) = 1.116$

which is very close to the initial pH measured with the pH meter. This test showed titration was a very reliable method to determine the acid concentration in the leaching solution.

Test 2

The pH change during the titration process is shown in Table 3.5.14.

Table 3.5.14 pH values during the titration process.

| pH | Added NaOH volume/ml |
|---|---|
| 1.33 | -- |
| 1.47 | 10 |
| 1.68 | 10 |
| 1.87 | 10 |
| 2.04 | 10 |
| 2.18 | 10 |
| 2.30 | 22 |

The acid concentration can be determined with the calculations shown below:

$C_{NaOH}$ = 0.1M

$V_{NaOH}$= 72ml

$V_{Acid}$= 30ml

We know,

$C_{NaOH} \cdot V_{NaOH} = C_{Acid} \cdot V_{Acid}$

0.1M×72 mL= $C_{Acid}$×30mL

$C_{Acid}$ =0.24 M

Test 3

The other titration test was conducted to determine the acid concentration in the leaching solution. The acid concentration can be determined with the calculations shown below. pH of the bioleaching solution is 1.16. Acid concentration can be determined with the calculations shown below.

$C_{NaOH}$= 0.1 M

$V_{NaOH}$= 42 ml

$V_{Acid}$= 10 ml

We know,

$C_{NaOH} \cdot V_{NaOH} = C_{Acid} \cdot V_{Acid}$

0.1M×42mL= $C_{Acid}$×10mL

$C_{Acid}$ =0.42M

The changes in the color of the bioleaching solution is shown in Figure 3.5.89.

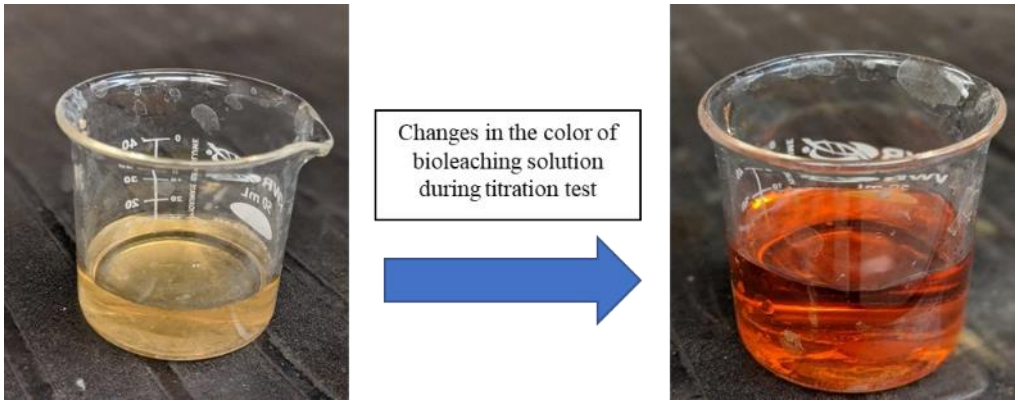



Figure 3.5.89. Changes in the color of the bioleaching solution during the titration process.

**Mass Balance Test and Calculations:** Mass balance was conducted during an 8.5-day running period. During this period of time, materials were used (in regular feeding) into the bioreactor and taken out of bioreactor twice a day. Considering the volume of the solution in the reactor, the residence time was 12 days. The feeding material was made by mixing 8 g of pyrite with 50 ml 9K medium (no Fe, no acid) and 50 ml DI water. Therefore, the actual solid percentage is about 7.4%.

Calculation of actual solid percentage:

Solid percentage = weight of individual component/total weight of solid

= (8)/(50+50+8)=8/108

=7.407

Hence, for other volumes, total % of pyrite or solid can be calculated. Considering 7.407% of solid, leftover feeding slurry of 1250 ml should contain:

$x/(x+1250) = 0.07407$

or x=100 gram of solid

However, during filtration for leftover slurry, only 67.65 g of pyrite/solid was found. It suggests that:

100-67.5=32.5 grams of extra solid were consumed in feeding. This solid must be in solution in bioreactor.

Corrected feeding solid: $(8\times17+32.5)/ (1700 +8\times17+32.5) =168.5/ (1700 +168.5)$

: 168.5/1868.5

: 9.017%

The weights of the input material and output material were evaluated for a mass balance tracking. First, the leftover feeding slurry was filtered to get 1250 ml liquid and 67.65 g of pyrite concentrate solid. In 1250 ml liquid, there should be 100 g pyrite, which meant we were feeding more than 7.4%. Corrected feeding slurry solid percentage is 9.017%. During this 8.5-day running, 1700 ml

of the feeding slurry was added to the reactor. The effluent was collected and filtered to get 1650 ml liquid (weights 1699 g, density: 1.03 g/ml) and 129.04 g solid. The solid percentage is 7.0%.

Mass balance:

- Input weight= solid weight + liquid weight=1700*9.017%+(1700-1700*9.017%/5) (at this step the density of solid was considered ~ 5)

=1700+153.28-30.65=1822.62

- Output: =1699+129.04=1828.04 g

Error = |(1828.04-1822.62)|/1822.62*100%=0.29%

We have also calculated the errors for the different range of density (range 4-6) and possible change in error, as shown in Figure 3.5.90.

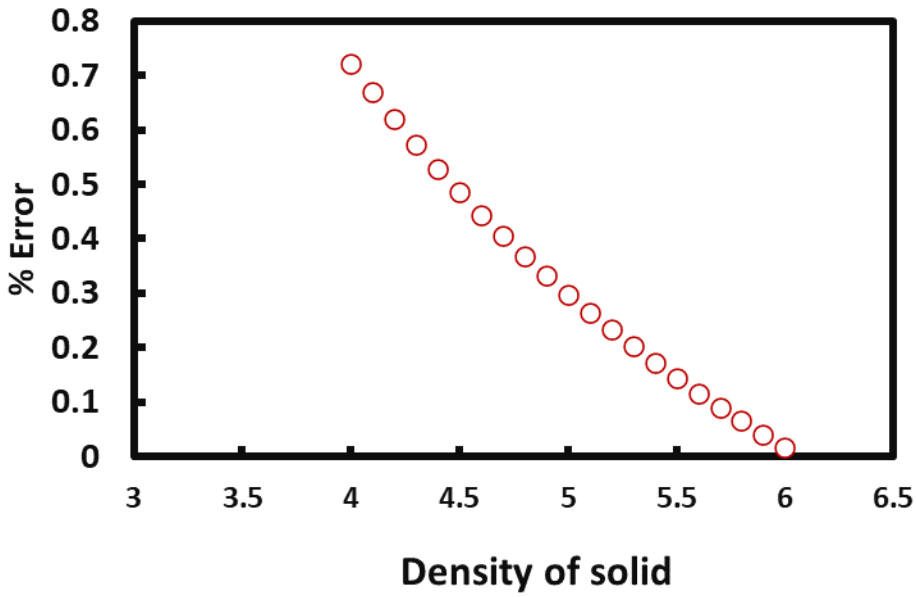



Figure 3.5.90. The errors for the different range of density (range 4-6).

It can be seen that at higher density of solid, % error will be reduced. At this stage, the solid composition and phase is not confirmed.

Note that the bacteria interacted directly with the mineral (in presence case, $FeS_2$), possibly by the extracellular secretion of an enzyme or by oxidation with an enzyme specific to sulfide minerals present on the cell wall. At longer reaction times, the mineral surfaces were covered with large amounts of extracellular polymeric substances and bacteria, indicating that significant numbers of bacteria were attached to the surfaces. Dissolution reactions are dependent on the amount of surface area that is available. However, as dissolution proceeds, both the particle size and the surface area are reduced.

Aspect of pH /Acid generation:

Total of 1650 ml of solution was collected.

The concentration $H^+$ can be calculated based on the pH and activity coefficient as follows:

$$C_{H+} = \frac{10^{-pH}}{\gamma_{H^+}} = \frac{10^{-1.33}}{0.434} = 0.107M$$

Here we have calculated the concentration, considering, average pH = 1.33, as shown in Figure 3.5.91.

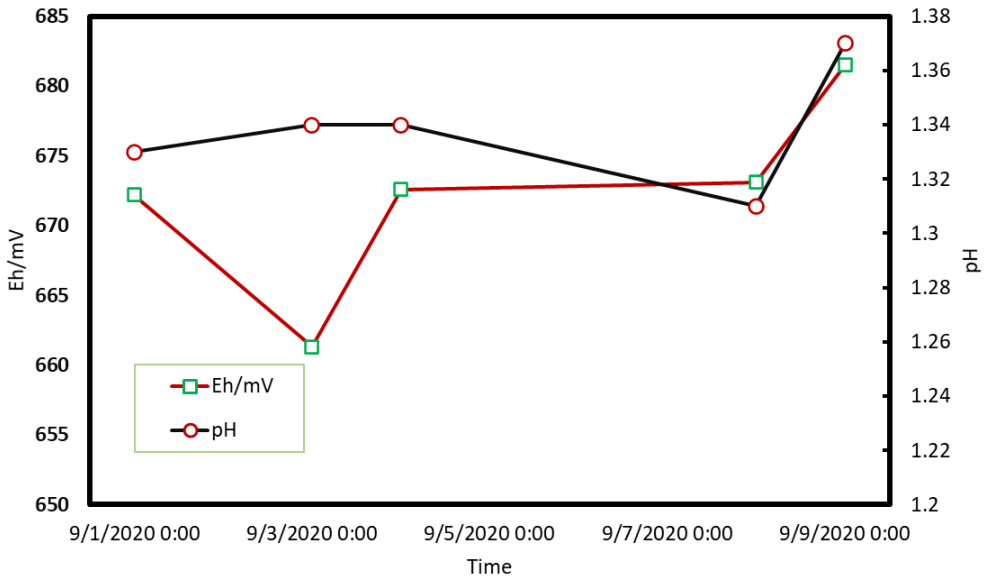



Figure 3.5.91. pH and Eh during the test.

During this operation period of the bioreactor with automatic feeding and effluent collection, the total of 168. 5 grams of solid was added and 1650 ml of solution was collected. The acid generation rate and efficiency was estimated based on the mass balance of the main input and output materials. So, the acid generation rate (amount of H+ generated per day) is:

0.107 mol/L × 1.65 L= 0.17655 mol

Rate=0.02077 mol/day (considering 8.5 day of operation/collection).

Mass Balance from Stoichiometry and amount of pyrite dissolved:

Assuming the overall reaction of the bioleaching process is:

$$4FeS_2 + 2H_2O + 15O_2 = 2H_2SO_4 + 2Fe_2(SO_4)_3$$

This reaction suggests 4 moles of $H^+$ can be generated from 4 moles of pyrite assuming complete dissociation. Hence, for 0.17655 mole of hydrogen, similar amount of pyrite will be consumed. This was equivalent to ~ 0.1765×119.98 ~ 21.056 grams of pyrite consumption. However, it was clear that ~ 168.5 gram of pyrite was added. Hence, the left-over pyrite (unreacted will be):

Pyrite (left-over/unreacted) = Initial pyrite-pyrite consumed (estimated from $H^+$ generation)

=168.5-21.05

=147. 5 grams

However, it can be seen that ~ 130 grams of solid was measured in effluent. The possibility of relatively less amount of left-over solid can be due to presence of other soluble impurities, or pyrite may be in suspended form in effluent. In case, consideration of soluble impurity can be used to estimate pyrite %:

Pyrite purity =130/147

~88%

It is also interesting to note that in case of consideration of pyrite dissolution without bacteria, relatively less amount of pyrite was dissolved.

**Aspect of jarosite formation:** At this stage, aspects of jarosite formation are not considered, as Jarosite formation occurs mainly at relatively higher pH range.

### *3.5.7 Composition Analysis of the Pyrite Concentrate and $H_2SO_4$ Consumption by $CaCO_3$*

To build a tight mass balance of the bioleaching solution, it is critical to determine the composition of the feeding material and the materials come out of the bioreactor. The back-titration method was used to measure the calcite percentage in the pyrite concentrate.

The procedures are listed below:

- Prepare 25 ml 0.5 M HCl solution
- Put 1 g pyrite concentrate in the HCl solution, wait for 30 min to allow the reaction between HCl and $CaCO_3$ complete.
- Filter the solution
- Use 0.5 M NaOH solution to titrate the solution to pH 7.0.
- Measure the volume of NaOH consumed.

After mixing the pyrite concentrate with the HCl solution, lots of bubbles were observed. This should be related to the reaction between $CaCO_3$ and HCl then the formation of $CO_2$.

Volume of 0.5 M NaOH consumed: 16 ml.

$n_{HCl\text{-}consumed}$=(25ml-16ml) $\times 10^{-3} \times$0.5 mol/L=0.0045 mol

$CaCO_3+2HCl=CaCl_2+H_2O+CO_2$

$m_{CaCO3}$=0.0045mol/2$\times$100.09 g/mol=0.225g

Percentage of $CaCO_3$ in pyrite concentrate:

0.225g/1g$\times$100%=22.5%

Based on the 22.5% calcite percentage in the pyrite concentrate, in each feeding, if all the $CaCO_3$ is reacted with $H_2SO_4$, amount of acid consumption is:

$$M_{H^+ consump} = \frac{8\,g \times 22.5\%}{157.94\,g/mol} \times 2 = 0.023 mol$$

### *3.5.8 Jarosite Formation Tests*

**Jarosite Formation Testing by Adding reagents:** To study the possible approach of removing Fe in the form of jarosite from the bioleaching solution, a series of tests were conducted to study the effect of various chemicals and their concentration on the removal effect. Test procedures included: 1) Sedimentation and decantation of 1 L of bioleaching solution from the new reactor; 2) Evaluation the effect of oxidation and iron addition (~ 300 ml solution sample was taken, that was oxidized to about 670 mV, then 10 g/l of ferrous sulfate was added, and additional oxidation was performed to raise the Eh to 670 mV. Because no precipitate formed, 10 g/l of additional ferrous sulfate was introduced and the solution was oxidized to achieve an Eh of 670 mV. The process was repeated until some jarosite formation was observed. 3) Evaluation of sodium and ammonium salt additions: After observation of precipitate formation from part 2, the solution was decanted and 40 ml of solution was placed in a series of test tubes to which different amounts of sodium sulfate (2 g/40 ml, 4 g/40 ml, 6 g/40 ml) and ammonium hydroxide (1 ml/40 ml, 2 ml/40 ml, 4 ml/40 ml, 6 ml/40 ml) were added.

In the case of sodium sulfate addition, no precipitate was formed after 24 hr. even when 6 grams of sodium sulfate were added (see Figure 3.5.92). However, a very interesting phenomenon was observed when these samples were left for 40 days. As can be seen from the picture of the test tube with 6 g of $Na_2SO_4$ addition after 40 days, a large amount of precipitate with an amber yellow color of jarosite formed and clearly, the color the of the solution became lighter compared with the sample after 24 hr., indicating a huge Fe concentration reduction in the sample. Therefore, the addition of sodium sulfate might be an effective was to remove the iron in the bioleaching solution in the form of jarosite. However, the reaction rate for this approach was not clear and more tests shown be done to determine the best condition of Fe removal from the solution. The addition of ammonium hydroxide. Pictures of the test solutions and the Eh/pH values after sedimentation are shown in Figure 3.5.92. It shows that the samples with 2ml or more of ammonium hydroxide addition show possible jarosite formation. Similar tests were conducted with potassium chloride (1 g/40 ml, 2 g/40 ml, 4 g/40 ml, 6 g/40 ml) and no precipitates were observed for this set of tests. Pictures of the test solutions are shown in Figure 3.5.93 (left 4 tubes). Another set of tests were conducted with potassium hydroxide (0.5 g/40 ml, 1 g/40 ml, 2 g/40 ml, 4 g/40 ml). Precipitates formed in all the tests and the pictures of the test solutions as well as the pH are shown in Figure 3.5.93 (right 4 tubes). All 4 tests show precipitates but the tests with 1 g/40 ml and 2 g/40 ml addition show possible jarosite formation. The pH and Eh values after adding these reagents are shown in

Table 3.5.15. Considering the formation and the color of precipitate as well as the pH range of 2-3 which in favors of jarosite formation, we can make a preliminary judgement on the effect of these reagents and the effect on jarosite formation. Precipitates from these jarosite formation tests are filtered and dried will be examined with XRD to determine the composition.

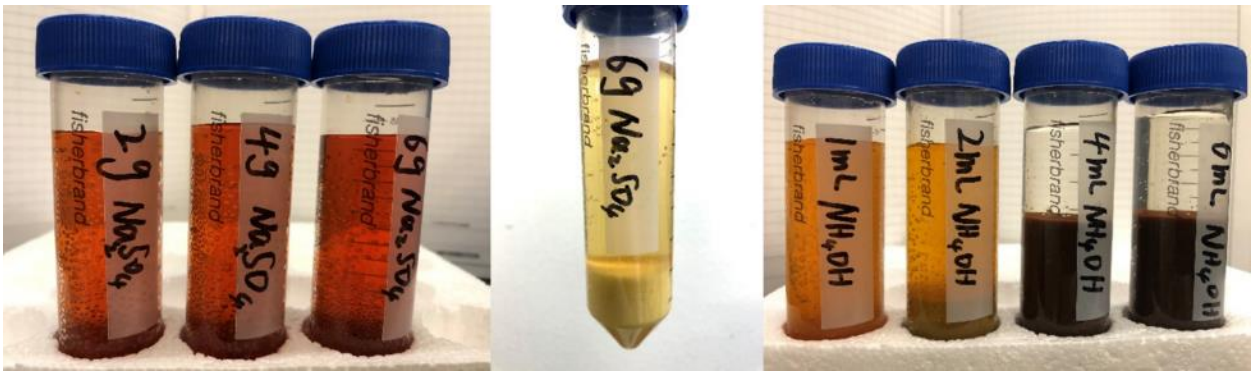



Figure 3.5.92. Digital images of the test solutions after sodium sulfate (left 3 test tubes), the sample with 6g $Na_2SO_4$ addition after 40 days (the middle test tube), and ammonium hydroxide (right 4 test tubes) additions and subsequent sedimentation.

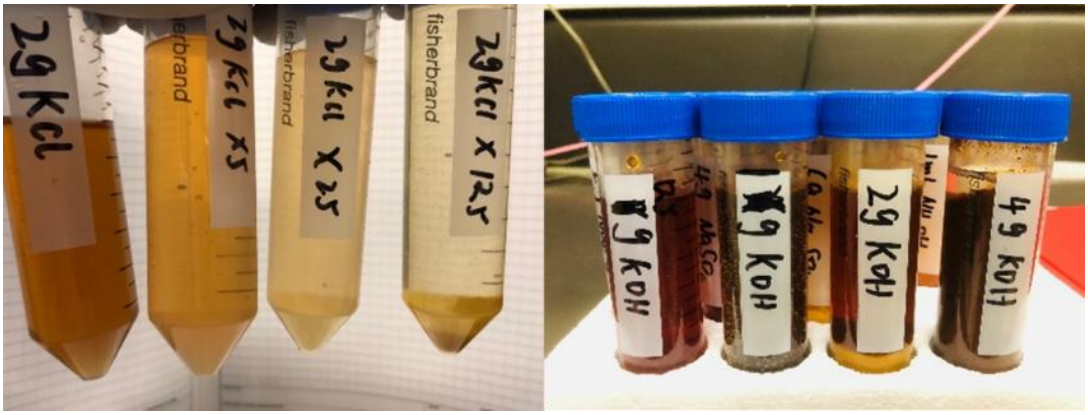



Figure 3.5.93. Digital images of KCl based test solutions (left image) after dilution; the bioleaching solution after addition of KOH(right image).

Table 3.5.15 pH and Eh values after adding different reagents to the bioleaching solution.

| Samples | pH | Eh/mV |
|---|---|---|
| 2 g $Na_2SO_4$-24 hr. | 2.06 | 627 |
| 4 g $Na_2SO_4$-24 hr. | 2.16 | 623 |
| 6 g $Na_2SO_4$-24 hr. | 2.23 | 620 |
| 6 g $Na_2SO_4$-40 days | 2.06 | -- |
| 1 ml $NH_4OH$-24 hr. | 1.93 | 649 |
| 2 ml $NH_4OH$-24 hr. | 2.3 | 620 |
| 4 ml $NH_4OH$-24 hr. | 9.01 | 86.1 |
| 6 ml $NH_4OH$-24 hr. | 9.62 | 49.7 |
| 2 g KCl-24 hr. | 1.59 | -- |
| 2 g KCl×5 dilution-24 hr. | 1.77 | -- |
| 2 g KCl×25 dilution-24 hr. | 1.91 | -- |
| 2 g KCl×125 dilution-24 hr. | 2.04 | -- |
| 0.5 g KOH-24 hr. | 1.68 | -- |
| 1.0 g KOH-24 hr. | 2.09 | -- |
| 2.0 g KOH-24 hr. | 2.4 | -- |
| 4.0 g KOH-24 hr. | 10.49 | -- |

By adding sodium salt to the bioleaching solution, Na-jarosite might form in the bioleaching solution. The precipitate formed in the test adding 6g $Na_2SO_4$ was examined with SEM and EDS. Figure 3.5.94 shows the picture of the precipitate and the SEM image of the precipitate. Compared with the jarosite precipitate formed in the bio-oxidation test, the precipitate form in this test was different. It looked like it consisted of small flakes rather than small particles. This was confirmed with the SEM image. The EDS analysis is shown in Figure 3.5.95. As it shows, the main elements in the precipitate is Fe, O, S, and Na, indicating that the precipitate was Na-jarosite.

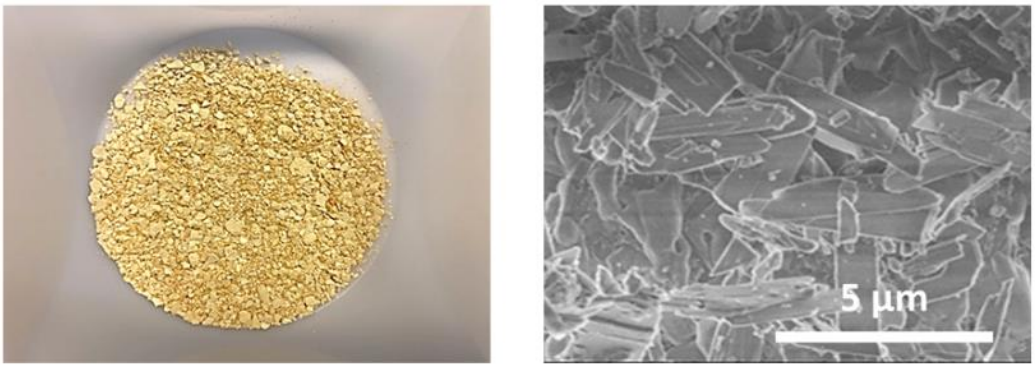



Figure 3.5.94. The picture and SEM image of the precipitate formed in the jarosite formation test by adding $Na_2SO_4$.

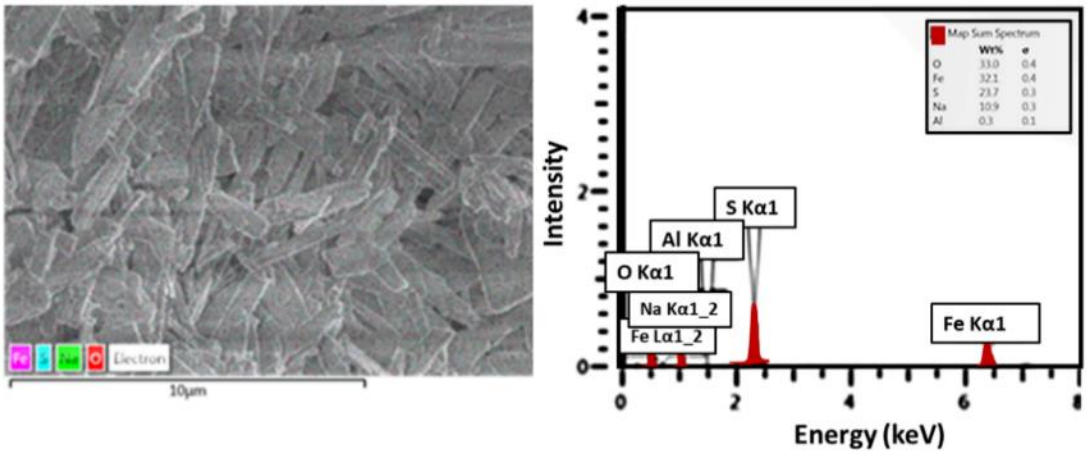



Figure 3.5.95. EDS analysis of the precipitate formed in the jarosite formation test by adding $Na_2SO_4$.

**<u>Jarosite Formation Modeling with Visual Minteq</u>:** To have a better understanding on the jarosite formation effect of reagent adding, equilibrium modeling was conducted to study the effects of pH and K and pH and Na for jarosite precipitation with 20 and 40 g/l dissolved $Fe^{3+}$. Similar to the experimental tests, the model studied the effect of pH and KOH addition, as well as the pH and NaOH addition. Plots were made to show the Fe3+ concentration and jarosite saturation index change with the composition change of the solution at equilibrium. The saturation index defined as the difference between log IAP(Ion activity product) and log Ks (solubility constant of the mineral at the given temperature). When undersaturation prevails, the saturation indexes are negative and given in blue. The modeling results are shown in Figure 3.5.96-Figure 3.5.98.

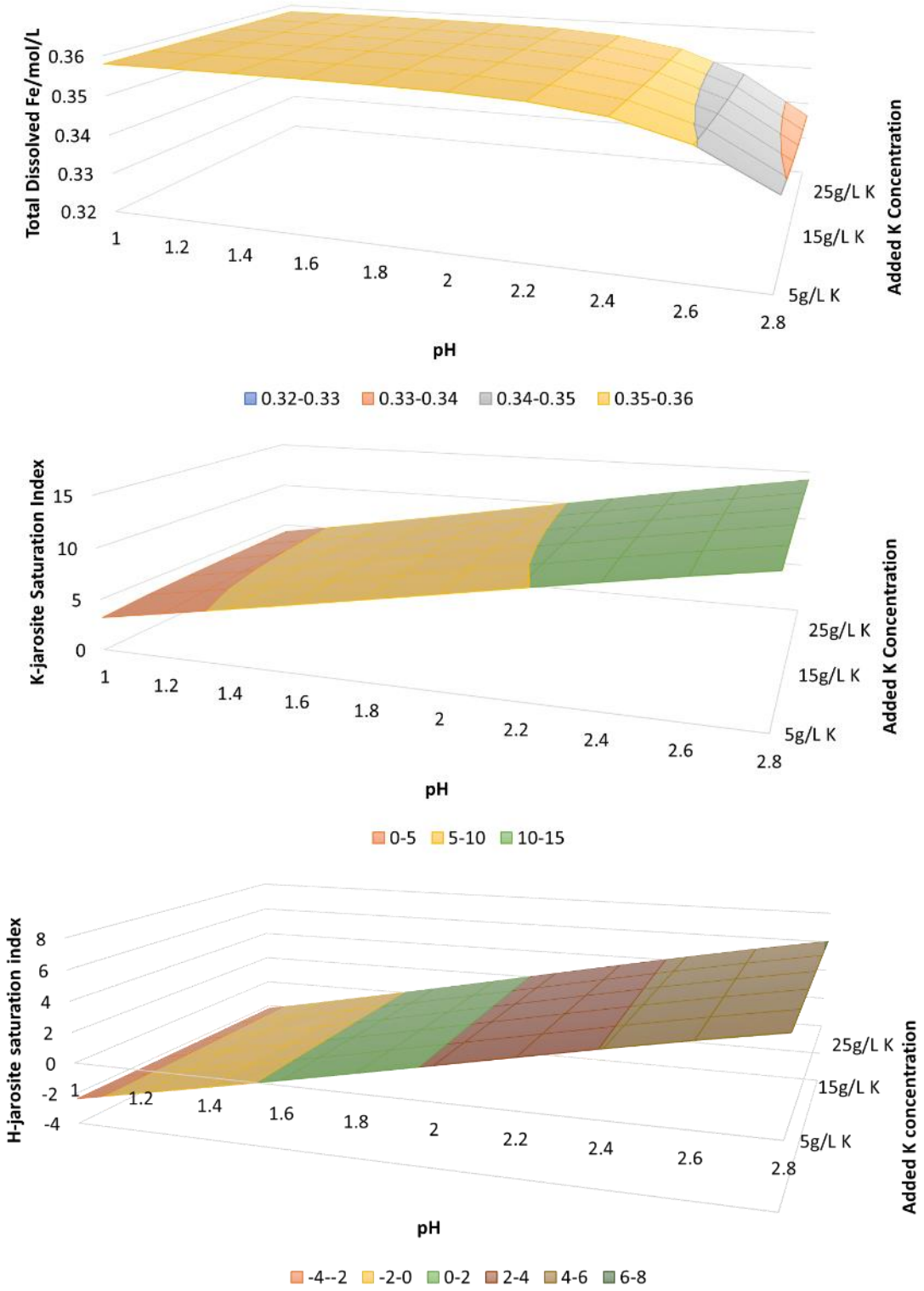



Figure 3.5.96. Effect of pH and K on the Fe concentration and jarosite formation when the initial $Fe^{3+}$ concentration is 20 g/L.

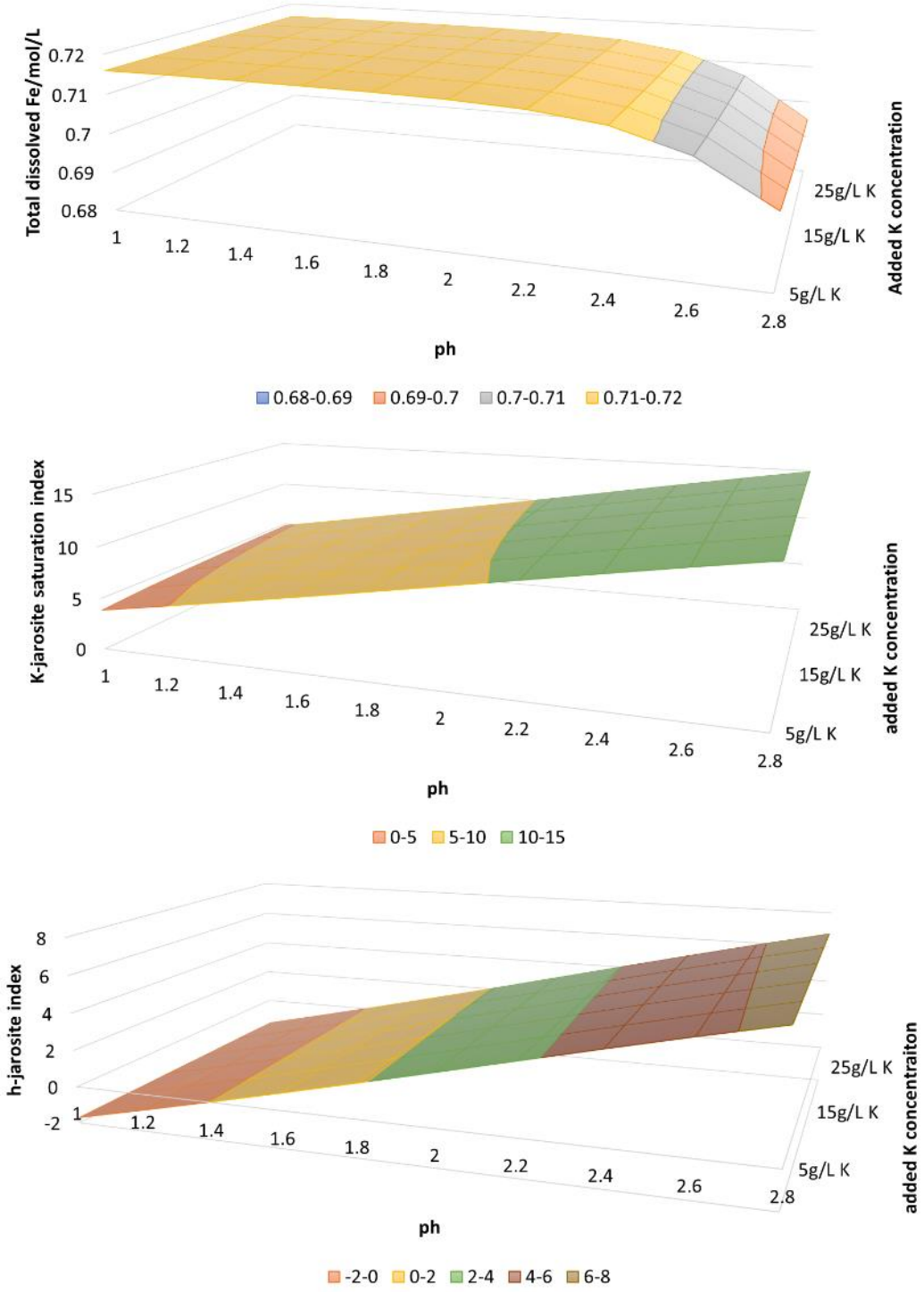



Figure 3.5.97. Effect of pH and K on the Fe concentration and jarosite formation when the initial Fe3+ concentration is 40 g/L.

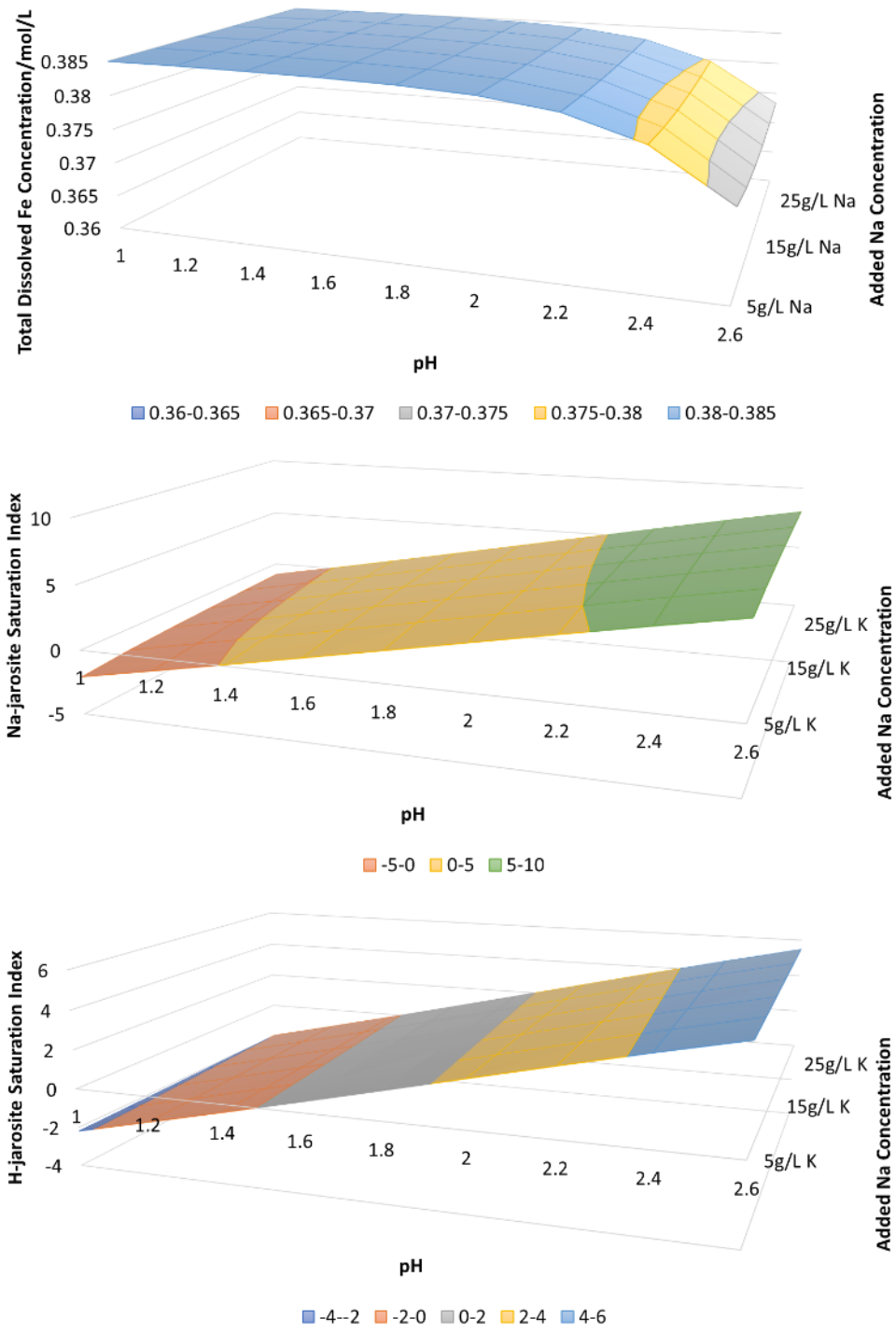
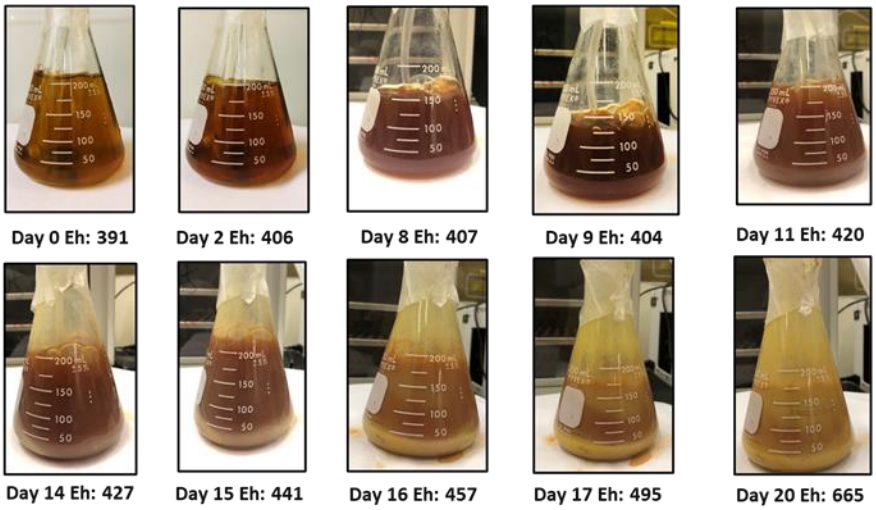



Figure 3.5.98. Effect of pH and Na on the Fe concentration and jarosite formation when the initial Fe3+ concentration is 20 g/L.

**Jarosite Formation with Bio-oxidation:** A 200 ml of the filtered bioleaching solution was collected from the new bioreactor. Based on the balance of feeding and effluent, the Fe concentration in the bioleaching solution was estimated to 0.5 M. The Fe concentration was adjusted to around 1.0 M by adding 27.8 g $FeSO_4 \cdot 7H_2O$ and 200 ml bioleaching solution to an Erlenmeyer flask. The solution was mixed with a magnetic stirrer and air was blown in the solution. Any change in the Eh change of the solution as well as formation of preciptiates was tracked during the test. Pictures of the bioleaching solution and the Eh value (mV) are shown in Figure 3.5.99.



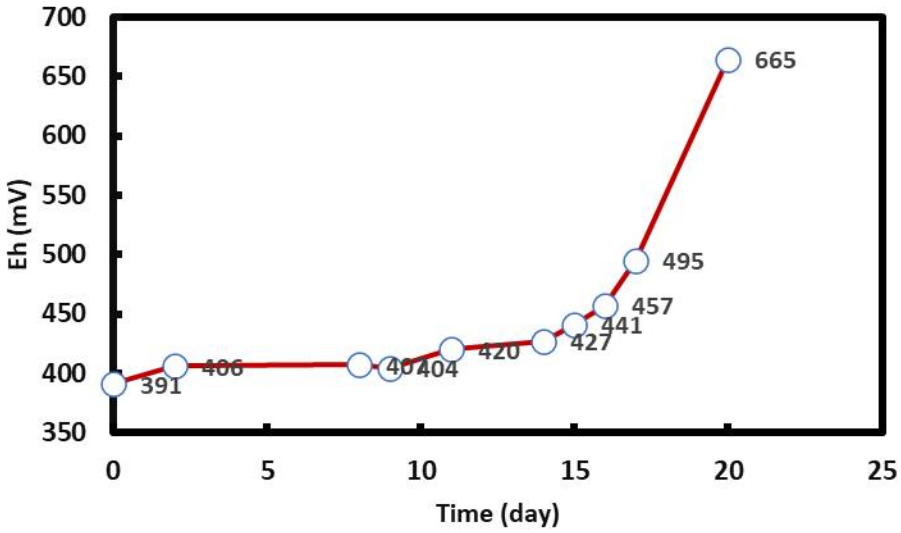




Figure 3.5.99. Pictures and Eh of the bioleaching solution after different time.

As can be seen from the pictures, in the first 11 days, the color and Eh of the solution did not change very much, and the solution was generally clear. After that, the solution became turbid with time indicating the generation of precipitates in the solution. Besides, the Eh of the solution increased with time faster and researched a very high value of 665 mV after 21 days, indicating

that most of the ferrous ions was oxidized to ferric ions, which might facilitate the generation of precipitate.

On August 10th, the solution was filtered to collect both the precipitate and the solution. The weight of the precipitate after drying is 3.66 g. The picture of the dried precipitate is shown in Figure 3.5.100. As can be seen, the color of the precipitate is amber yellow, which is the typical color of the jarosite. This is confirmed with SEM, EDS and XRD analysis, as shown in Figure 3.5.101 and Figure 3.5.102. From the EDS anslysis, it can be seen that the main elements in the precipitates were O, Fe, and S. This was consistent with the chemical composition of the H-jarosite. XRD analysis further confirmed the composition of the precipitate to be mainly jarosite because all the peaks of the sample match well with the peaks of the jarosite.



Figure 3.5.100. Picture of the precipitate from the jarosite formation test with bio-oxidation.

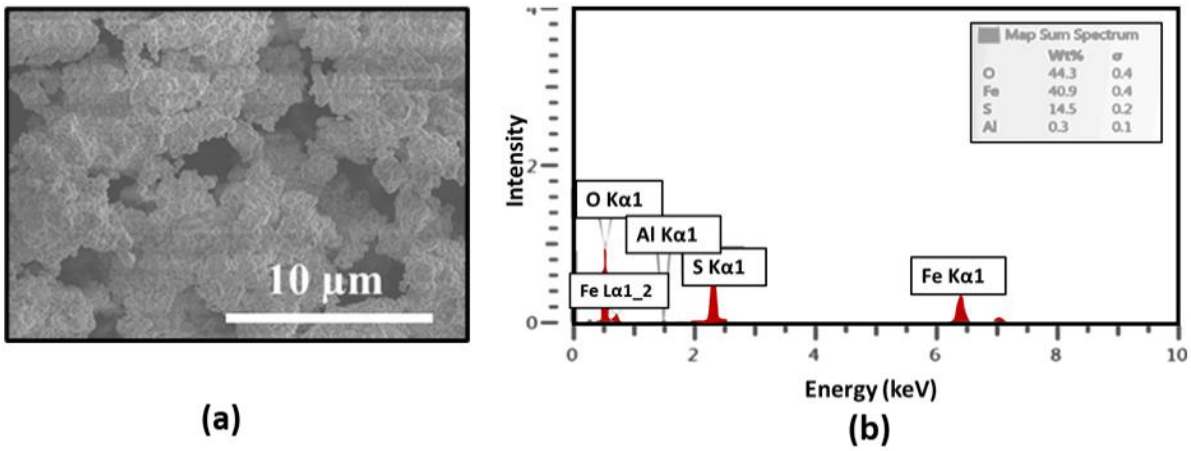



Figure 3.5.101. SEM and EDS analysis of the precipitate from the jarosite formation test with bio-oxidation.

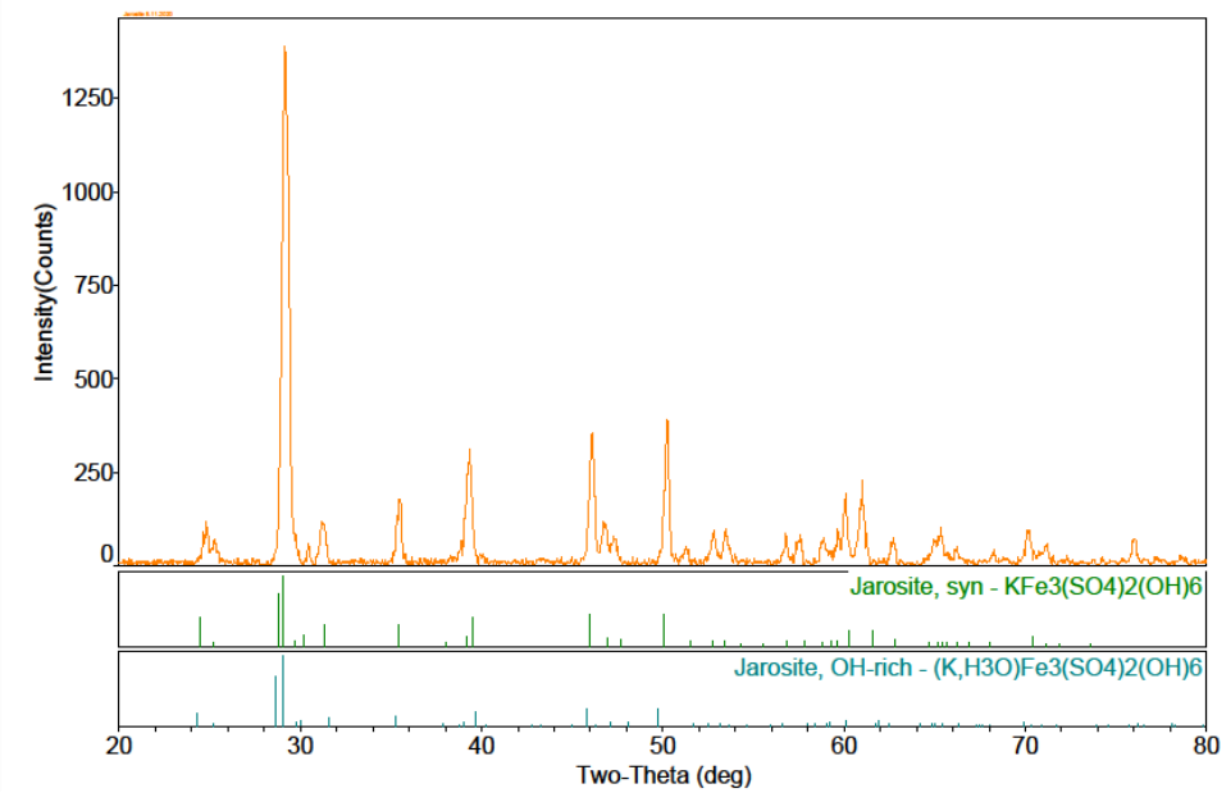



Figure 3.5.102. XRD analysis on the precipitate from the jarosite formation test with bio-oxidation.

Since there is not too much K, Na, or NH4 in the solution, we assumed most of the jarosite was hydronium jarosite ($(H_3O)Fe^{3+}{}_3(SO_4)_2(OH)_6$). Based on the mass balance, the amount of the Fe in the precipitate is calculated as:

$$n_{Fe} = 3\frac{m_{precipitate}}{M_{\text{hydronium jarosite}}} = 3\frac{3.66\,g}{480.74\,\text{g/mol}} = 0.023\,mol$$

In the initial solution, the amount of Fe is about 0.2 mol; therefore, about 11.5% of the Fe was precipitated in the form of jarosite during the bio-oxidation process.

As stated before, the solution was filtered, and the precipitate was collected. While the liquid was put back to the flask to continue the test. 5 g of $FeSO_4 \cdot 7H_2O$ and 100 ml 9K medium was added to the solution. Figure 3.5.103 shows the pictures of the solution at different time. After the other 20 days, precipitate formed again in the solution. After filtration and drying, the weight of the precipitate was 2.02 g. The amount of Fe precipitated out in this test is:

$$n_{Fe} = 3\frac{m_{precipitate}}{M_{\text{hydronium jarosite}}} = 3\frac{2.02\,g}{480.74\,\text{g/mol}} = 0.013\,mol$$

Therefore, 6.5% of the Fe was precipitated in the form of jarosite during the continued bio-oxidation process.

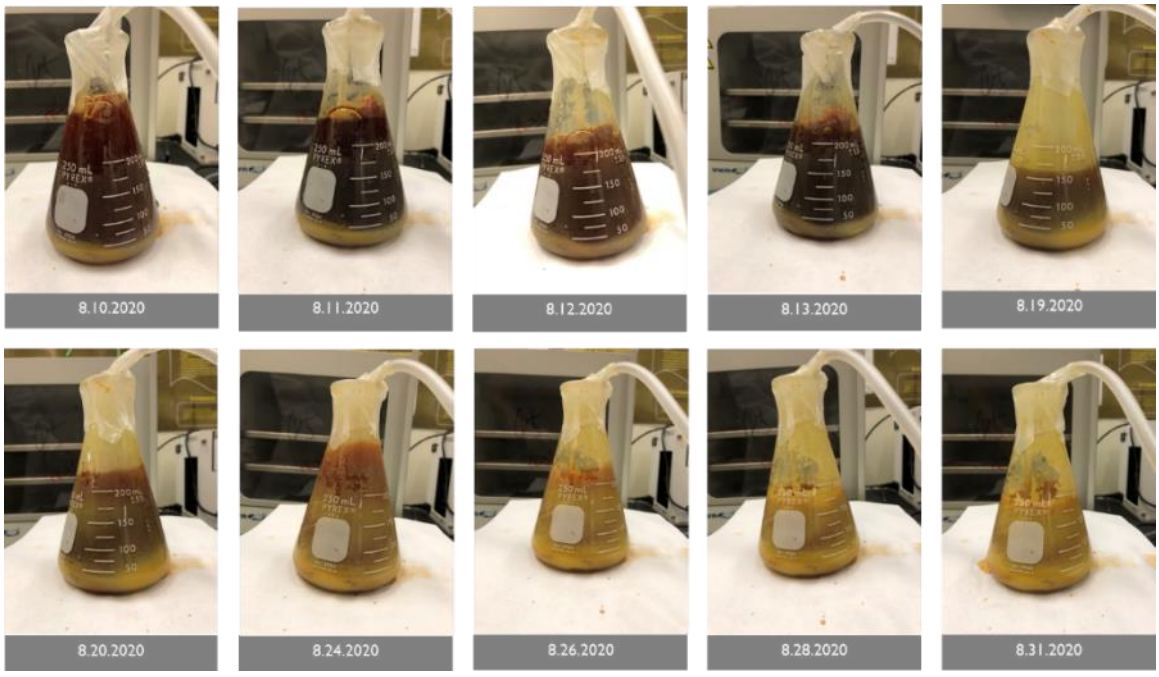



Figure 3.5.103. Pictures of the bioleaching solution after different time in the continued jarosite formation test.

### *3.5.9 REE Bioleaching Tests*

**Coal Waste Bioleaching Test 1:** Two samples of coal waste (Fireclay coarse refuse roaster feed and Dotiki coarse refuse roaster feed) were tested to evaluate their potential for heap bioleaching using bioleaching solution accumulated from other bioreactors. Pictures of the bioleaching tests were shown in Figure 3.5.104 and Figure 3.5.105 for Fireclay and Dotiki samples, respectively. The experiments were conducted with overhead stirring at 1200 rpm at 25ºC using 2.8 liters of solution and 1.2 kg of feed. The leaching test of the Fireclay refuse was conducted for 48 hrs. And the leaching test of the Dotiki refuse was conducted for 120 hrs. Samples were collected at different time intervals, and the final leaching solutions were saved after filtration. ICP-MS tests were conducted on these leaching samples to determine the concentration of REE elements in the solution and the leaching kinetics.



Figure 3.5.104. Pictures of the Fireclay coarse refuse roaster feed leaching test.




Figure 3.5.105. Pictures of the Dotiki coarse refuse roaster feed leaching test.

The pH and Eh change during the Dotiki tests, as well as the adjustment made, are shown in Table 3.5.16. This information shows the consumption of acid and $Fe^{3+}$ in the bioleaching solution during the leaching tests.

Table 3.5.16 The pH and Eh change during the Dotiki leaching test as well as the adjustment made.

| Time | pH | | Eh/mV | | Note |
|---|---|---|---|---|---|
| | Measured | Adjusted to | Measured | Adjusted to | |
| 0 hr | 1.33 | --- | 538 | --- | --- |
| 1 hr | 1.38 | --- | 519 | --- | --- |
| 2 hr | 1.38 | --- | 508 | 640 | 18 ml $H_2O_2$ |
| 1 day | 1.45 | 1.24 | 462 | 624 | $H_2SO_4$<br>20 ml $H_2O_2$ |
| 2 day | 1.64 | 1.28 | 476 | 621 | $H_2SO_4$<br>20 ml $H_2O_2$ |
| 3 day | 1.35 | 1.27 | 408 | 612 | $H_2SO_4$<br>60 ml $H_2O_2$ |
| 4 day | 1.5 | 1.3 | 485 | 623 | $H_2SO_4$<br>20 ml $H_2O_2$ |

The concentrations of all the rare earth elements in the process samples from the Fireclay refuse and the Dotiki refuse are shown in Figure 3.5.106 and Figure 3.5.107, respectively. The leaching tests of both the samples yielded similar concentration levels of REE in the final solution. For the Fireclay refuse, the first four elements of highest concentrations are Ce, Nd, Y, and La. While for the Dotiki refuse, the first four elements of highest concentrations are Y, Ce, Nd, and Sc.

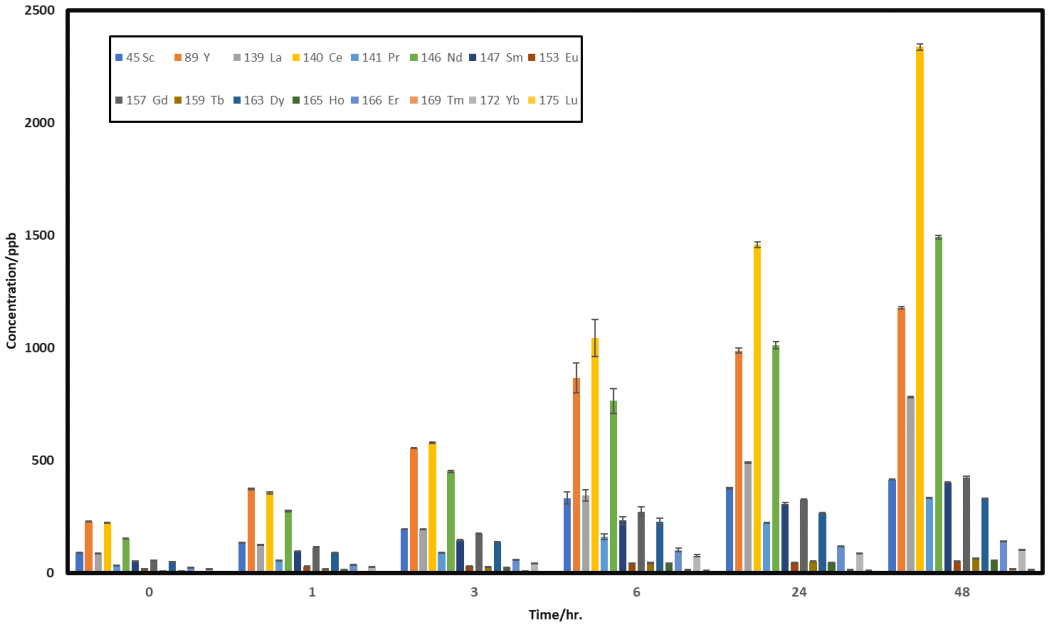



Figure 3.5.106. Rare earth elements concentrations in the leaching samples collected during the leaching process of Fireclay refuse.

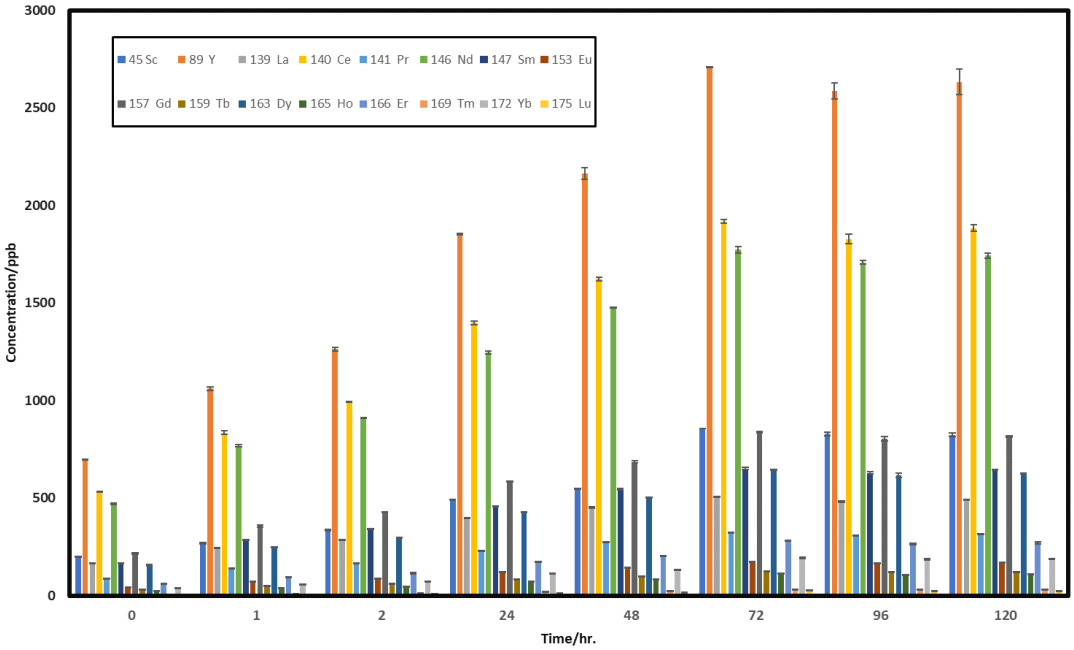



Figure 3.5.107. Rare earth element concentrations in the leaching samples collected during the leaching process of Dotiki refuse.

The concentrations of all the REEs were summed up and plotted versus time for both tests to show the leaching kinetics, as shown in Figure 3.5.108 and Figure 3.5.109. As can be seen from these figures, the leaching was rapid initially. After that, it becomes almost linear. For both samples, the

total concentration of REEs reached 8-9 ppm after 48 hours of leaching. For the Dotiki sample the concentration stabilized after three days. The final concentration of the Dotiki refuse leaching solution is about 10.9 ppm. It is anticipated that the Fireclay sample would achieve a similar final concentration after 3 days also. The overall recoveries based on solution analyses is, therefore, likely to be equivalent to 25.4 ppm REEs in the feed. Thus, if the feed contains about 250 ppm REEs, the typical recovery for heap bioleaching is likely to be about 10 %.

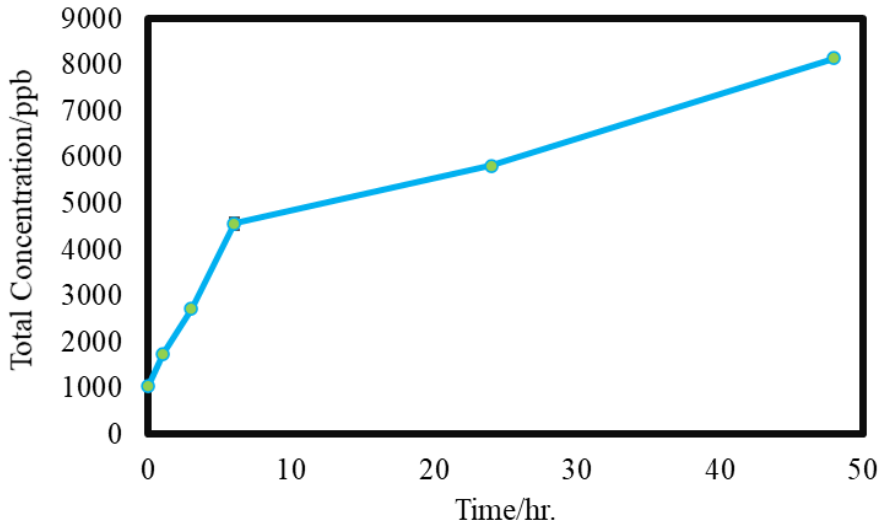



Figure 3.5.108. Total REEs concentration vs. time in the Fireclay refuse leaching test.

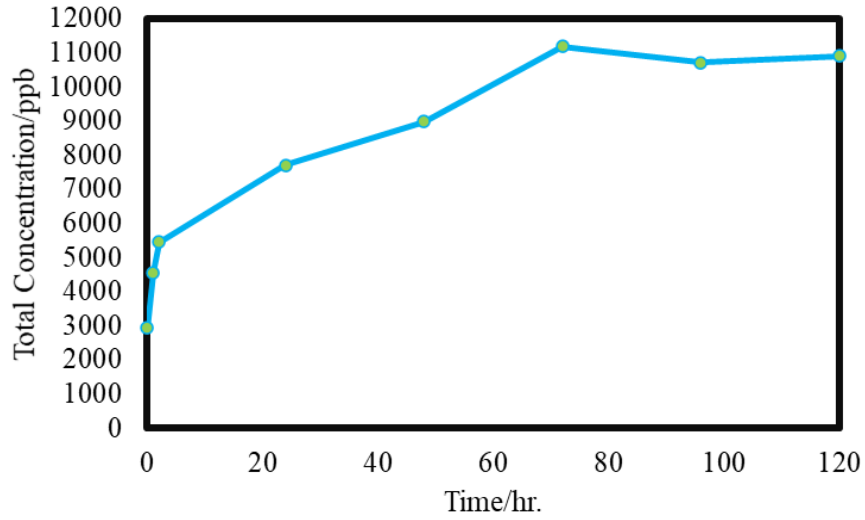



Figure 3.5.109. Total REEs concentration vs. time in the Dotiki refuse leaching test.

**Coal Waste Bioleaching Test 2:** Leaching experiment was conducted for 3 different coal waste samples Dixon clay above 8A seam, Dixon 8A seam, and Gilmore 8A seam by using bioleaching solution collected from 1.5 L bioreactor. Figure 3.5.110 shows the results for Dixon clay above 8A seam. It can be seen that for Dixon clay, highest recovery of REE was acquired after 4 hours and after that recovery rate of REE was stable and cerium has the highest recovery percentage. Figure

3.5.111 shows the results for Dixon 8A seam. It was found that for Dixon 8A seam highest recovery of REE was acquired after 24 hours and after that recovery rate of REE was stable and cerium had the highest recovery percentage. Figure 3.5.112 shows the results for Gilmore 8A seam. Highest recovery of REE was acquired after 168 hours and after that recovery rate of REE was stable for Gilmore 8A seam and cerium has the highest recovery percentage. Figure 3.5.113 shows the percentage of elements distribution at the last hours of leaching for all four samples.

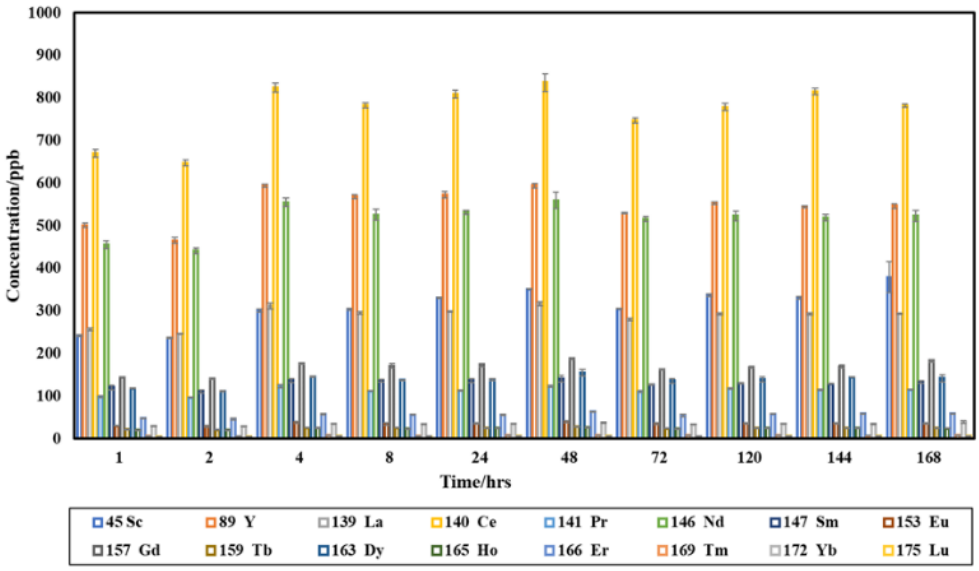



Figure 3.5.110. Leaching Experiment results for Dixon Clay Above 8A seam.

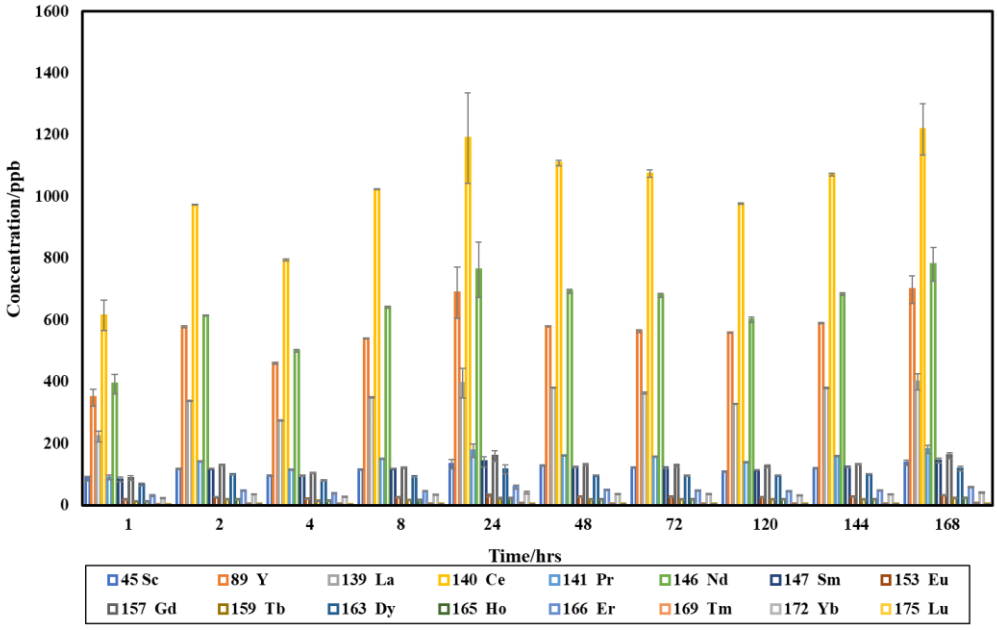



Figure 3.5.111. Leaching Experiment results for Dixon 8A seam.

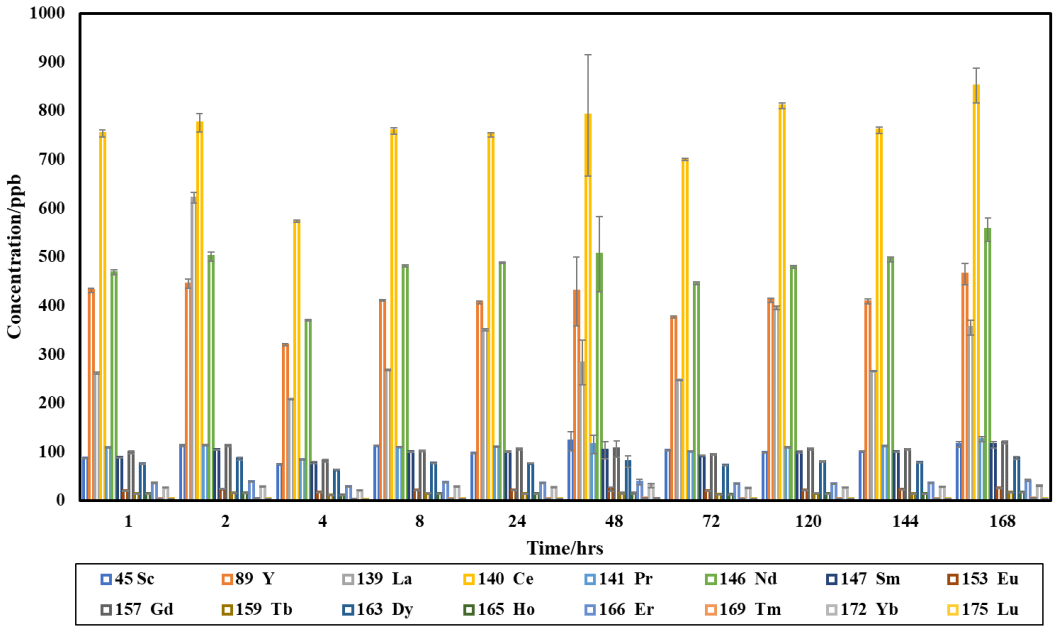



Figure 3.5.112. Leaching Experiment results for Gilmore 8A seam.

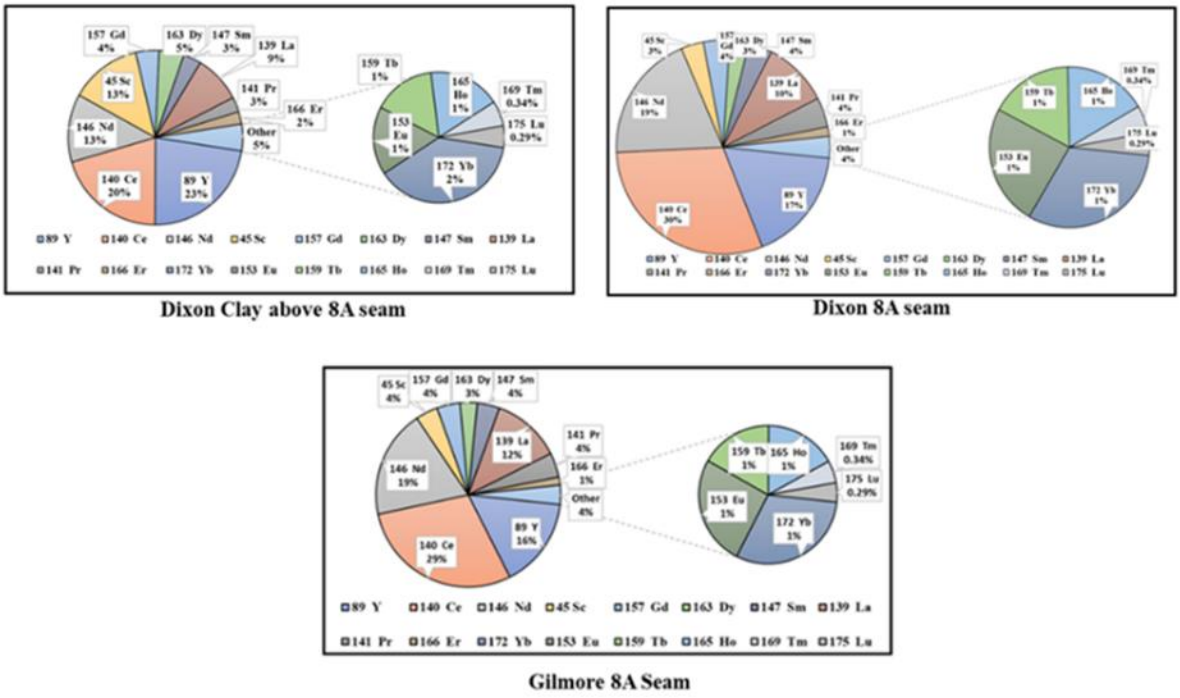



Figure 3.5.113. Percentage of elements distribution at the last hours of leaching for all three samples.

**Work performed – January 1st, 2021 to June 17, 2021:** For this period, the pH and Eh vs. time behavior are shown in Figure 1. For this period, the effect of feeding rate also was frequently examined. It was checked by doing the ferrous oxidation test using ferrous sulfate, and study

results are shown in Figure 2. Different feeding rates were applied in the bioreactor to increase the bioreactor throughput and study the effect of feeding rate (residence time) on the performance of the Bioreactor. At the beginning of January, a 1.5 L bioreactor was fed with a residence time of ~ five days with 6% solids feeding. A fine pyrite feeding continued with a frequent feeding mechanism. pH and Eh were at the optimum level in this reactor.

On January 5, a new 3L bioreactor was installed. After installing the bioreactor, it took nearly three days to start feeding with 6% fine pyrite with a residence time of 6 days. Automatic feeding was formed on January 29 with 12 days of residence time. Later, residence time was changed to 8 days on February 2 and six days on February 4. It can be observed from Figure 3.5.114 that pH has been increased after switching to a 6-day residence time (RT). On February 11, automatic feeding was stopped. On February 12, pH was adjusted, and then 8% solids feeding was started with eight days of residence time. Frequent feeding was stopped at this moment, and a regular feeding pattern was followed. This feeding parameter was continued until March 4th. During this period, pH and Eh were found stable. On March 5th, feeding parentage was changed to 6% solidss from 8% solids with six days of residence time. After observing the stability in pH and Eh, residence time was changed to 4 days with a frequent feeding mechanism. The frequent feeding of 8 times/ day (70 ml each time) was applied to provide a more stable $CO_2$ supply to the bacteria. This parameter continued until the end of March. No significant changes were observed in regular Eh and pH parameters during this period. From the very beginning of April to June 17th, fine pyrite was fed into the bioreactor. Stirring Speed ~ 500rpm and airflow rate ~ 1lpm were constant for this operation period. From April 1st to April 23rd, 6% solids was provided with a residence time of 4 days. Eh was moreover stable at this period. But there was an increment observed during the use of this parameter.

Later on, on April 23$^{rd}$ , residence time was reduced to 3.5 days. After starting operation with 3.5 days of RT, pH was decreased using some sulfuric acid. After that, a slight reduction was observed in the pH range, which remained stable for the entire time. In this case, a relatively higher average Eh was maintained at the last segment of this performance period after continuing the operation. A drop in Eh and pH can be seen in the Figure 3.5.114 on May 11th. There were some obstacles in the feeding connection during that day, and the bioreactor was not fed properly. When this problem was fixed, the bacteria recovered very quickly. All the changes in residence time and key impactors over this period have been pointed in Figure 3.5.114.

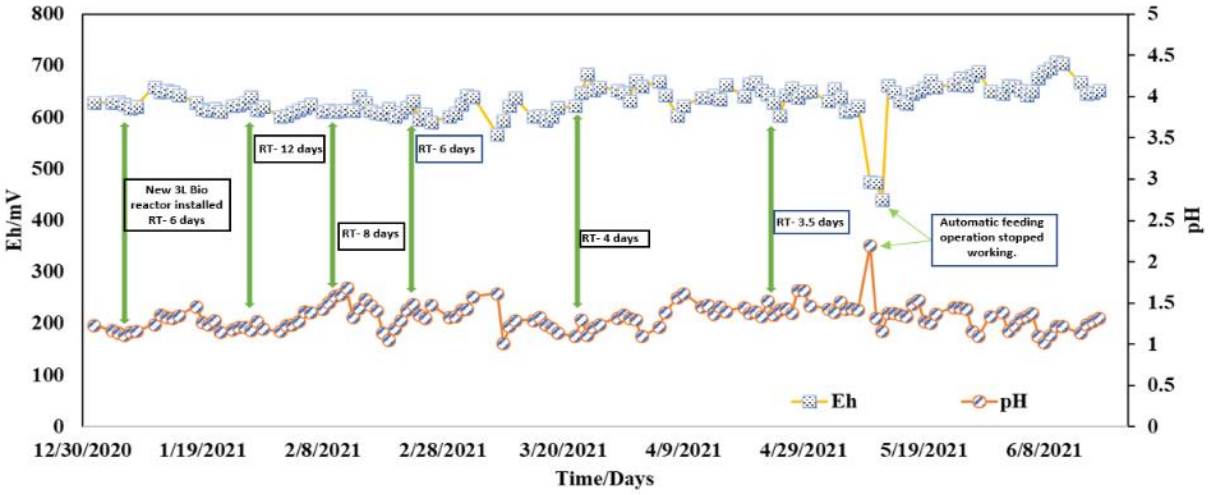



Figure 3.5.114: Changes in Eh and pH from January 1, 20201 to June 17, 2021.

The bioactivity of the bacteria was measured with the ferrous oxidation test. The initial Eh before the measurement and the average ferrous oxidation rate is shown in Figure 3.5.114. The ferrous oxidation rate fluctuated with time due to various studies' changes of operating conditions, as can be seen, in early January after installing the new Bioreactor and after the stabilizing period. Comparing the pH and the ferrous oxidation rate, the pH increase caused by the increasing feeding rate is not because of the reduction of the ferrous oxidation rate. Overall biooxidation rate dropped after using the modified composition of 9K medium made with only $(NH_4)_2SO_4$ and $K_2HPO_4$ with 50ml:50ml ratio with DI water, and then it was increased though. At the period of 8% solids feeding with the residence time of 6 days, a ferrous oxidation rate was observed, which is 2.652 g/l/hr. After switching 6% solids with a residence time of 6 days, a drop in the biooxidation rate has been observed. Also, it can be seen there were several fluctuations in the ferrous oxidation rate curve which is the result of an operational error in the Bioreactor. After switching to 4 days of residence time with a frequent feeding mechanism, It can be seen that during the 6% fine pyrite solids feeding with a residence time of 4 days, the biooxidation rate was in the range of 1.6 to 1.9 g/l/hr. There were fluctuations in a few other rates, mainly due to operational error. After starting 3.5 days residence time, the biooxidation rate has been stable for a standard period. All the changes in solids feed concentration and airflow rate, and other operational parameters for this period have been shown in Figure 3.5.115.

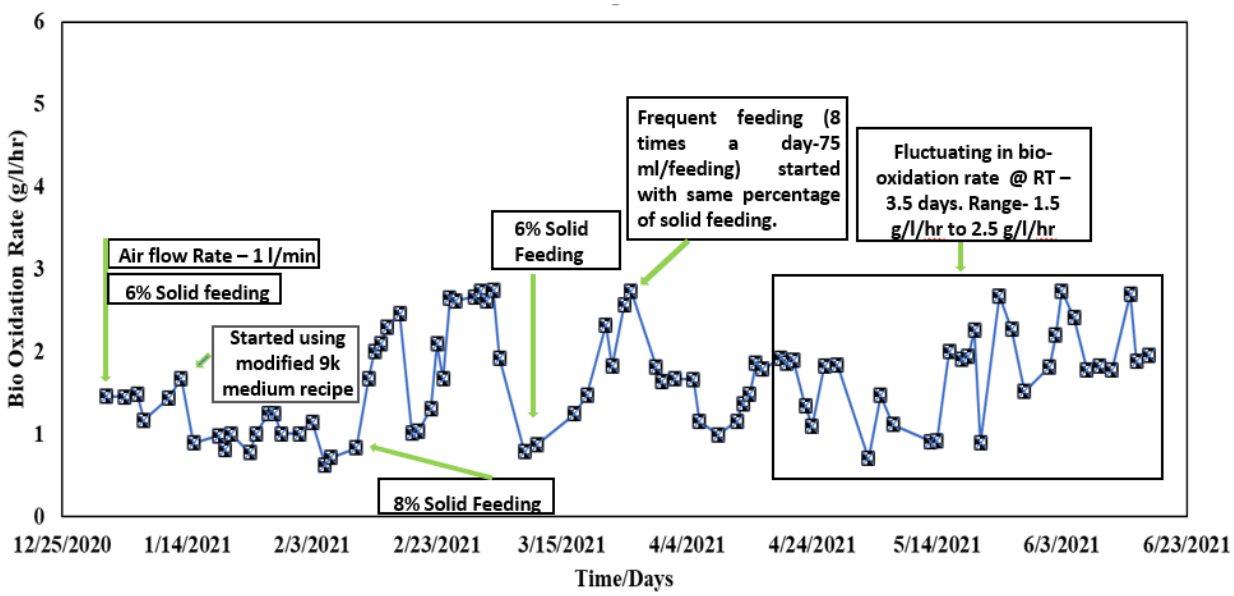



Figure 3.5.115: Changes in biooxidation rate from January 1, 2020 to June 17, 2021.

**Timeline – June 18 2021 to December 14, 2021:** Experimental matrix or plan 1 shown in Table 1 is the first part of steady-state parameter optimization. The team examined the effect of various parameters for the efficient operation of the bioreactor (see experimental plan in Table 1). These different parameters discussed were residence time, solids concentration, airflow rate, and nutrients. After each change in operating parameters in the Bioreactor, the process metrics were carefully observed to ensure steady-state operation before collecting data for biooxidation performance (ferrous ion oxidation rate), Eh, pH, and overall acid generation. Each set of parameters took approximately two weeks to evaluate due to the time required to reach a steady-state and appropriate exchange of slurry from the previous set of parameters. An experimental plan consisting of 10 different operating parameters with two repeat tests was developed, as presented in table below. The PIs and team evaluated eight sets of parameters and tried to maintain a steady state in the Bio-Reactor for at least two residence times.

Table 3.5.17 Experimental Matrix 1

| Experiment Number | Residence Time ( days ) | Solids Concentration (%) | Air Flow Rate (lpm) | Remark |
|---|---|---|---|---|
| P1 | 4 | 6 | 1 | Completed |
| P2 | 6 | 6 | 1 | Completed |
| P3 | 8 | 6 | 1 | Completed |
| P4 | 6 | 3 | 1 | Completed |
| P5 | 6 | 6 | 1 | Repeat |
| P6 | 6 | 9 | 1 | Completed |
| P7 | 6 | 6 | 0.5 | Completed |
| P8 | 6 | 6 | 1 | Repeat |
| P9 | 6 | 6 | 1.5 | Completed |
| P10 | 2.5 | 6 | 1 | Low Residence Time-Completed |
| P11 | 6 | 6 | 1 | Nutrition -STD 9K medium- Completed |
| P12 | 6 | 6 | 1 | Nutrition with lower (50% by volume) elemental concentration - Completed |

P1 was the first parameter set that was applied. Steady-state data was collected from July 5 to July 15. pH, Eh, and biooxidation rate was stable, and the range was good. After starting P2 on August 16th, the Eh increased, and the pH decreased compared to the other parameters, while the biooxidation rate was similar. Later, P7 conditions were applied. During the operation of this parameter, Eh and pH were like the previously found data. But there was a drop in biooxidation rate. This parameter has the lowest airflow rate, which might be why this continuously low biooxidation rate. P4 conditions were later applied. During the operation using P4 parameters, there was a considerable increase in pH and Eh, and some fluctuations were continued. Then it was repeated from September 29th to October 29, 2021. The steady-state parameter was achieved, and necessary data was collected. During P10 operation, a large change in overall operation was anticipated because this was the shortest residence applied in this entire biooxidation process. Daily measurements fluctuated a lot, but it stabilized after 13 days, and necessary data was collected. P4 and P3 was also repeated to get more steady-state data until November 24th. Figure 3.5.116 shows the overall changes in Eh and pH and biooxidation rate for this period.

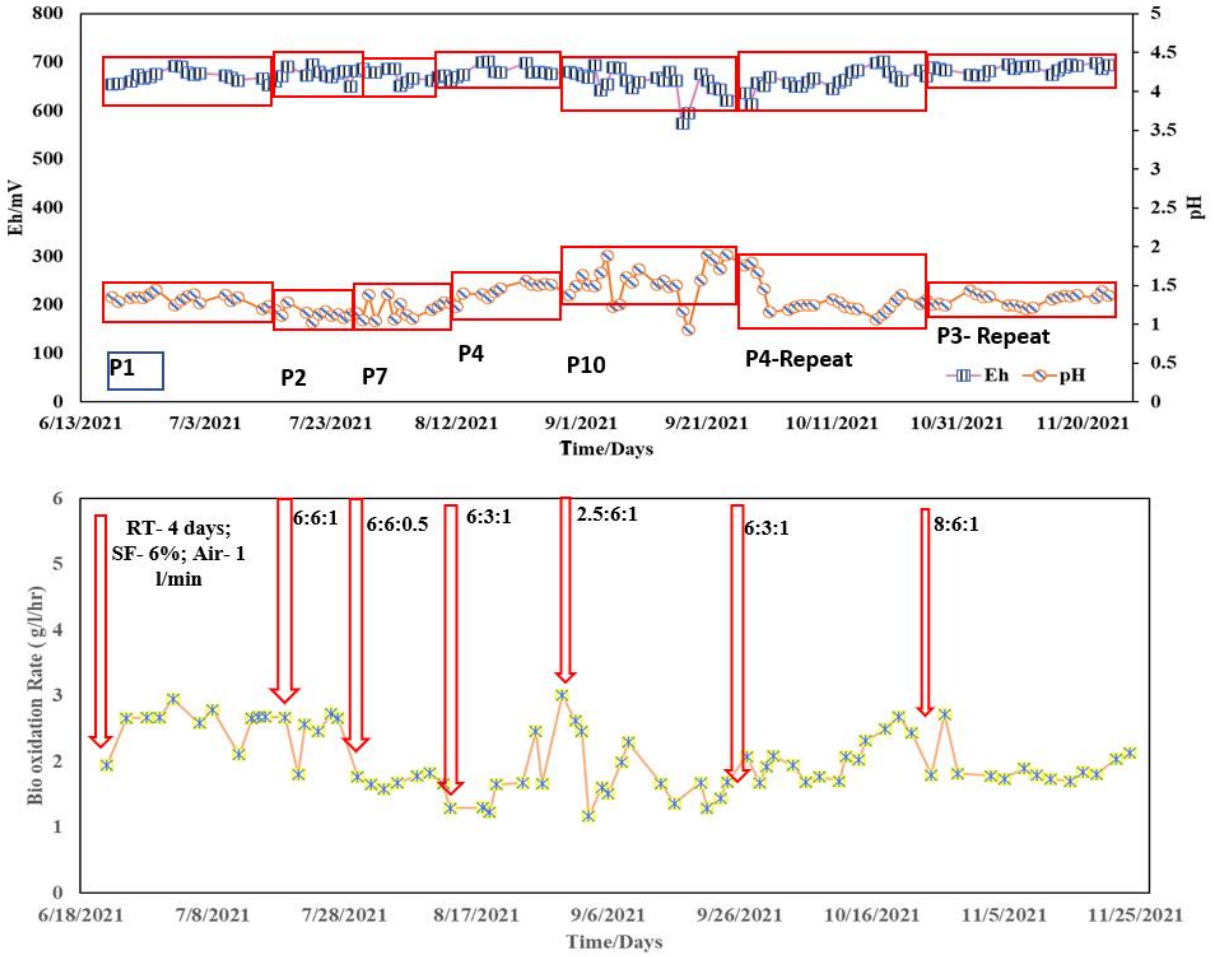



Figure 3.5.116: Changes in Eh, pH & Biooxidation rate From June 18 to November 24 of 2021 at steady state parameter optimization part (Following experimental matrix 1)

Subsidiary information: Experimental matrix or plan 2 shown in table below is for the second part of steady-state parameter optimization.

Table 3.5.18 Experimental Matrix 2

| Parameter | Residence Time | Feeding Solids % | Air Flow Rate (lpm) | Temperature | Stirring Speed |
|---|---|---|---|---|---|
| K1 | 4 | 6 | 1 | 35 | 700 |
| K2 | 4 | 6 | 1 | 35 | 300 |
| K3 | 4 | 6 | 1 | 35 | 500 |
| K4 | 4 | 6 | 1 | 35 | 800 |
| K5 | 4 | 6 | 1 | 25 | 500 |
| K6 | 4 | 6 | 1 | 30 | 500 |
| K7 | 4 | 6 | 1 | 35 | 500 |
| K8 | 4 | 6 | 1 | 40 | 500 |

**Timeline – November 29th to December 31st of 2021**: Table above shows the second part of the experimental matrix made to analyze the effect of stirring speed and temperature on biooxidation rate. The first parameter was to run was K1 with 700 rpm. Steady-state was achieved, and necessary data and samples were collected comparatively higher yet steady than the pH range of part 1 experiments. Eh, and Biooxidation Rate was in a very significant range. Then on December 14, K3 was continued. After running this parameter, a severe drop was observed in Biooxidation Rate because most of the pyrite particles precipitated in the bottom of the reactor due to the deficient stirring Speed. pH was unstable and high at an alarming range when Eh was unharmed. Then K4 -800rpm was applied and continued until this report was made. Biooxidation rate, eh, and pH were still stabilizing. During this operation period, other parameters that have already been optimized during the part 1 experiment were stable. Their ranges and overall changes in biooxidation rate, ph, and Eh, have been shown in Figure 3.5.117: Overall changes in biooxidation Rate, Eh and pH from November 29th to December 31st of 2021.

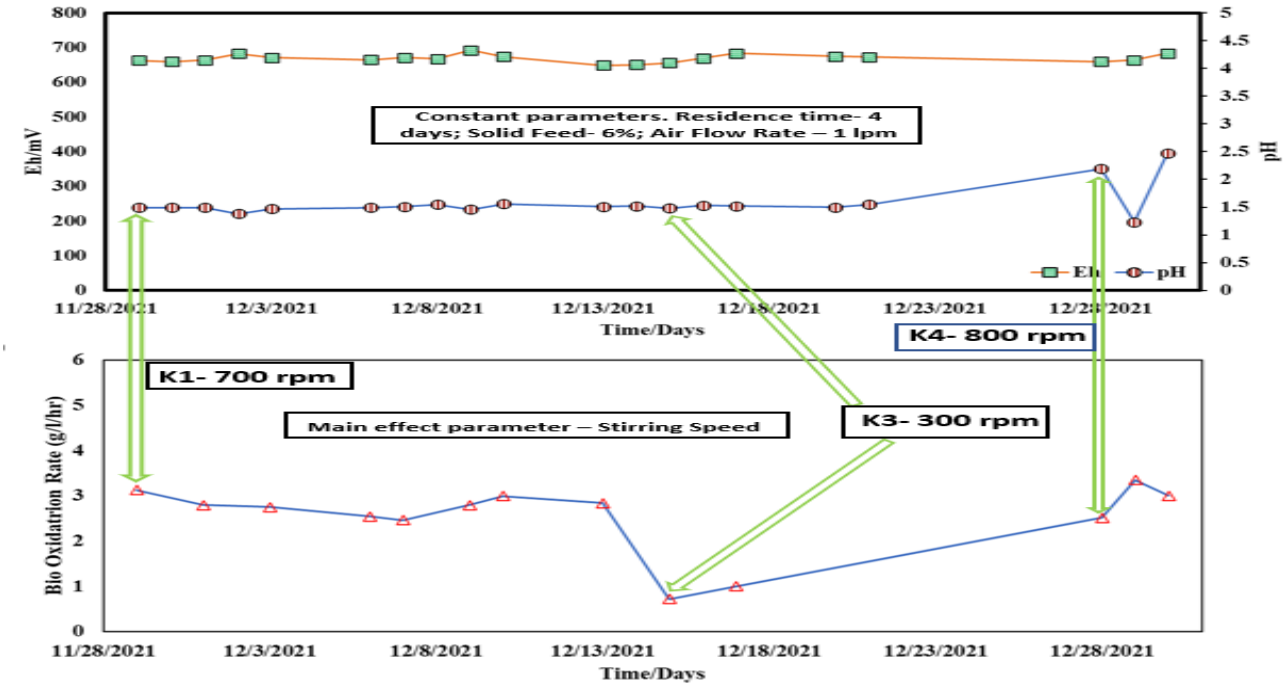



Figure 3.5.117: Overall changes in biooxidation Rate, Eh and pH from November 29th to December 31st of 2021.

**MCEB 2474 Bio reactor Update - Timeline 1 – January 1 to June 17 of 2021:** For this reporting period, only fine prite was fed into this bioreactor. From January 1, 2021, to March 19, 2021, only 10% solidss was provided into the bioreactor with six days of residence time. Later, the feeding solidss concentration was reduced to 8% in this bioreactor, and pH mainly was unstable and required adjustment with sulfuric acid for some time. During this period, some incidences such as feeding tube clogging, irregular airflow, water-connection loss were noted but by March 15th, these problems were resolved, and it can be seen that the biooxidation rate was increasing. pH was also in a steady-state, and Eh is decreasing slowly. Figure 3.5.118: Changes in Eh and pH from January 1st to June 17th of 2021 and Figure 3.5.119: Overall Changes in biooxidation Rate from Jan 1st to June 17th of 2021. show the results for this bioreactor. The bioreactor was operated with a stirring speed of 500rpm and an airflow rate of 1 lpm from April 1st to April 21st. Only 8% solidss was used in the bioreactor with four days of residence time. A higher pH was observed during this period in the later part of this parameter application. Eh was relatively higher. Later, the substantial concentration was switched to 12% with six days of residence time. With these sets of parameters, the bioreactor's performance was stable. pH was reduced to 1.26 from 1.46. Eh was appropriate. After getting steady-state data from these parameters, residence time was reduced to 4 days.

By conducting a ferrous oxidation test, bacterial activity was evaluated by putting $FeSO_4 \cdot 7H_2O$ into the 1.5 L bacterial solution in the reactor. At the same time, Eh and pH were measured without adding any feeding solution before starting the oxidation test $FeSO_4 \cdot 7H_2O$. This test was conducted at different bioleaching processes using different amounts of $FeSO_4 \cdot 7H_2O$. During this oxidation test, the same operating parameters were maintained. The Eh vs. time was measured and recorded for every test. The ferrous ion consumption rate vs. time inclinations had been portrayed

by following the Eh data. Figure 3.5.119 shows all the trials and plotted data for biooxidation evaluation for different conditions. It can be seen from Figure 3.5.119 that a more steady-state result of biooxidation when 12 % solids was fed into the reactor with the residence time of 4 days compared to 6 days of residence time. 8% feeding showed a steady-state for two sets of parameters.

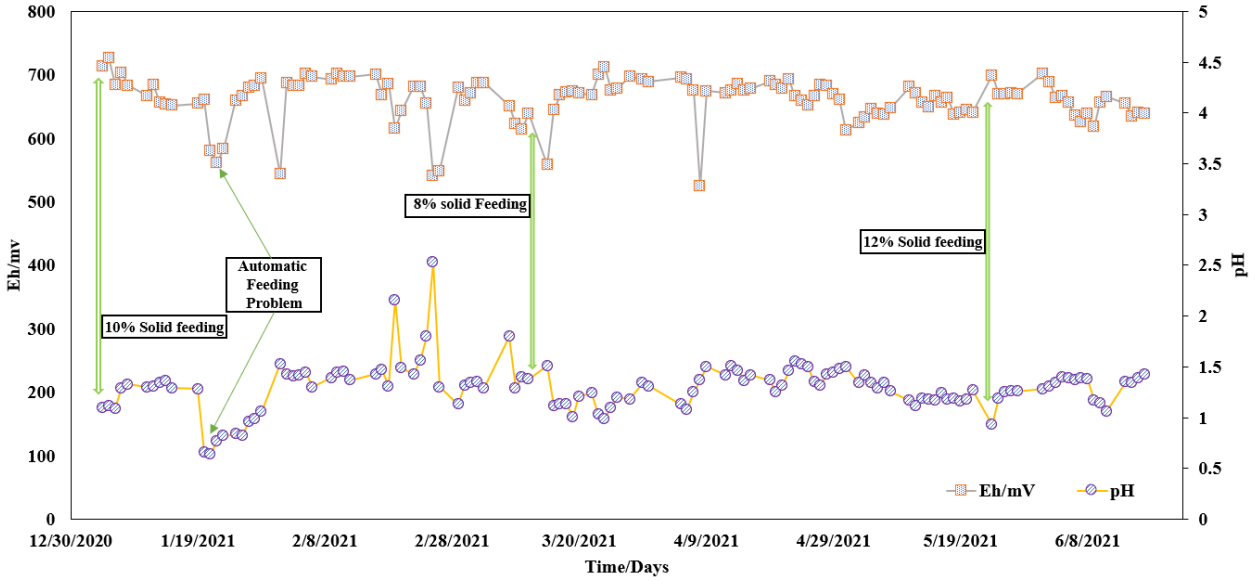



Figure 3.5.118: Changes in Eh and pH from January 1st to June 17th of 2021

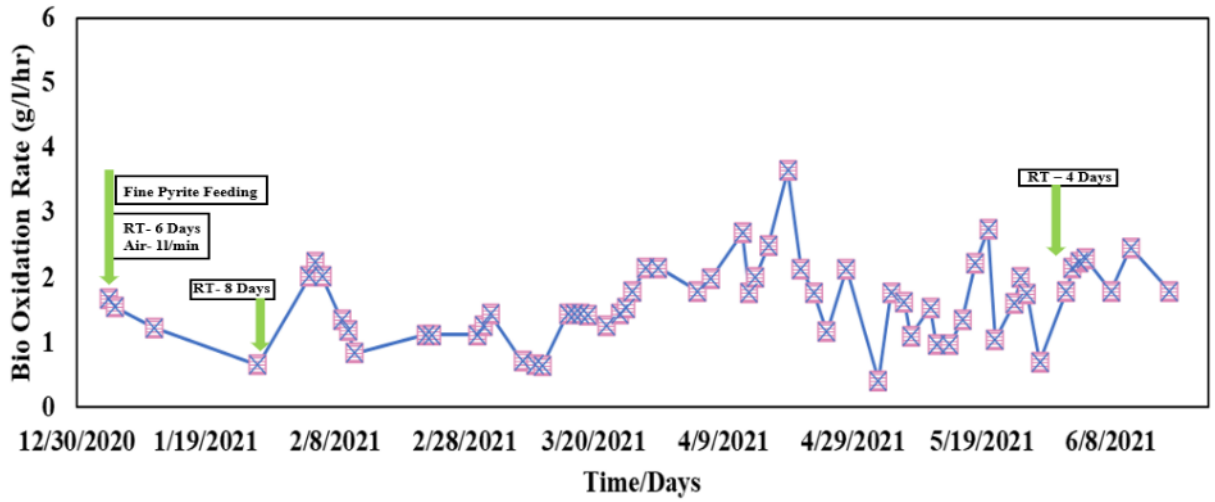



Figure 3.5.119: Overall Changes in biooxidation Rate from Jan 1st to June 17th of 2021

**Timeline 2: From June 18 to December 12 of 2021:** Before applying any parameter from the experimental matrix, one Bioreactor was brought under a steady state. Then Parameter set 3 (P3) was used in this bio-reactor on June 25, and this experiment was continued until July 29. Steady-state data was collected from July 15th to July 29th. All experiments plan were made in such a way so that each parameter should run for at least two residence times in a steady state. The data for this parameter was collected for 16 days (residence time - 8 days). Subsequently, P6 parameters were applied in the reactor data were collected for 12 days. The biooxidation rate was higher than

previous parameters, but there were no significant differences in pH and Eh. Later, P9 was applied, and data were collected from 16th to 28th August for twelve days (RT-6days). During test P9, pH, biooxidation rate, and Eh were close to the previous parameters. Then P11 parameters were introduced to the system, observing initial fluctuations that stabilized to steady conditions by September 9th. Data were collected from September 9th through 21st.

After changing to P12 conditions, the biooxidation rate dropped, and the Eh was stable. There was a decrease in the pH. P12 conditions have been applied in the reactor. It took some time to be stabilized, but after achieving the required range, biooxidation, Ph, and Eh were in a stable state range; after that, P10 was repeated for 5days in this reactor to justify the fastest residence time output of the WBB 419 reactor of the same parameter. Then P9 was applied. With the highest amount of air supply during the operation of this parameter, a rock-solids steady-state was observed in Eh, ph, and biooxidation rate data. Figure 3.5.120 shows the general changes in steady-state Eh, pH, and biooxidation rate changes at this period.

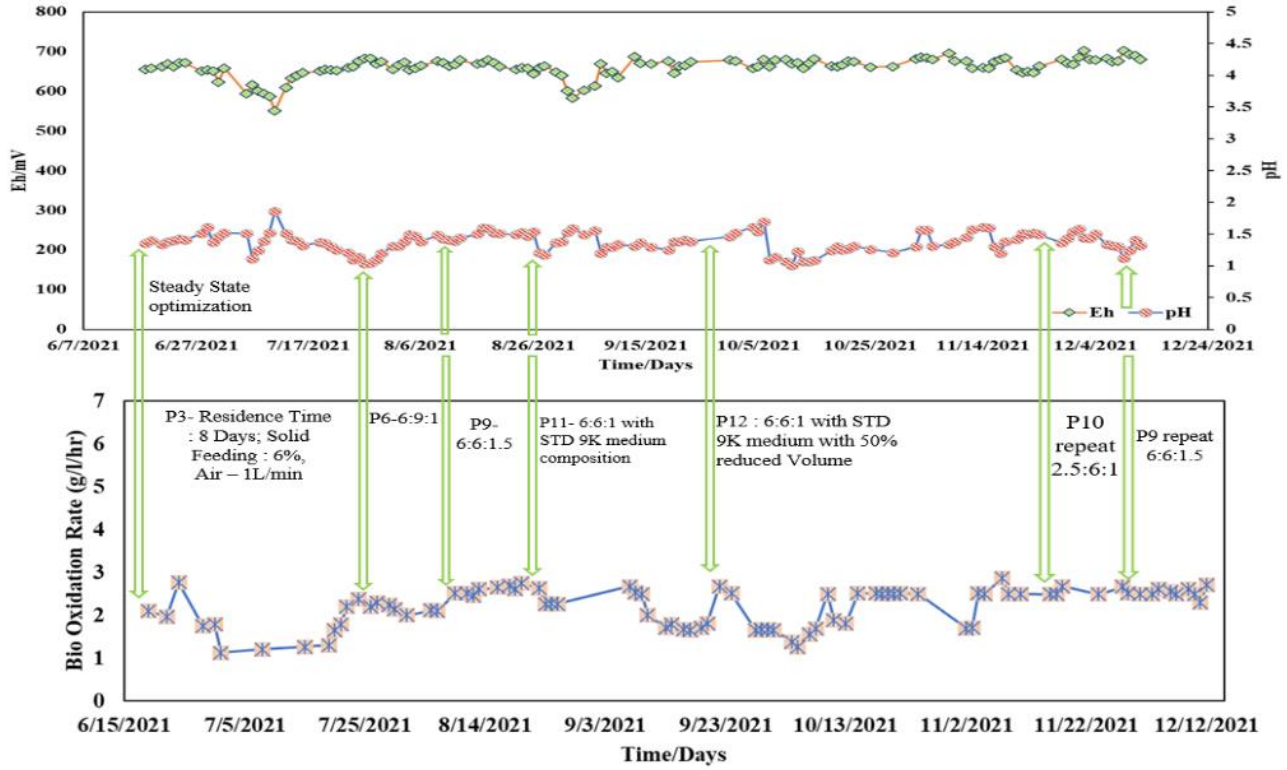



Figure 3.5.120 :Changes in Eh, ph and Biooxidation rate from June 13th to December 12th of 2021

**Timeline 3 – 13th December to end of the Year:** In this time period, K5 conditions were applied with the temperature reduced ot 25°C and other parameters were constant. It was close to the steady-state until this report was made. This parameter will be changed once all the necessary information is collected. Figure 3.5.121 shows the results.

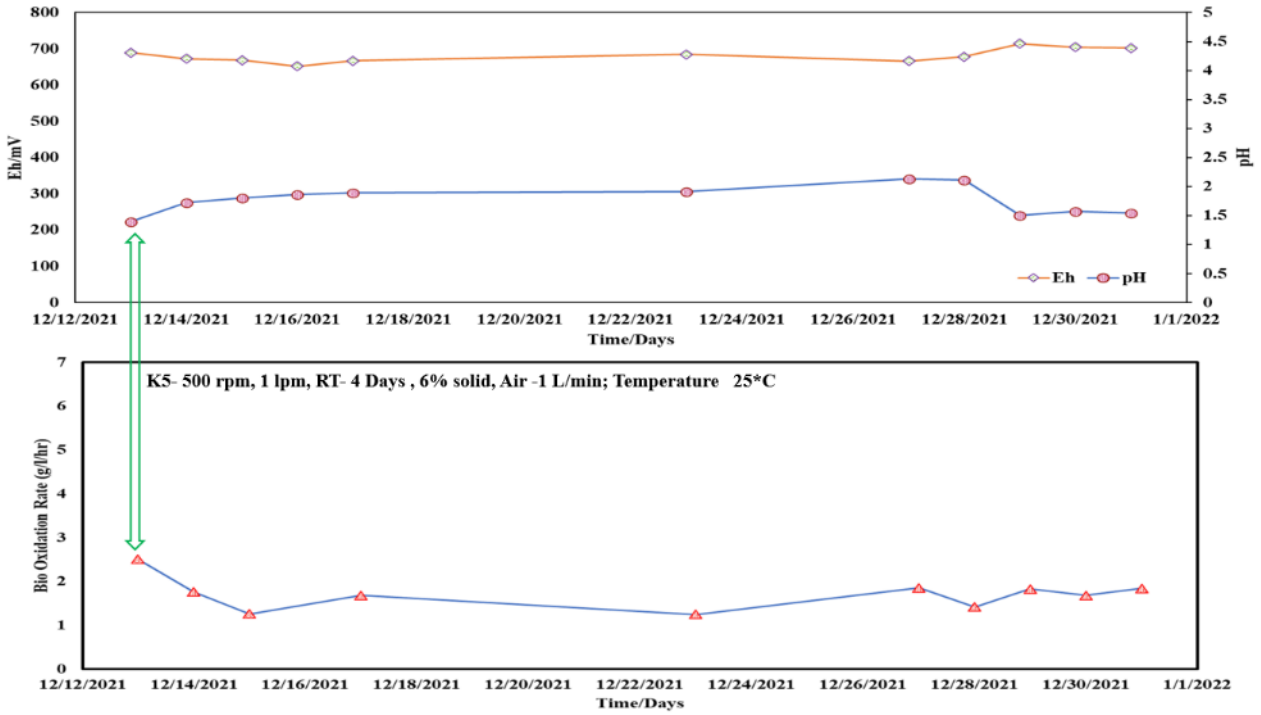



Figure 3.5.121: Overall changes in Eh, ph and biooxidation rate of K5 parameter.

Average pH, Eh, and biooxidation rate for different parameters for the entire operation period are shown in Figure 3.5.122, Figure 3.5.123, and Figure 3.5.124. It is helpful to observe that during the application of P9 conditions, pH and biooxidation rates started going down. The steady-state has been continued for at least two residence periods. So P9 might need to be re-evaluated to validate the data collected in this period. A possible reason would be the automatic timer control problem on August 27th. P1 and P2 conditions seem favorable regarding pH and biooxidation rates, though detailed analysis is needed.

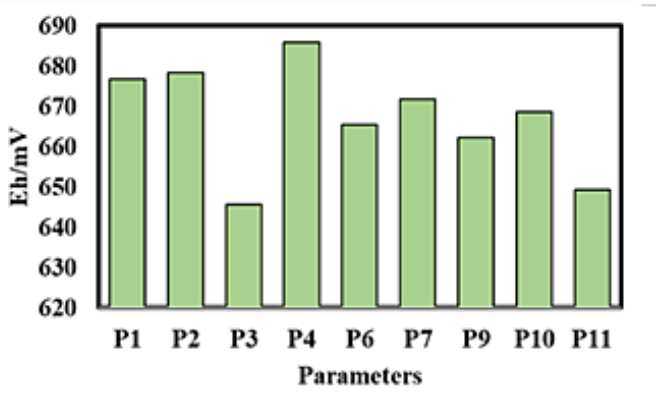



Figure 3.5.122: Average Eh for different tests (parameters are shown table 1).

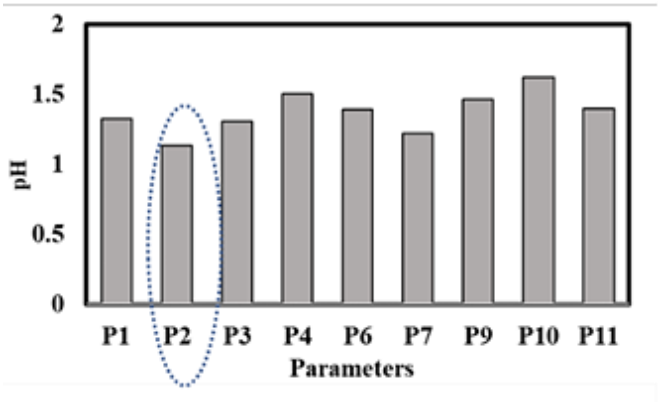



Figure 3.5.123: Average pH for different tests (parameters are shown table 1).

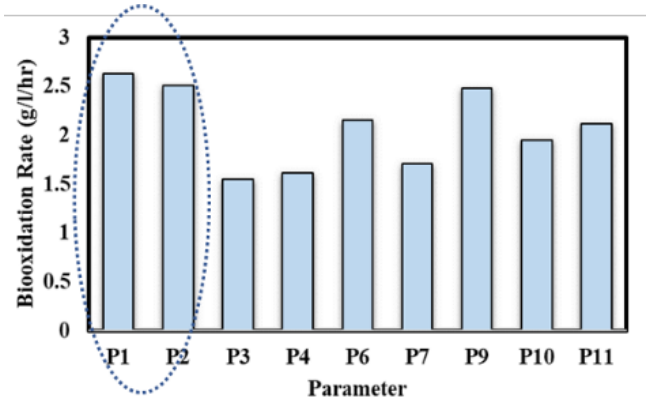



Figure 3.5.124: Average biooxidation rate for different tests (parameters are shown table 1).

**Assembling and Installation of new 3L Bioreactor:** A new 3L bioreactor was purchased from Chemglass Life Sciences. Other accessories, such as the circulating water heating, air injection, and the automatic feeding system, were connected to the new reactor (See Figure 10). All the equipment's parts were assembled in the University of Utah labs. Similar bioreactor startup procedures were followed. The team performed the necessary procedures to start the bioreactor, including cleaning, installing all parts, re-filling new solutions, maintaining temperature (inside the reactor ~35°C), pH control (~1.4 using $H_2SO_4$), and providing the supply of 9k medium. After installing the new Bioreactor, it was filled with 1.5 L of stored bioleaching solution from the old 1.5L in the Bioreactor. Overhead stirrer speed was set up at ~500 rpm, and gas flow was set up at one l/min to keep the same parameters in both bioreactors. Only feeding was started with eight times per day frequent feeding with the residence time of 6 days. No effluent extraction was done until the bioreactor solution level reached 2.4L. At that point, the automatic feeding and effluent extraction process were started. A timer was adjusted to feed a 50 ml feeding slurry in the reactor and take out a 50 ml effluent solution per feeding. Eh, and pH changes were also tracked.

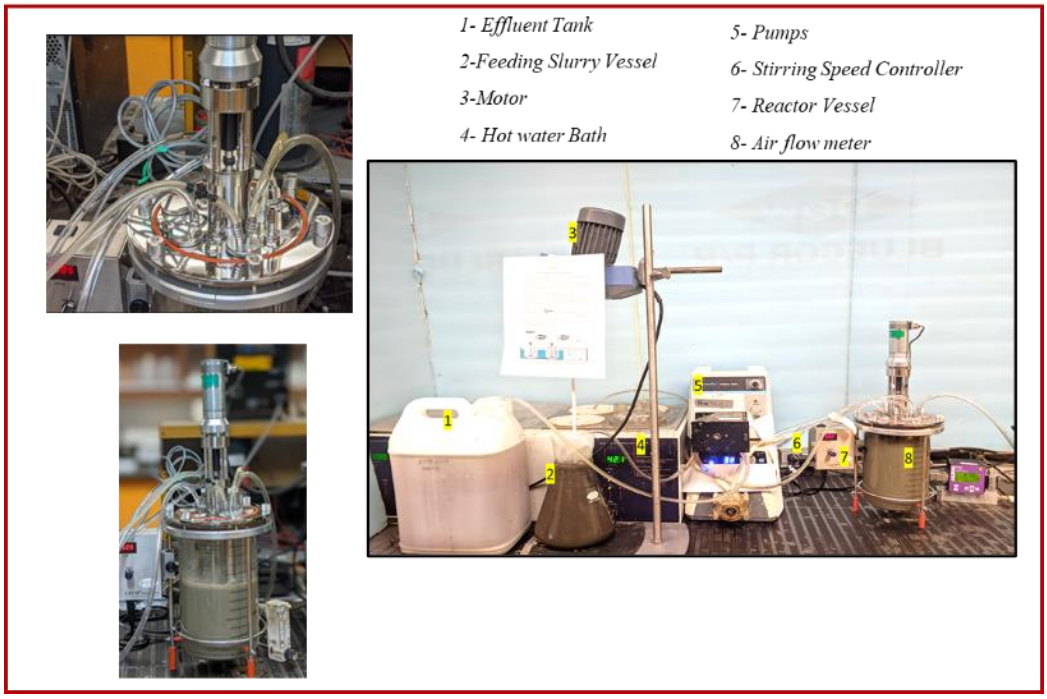



Figure 3.5.125: Digital Images of New 3L Bio-Reactor and experiment-setup

**Designing and Analysis of Single-factor experiments:** Statistics-based experimental design techniques are instrumental in the engineering world for improving the performance of a manufacturing process. They also have extensive application in the development of new methods. Most procedures can be described in several controllable variables, such as temperature, pressure, and feed rate. By using designed experiments, engineers can determine which subset of the process variables most influence process performance. With the single factor experiment, the data from the investigation could determine which solids concentration rate should be used to provide maximum biooxidation rate.

Some typical applications of statistically designed experiments in engineering design include

1. Evaluation and comparison of basic design configurations
2. Evaluation of different materials
3. Selection of design parameters so that the product will work well under a wide variety of field conditions (or so that the design will be robust)
4. Determination of key product design parameters that affect product performance

In this design of the experiment, two different methods were investigated. One was the 'random model effect,' and another was the 'fixed effect model'. During the random model effect, data points of ferrous Oxidation were collected randomly to see the impact of solids concentration in biooxidation rate. For the fixed effect model, data points for biooxidation rate were collected at the steadiest state of the bioreactor performance. Our team is interested in testing the effect of different kinds of treatments in operation. To test the hypothesis, we need to look into the ANOVA table. If the p-value is lower than the confidence level, the Alternative Hypothesis is true.

**Alternative Hypothesis; H1** = The Hypothesis that we are interested in proving (For our experiment, we want to confirm that the solids concentration difference has a significant effect on biooxidation rate)

**Null Hypothesis; H0** = The complement of the Alternative Hypothesis.

So here if:

P value > $\alpha$ (confidence level): Accept Null Hypothesis: Conclude H0

P- Value < $\alpha$ (Confidence level): Accept Alternative Hypothesis. Conclude H1

"The ANOVA partitions the total variability in the sample data into two parts. Then, the hypothesis test is based on a comparison of two independent estimates of the population variance. The total sum of squares describes the total variability in the data:

$$SS_T = \sum_{i=1}^{a}\sum_{j=1}^{n}(y_{ij} - \bar{y}..)^2$$

The expected value of the treatment sum of squares is:

$$E(SS_{Treatments}) = (a-1)\sigma^2 + n\sum_{i=1}^{a}\tau_i^2$$

and the expected value of error sum of squares is:

$$E(SS_E) = (a-1)\sigma^2$$

The mean square for treatments is:

$$MS_{Treatments} = \frac{SS_{Treatments}}{a-1}$$

The error mean square is:

$$MS_E = \frac{SS_E}{a(n-1)}$$

The equality of treatment means is:

$$F_0 = \frac{\frac{SS_{Treatments}}{a-1}}{\frac{SS_E}{a(n-1)}} = \frac{MS_{Treatments}}{MS_E}$$

**Multiple Comparisons Following the ANOVA:** The ANOVA doesn't identify which means are different. Methods for investigating this issue are called multiple comparisons methods. Many of these procedures are available. Here we describe a very simple one, Fisher's least significant difference (LSD) method provides a comparison. In the LSD method if the LSD value is greater than the difference in the pairs of the average of

replicants then those pairs of treatments are significantly different.

$$|\bar{y}_i - \bar{y}_j| > LSD$$

LSD, the least significant difference, is defined as:

$$LSD = t_{\alpha/2,a(n-1)}\sqrt{\frac{2MS_E}{n}}$$

where,

$\alpha$ = Confidence level

$a$ = Number of treatments

$n$ =Number of replicants

$MS_E$ = Mean Square of error

Single Factor design of experiments – Random Model effect

The data table has been shown in Table 3.5.19: Data table and ANOVA table for RANDOM effect model. It can be seen that ferrous ion oxidation rate data was collected randomly and ANOVA test were done to find out if there was a significant difference in effect of solids concentration. From the ANOVA table P- value was lower than the confidence level 95% (alpha-0.05) so we accept H1 and conclude that solids concentration has a significant effect on biooxidation rate.

Table 3.5.19: Data table and ANOVA table for RANDOM effect model.

| | Ferrous Oxiation Rate | | | | | |
|---|---|---|---|---|---|---|
| Solid Percentage | Day 1 | Day 2 | Day 3 | Day 4 | Day 5 | Day 6 |
| 5 | 1.01 | 0.674 | 1.31 | 0.84 | 1.03 | 0.976 |
| 6 | 1.004 | 1.006 | 1.147 | 0.787 | 0.874 | 1.249 |
| 8 | 1.22 | 1.444 | 1.313 | 2.096 | 1.674 | 2.652 |
| 10 | 1.225 | 2.232 | 1.546 | 1.433 | 1.258 | 1.116 |

**ANOVA Table**

| Source | DoF | SS | MS | F0 | P-value |
|---|---|---|---|---|---|
| Treatment | 3 | 2.43633767 | 0.812112556 | 28.23614788 | 2.0419E-06 |
| Blocks | 5 | 0.14380725 | 0.02876145 | 0.169411208 | |
| Error | 15 | 2.54659508 | 0.169773006 | | |
| Total | 23 | 5.12674 | | | |

Confidence level – 95%

**Fisher's LSD method to check Significant difference in the treatments:** Since we already learned that there is difference in the solids concentration. So, LSD fisher method has been applied to see differences. According to the law a two-sided alternative hypothesis, the pair of means Ui and Uj would be declared significantly different if

$$\left|\bar{y}_{i.} - \bar{y}_{j.}\right| > LSD$$

From table below it can be seen that difference of solids concentration 5 vs 8; 5Vs 10; 6 vs 8 and 8 vs 10 values were higher than LSD ($t_0$) value. So, it can be concluded that there was significant difference in these pairs. Also, a visual representation plot has been showed in Figure 6. From this plot we can see that 8% percent of Solids had the most significant effect on Biooxidation rate and

solids percent of 5 had the lowest effect on biooxidation rate when the data set was collected randomly.

Table 3.5.20: LSD table

| Solid percentage | Treatment effects / average | DR | Difference | | LSD |
|---|---|---|---|---|---|
| 5 | 0.973333333 | 5v6 | 0.037833333 | < | 0.660 |
| 6 | 1.011166667 | 5v8 | 0.759833333 | > | 0.660 |
| 8 | 1.733166667 | 5v10 | 0.759833333 | > | 0.660 |
| 10 | 1.468333333 | 6v8 | 0.722 | > | 0.660 |
| | | 6v10 | 0.457166667 | < | 0.660 |
| | | 8v10 | 0.264833333 | > | 0.660 |

**Normality check:** To see if the data was normally collected or distributed, a normally check has been done and the normality plot has been presented in Figure 3.5.126. Since there was observation of linear relationship so it can be derived that data was normally distributed.

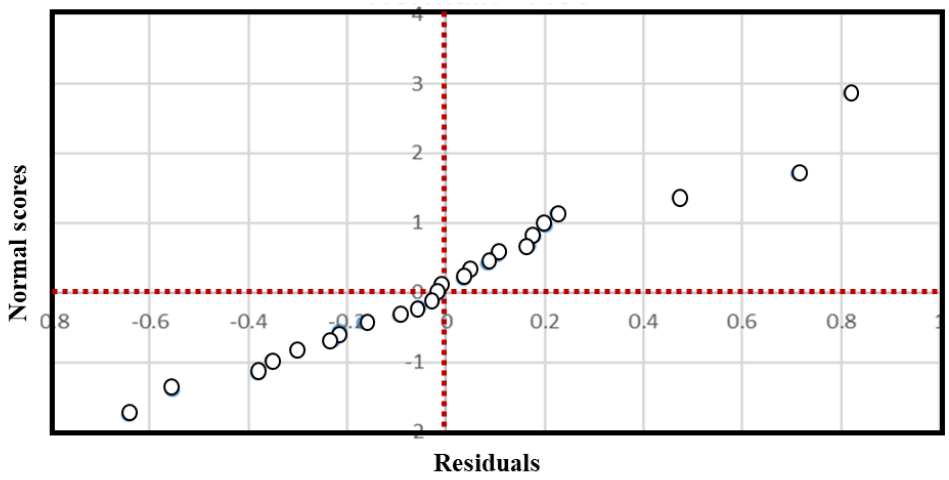



Figure 3.5.126: Normality check plot

**Fixed Effect Model Test:** The data table has been showed in Table 3.5.21. Ferrous oxidation rate data was collected by following the days when the bioreactor operation was in super steady state and provided highest rate of bio- Oxidation. ANOVA test has been done to find out if there is any significant difference in effect of solids concentration. From the ANOVA table; - P- Value was lower than the confidence level 95% (alpha-0.05) so it accepts H1 and concludes that there is at least one or more than one solids concentration has significant effect on biooxidation rate.

Table 3.5.21: Data table and ANOVA table for fixed effect model.

DATA Table

| Solid Percentage | Changes in Bio Oxidation Rate | | | | | Total | Average |
|---|---|---|---|---|---|---|---|
| 5 | 0.485 | 0.523 | 1.286 | 0.713 | 1.388 | 4.395 | 0.879 |
| 6 | 1.924 | 1.249 | 1.476 | 2.327 | 1.826 | 8.802 | 1.7604 |
| 8 | 2.46 | 1.02 | 1.04 | 1.13 | 2.096 | 7.746 | 1.5492 |
| 10 | 2.012 | 2.232 | 2.004 | 1.339 | 1.182 | 8.769 | 1.7538 |
| | | | | | | | |
| | | | | | | | |
| | | | | | | | 1.4856 |
| | | | | | | | grand mean |

ANOVA table

| Source | DOF | SS | MS | F | P-value |
|---|---|---|---|---|---|
| Treatment | 3 | 2.597274 | 0.865758 | 3.35455833 | 0.04528953 |
| Error | 16 | 4.1293448 | 0.25808405 | | |
| Total | 19 | 6.7266188 | | | |

Fisher's LSD method to check Significant difference in the treatments: From Table 3.5.22 it can be seen that difference of solids concentration (1-5% Solids; 2- 6 % solids; 3- 8% solids and 4-10% solids) between 5 vs 10 and 5 vs 6 values were higher than LSD ($t_0$) value. So, it can be concluded that there was significant difference in these pairs.

Table 3.5.22 LSD table for solids effects

| Diif. Range | Diff | Comp. | LSD |
|---|---|---|---|
| 4 vs 3 | 0.2046 | < | 0.681125 |
| 4 vs 2 | 0.0066 | < | 0.681125 |
| 4 vs 1 | 0.8748 | > | 0.681125 |
| 3 vs 2 | 0.2112 | < | 0.681125 |
| 3 vs 1 | 0.6702 | < | 0.681125 |
| 2 vs 1 | 0.8814 | > | 0.681125 |

Effect of operating parameters on the oxygen consumption rate on new 3L Bioreactor

Response surface design is a way of presenting the data, primarily visualizing the research findings. For Instance, if we have two independent variables, namely rotation speed and airflow rate, and we want to evaluate the effect on the oxygen consumption rate, the needed visual representation of these corresponding data is called response surface methodology. The response surface was analyzed in Minitab. The primary reason for the response surface was to find not just one number of rotation speed and/or air flow rate. We wanted to see a range of regions where rotation speed or air flow rate would be more productive. The contour plot and the surface plot of the optimization test are shown in Figure 3.5.127. There was one optimized oxygen consumption rate under one condition. This test shows us the direction of optimization on the operating parameters.

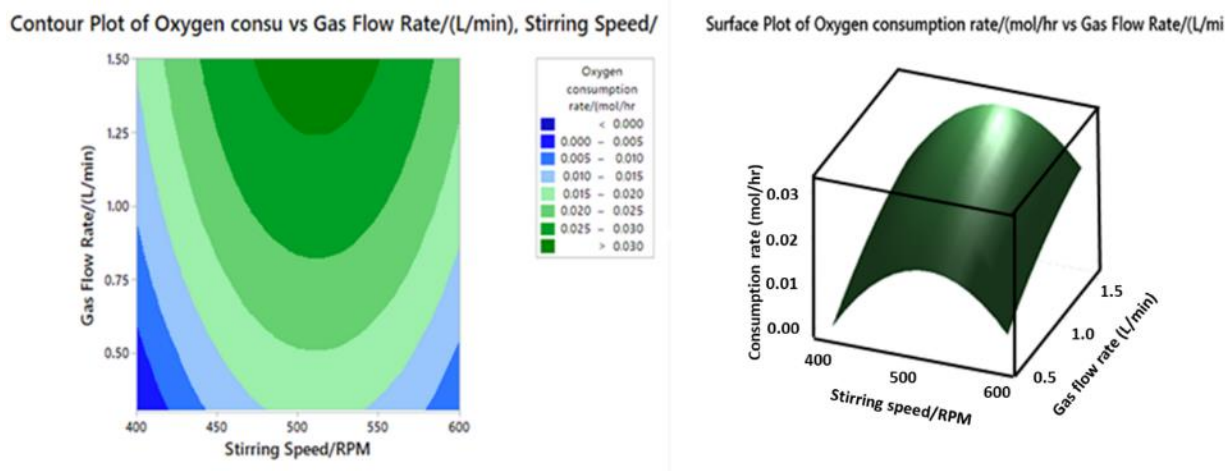



Figure 3.5.127: The contour plot and the surface plot of the optimization test.

**Titration Test for the Bioleaching Solution:** Titration tests were conducted throughout the period to determine the acid concentration in the leaching solution. The test procedures are listed below.

(1) Preparation of 0.5 M NaOH solution by dissolving NaOH pellet into DI water.
(2) 30 ml of the leaching solution from the Bioreactor was taken and it was filtered to remove solids particles.
(3) Use of a burette and that was filled with the 0.5 M NaOH solution, initial reading on the burette was recorded.
(4) Use of 20 mL of the filtered leaching solution to an Erlenmeyer flask and putting it under the burette.
(5) Opening of the valve of the burette and start titration, tracking the pH of the solution with a pH meter till the pH reading reach 2.2, closing of valve at this reading.
(6) Recording of the final reading on the burette.

With this procedure, one titration test was conducted. The acid concentration has been showed for 6 different titration tests in Table 3.5.23 .

Table 3.5.23 Comparison of dates with pH and acid concentration

| Date | Initial pH | Acid Concentration |
|---|---|---|
| 1/1/2021 | 1.23 | 0.225M |
| 1/6/2021 | 1.09 | 0.250M |
| 1/14/2021 | 1.32 | 0.175M |
| 1/27/2021 | 1.18 | 0.275M |
| 2/2/2021 | | |
| 2/10/2021 | 1.59 | 0.210 M |
| 2/24/2021 | 1.42 | 0.175 M |
| 3/3/2021 | 1.42 | 0.150 M |
| 3/19/2021 | 1.14 | 0.250 M |

**Mass balance analysis:** Mass balance analysis was conducted for the bioreactor, monitored for a 3.5-days running period. During this period, materials were used (in regular feeding) into the bioreactor and taken out of bioreactor four times a day. The feeding material was made by mixing

6 g of pyrite with 50 ml 9K medium (no Fe, no acid) and 50 ml DI water. Therefore, the actual solids concentration was about 5.74%.

- ❑ The weights of the input and output materials were evaluated for mass balance tracking. The input material and output material were evaluated for 3.5 days residence time with 6% solids feeding.
- ❑ Leftover feeding slurry was filtered, which resulted in 900 ml liquid and 63.675 g pyrite solids. In 900 ml it should have 54 g pyrite whereas we found 63.675 g. It indicates that feeding was less than 6% solids. The corrected feeding slurry solids concentration that went into the bioreactor was 5.57 %
- ❑ The effluent was filtered that resulted in 2250ml liquid (Weight ~2299 g). The solution density was 1.11 g/ml. Solids weight -129.15 g. Solids concentration – 5.74 %.
- ❑ Mass balance calculation is as follows:

Input weight= solids weight + liquid weight

=2250×5.57% + (2250-2250×5.57%/5) (Density of solids was considered ~ 5)

= 125.325+2275.065

=2400.39g

=Output: = 2299+129.15

=2428.15g

Error = (2428.15-2400.39)/2400.39×100%

=1.156%

This error concentration can vary for different solids density range.

**Estimation of the acid generation rate in the bioreactor:** During the standard operation period of the bioreactor at 3.5 days residence time with automatic feeding, 9.6 g pyrite ($FeS_2$) was supplied to the bioreactor with 80 ml 9K medium and 80 ml DI water to the bioreactor 4 times a day. Simultaneously, 160 ml of the leaching solution in the bioreactor was pumped to the effluent vessel four times a day. Therefore, the acid generation rate and efficiency can be estimated based on the mass balance of the main input and output materials.

The average pH was found to be 1.23 during the period of mass balance analysis. Based on the fact that 400 mL of acid solution (with pH 1.23) was generated per day, the amount of acid can be calculated.

Bio Leaching Solution form WBB 419 Bioreactor

Initial pH-1.23

$C_{NaOH}$= 0.5 M

$V_{NaOH}$= 11.5 ml

$V_{Acid}$= 20 ml

We know,

$C_{NaOH} \cdot V_{NaOH} = C_{Acid} \cdot V_{Acid}$

$0.5M \times 11.5\ mL = C_{Acid} \times 20mL$

$C_{Acid} = 0.2875M$

Therefore, the acid generation rate (amount of $H^+$ generated per day) was:

$0.2875mol/L \times 0.4L = 0.115$ mol/day in the bioreactor.

Furthermore, based on the overall reaction of the bioleaching process shown in Eq. (1), the efficiency of $FeS_2$ transformation can be calculated. Theoretically, for 1 mole of $FeS_2$ consumed, there is 1 mole of $H^+$ generated in the Bioreactor.

$$4FeS_2 + 2H_2O + 15O_2 = 2H_2SO_4 + 2Fe_2(SO_4)_3 \quad (1)$$

The amount of substance in 38.4 g pyrite feed is:

$$m_{FeS_2} = \frac{\frac{38.4}{\text{g}}}{120(g/\text{mol})} = 0.32 \text{ mol.}$$

Therefore, the theoretical $H^+$ generation from $FeS_2$ was 0.32 mol. However, much of the acid produced from pyrite was consumed by the large amount of calcite, which was usually near or above 20 % of the total weight (0.2*38.4 g feed /day = 7.7 grams or 0.077 moles, which require 2 moles of acid per mole of calcite for a total of 0.154 moles of acid consumption by calcite). Some acid was also consumed by other minerals.

**Consistency Check on Feeding Solution:** To maintain a steady-state condition in the bioreactor, the correct feeding solution should feed into the system. To check that, a consistency check was done. At 3.5 days residence time, this experiment was conducted.

During this RT period four times (160 ml with 6% Solids – 9.6 g), feeding per day was continued in a day. So, 14 samples were collected to complete the residence time. First, all the samples were collected by setting up an automatic feeding solution collection system. Then the solution was filtered, collected and measured (feeding liquid and solids). Figure 3.5.128 shows the amount of liquid and solids that were found in 14 feedings.

Calculation and Results:

The calculated amount of liquid in the feeding solution was: 160 ml

The computed amount of solids in the feeding solution was: 9.6g.

Checked the consistency during the residence time of 3.5 days

Number of feeding analyzed – 14 (4 times a day)

The average amount of liquid- 157.85 ml

Average amount of solids – 9.28

Corrected solids concentration – 9.67 which is very close to the corrected solids concentration from the mass balance .

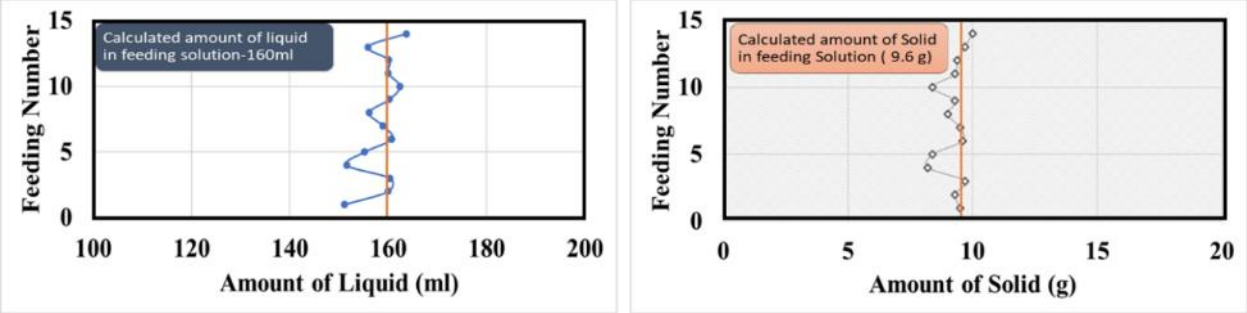



Figure 3.5.128: Consistency check on feeding solution and amount of solids calculation

The average amount of liquid and solids found during the experiment was reasonably close to the calculated amount of liquid which means there is consistent feeding.

**Magnetite analysis on new feeding solution:** Two buckets of new fine pyrite slurry were received. They were dried later and splatted and mixed. Subsequently, 100 g sample was collected to evaluate the magnetite concentration in the new feeding solution. It was found that the new pyrite sample has approximately 1.07 % magnetite in it. Figure 3.5.129 shows the magnetite characterization of the new feed solution.

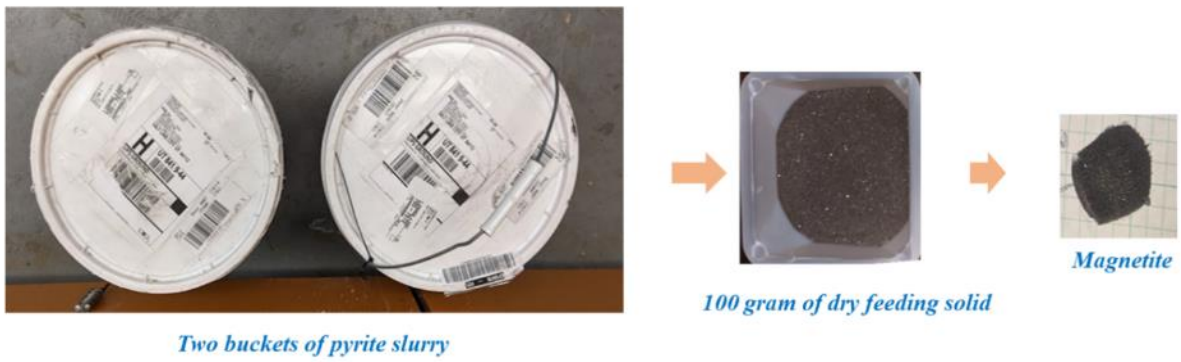



Figure 3.5.129: Magnetite analysis on new feeding sample.

Summary of processing parameters and effect of residence time in biooxidation rate ( three steady state condition): Below is the list of all the parameters that were used in the first part of the testing.

WBB 419 – Bio reactor ( 100 % and 20% pure pyrite parameters not included)

60% Pure Pyrite – 1.5 L Bio Reactor

Table 3.5.24 fine particle concentrate with approximately 60 % pure pyrite parameter data set

| 60% Pure Pyrite – 1.5 L Bio Reactor | | | |
|---|---|---|---|
| Residence time (days) | Solid percentage | Stirring speed (rpm) | Air flow rate (lpm) |
| 15 | 7 | 680 | 0.5 |
| 15 | 5 | 680 | 0.5 |
| 7.5 | 5 | 680 | 0.5 |
| 5 | 5 | 680 | 0.5 |
| 3.75 | 5 | 680 | 0.5 |
| 15 | 7 | 680 | 0.3 |

| Fine Pyrite – 1.5 L Bio Reactor | | | |
|---|---|---|---|
| Residence time (days) | Solid percentage | Stirring speed (rpm) | Air flow rate (lpm) |
| 7.5 | 5 | 680 | 0.3 |
| 5 | 5 | 680 | 0.5 |

Table 3.5.25 Fine Pyrite – New 3L Bio reactor parameters

| Fine Pyrite – new 3L Bio Reactor | | | | |
|---|---|---|---|---|
| Residence time (days) | Solid percentage | Stirring speed (rpm) | Air flow rate (lpm) | Days of operation |
| 12 | 6 | 500 | 1 | 26 |
| 6 | 8 | 500 | 1 | 32 |
| 4 | 6 | 500 | 1 | 28 |
| 3.5 | 6 | 500 | 1 | 54 |

Three blue marked parameters has been collected for making a fixed effect model design of experiment to analyze the effect of different residence time on biooxidation rate.

Table 3.5.26: MCEB 2474 bio reactor 60% pure pyrite

| 60% Pyrite –3L Bio Reactor MCEB 2474 | | | | |
|---|---|---|---|---|
| Residence time (days) | Solid percentage | Stirring speed (rpm) | Air flow rate (lpm) | Days of operation |
| 10 | 8 | 500 | 1 | 94 |
| 7.5 | 8 | 500 | 1 | 41 |
| 4 | 8 | 500 | 1 | 23 |
| 6 | 8 | 500 | 1 | 12 |
| 6 | 10 | 500 | 1 | 14 |

Table 3.5.27: Fine pyrite data set information

| Fine Pyrite –3L Bio Reactor MCEB 2474 | | | | |
|---|---|---|---|---|
| Residence time (days) | Solid percentage | Stirring speed (rpm) | Air flow rate (lpm) | Days of operation |
| 6 | 10 | 500 | 1 | 34 |
| 8 | 10 | 500 | 1 | 42 |
| 6 | 8 | 500 | 1 | 36 |
| 6 | 12 | 500 | 1 | 38 |
| 4 | 12 | 500 | 1 | 17 |

In this set of parameters, 100% pure pyrite and 20% pure pyrite parameters were not included. The parameters of 6% fine pyrite feeding with stirring speed of 500 rpm and air flow rate of 1 lpm with three different residence times were asessed. The effect of the parameters on the biooxidation rate was evaluated. A fixed effect model design of experiments was utilized where 6 steady-state data points for ferrous oxidation rate were collected and analyzed.

**Alternative Hypothesis; H1** = The Hypothesis that we were interested in proving (For our experiment, we want to examine if the solids concentration difference has a significant effect in biooxidation rate)

**Null Hypothesis; H0** = The complement of the alternative Hypothesis.

So here

P value > Alpha (confidence level): Accept Null Hypothesis: Conclude H0

P- Value < Alpha (Confidence level): Accept Alternative Hypothesis. Conclude H1. Figure 7.130 (a) and (b) shows the data set table and ANOVA table.

To test the Hypothesis, we need to look into the ANOVA table. It can be seen that the p-value was lower than the confidence level, so we accept the alternative Hypothesis and conclude H1. This suggested that residence time has a significant effect on biooxidation rate.

The ANOVA doesn't identify which means were different. Methods for investigating this issue were called multiple comparison methods. Many of these procedures were available. Here we describe a straightforward one, Fisher's least significant difference (LSD) method in the LSD method; if the LSD value is greater than the difference in the pairs of the average of replicants, then those pairs of treatments are significantly different.

$$|\bar{y}_{i\cdot} - \bar{y}_{j\cdot}| > \text{LSD}$$

LSD, the least significant difference, is

$$\text{LSD} = t_{\alpha/2,a(n-1)}\sqrt{\frac{2MS_E}{n}}$$

Where

alpha (α) = Confidence level

a = Number of treatments

n =Number of replicants

MSe = Mean Square of error

Fisher's LSD method to check Significant differences in the treatments:

From Figure 3.5.130 (c), it can be seen that the difference between RT 3.5 and 4 days was lower than the LSD value, which indicated that these two parameters do not have any significant difference. Still, the difference between RT 12 with four days and five days was lower than the LSD value that means they were significantly different. Also, a visual representation plot has been shown in Figure 10. From this plot, we can notice that RT of 4 days had the highest oxidation rate while RT of 12 days had the lowest. This kind of statistical experiment was a helpful method to analyze these kinds of data of the experiments.

| RT | | | Bio oxidation Rate | | | | Total | Average |
|---|---|---|---|---|---|---|---|---|
| 12 | 1.004 | 1.006 | 1.147 | 1.627 | 0.94 | 1.714 | 7.438 | 1.23966667 |
| 4 | 2.765 | 2.539 | 1.824 | 1.869 | 1.794 | 1.919 | 12.71 | 2.11833333 |
| 3.5 | 1.346 | 1.475 | 1.914 | 2.2 | 2.414 | 1.701 | 11.05 | 1.84166667 |
| | | | | | | | | 1.7332222 |
| | | | | | | | | Grand mean |

(a)

| Source | DOF | SS | MS | F | P-value |
|---|---|---|---|---|---|
| Treatment | 2 | 2.42200711 | 1.211003556 | 7.78438009 | 0.00479845 |
| Error | 15 | 2.333526 | 0.1555684 | | |
| Total | 17 | 4.75553311 | | | |

(b)

$|\bar{y}_{i\cdot} - \bar{y}_{j\cdot}| > \text{LSD}$

LSD 0.48537245

**Least Significant Difference for Multiple Comparisons**

$$\text{LSD} = t_{\alpha/2,a(n-1)}\sqrt{\frac{2MS_E}{n}}$$

| RT | treatment effect | | | | | |
|---|---|---|---|---|---|---|
| 12 days | 1.239666667 | 0 | 3.5 days vs 4 days | 0.27666667 | < | 0.48537245 |
| 4 days | 2.118333333 | 0 | 3.5 days vs 12 days | 0.602 | > | 0.48537245 |
| 3.5 days | 1.841666667 | 0 | 4 days vs 12 days | 0.87866667 | > | 0.48537245 |

(c)

Figure 3.5.130. (a) Data table; (b) ANOVA Table and (c)LSD table for residence time analysis

**Standardization of both reactors:** From April 19, both the bioreactors were run the same operational conditions to help ensure the reactors were standardized. Both reactors shared the feedstock culture of bacteria by mixing some of the reactor solution from one reactor into the other. The identical parameters were applied to both bioreactors. A similar rotational speed, airflow rate, and solids concentration were maintained in the reactor. The results had been evaluated to determine if the same operating conditions in two reactors in two labs give the same biooxidation rate, Acid generation, and pH. The same set of manual airflow meters was introduced to both bioreactors simultaneously. Figure 3.5.131 shows the digital images of both bioreactors with a new set of the manual airflow meter.




Figure 3.5.131. The digital images of both bioreactors with new set of manual air flow meter.

**<u>SEM images of the pyrite concentrate and effluent solids of different parameter</u>:** Only the fine pyrite concentrate has been used during the bioreactor steady-state analysis. Four different magnifications were used for imaging. It can be seen that various particles had regular surfaces with less variation in size. A significant amount of structured crystal particles, which were supposed to be pyrite particles, were present in the sample with some white-colored particles, silica, and rough structured particles, mostly clay. Higher magnification images were collected to assess the structural differences between crystalline and rugged white particles. It was observed that there was a stack of crystalline and porous surface particles and a generous amount of other rough surface particles. Figure 3.5.132 shows SEM imaging results of different pyrite particles at different magnifications.

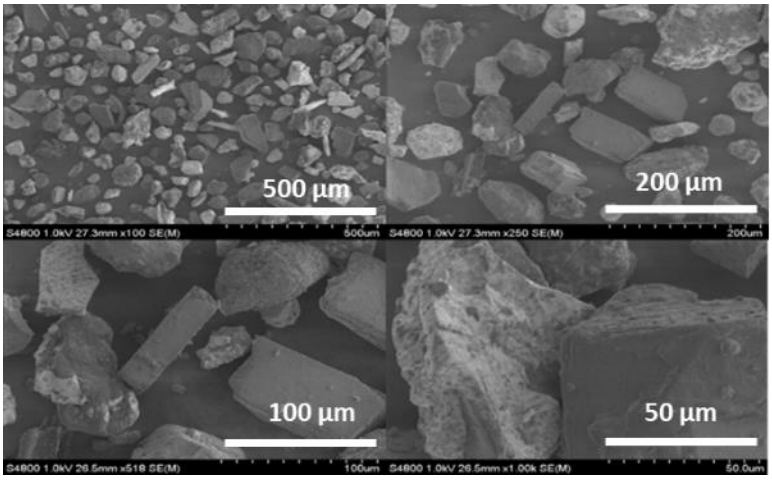



Figure 3.5.132. SEM images of new Fine Pyrite Concentrate at different magnifications.

Figure 3.5.133 shows the effluent solidss' SEM images from P1, P2, and P7 tests. It was observed that the micromorphological structure of the solids was different than the pyrite feed particles, as various rough interfaces are visible in all particles. For the P2 sample, it was easy to see a small amount of crystalline structure still present compared to the P7 effluent sample. Thus, there was evidence that most pyrite particles have been oxidized during the application of P2 conditions.

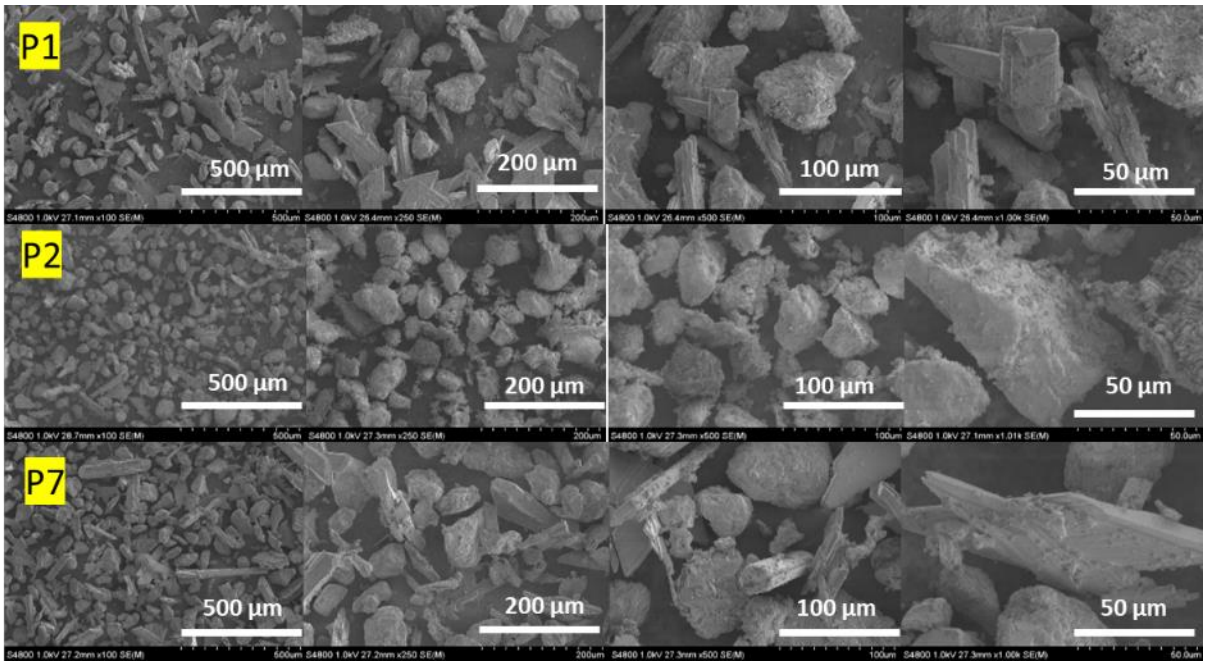



Figure 3.5.133: SEM images of effluent solidss from P1, P2, and P7 conditions.

P4, P6, and P10 effluent solids identify that most of the solids effluent particle of P4 and P6 had been migrated to rugged surface from crystal and smooth texture of feed pyrite particle. This is clear identification of oxidation of pyrite particles at a higher concentration. But when we look into the P10 solids, we can see that some smooth and unreacted surface particles were still present. So there was a chance that a significant portion of unoxidized pyrite came out with the effluent solution without being oxidized. P7, P11, and P12 have been shown. It can be seen how the pyrite surfaces were affected during the biooxidation during P7 and P11 operation. P12 images have been unclear due to the operational error. Figure 3.5.134 shows the results.

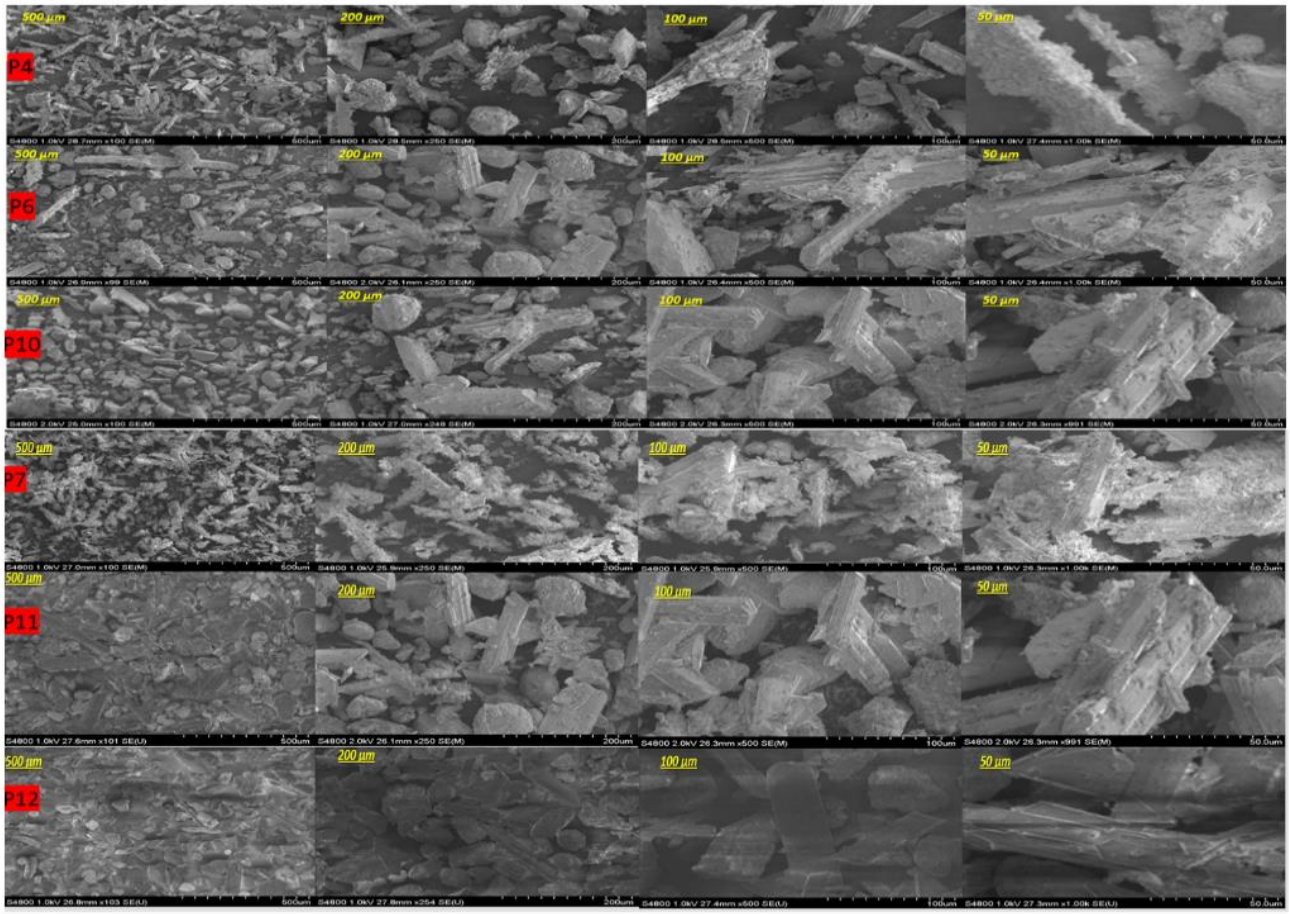



Figure 3.5.134. SEM images of effluent solidss from P1, P2, and P7 conditions.

**EDS mapping and point analysis of the Fine pyrite concentrate:** Figure 3.5.135 shows the EDS mapping and point analysis of the pyrite sample. It can be seen that a significant number of $FeS_2$ particles was observed because Fe and S can be traced in identical points. Also, silica, clay materials, and calcite were determined by identifying their positions in the map images. From the point analysis, it can be seen that there were traces for Fe, S, C, Ca, Al, Si, and K.

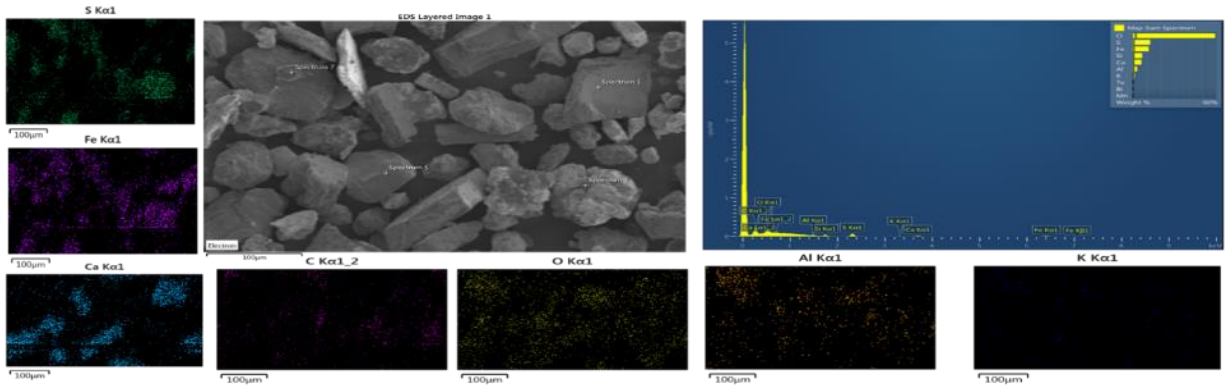



Figure 3.5.135. EDS Mapping and point analysis results for Fine Pyrite.

**Elemental Mass balance analysis of steady-state parameter:** Mass balance analysis was done for different steady-state experimental conditions (P1, P2, P10). For such research, the total input feed (input mass of various elements from pyrite concentrate + total mass of dissolved ingredients) and output (final mass of multiple features in effluent solidss, total dissolved mass in collected solution, and final total dissolved mass of elements in bioleaching solution) were monitored. In most standard cases, the expected ratio of whole input feed and output for targeted details should be 1. In most cases, it can be noticed that this ratio is close to 1. The team also calculated the relative error (%) based on the following formula:

$$Error\ (\%) = \frac{Output\ mass - input\ mass}{input\ mass} \times 100$$

It can be seen that in most cases, the error was small.

The input/output mass ratio is plotted for different experimental settings, and the % error is also shown (See Figure 3.5.136-7.149). Some of the tests need to be repeated, and hence, data are not presented here. It can be seen that the mass balance seemed to be complete within reasonable error.

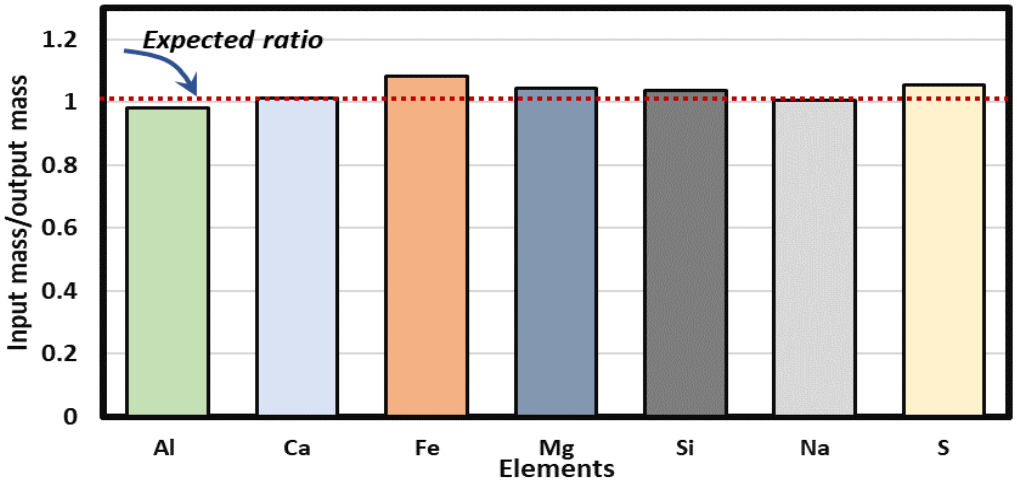



Figure 3.5.136. The ratio of input vs. output mass of various elements for experiment P1

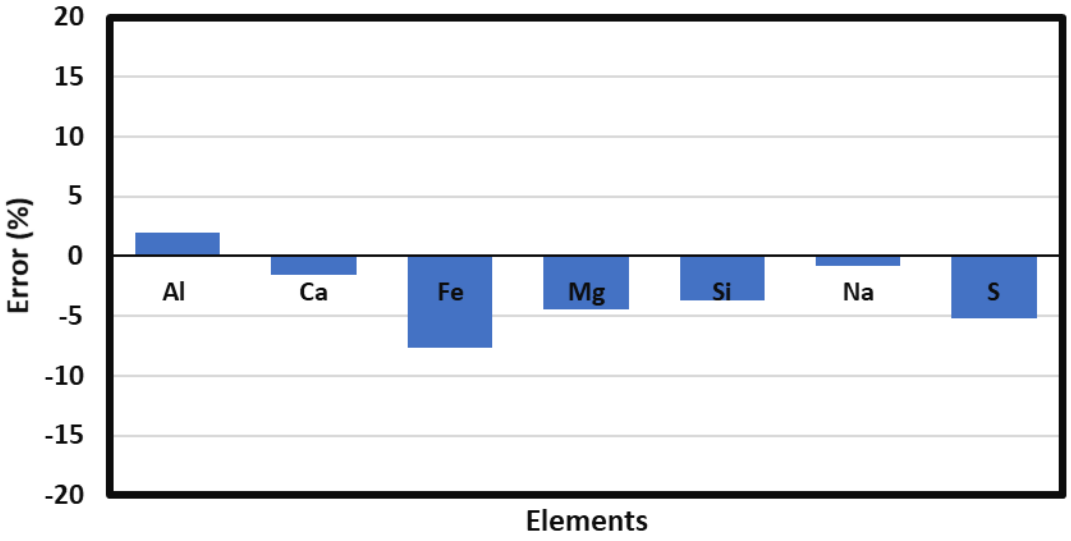



Figure 3.5.137. % Error for mass balance for various elements for experiment P1

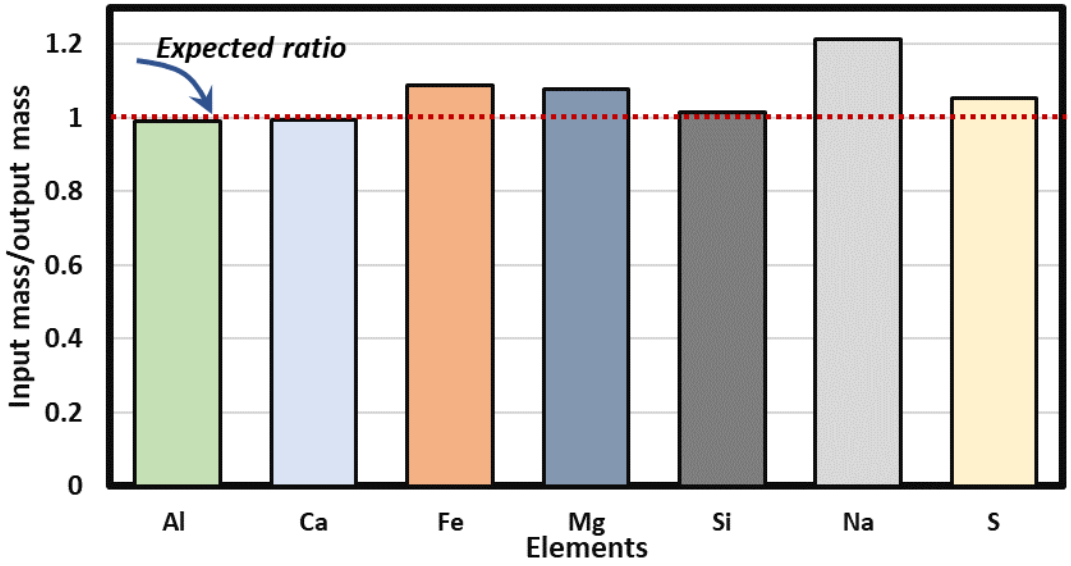



Figure 3.5.138. The ratio of input vs. output mass of various elements for experiment P2

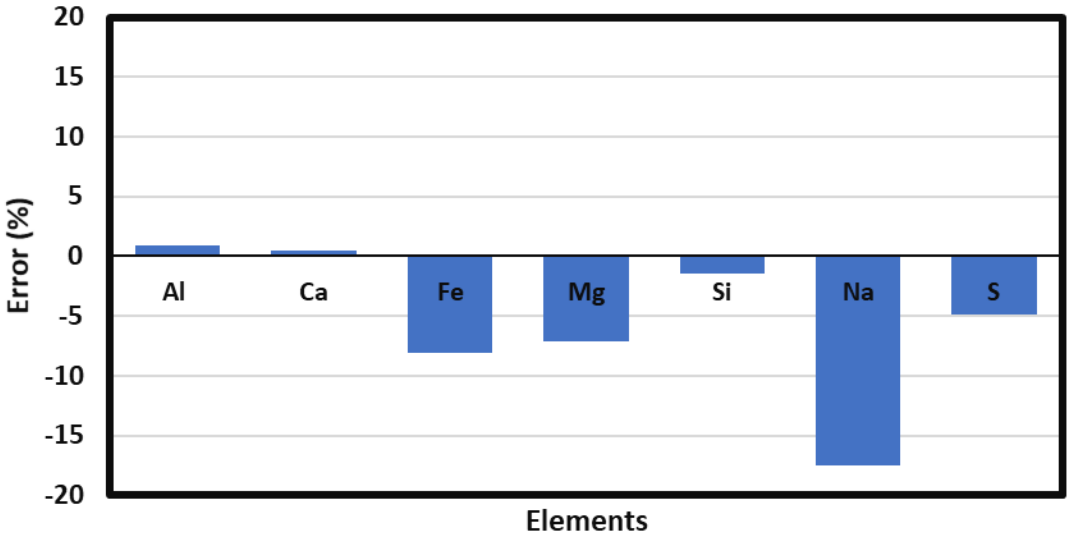



Figure 3.5.139. % Error for mass balance for various elements for experiment P2

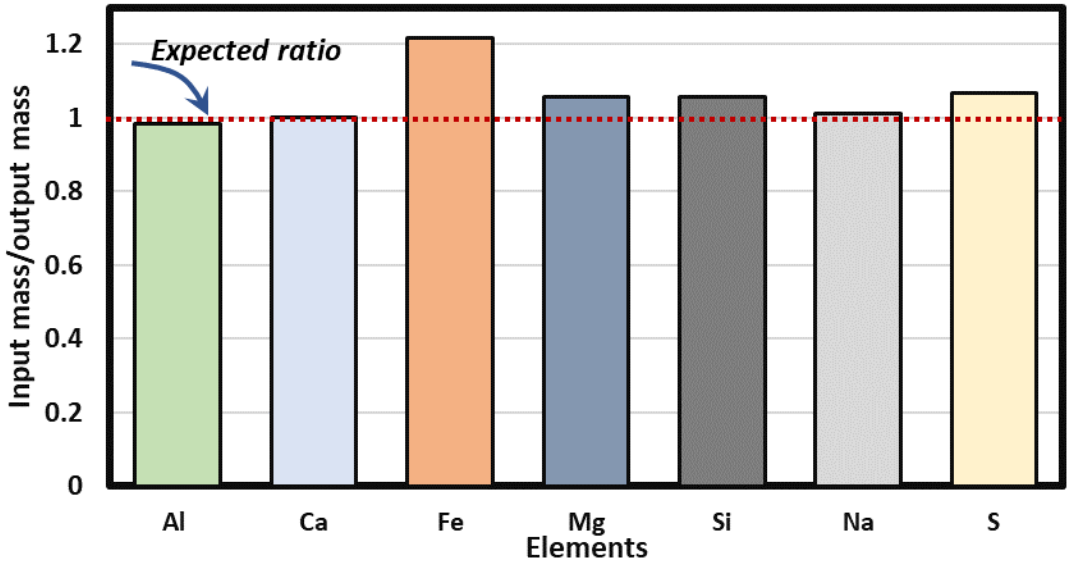



Figure 3.5.140. The ratio of input vs. output mass of various elements for experiment P3

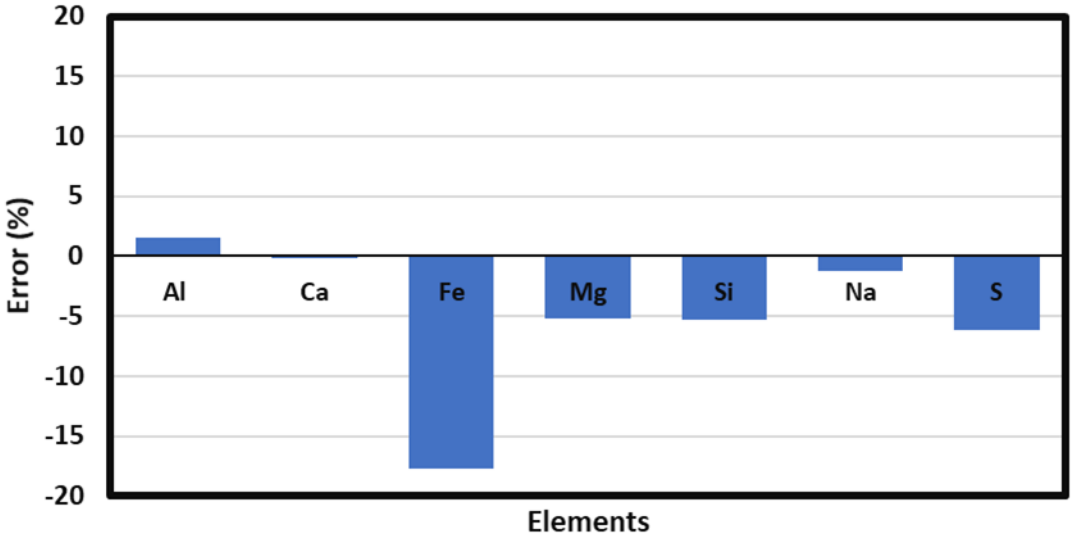



Figure 3.5.141. % Error for mass balance for various elements for experiment P3

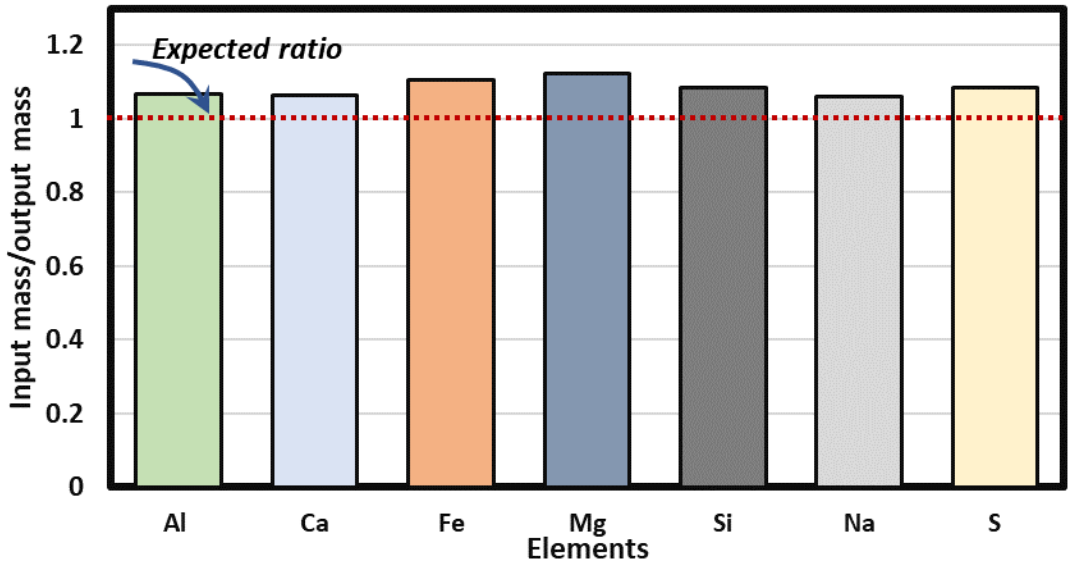



Figure 3.5.142. The ratio of input vs. output mass of various elements for experiment P4

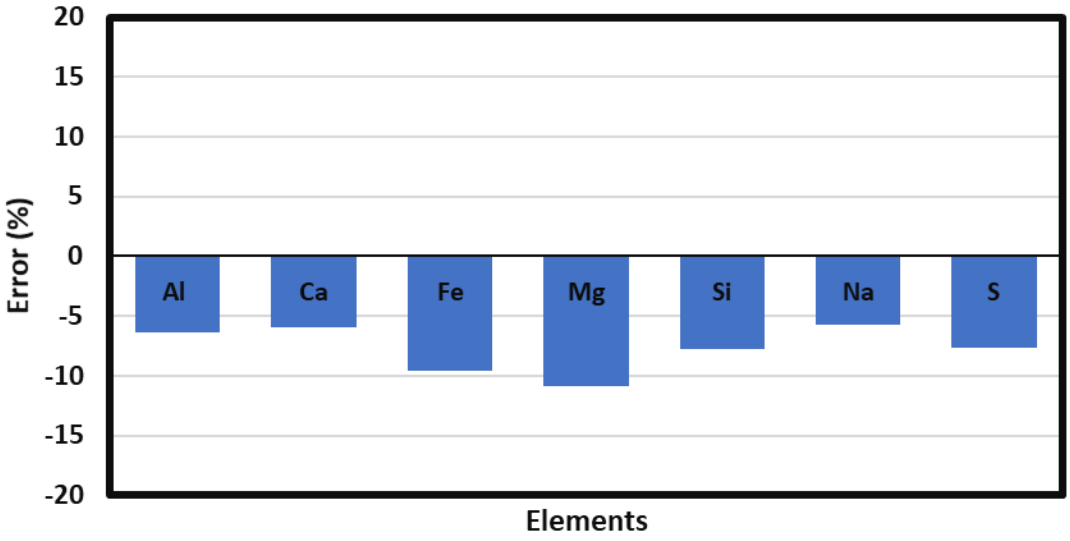



Figure 3.5.143. % Error for mass balance for various elements for experiment P4

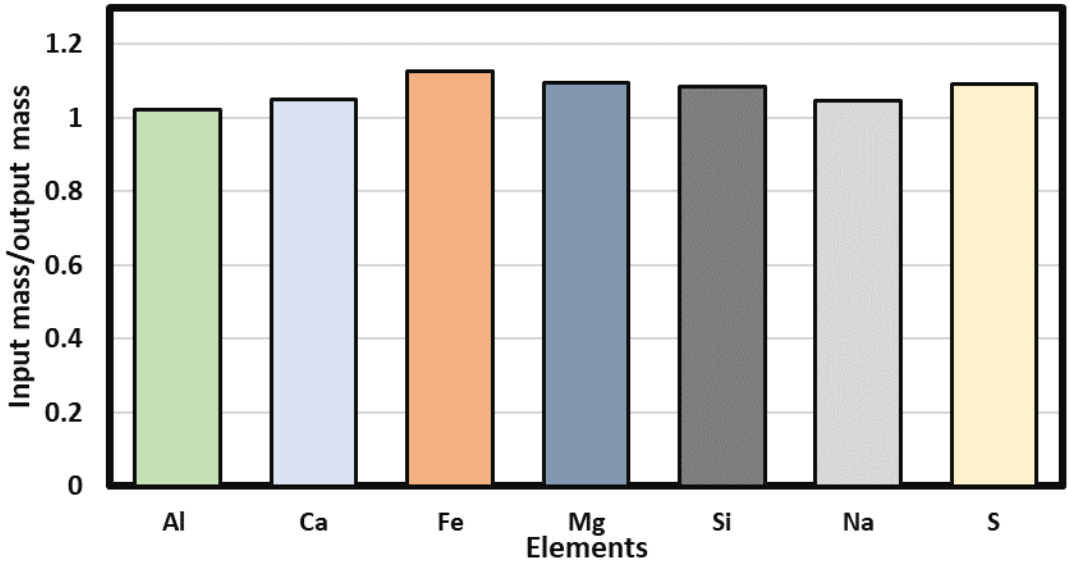



Figure 3.5.144. The ratio of input vs. output mass of various elements for experiment P6

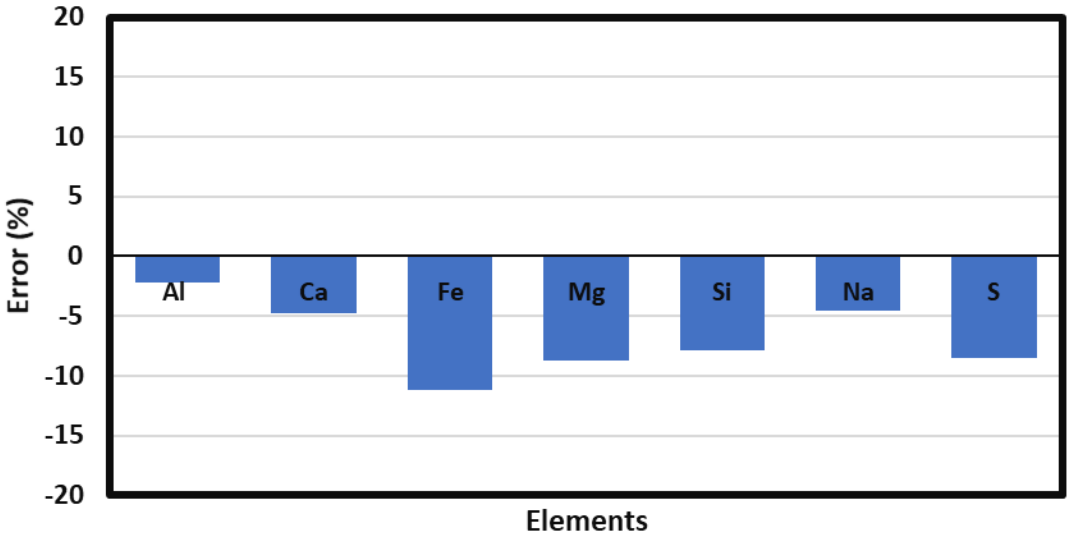



Figure 3.5.145. % Error for mass balance for various elements for experiment P6

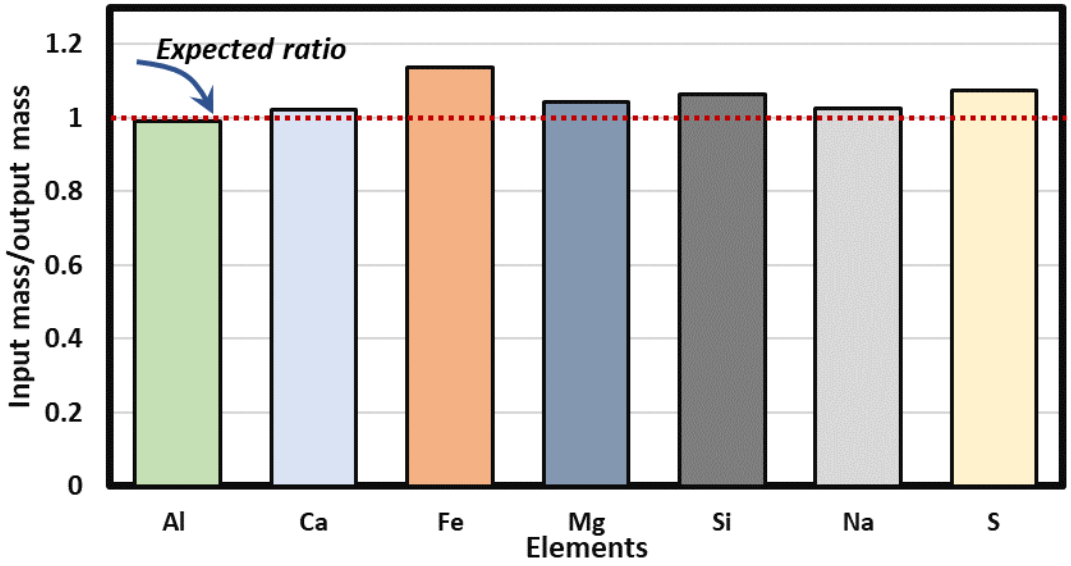



Figure 3.5.146. The ratio of input vs. output mass of various elements for experiment P7

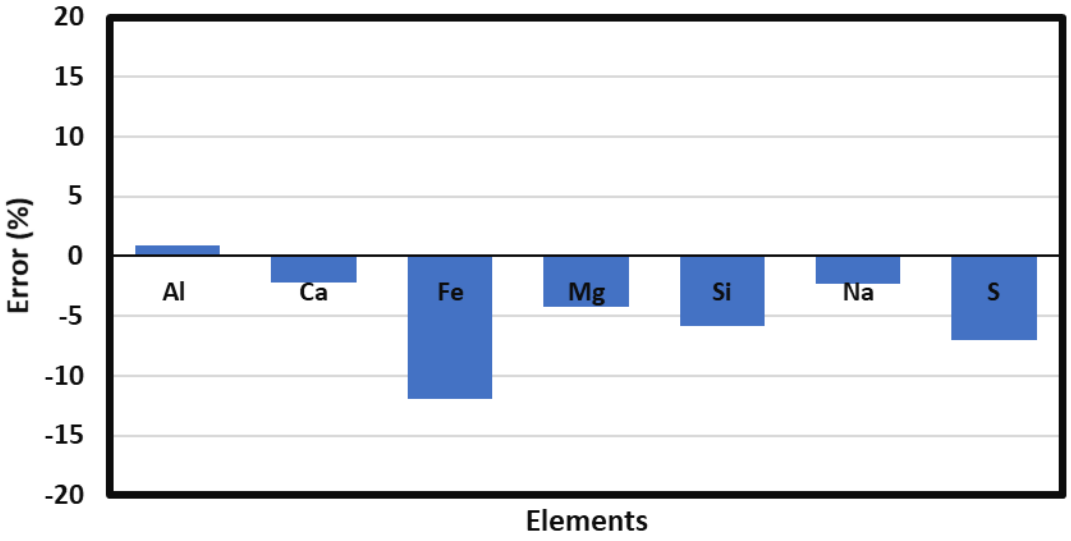



Figure 3.5.147. % Error for mass balance for various elements for experiment P7

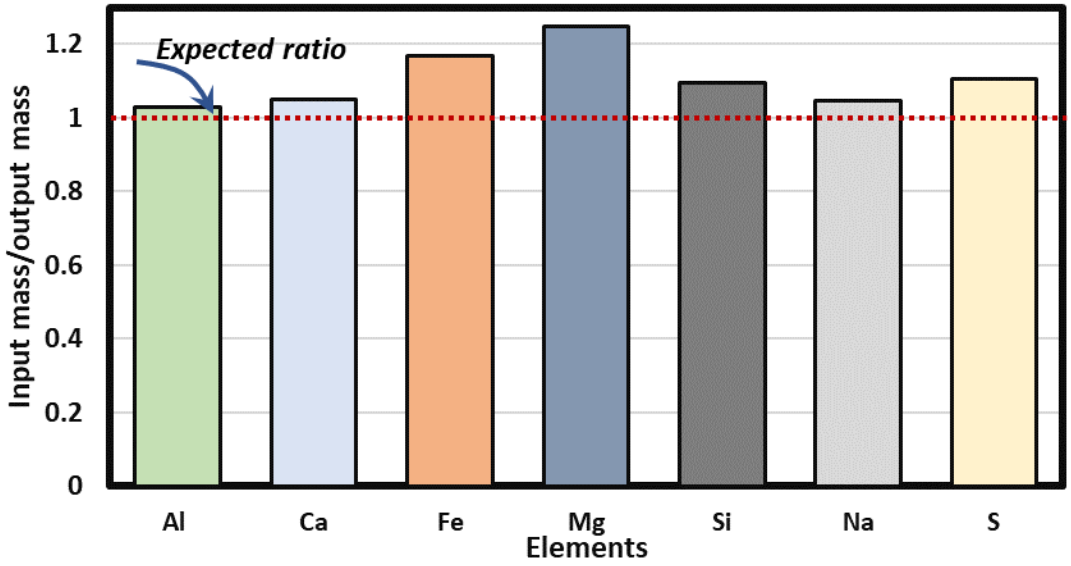



Figure 3.5.148. The ratio of input vs. output mass of various elements for experiment P10

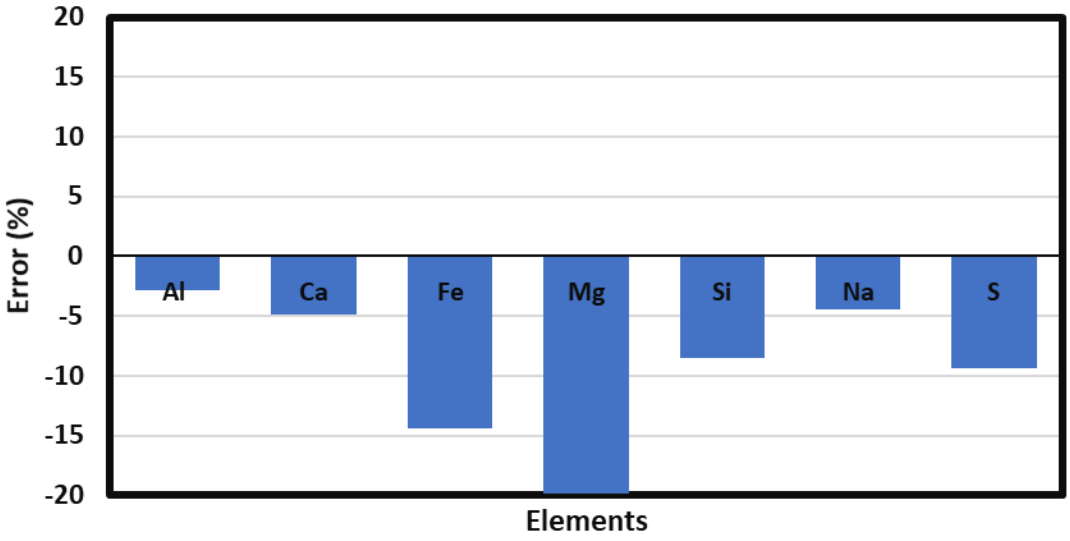



Figure 3.5.149. % Error for mass balance for various elements for experiment P10

**ICP analysis results:** ICP analysis was conducted to determine the elemental concentration in pyrite concentrate and effluent solids. Only a new batch of pyrite was applied during the steady-state operation. American West Analytical Laboratory did ICP analysis of the latest pyrite and effluent solidss collection. ICP analysis of an old pyrite batch was done at a University of Utah facility. Figure 3.5.150 the essential elemental concentration difference between the old and new batch of feed pyrite. It can be seen that along with a high concentration of Fe and S, and there was an adequate amount of Ca and a meager amount of Al, Mg, Si, and Na. Since there is no massive difference in the elemental concentration in Fe and S in varied pyrite samples, two aspects can be evaluated. First, there is not much difference between the two pyrite samples. Second: Some steady-state data from the old pyrite bath can be compared with the current pyrite data if needed.

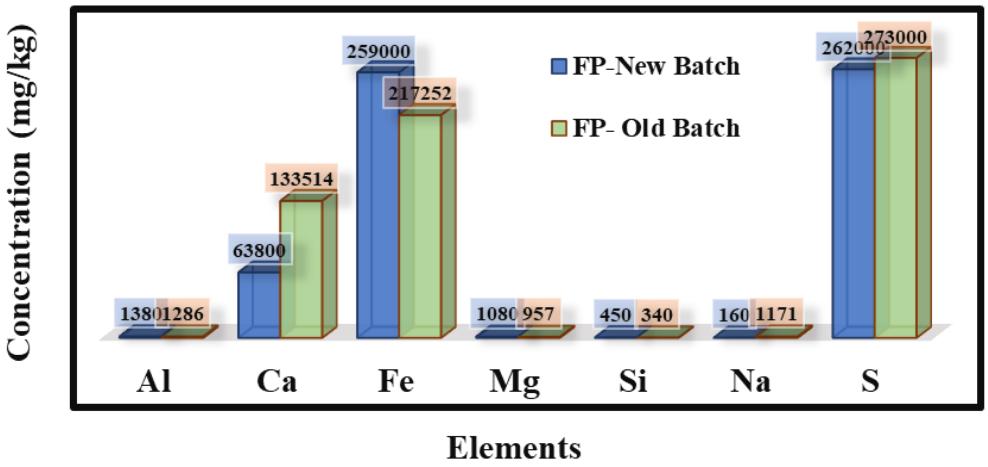



Figure 3.5.150. ICP analysis result of the new batch and old batch of Pyrite.

The elemental concentration in effluent solids of different parameters is shown in Figure 3.5.151. The changes were observed mainly in Fe, S, and Ca concentration. Fe concentration is lower in P3 effluent solids than P1 sample, whereas S concentration is higher in P6 solids. Ca concentration is highest in P3, and P2 has the lowest Ca concentration. These results only show the elemental concentration in effluent solidss. Samples have already been sent for analysis to complete the mass balance.

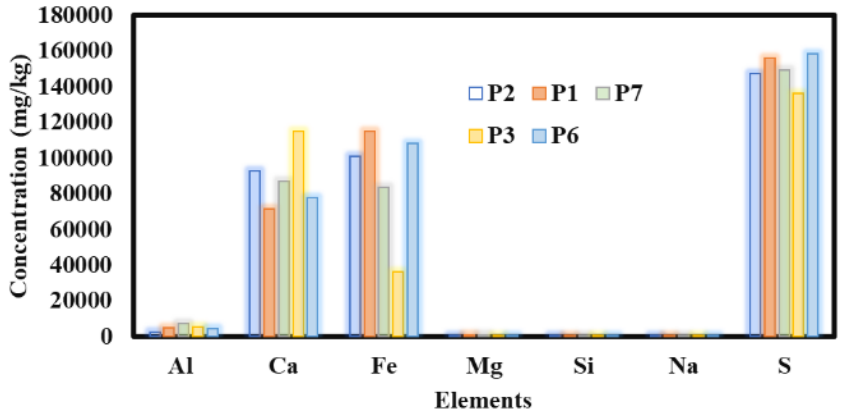



Figure 3.5.151. ICP analysis result of effluent solids of different parameters.

**Acid generation rate of the active bioleaching solution from the bioreactor:** To perform the acid generation rate from biooxidation of coal waste pyrite, Its significant to know the actual acid concentration in the bioleaching solution. A titration test was conducted to determine the acid concentration in the leaching solution. The team had applied different parameters, including residence time, solids concentration, and airflow rate (to evaluate the effect on biooxidation rate) and collected effluent solution at the most steady-state to conduct a titration test. Acid concentration found from the titration test at different parameters is shown in Table 3.5.28.

Calculation: $C_{NaOH} \cdot V_{NaOH} = C_{Acid} \cdot V_{Acid}$

Table 3.5.28 Acid Concentration determined for different test parameters

| Parameter | Acid Conc. (M) |
|---|---|
| P1 | 0.175 |
| P2 | 0.175 |
| P3 | 0.1875 |
| P4 | 0.105 |
| P6 | 0.2 |
| P7 | 0.1625 |
| P9 | 0.173 |
| P10 | 0.1 |
| P11 | 0.1625 |

Based on the acid concentration data table, it can be easily seen that the molarity of a solution was highest during the application of P6 conditions. In contrast, the lowest acid concentration was found in Parameter 2. Figure 3.5.152 shows the changes in molarity (acid concentration) during different parameter applications.

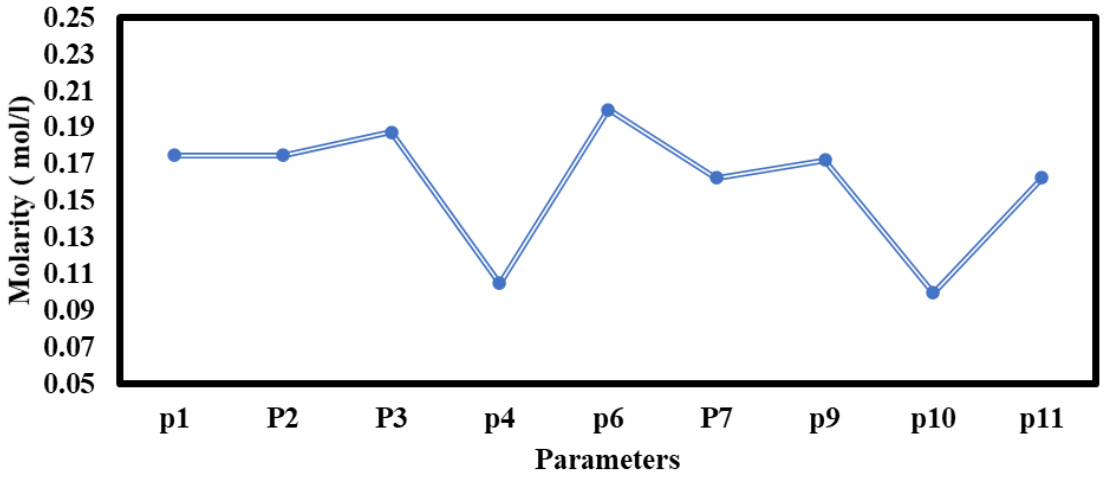



Figure 3.5.152. Acid concentration based on titration tests for different parameters.

Net acid generation can be calculated from the molarity of the solution. For instance, the net acid generation rate during P1 conditions can be determined. During this operation period of the P1, 600 ml of pyrite solution was fed into the reactor. Simultaneously, 600 ml of the leaching solution was pumped to the effluent vessel. So the net acid generation rate for this parameter would be

The acid generation rate (amount of $H^+$ generated per day) is:

Acid conc. ( TT)× amount of leaching solution collected every day

= 0.175 mol/l *0.6 l/day

By following these procedures net acid generation was calculated for all the parameters and plotted in Figure 3.5.153. Interestingly, P10 and P1 have generated the most acid, whereas P4 produced the least.

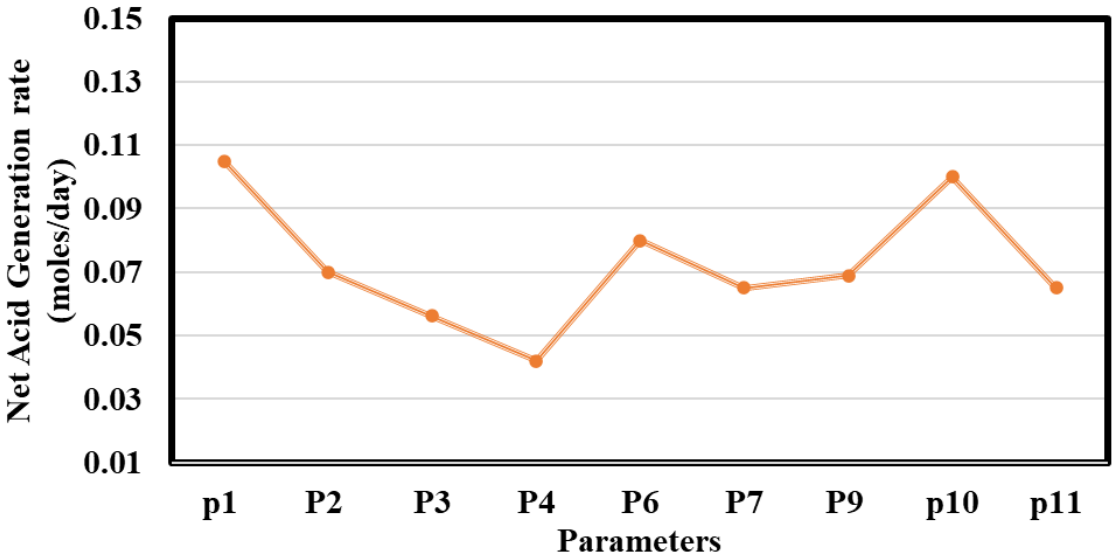



Figure 3.5.153. Acid generation rate (moles/day) based on titration tests for different parameters.

**Calcite titration on fine pyrite:** Calcite titration was conducted to determine the $CaCO_3$ concentration in the fine pyrite concentrate. The workflow chart is shown below

1. Prepare of 20 ml of 0.5M HCl
2. Grind fine pyrite concentrate carefully to make them homogenous
3. Use of 1g pyrite conc. With the HCl Solution
4. Allow 24 hrs for the reaction competition between HCl and $CaCO_3$
5. Filter the solution
6. Use of 0.5 M NaOH solution to titrate the solution at pH 7.0 ( pH 7 is neutral pH so pH 7 is the target pH to determine the endpoint of the total conversion of calcium carbonate to calcium chloride)
7. Measure the volume of NaOH consumed and perform final calculation

$CaCO_3 + H_2O = Ca(OH)_2 + CO_2$

$Ca(OH)_2 + 2HCl = CaCl_2 + 2H_2O$

Balanced equation: $CaCO_3 + 2HCl = CaCl_2 + CO_2 + H_2O$

Calculation

Amount of 0.5 M NaOH to titrate the solution- 8.5 ml

nHCl- Consumued= (18-8.5)*10^-3*0.5 mol/l =0.0475 mol

In the balance reaction we can see that molar ratio of $CaCO_3$ and HCl is 1:2.

So the amount of $CaCO_3$ in fine Pyrite would be m $CaCO_3$ =0.00475/2*100.09 g/mol =0.238 g

Concentration of $CaCO_3$ in Fine Pyrite Concentrate= 0.225/1g*100% =23.77%

**XRD analysis on fine pyrite and effluent solids:** A XRD analysis has been performed on Pyrite and effluent solidss to know the common compounds available in the fine pyrite concentrate. XRD analysis results of fine Pyrite show that the picks of the compounds with the Experimental pick of fine pyrite concentrate were visible that there are lots of pyrite pick matches with the experimental, which means there are lots of pyrite compounds available in the Concentrate. Also, a few pick of calcite, silica, and magnetite matches the practical selection. There is no trace of Pyrite in P1. Still, in P10, some fraction of pyrite was found, representing the successful biooxidation performance during P1 operation, and P10 with the lowest residence time could not oxidize all the pyrite. Figure 3.5.154 and Figure 3.5.155 show the results.

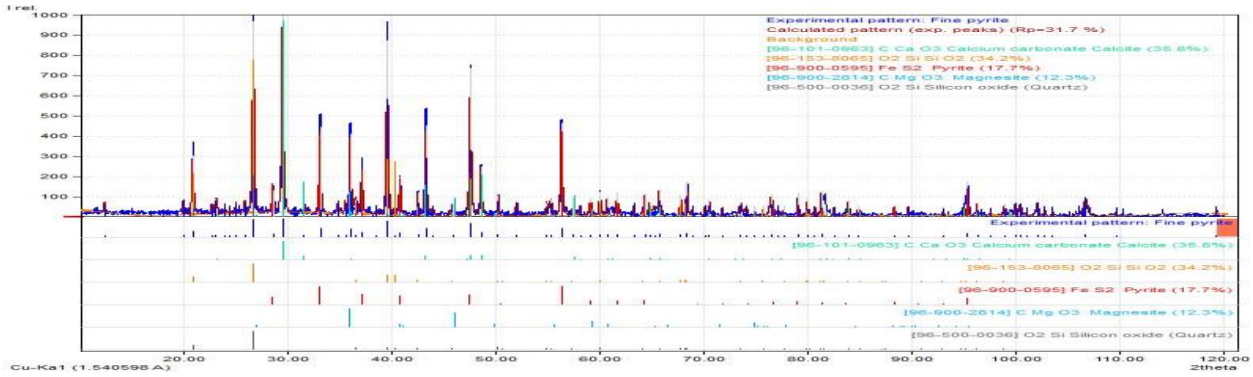



Figure 3.5.154. XRD analysis result of Fine Pyrite

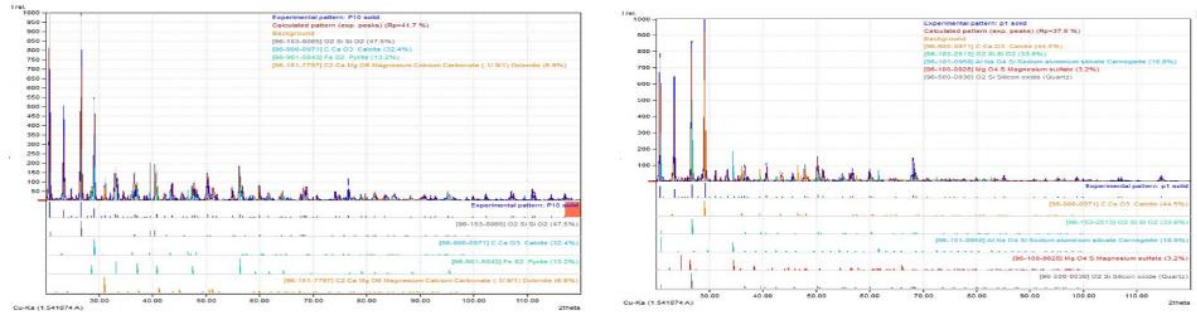


Figure 3.5.155. XRD analysis result of P1 and p10 effluent solids.

**Pyrite Usage Efficiency @ different operating Parameters**: Assuming the overall reaction of the bioleaching process is:

$$4FeS_2 + 2H_2O + 15O_2 = 2H_2SO_4 + 2Fe_2(SO_4)_3$$

This reaction suggests 2 moles of sulfuric acid can be generated from 4 moles of pyrite concentrate. Here we are assuming complete dissociation. For instance, during the application of P1, we generated 0.105 moles of sulfuric acid every day. According to the Equation, we needed 0.1052*2*119.98 = 25.19 g of Pyrite to develop this much acid (neglecting acid consumption by other minerals). During  This time, we used 36 g of pyrite every day. So from this perspective, pyrite usage efficiency would be 25*100/36=69.44% . Figure 3.5.156 shows the results.

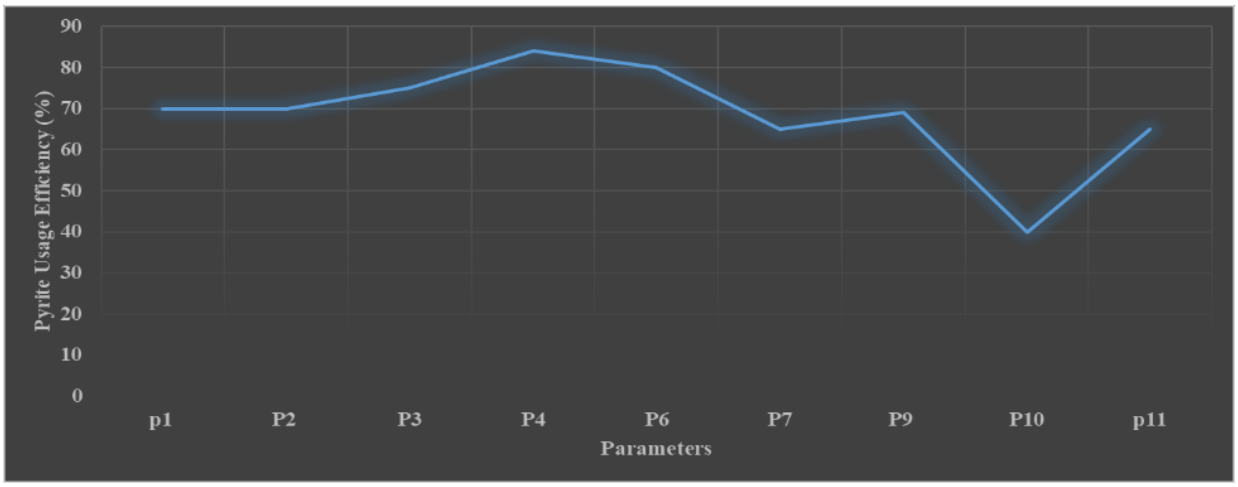



Figure 3.5.156. Pyrite usage efficiency curve of different parameters (neglecting calcite acid consumption).

**Compound concentration in pyrite feed and effluent solution:** The compound ( predicted ) concentration has been calculated for pyrite (feedstock concentrate) and effluent solution. The calculation had been conducted based on the elemental concentration found from the ICP-MS analysis. After observing the Elements concentration thoroughly and following the predicted reaction equation, a few compounds were expected, and then their mass concentration was calculated. Fine pyrite concentrate has 70% of FeS2; 28.82% of calcite, and 1% silica and some alumina. The XRD data of fine pyrite can also justify this value. Some pyrite was still an available effluent solids sample.

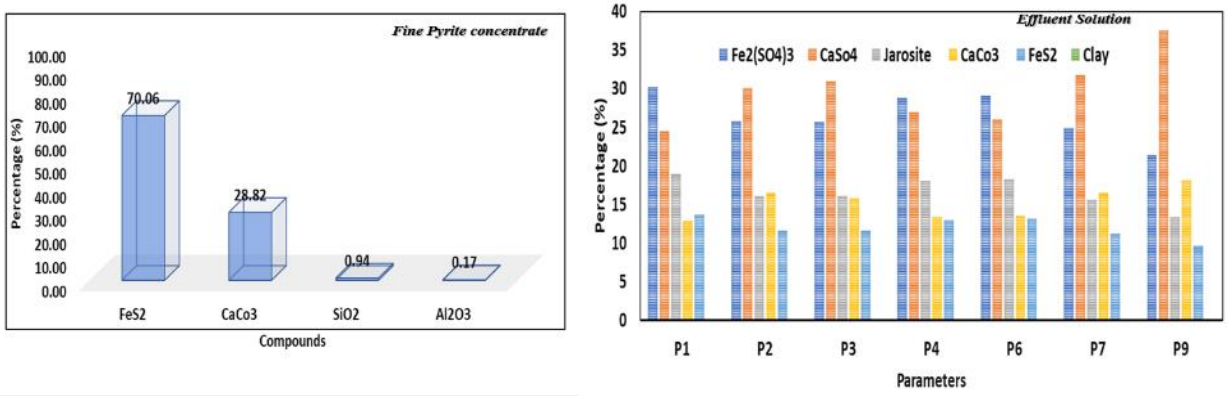



Figure 3.5.157: Compound concentration

**Effect of Operating Parameter on Biooxidation Rate:** The following plot shows the effect of operating factors on the bio-ooxidation rate differences that had been established between P1 to P9. The airflow rate was the most significant factor, followed by residence time and solids feed. A central composite design was made to analyze the effect of individual elements. It is imperative to know how different factors could interact to deliver the best biooxidation performance. Based on this analysis, it's easier or effective to show which elements can be the most effective pair to provide the best biooxidation rate. A 3-factor factorial design had been conducted to make this design of experiments. Figure 48 A indicates the relationship between residence time and solids

concentration. It can be seen that when residence time was four days and the solids portion was 9%, it gave the highest biooxidation performance. Figure 48 B indicates that the biooxidation rate was highest when the residence was four days, and the airflow rate was 0.5 l/m. It is an important finding that airflow rate could be reduced with faster residence time to get more biooxidation performance. Figure 1C shows the interaction between solids concentration and airflow rate where 9 percent solids would give the highest biooxidation performance with an airflow rate of 1.5 Lpm, and it is clear that there is no significant difference between 1.5l/m and one l/m air flow rate.

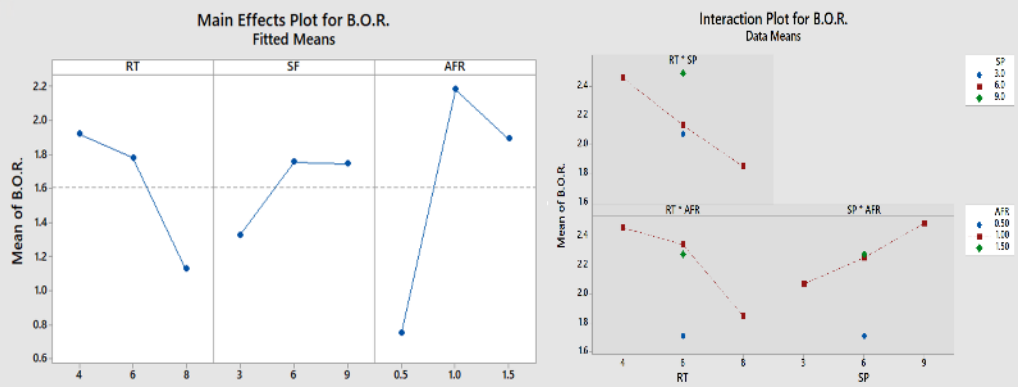



Figure 3.5.158. Effect of operating parameters on biooxidation rate.

**Analysis of Bacteria:** Genome sequencing was performed to understand the species of microorganisms present in different operating conditions. Bacterial concentrations were also analyzed through these sequencing methods. An HPC test was conducted to see how the bacterial colonies can be formed under other conditions. Figure 3.5.159 shows the presence of *Acidithiobacillus*, *Leptospirillum*, and *Sulfolobacillus* bacteria.

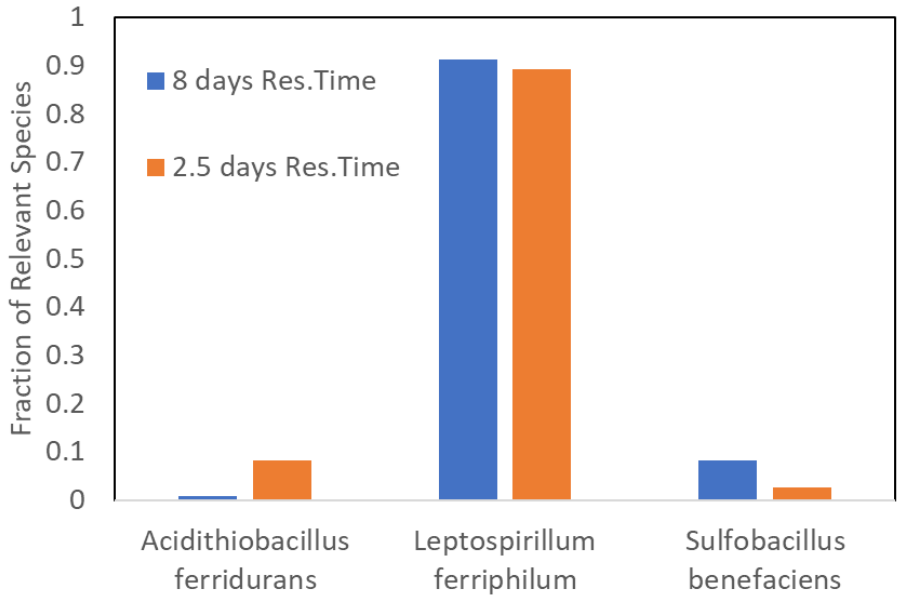



Figure 3.5.159. Genome sequencing data from samples originating from bioreactor solutions from 2.5 and 8 days residence time tests at 500 RPM, 1 l/min air flow, 35ºC, and 6 wt % solids conditions showing the relative fraction of species that are generally identified with biooxidation of pyrite.

Also, oxidized pyrite solids were investigated through a high magnification SEM imaging system to view potential bacterial attachment to the pyrite surface. These SEM images are presented in Figure 3.5.160.

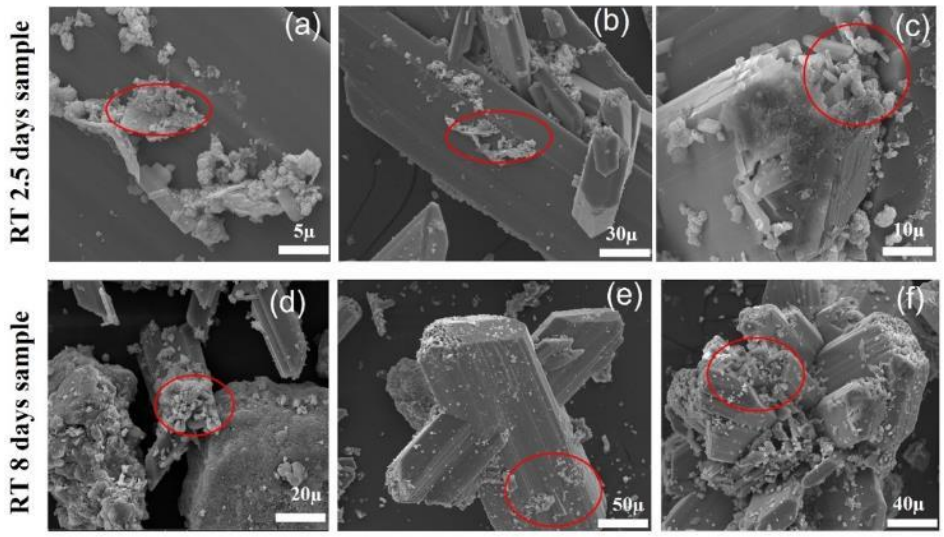



Figure 3.5.160. View of magnified pyrite surface from samples originating from bioreactor solutions from 2.5 and 8 days residence time tests at 500 RPM, 1 l/min air flow, 35ºC, and 6 wt % solids conditions showing the relative fraction of species that are generally identified with biooxidation of pyrite.

**Small Pilot Plant reactor overview:** A 250-gallon pilot scale bio-reactor was setup in the pilot plant in Kentucky facility. The University of Utah team helped to get the reactor going and sent solution feedstock samples with bacterial cultures to enable its start up. The team also helped with continuing work at the pilot plant with the bioreactor operation.

Initial parameters of starting the initial small reactor are shown below. Figure 49 shows some digital images of bioreactor operation in pilot plant.

- Solution volume was adjusted to 90 gallons by draining extra water from 250-gallon tank. 7.5 gallons of Bio leaching solution was added to that.
- 3.5 gallon of 9K media solution (using $FeSO_4$) was also added to the bioleaching solution. The composition of 9K media was similar to that used in lab scale bioreactor.
- Once the bioleaching solution was filled, feeding was started with 10 days of residence time and 6% solidss.
- Initial pH adjustment was done using $H_2SO_4$.
- Stirring Speed: 450 rpm (A giant motor was used, as can be seen in picture)
- Air Flow rate: 4 to 5 cubic meter per minute
- Temperature- 32.3$^o$ C




Figure 3.5.161. Digital images of pilot plant reactor operation.

The team monitored PILOT scale bioreactor over the period of time. A periodic Eh and pH measurements has been done to evaluate the performance of the reactor. Figure 7.162 shows this trend for a period of ~ 1 month. It can be seen that average pH in this duration was ~ 1.25 whereas Eh was very high, greater than 650 mV in most of the cases. In few cases, pH was adjusted (when pH exceeds 1.9 or so) using sulfuric acid. pH and Eh probes were calibrated frequently in order to avoid any error. Bio-oxidation performance of reactor was also measured in regular interval and found that ferrous bio-oxidation rate was high (see Figure 7.163) in the beginning ~ 1.65 g/l/hr

which reduced over the period. Overall, the performance of bio-reactor was satisfactory as can be seen the consistency in Eh and pH. The resident time was changed to 8 days on 1/29/2021. A rough calculation indicates (considering the volume of solution ~ 100 gallons i.e. 378 liters and average pH 1.125), the presence of ~ 19 moles of $H^+$ ; This will be equivalent to ~ 9.5 moles of sulfuric acid. In case of consideration of full capacity i.e. 250 gallons, this quantity will be equivalent to ~ 24 moles of sulfuric acid.

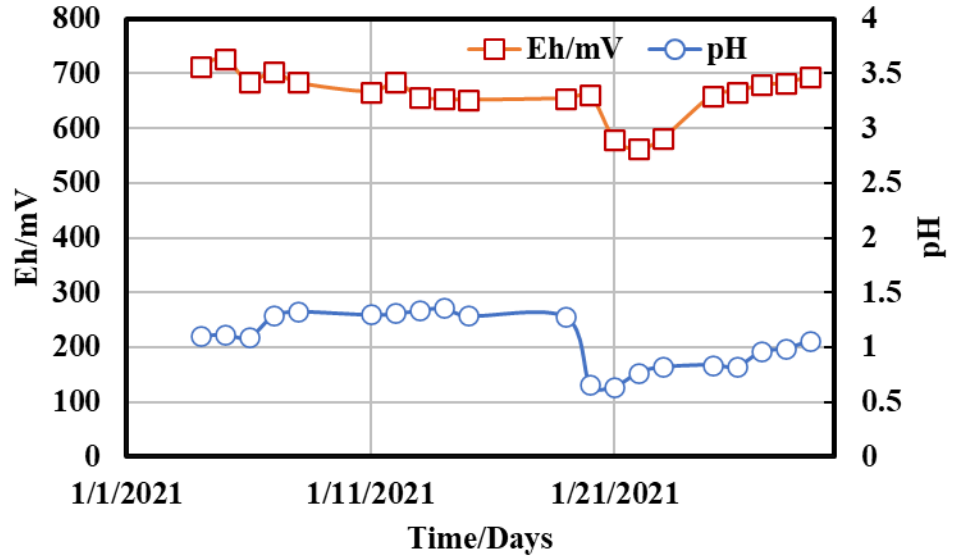



Figure 3.5.162. Eh and pH vs. time behavior of pilot scale reactor

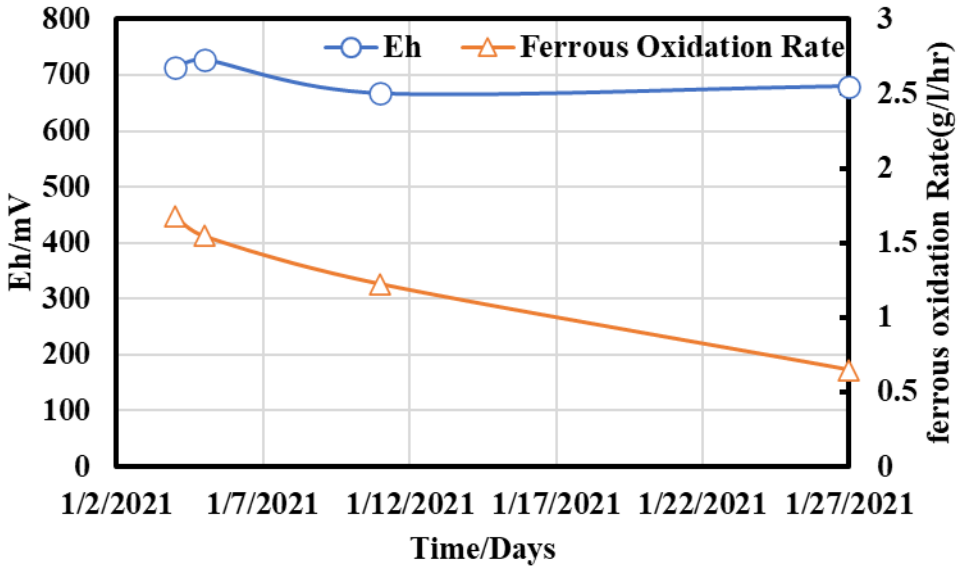



Figure 3.5.163. Eh and biooxidation rate vs. time behavior of pilot scale reactor

## 3.6 OPTIMIZATION OF ROASTING & LEACHING (TASK 8)

A systematic study was performed to evaluate the effects of roasting reagents, reagent dosage, roasting temperature, duration and atmosphere on the leaching recoveries of REEs especially Heavy Rare Earth Elements under standard conditions. The conditions under which maximum recoveries of the heavy and critical REEs are obtained were identified. Mineralogical changes caused by roasting was investigated by x-ray diffraction (XRD), scanning electron microscopy coupled with energy dispersive x-ray (SEM/EDX) spectroscopy and thermogravimetry and differential scanning calorimeter (TGA-DSC). Leaching optimization was performed once the optimum roasting conditions were identified. Acid concentration, solid concentration, salt concentration, leaching temperature and duration were varied to determine the leaching conditions providing satisfactory recoveries of heavy and critical REEs with minimum operational and capital costs.

### *3.6.1 TGA-DSC Studies*

**Installation and training of TGA-DSC Simultaneous Thermal Analyzer:** To evaluate the effect of roasting temperature, duration and atmosphere on the leaching recoveries of REEs and critical elements, a Simultaneous Thermal Analyzer (STA) was purchased and installed at the University of Kentucky. Simultaneous TGA-DTA/DSC measures both heat flow and weight change of a sample as a function of temperature and time under a controlled atmosphere. The setup of the equipment is shown in Figure 3.6.1. The STAPT1600 manufactured by LINSEIS, Germany provides information about the mineralogical phase transformations caused by roasting and roasting kinetics and identifies the conditions under which maximum recovery values of heavy and critical REEs can be obtained.

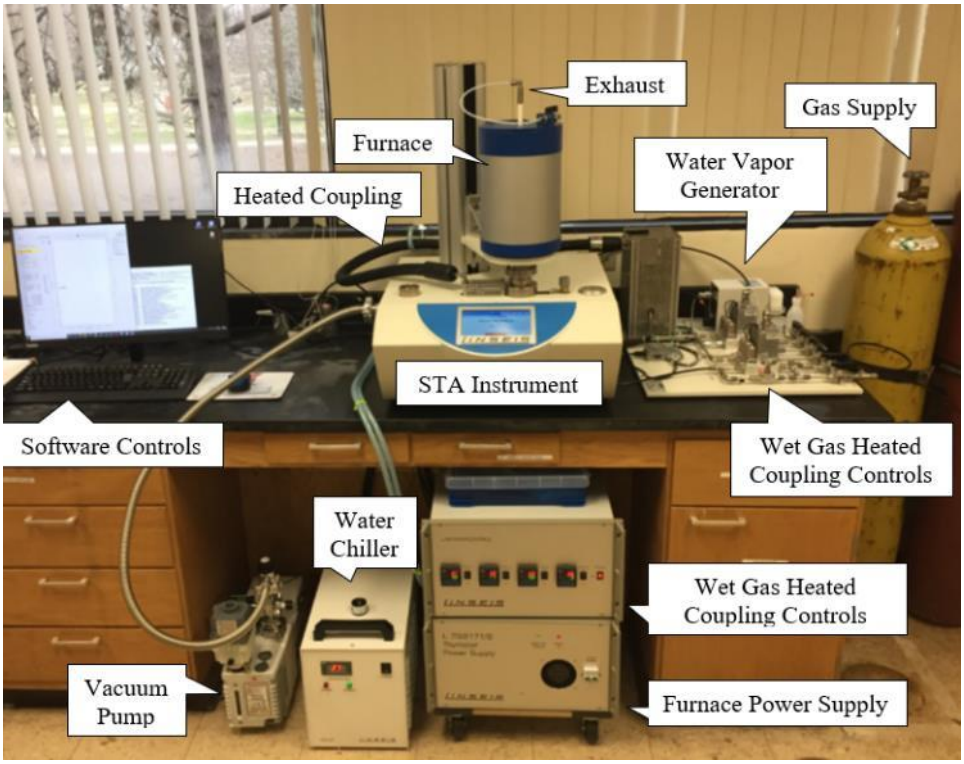


Figure 3.6.1. LINSEIS STA with water vaporizer with 5 sample gases and 1 balance purge service at the University of Kentucky.

The STA hardware installation was completed by making all the appropriate communication, power, gas, and water connections between the system components. The equipment uses a 208/ 16A / single-phase power supply. The vacuum pump draws a vacuum inside the furnace and the balance backfills with the desired gas before a run to control the environmental conditions inside the chamber. A closed-loop water chiller with an internal flow switch is connected to the furnace collar. The water vapor generator with heated coupling was installed which allows for the use of wet gas into the furnace and provides control on the humidity inside the equipment. The gas mixer is equipped with two mass flow controllers (MFCs). The MFC controls the flow of Purge gas and Gases 1-5 flowing into the equipment. The setup can be used for any combination of gas concentrations to achieve the desired chamber environment (Figure 3.6.2). The MFCs were checked for correct flow actuation by placing the effluent leg in a water beaker while turning flow ON/OFF in the software. Oxygen is connected via two-staged brass regulators as gas 1 and Nitrogen was used as the purge gas and gas 2. Argon was set up for gas 3 and two reducing gases were connected to gases 4 and 5. A variety of TGA and DSC sensors along with small and large sample crucibles allow for a wide range of applications. The entire system is operated by a software preloaded on a Think Centre windows 10 computer for fully automatic system control, operation, process sequencing, and data analysis. The specifications of the equipment are given in Table 3.6.1.

After completion of hardware installation, training focused on the installation of the DSC/ TGA sensor, leveling of the TGA microbalance arm and the basic software operations and data analysis was completed. Best practices for using the unit with water vapor were also identified. Wet gas was introduced in the unit when the temperature was greater than the dew point of the wet gas to prevent the condensation of water vapor on the furnace collar.

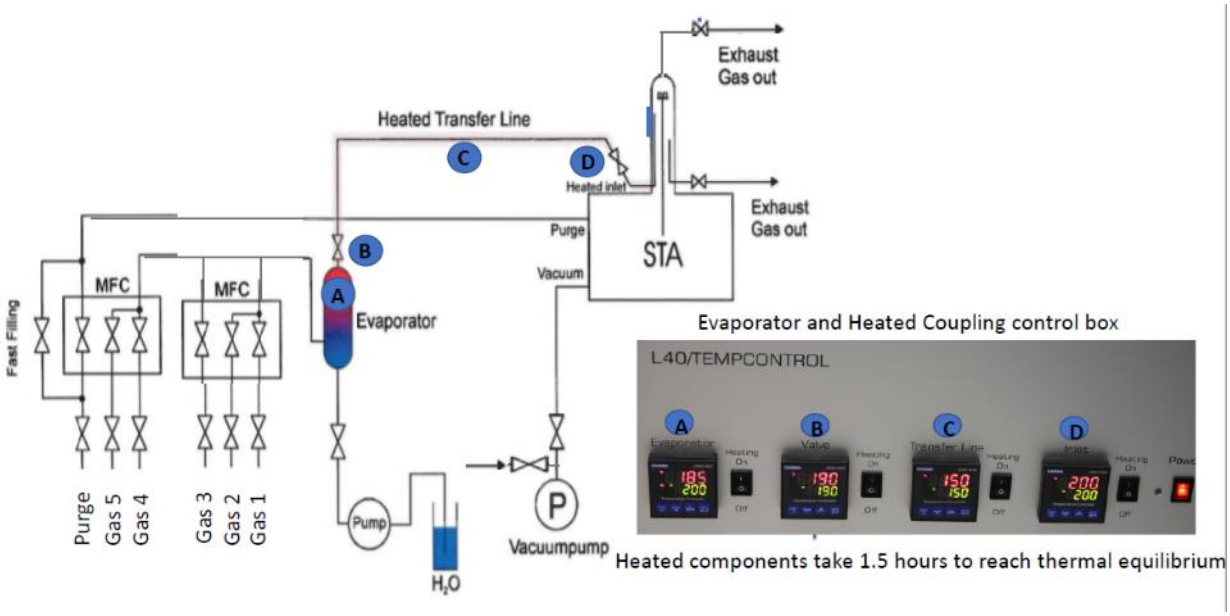



Figure 3.6.2. GasService and heated wet gas transfer map.

Table 3.6.1. Technical Specifications of the STA1600.

| | | | |
|---|---|---|---|
| **Temperature range** | **-150 up to 2400°C** | | |
| **Vacuum** | **$10^{-5}$ mbar (depends on vacuum pump)** | | |
| **Pressure** | **up to 5 bar (optional)** | | |
| **Heating rate** | **0.01 up to 100K/min (depends on the furnace) from 0.001°C/min on request)** | | |
| **Temperature precision** | **0.001°C** | | |
| **TG** | 1 | 2 | **3** |
| **Resolution** | 0.025 µg | 0.1 µg | **0.1 µg** |
| **Sample weight** | 5 g | 25 g | **35 / 50 g** |
| **Measuring range** | 25 / 2500mg | 25 / 2500mg | **35000 mg** |
| **DSC** | | | |
| **DSC-sensors** | **E / K / S / B / C (C = DTA only)** | | |
| **DSC resolution** | **0.3 / 0.4 / 1 / 1.2 µW** | | |
| **Calorimetric sensitivity** | **approx. 4 / 6 / 17.6 / 22.5 µW** | | |
| **DTA** | | | |
| **DTA-resolution** | **0.05 µV** | | |
| **Sensitivity** | **1.5 µV/mW** | | |
| **DTA-measuring ranges** | **250 / 2500 µV** | | |

**Parametric Study of the TGA-DSC Simultaneous Thermal Analyzer:** In the first set of experiments, scoping tests were conducted on the 1.80 x 2.00 specific gravity (SG) fraction of the Fire Clay coarse refuse to determine the effect of heating/ramp rate. Thermogravimetric analysis was performed for the temperature range 25-1100°C at varying heating rates ranging from 1-15°C/min under oxidizing condition ($O_2$ flow rate=100scc/min). A 4 gram sample was loaded into the crucible and placed into the STA chamber. A vacuum was pulled inside the chamber and was subsequently backfilled with inert gas before starting the analysis. The relative weight loss with respect to temperature and time was recorded as a response variable (Figure 3.6.3). The mass loss in the 100-200°C range corresponded to the removal of surface and inherent moisture. The weight loss post 300°C was attributed to the removal of volatiles and combustion of the carbon matter. The decomposition or dehydration reactions for the constituent mineral phases cannot be identified and was masked by the exothermal reaction of the carbon oxidation. Heating rates faster than 5°C/min provided incomplete oxidation of the sample upon completion of the test. Slower heating rates corresponded to complete ashing of the sample. Low heating rates ensure that the sample is under thermodynamic equilibrium. For higher heating rates, the reaction was kinetically controlled. Based on the data, a heating rate of 1-2°C/min was used as a baseline for the DSC tests in this project. The temperature range was narrowed to 25-700°C to save on test time. TGA tests with even lower ramp rates (<1°C/min) were conducted to precisely determine the ramp rate equilibrium.

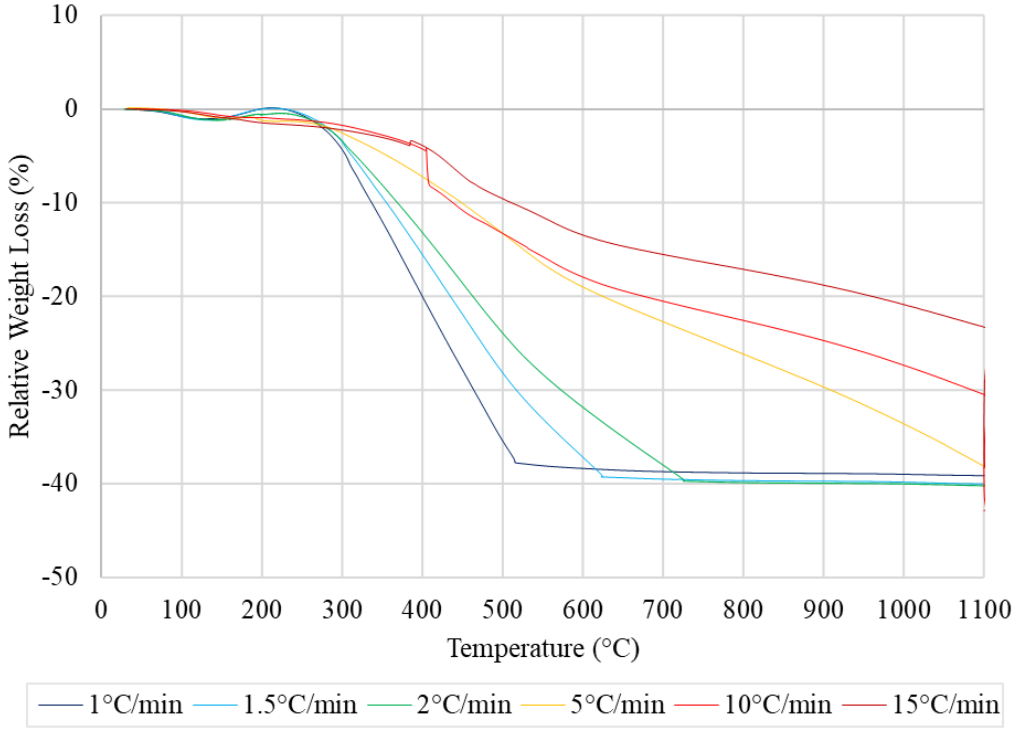



Figure 3.6.3.Thermogravimetric curves for different ramp rates for 1.80x2.00 SG fraction of the Fire Clay coarse refuse. O2 flow rate= 100 scc/min.

Secondly, scoping tests were also carried out to determine the optimum rate of flow of gases and the weight of sample to be analyzed in a test. Based on mole balance, it was found that a gas flow rate of 100 scc/min ensured saturation of the camber for complete oxidation. The calculated value was also validated by experiments, which resulted in complete combustion. Additionally, the TGA microbalance is capable of measuring a weight change of +/- 2500 mg during an analysis. Therefore, to produce oxidation product for low ash material for leaching and XRD/XRF analysis, sample weight of 4-gram (pulverized, 80 mesh) per run was calculated to be within the balance measurement range. The weight change resulting from the removal of carbon and volatiles along with the other chemical reactions should be within the measurement range for all the density fractions.

Based on the results from the heating rate studies, TGA analysis was performed for the density fractions of the two feed sources. The samples were analyzed at 2°C/min heating rate in 25-1100°C temperature range. The TGA curves for Fire Clay coarse refuse are compiled in Figure 3.6.4. The TGA analysis shows that the oxidation reaction was completed in the 600ºC-700°C range for all density fractions. Thus, an upper bound of 700°C was used for subsequnet DSC tests for all density fractions of the Fire Clay-CR. For the density fractions of the West Kentucky No. 13 coarse refuse material, the oxidation reaction was completed in the 700-800°C range (Figure 3.6.5). Thus, 800°C was used as the range for additional DSC analyses. The difference between the two feed sources can be attributed to the rank of the coal and its composition and age of the deposit.

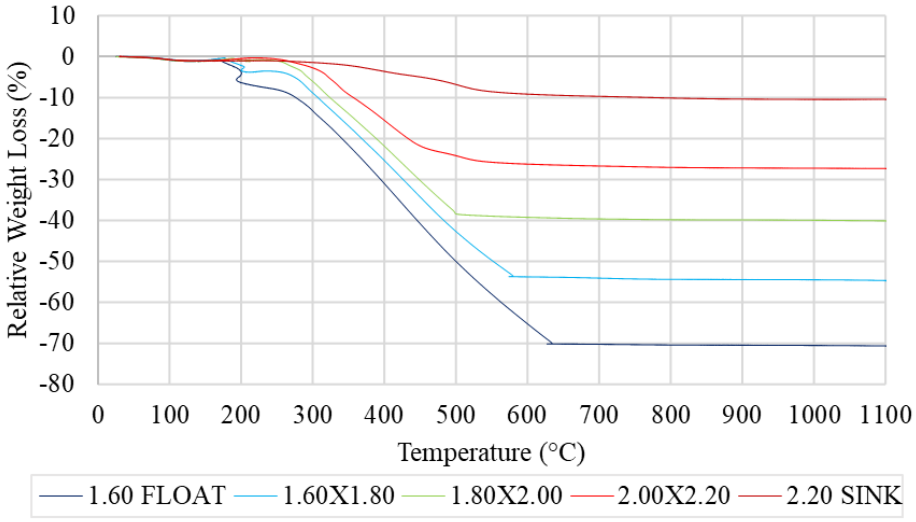



Figure 3.6.4. Thermogravimetric curves for the density fractions of the Fire Clay coarse refuse samples. Ramp Rate= 2°C/min, O2 flow rate= 100 scc/min.

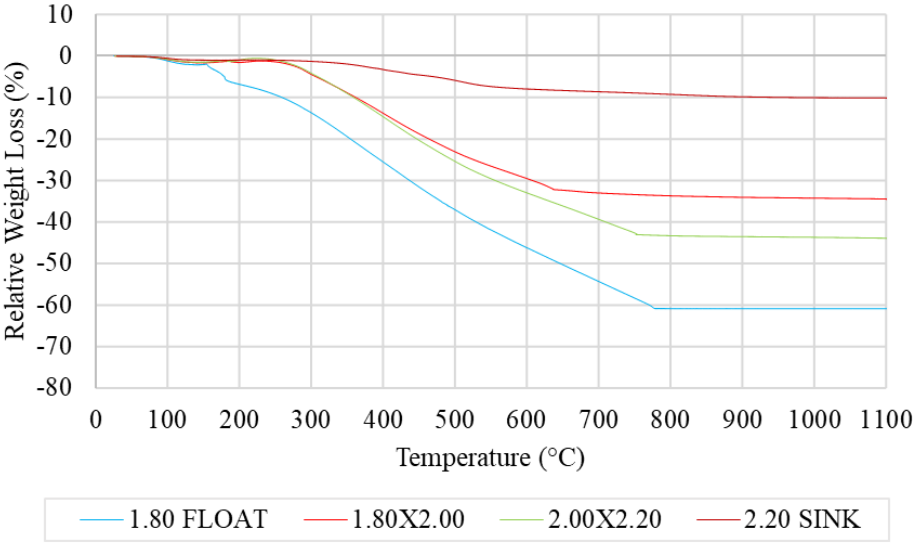



Figure 3.6.5.Thermogravimetric curves for the density fractions of the WK No.13 coarse refuse samples. Ramp Rate= 2°C/min, O2 flow rate= 100 scc/min.

**TGA-Differential Scanning Calorimetry (DSC) Tests:** DSC tests using the LINSEIS TGA-DSC Simultaneous Thermal Analyzer (STA) were performed on the density fractions of both coarse refuse samples (Fire Clay: 1.60 Float, 1.60x1.80, 1.80x2.00, 2.00x2.20, 2.20 Sink) and (West Kentucky No.13: 1.80 Float, 1.80x2.00, 2.00x2.20, 2.20 Sink). TGA-DSC analysis was performed for a temperature range of 25ºC-1000°C at a 10°C/min heating rate under oxidizing conditions ($O_2$ flow rate=100scc/min). Samples of 50 mg were loaded into the crucible and placed into the STA chamber. An empty reference crucible was placed on the second thermocouple junction of the DSC sensor. The DSC sensor metal frame is made from platinum rhodium. A single platinum rhodium wire is attached to the middle of the metal frame between the two positions. This thermocouple

leg is shared between the sample and the reference side. Underneath the frame at each position, a platinum wire is attached to complete the thermocouple junction. This makes a total of three wires from the sensor tip to the base (Figure 3.6.6). A vacuum was pulled inside the furnace and the balance chamber was subsequently backfilled with inert gas before starting the analysis.

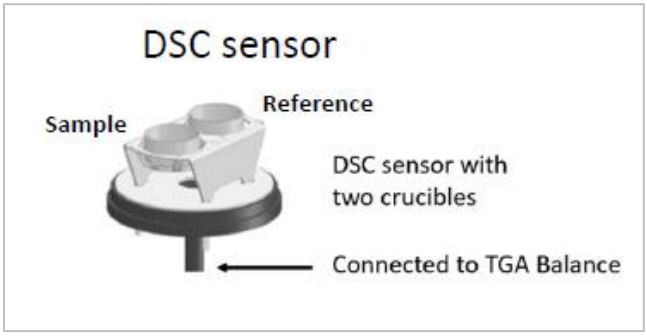



Figure 3.6.6. DSC Sensor setup for LINSIES STA analysis.

The relative weight loss with respect to temperature and a DSC signal to quantify the phase change enthalpy was recorded as a response variable in the TGA-DSC analysis. The TGA-DSC curves for Fire Clay coarse refuse material are compiled in Figure 3.6.4. The data shows that the oxidation reaction is completed in the 600-700°C range for all density fractions. This observation correlates well with the findings of the scoping TGA tests reported previously. Two distinct reactions/phase transformations in the Fire Clay material are observed for all density fractions.

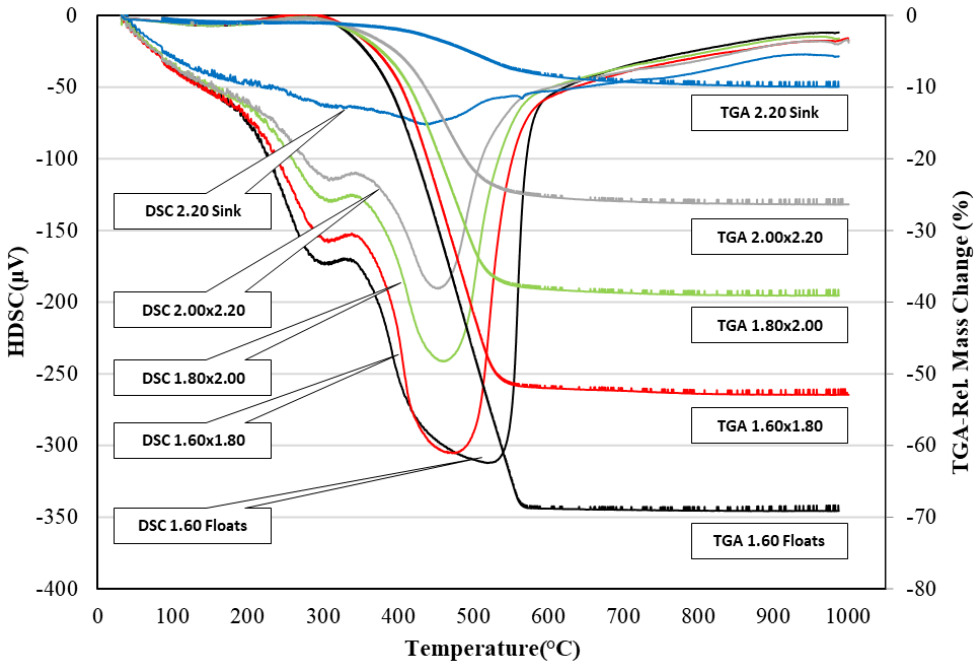



Figure 3.6.7. TGA-DSC curves for the density fractions of the Fire Clay coarse refuse samples. Ramp Rate= 10°C/min, O2 flow rate= 100 scc/min.

For the density fractions of the West Kentucky No.13 coarse refuse material, the oxidation reaction was completed in the 700-800°C range (Figure 3.6.8). DSC tests show areas of correlation between

coal samples and mineral samples. The additional peak in the West Kentucky No. 13 data relative to those obtained from the analysis of Fire Clay material may be due to the presence of a significant amount of coal pyrite. Correlating the baseline TGA data of the coal sources with the mineral phases (identified with XRD) found in the two sources suggests that thermal changes in clays, such as kaolinite and illite, and coal pyrite can be associated with the improvements in REE recovery by roasting. The difference between the two feed sources can be attributed to the rank of the coal and its composition and age of the deposit. Tests on pure mineral phases were subsequently performed to correlate the association of REEs with mineralogy in coal samples. This can potentially lead to the selection of an appropriate chemical additive to provide access to locked REEs.

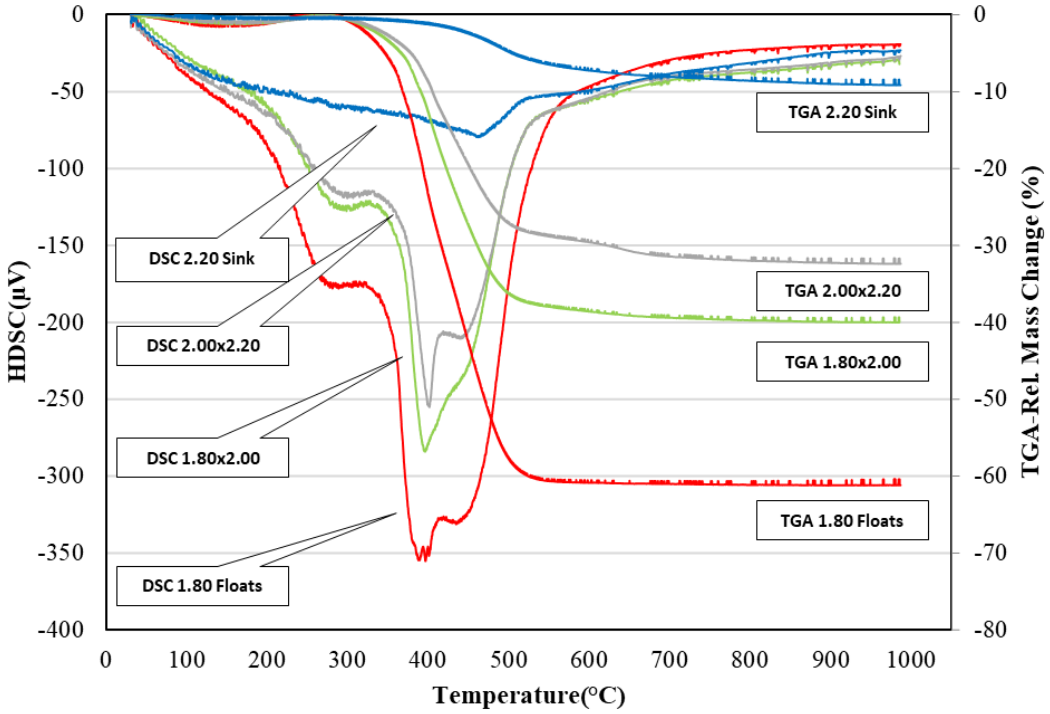



Figure 3.6.8. TGA-DSC curves for the density fractions of the West Kentucky No.13 coarse refuse samples; ramp rate= 10°C/min, $O_2$ flow rate = 100 scc/min.

**Fundamental Studies on Pure Mineral Analysis:** DSC analysis using the LINSEIS STA were performed under inert, oxidizing and reducing atmospheres on the pure mineral species commonly found in the West Kentucky No.13 and Fire Clay deposits. Pure samples of kaolinite, illite, quartz, calcite, and pyrite were procured and ground to finer than 177 microns. Additionally, samples of REE containing conventional minerals such as monazite and bastnaesite were purchased and evaluated using DSC and TGA analysis. The DSC tests on the pure minerals followed by XRD and XRF analysis provided an understanding of the thermal decomposition and/or phase transformations of the pure minerals. This data was correlated with the changes seen on the different density fractions of the West Kentucky No.13 and Fire Clay materials to determine the phase transformations responsible for the change in REE recovery after roasting. The data was utilized to assist with the identification of the mode of association for the REEs in the different

density fractions of the two coal feedstocks and to understand the effect of roasting and roasting kinetics.

TGA analysis was performed under oxidizing atmosphere on the pure mineral species commonly found in the West Kentucky No.13 and Fire Clay deposits. The heating rate was set at 2°C/min. High purity samples of kaolinite, illite, quartz, calcite, and pyrite were procured from Ward Science and pulverized to finer than 177 microns. Additionally, samples of REEs associated with conventional minerals such as monazite and bastnaesite were also subjected to TGA analysis to determine the temperature range for thermal oxidation, decomposition, dehydration and phase transformation reactions.

Based on the characteristic TGA curves as shown in Figure 3.6.9, quartz remains inert and does not undergo any change. Clays like kaolinite and illite show a mass change between 100-600°C which can be attributed to the loss of interlayer water and dehydration. Pyrite oxidizes, releases $SO_2$ and produces iron oxide. The decomposition of calcite to lime and $CO_2$ is prominently seen for both crystalline and amorphous calcite samples. Based on the temperature range obtained for individual minerals, DSC analysis focused on the range where the mass change is observed to identify the type of reaction responsible for the change. This data can then be correlated with the changes seen on the different density fractions of the West Kentucky No.13 and Fire Clay materials to determine the phase transformations responsible for the change in REE recovery after roasting.

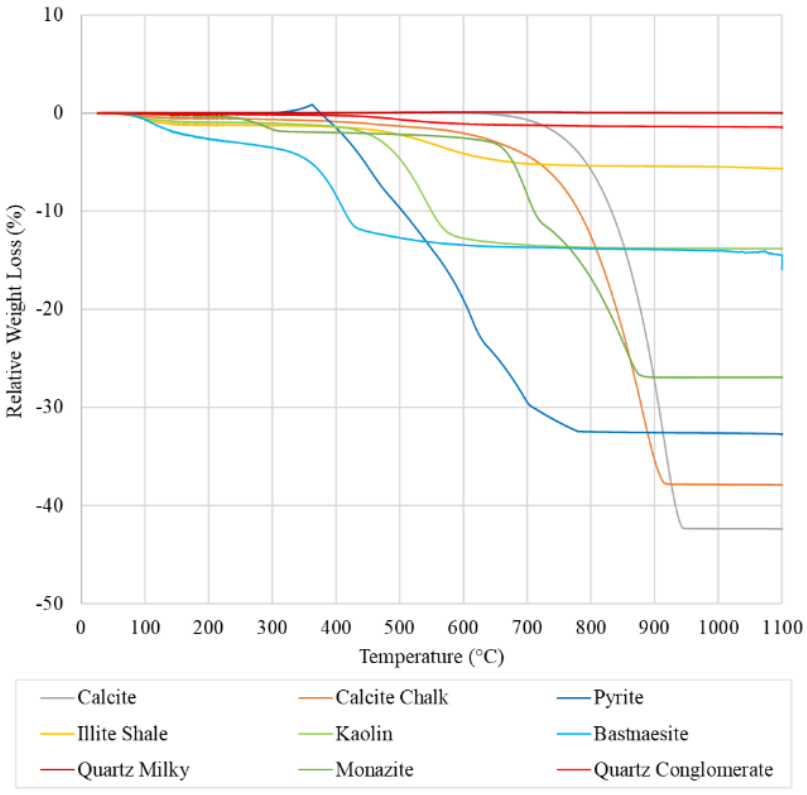



Figure 3.6.9. Thermogravimetric curves for the mineral samples. Ramp Rate= 2°C/min, O2 flow rate= 100 scc/min.

### *3.6.2 Roasting Parametric Effects*

***Oxidative environment effect on roasting:*** The LECO 701 thermogravimetric analyzer was used to produce oxidation products for all density fractions of both the feed sources under oxidative and inert atmosphere for subsequent leaching studies. The following oxidation products were obtained by programming the heating rate, hold time, temperature and gas flow rates in the TGA:

1. Direct ashing at 600°C under oxidizing atmosphere.
2. Volatiles removed at 600°C under inert atmosphere.
3. Volatiles removed at 600°C under inert atmosphere followed by ashing under oxidizing atmosphere at 600°C.

Nitrogen was used as the purge and inert gas in the furnace for the removal of volatile matter. Additionally, lids were used on crucibles to limit exposure to any atmospheric oxygen. The oxidation products were subjected to acid leaching with 1.2 M sulfuric acid at 1% solids and 75°C for one hour to evaluate the best set of conditions for maximum REE recovery. The results were helpful to understand the role of volatile matter in the roasting process and the liberation/oxidation of REEs from the feed. This data was also used for designing a high-capacity tube roaster with two (volatile removal followed by ashing) or single stage roasting (direct ashing).

Acid leaching tests with 1.2 M sulfuric acid at 75°C for one hour were used as a standard test for evaluating the performance of the roasting products. Tests were conducted to understand the effect of having an intermediate cooling step between removal of volatiles in inert atmosphere and ashing of the remaining sample under oxidizing atmosphere. Both the samples achieve the same final ash content after the test run as shown in Figure 3.6.10. It was found that the total REE in solution after one hour of acid leaching for intermediate cooling was 3.03 ppm in solution while the solution obtained from material that was not cooled between volatile removal and ashing contained 3.09 ppm. Therefore, no significant difference was observed between the two methods.

For the acid leaching products, the slight increase in the REE content in the solutions produced from de-volatilized samples relative to the untreated feed may be a result of increased porosity of the samples (Table 3.6.2). The ash products that were de-volatized and not de-volatized, however, do not exhibit a significant difference in leaching performance. Therefore, based on the data, it can be concluded that de-volatilization does not have a significant positive/negative effect such as sintering on the oxidation products.

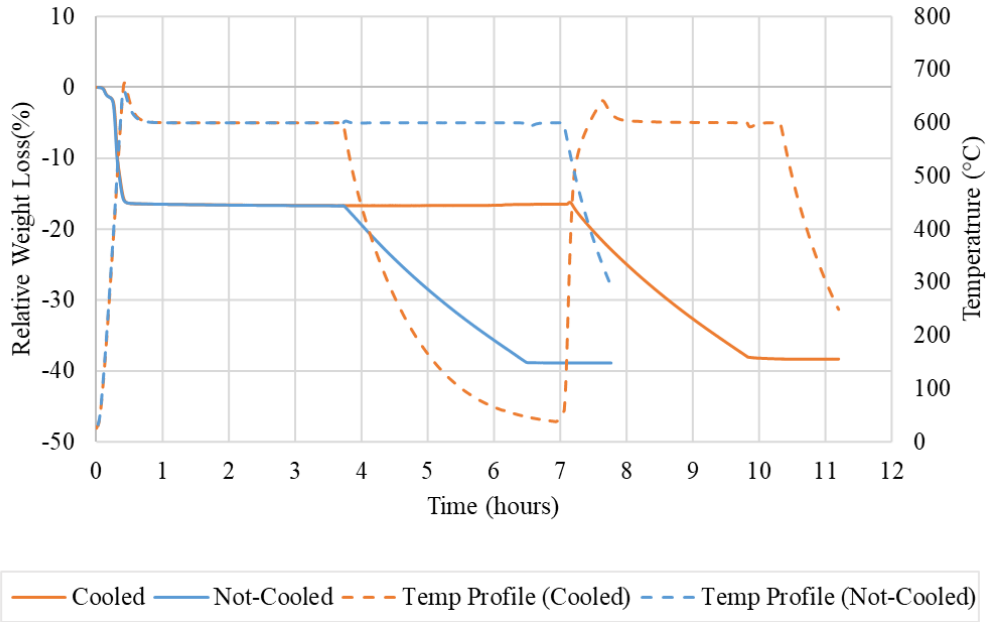



Figure 3.6.10. Thermogravimetric curves for 1.80 x 2.00 SG of the Fire Clay coarse refuse that was de-volatilized and then ashed.

Table 3.6.2. Total REE contents in the leach solution produced from the roasted products of the Fire Clay and West Kentucky No.13 coarse refuse materials.

| Fire Clay Coarse Refuse | | | | |
|---|---|---|---|---|
| Specific Gravity | Feed | No Volatiles(600°C) | No Volatiles + Ash (600°C) | Ash (600°C) |
| 1.60 Floats | 0.39 | 0.62 | 4.71 | 4.48 |
| 1.60x1.80 | 0.85 | 0.74 | 2.87 | 2.79 |
| 1.80x2.00 | 1.21 | 1.39 | 3.39 | 3.36 |
| 2.00x2.20 | 1.04 | 1.76 | 3.31 | 3.35 |
| 2.20 Sink | 0.42 | 1.80 | 1.87 | 1.45 |
| West Kentucky No.13 Coarse Refuse | | | | |
| Specific Gravity | Feed | No Volatiles(600°C) | No Volatiles + Ash (600°C) | Ash (600°C) |
| 1.80 Floats | 0.22 | 0.54 | 2.27 | 2.43 |
| 1.80x2.00 | 0.46 | 2.67 | 4.43 | 4.30 |
| 2.00x2.20 | 0.53 | 2.50 | 3.62 | 3.51 |
| 2.20 Sink | 0.39 | 1.38 | 1.45 | 1.88 |

***Pyrite oxidation effect*:** A study was performed on the 2.20 SG sink fraction of the West Kentucky No. 13 material to minimize the contaminant ions in the pregnant leach solution by roasting pyrite to magnetic oxides. The density fraction contained 82% ash and had a higher concentration of LREEs than HREEs (Figure 3.6.11). This density fraction was found to contain about 10% sulfur in the feed and a pyrite content of 9% as identified by XRD analysis (Table 3.6.3). The pyrite contained in the coal-derived sample is the potential source of iron contamination in the acid leaching process. The investigation is based on the hypothesis that pyrite can be oxidized and

converted to magnetite and/or hematite under controlled roasting conditions and the magnetic properties of pyrite oxidation products can be used to separate them from the feed going into the leach circuits. The reduction in iron contamination in the pregnant leach solution would have potential benefits in cost and operational ease in the downstream solvent extraction and selective precipitation processes.

The density fraction was roasted at different temperatures ranging from 100-1000°C for 2 hours in a muffle furnace. The heating ramp rate was set at a constant 10°C/min. The roasting products were analyzed for REE contents and were subjected to XRD analysis to determine the mineral composition. The relative concentration of pyrite, hematite, and magnetite in the roasted products are summarized in Table 3.6.3.

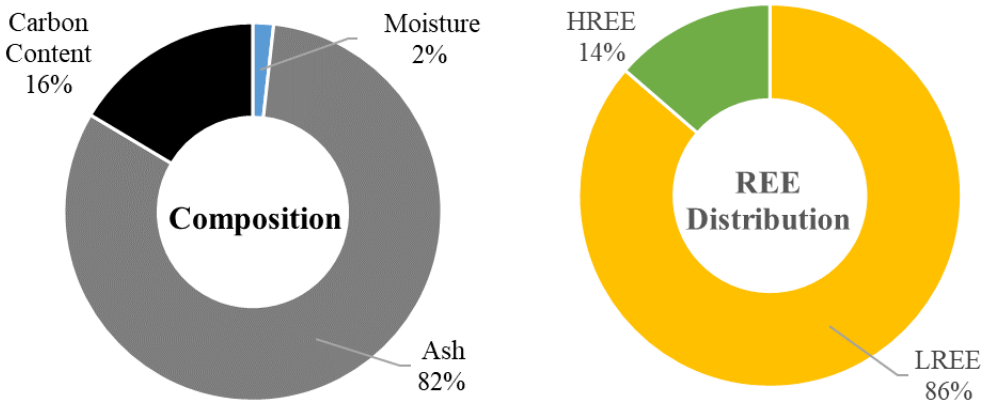



Figure 3.6.11. Composition and REE makeup of the 2.20 sink fraction of WK No.13 material.

Table 3.6.3. The concentration of sulfur, pyrite, hematite, and magnetite in the 2.20 SG sink fractions of the West Kentucky No.13 coarse refuse and associated roasted products at various temperatures.

| Sample | % Sulfur | Pyrite (%) $FeS_2$ | Hematite (%) $Fe_2O_3$ | Magnetite (%) $Fe_3O_4$ |
|---|---|---|---|---|
| Feed | 9.7 | 9.0 | 0.0 | 0.0 |
| 100°C | 11.1 | 9.1 | 0.0 | 0.0 |
| 200°C | 10.6 | 9.3 | 0.0 | 0.0 |
| 300°C | 8.5 | 4.0 | 0.0 | 0.0 |
| 400°C | 2.0 | 0.0 | 6.5 | 0.0 |
| 500°C | 0.9 | 0.0 | 5.7 | 0.5 |
| 600°C | 0.7 | 0.0 | 9.6 | 1.6 |
| 700°C | 0.5 | 0.0 | 7.7 | 1.0 |
| 800°C | 0.4 | 0.0 | 8.2 | 0.7 |
| 900°C | 0.3 | 0.0 | 11.2 | 0.0 |
| 1000°C | 0.2 | 0.0 | 17.8 | 0.0 |
| 1100°C | 0.1 | 0.0 | 23.6 | 0.0 |

It was found that complete oxidation of pyrite occurs at 400°C. Magnetite is formed as a product between 500-800°C. Due to abundant oxygen, hematite was the major oxidation product of pyrite. These roasting products were subjected to a standard acid leach test with 1.2 M $H_2SO_4$ at 75°C for 5 hours to determine recovery. High REE leaching recoveries were found for 500-800°C temperature range. The contaminant iron dissolution however depended on the roasting temperature. Maximum iron concentration was found at 400°C, which decreased significantly with a rise of roasting temperature. This suggests that the iron in hematite at 400°C is more soluble than the hematite formed at 800°C. This may be due to the difference in the crystalline structure, which changes the magnetic and leaching properties. Additionally, the roasting products of 400°C were highly magnetic when subjected to a strong neodymium magnet. The correlation of LREEs with aluminum was found to be 0.93 suggesting the possible association of LREEs with clays. The data also suggests no association of REEs with pyrite. Based on the current data, flowsheet modification can be formulated using a magnetic separator to remove magnetic roasting products and the decision to use a second stage roasting can be made considering the economics of the process. The $SO_2$ produced by the combusting pyrite rich feed may be used to produce sulfuric acid.

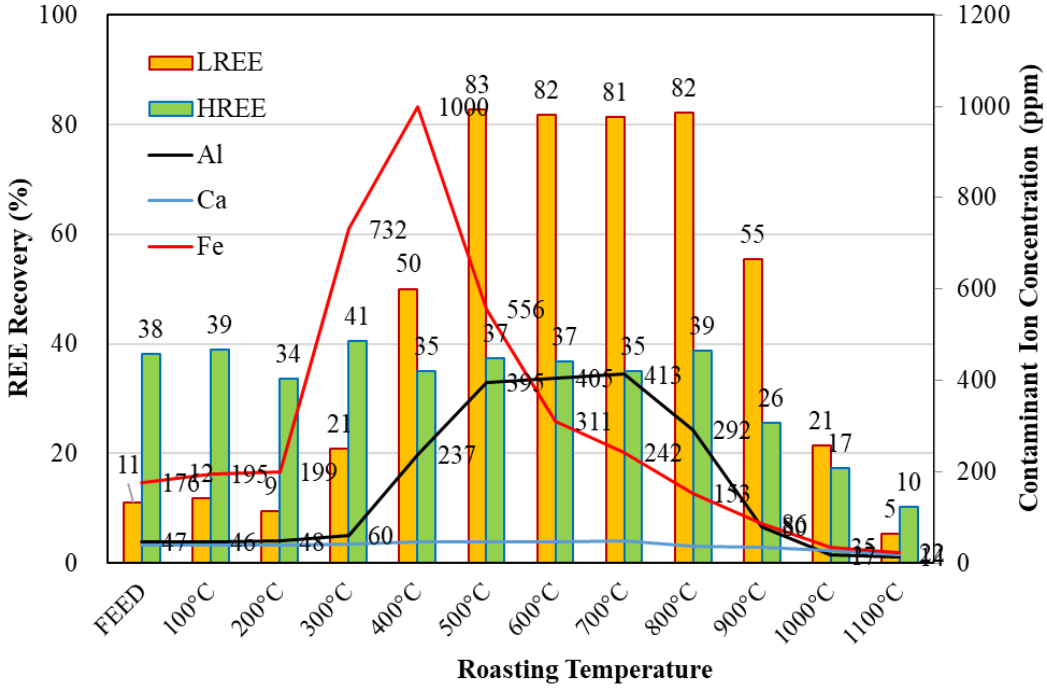



Figure 3.6.12. REE leaching recoveries for roasting products of 2.20 sink density fraction of WK No.13 material. (1.2 M H2SO4, 75°C for 5 hours)

### *3.6.3 Sulfuric Acid Production*

The process of using genetic sulfuric acid aid in the roasting process was explored for the enhancement of REE recovery. The 2.2 SG float Fire Clay course refuse coal with the addition of mineral pyrite at 1:0.5 ratio was used as a feed for the test. The tests were performed in both a muffle furnace and TGA-DSC. The experiments performed in the muffle furnace included roasting of a blank coal sample and coal+pyrite mixture. This test plan provided a basis for performance

evaluation. The roasting conditions used in the furnace were 600 °C and 2-hr dwell time with the ramp rate of 10 °C/min. The TGA-DSC experiments were performed at temperatures of 750 °C in the presence of $O_2$ and water vapor. The gas flow rates used in the TGA-DSC experiment are shown in Table 3.6.4**.**

Table 3.6.4. Individual flow rates used in the experiment.

| | |
|---|---|
| **$N_2$ Rate** | 75 $\boldsymbol{\mu l/min}$ |
| **$O_2$ Rate** | 75 $\mu l/min$ |
| **Water Equivalent Rate** | 25 $\mu l/min$ |

Method:

a. TGA-DSC
   i- Fire Clay 2.2 SG sink material and mineral pyrite mixture was heated to 400°C at 10°C/min in the presence of $N_2$ as a purge gas.
   ii- At 400°C, oxygen and water mist were introduced for the combustion of carbon as well as the conversion of $SO_2$ to $SO_3$.
   iii- The dwell time of 4-hr at 550°C ensured the complete combustion of carbon and pyrite decomposition in the coal.
   iv- At 750°C, a dwell time of 4-hr provided sufficient time for the complete production of sulfuric acid and its reaction with REEs.
b. Muffle Furnace
   i- Fireclay 2.2 SG sink material and mineral pyrite mixture were roasted at 600°C and a 2-hr dwell time with a ramp rate of 10 °C/min.
   ii- The blank coal sample was also roasted at 600°C and a 2-hr dwell time with a ramp rate of 10 °C/min.

**Results and Discussion:** The formation of sulfuric acid occurs by the oxidation of $SO_2$ to $SO_3$ in the presence of hematite as a catalyst. The freshly formed $SO_3$ is then converted to $H_2SO_4$ via an additional water scrubbing stage. This provides a possibility to enhance REE recovery through the generation of sulfuric acid in the reaction chamber. To that end, tests were performed and the results are shown in Figure 3.6.13. The leaching recovery of the pyrite and coal sample roasted in the muffle furnace is approximately 9% lower in comparison to the blank coal sample when roasted under standard conditions. The TREE recovery is 58.56% and 49.77% for blank, and coal and pyrite mixture, respectively. Comparatively, TREE recovery of the samples roasted in a controlled atmosphere in TGA-DSC seems to be adversely affected by the addition of pyrite and water vapor. The recovery, in this case, drops to a low of 37.54%. A possible explanation for this decrease in the recovery may be the unsuccessful formation of sulfuric acid.

It has been shown in the previous TGA-DSC studies that pyrite starts decomposing at approximately 400°C. Coincidentally, the carbon also starts to burn at around the same temperature range. This might be due to the oxygen consumption by carbon for combustion in an oxygenated environment, leaving little to no products for $SO_3$ conversion. Furthermore, $SO_2$ to $SO_3$ reaction takes place in the presence of a catalyst. The majority of pyrite in coal converts to hematite at

temperatures above 700°C. At lower temperatures, the presence of hematite is little, if not negligible. As the pyrite decomposition starts at 400°C, it can be assumed that $SO_2$ formation also starts around the same temperature range. As previously stated, the hematite content present in the system is minimal at that temperature. Therefore, without the presence of a catalyst, $SO_2$ to $SO_3$ conversion is unsuccessful. As the temperature is increased up to 600°C, the majority of the pyrite decomposes to hematite. However, sulfur-containing gases have already escaped the system at that point through the exhaust valve. Therefore, it is highly probable that $SO_2$ to $SO_3$ conversion did not take place. Furthermore, there is also a possibility that the water present in the system was insufficient.

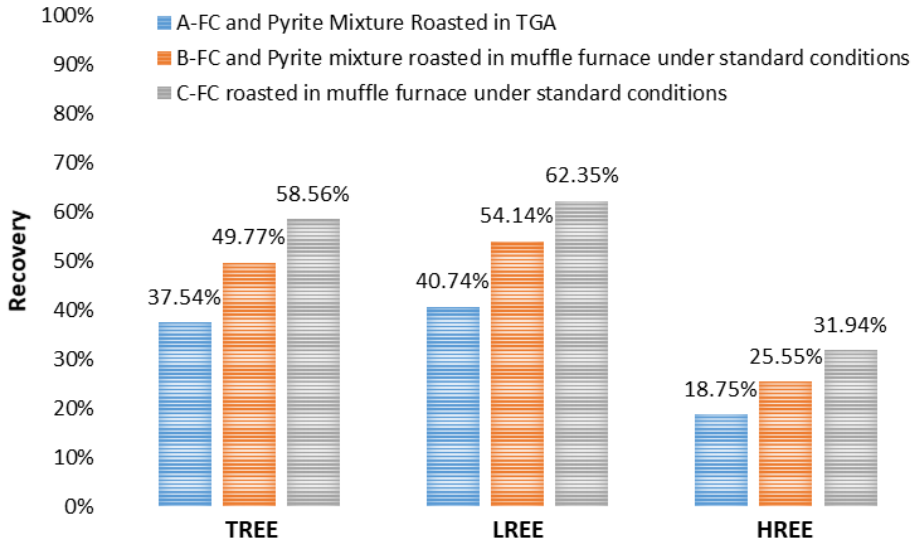



Figure 3.6.13 REE recovery of from Fire Clay 2.2 SG sink (A-FC) and pyrite mixture (1:0.5) roasted for 4-hrs at 550°C and 750 °C with water mist and oxygen being introduced at 400°C for the complete process B- FC and pyrite mixture (1:0.5) roasted at 600°C for 2-hrs C- FC sample roasted at 600°C for 2hrs.

### *3.6.4 Leaching Studies*

**Baseline Leachability Characteristics:** Baseline acid leaching tests with 1.2 M Sulfuric acid at 75°C for five hours were performed on various density fractions of the West Kentucky No.13 and Fire Clay coarse refuse materials. This feedstock was density fractionated by SGS (commercial lab). The total REE recovery values for the West Kentucky and Fire Clay Samples are shown in Figure 3.6.14. The TREE leaching recovery of the four density fractions of the WK No.13 material was less than 30% (Figure 3.6.14 1-A). The total REE recovery gradually increased with an increase in reaction time. The baseline TREE recovery for Fire Clay material was found to be higher than WK No.13. The total REE recovery for the 2.20 sink fraction of Fire Clay was found to be the lowest. Overall, the Fire Clay sample exhibits better leaching characteristics than the WK No.13 sample.

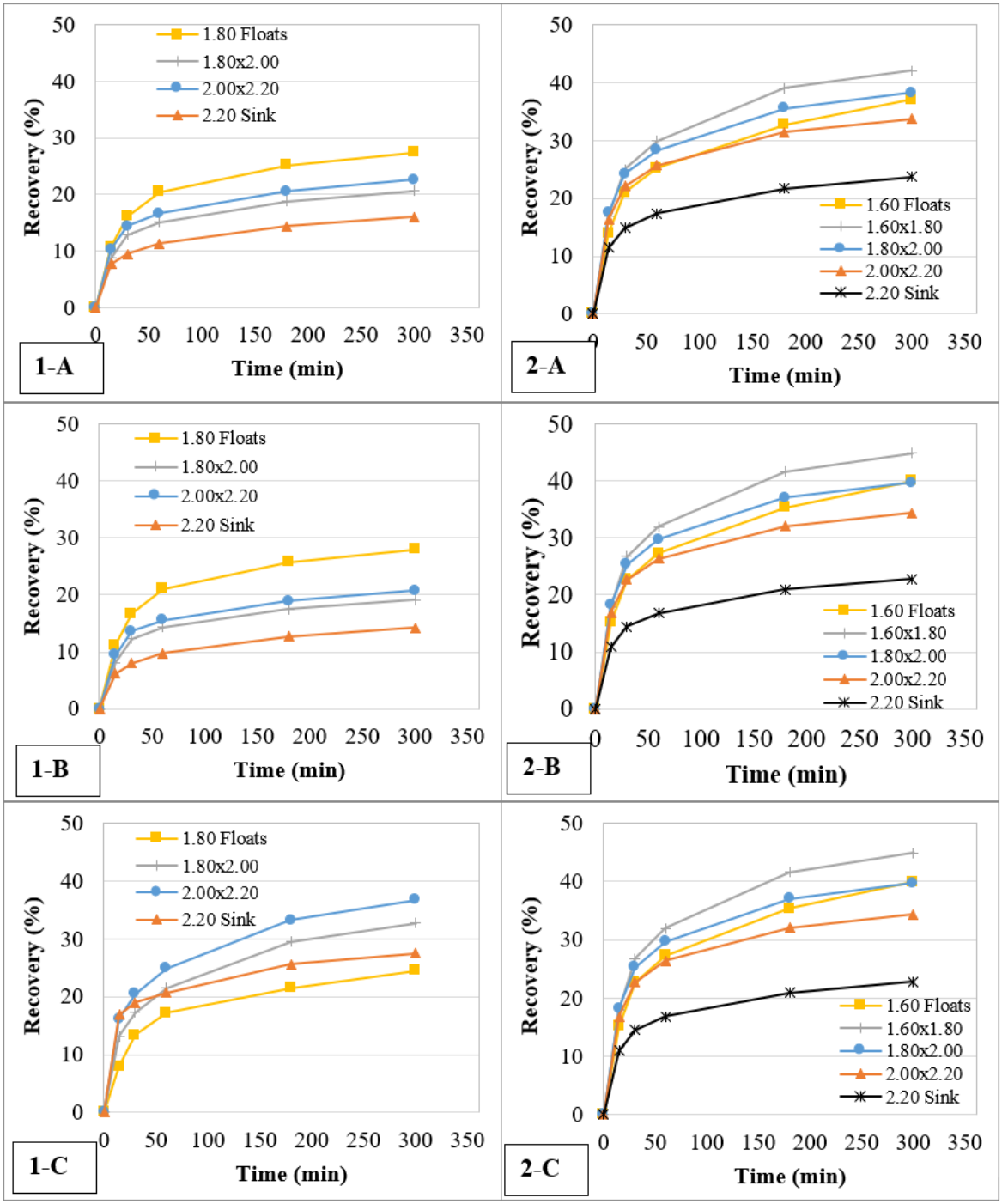



Figure 3.6.14. REE leaching recoveries for (1) West Kentucky No.13 and (2) Fire Clay coarse refuse density fractions. (A)TREE, (B) LREE and (C) HREE. (1.2 M $H_2SO_4$, 75°C for 5 hours).

### *3.6.5 Optimization of Blank Roasting*

A test program using a Box-Behnken statistical design was performed to quantify the main parameters and their interaction effects on the response variables. It also allows the development

of a mathematical model to describe the process by assessing the statistical significance of the independent factors as well as the significance of the interactions between the factors. The set of parameter values that provide an optimum performance can also be identified using multivariate analyses. The three independent input parameters selected for the study were temperature (°C), heating rate (°C/min), and hold time (min). The lower and upper bounds of the variables are presented in Table 3.6.5. The temperature range for the tests was selected based on the results of the parametric tests. The maximum value for heating ramp rate was selected based on a controlled experiment where the increase in furnace temperature was measured with time. The maxima for hold time was based on the studies presented previously by Zhang where a hold time of 120 minutes provided complete oxidation of the feed (W. Zhang & Honaker, 2020). The recovery (%) of REEs and the concentration of major contaminant ions (Al, Ca, and Fe (ppm)) were measured as response variables.

Table 3.6.5. Levels of Independent variables and their ranges for Box-Behnken design.

| Factor | Name | Units | Minimum | Maximum | Mean |
|---|---|---|---|---|---|
| A | Temperature | °C | 400 | 800 | 600 |
| B | Ramp | °C/min | 2 | 10 | 6 |
| C | Hold Time | min | 0 | 120 | 60 |

The three-factor, three-level Box-Behnken experimental configuration formulated using Design Expert-12 software required 17 test runs for the identification of significant factors and development of empirical models to relate REE recovery and contaminant concentration with the significant input parameters. The runs were randomized to minimize systematic error and ensure that the results were probabilistically independent. The significance of the empirical model was tested based on the p-values obtained from the Analysis of Variance (ANOVA). Empirical equations were built to predict the response based on the individual variables and their interactions. The significance of each parameter and their interactions were tested against the null hypothesis that the coefficient for the variable is null. Terms with a p-value less than 0.05 were considered significant. All insignificant parameters were removed from the final model. High adjusted $R^2$ value was used as a measure to determine the efficacy of the best-fit model to the data. Additionally, the models were tested against the following assumptions of regression:

1. Residual errors for the selected models were independent;
2. Residual errors for the selected models were distributed normally with a mean value of zero; and
3. The variance of errors for the selected models was constant across all the observations.

**Results and Discussions:** The calcination tests at varying temperatures were conducted to establish the upper and lower bound variables in the subsequent calcination optimization studies. The leaching recovery for light and heavy REEs along with the concentration of Al, Ca, and Fe for the calcination products of all the density fractions of Baker seam coal at temperatures ranging from 100-1000°C is provided in Figure 3.6.17-Figure 3.6.18. About 9-17% of the LREEs and 32-43% of the HREEs were leached from the untreated density fractions of the West Kentucky No. 13 seam material. The higher recovery of the heavy REEs from the untreated 2.20 SG sink fraction (Figure 3.6.18) as compared to the LREEs contributes to a higher concentration of HREEs in the

acid mine drainage from the West Kentucky No.13 rejects (W. Zhang & Honaker, 2020). Significant benefits in the HREE recoveries were achieved after calcination from the 1.60 SG float fraction when the organic matter was fully removed, and the mineral matter was liberated at temperatures 400-600°C. As explained by Zhang and Honaker, the organic removal from the light density fraction by calcination results in the liberation of the finely dispersed mineral matter, thereby providing enhanced interaction between the lixiviant and the mineral surface (W. Zhang & Honaker, 2019). There is also the possibility of organic association of REEs with humic acids in coal which form complexes with REE ions (Erdogan et al., 2007). Based on the leaching characteristics exhibited by the HREEs there seem to be multiple modes of association as reflected by the leaching results of most density fractions. Organically associated HREEs are released by the combustion process. However, a large portion of HREEs were unaffected by calcination at high temperatures and would require an acid baking process for chemical liberation (Nawab et al., 2022b). This could be a signature of the presence of xenotime in the mineral matrix.

However, the HREEs in the density fractions >1.80 SG showed no significant improvement in leaching kinetics and characteristics (Figure 3.6.17). This would suggest that a majority of HREEs in the high ash parting material of the West Kentucky No.13 seam are hard to leach and/or strongly bound in minerals that are not affected by thermal treatment (Ji, Li, & Zhang, 2022b). The benefits in leaching characteristics of the LREEs for all the density fractions corresponded to the increase in temperature from 500-800°C. A similar increase in the concentration of aluminum was also observed suggesting a possible co-existence of LREEs in the clays associated with the coal deposits. Calcium in the PLS primarily originated from the calcite present in the coal deposit. This calcite is highly soluble and dissolves readily in an acidic solution. The decrease in leaching recoveries of all the species at temperatures greater than 800°C may be due to the sintering of the clay surfaces that makes the REEs hard to leach. Iron contamination in the PLS solution was found to be maximum from the leaching of samples calcinated at a temperature of 400°C. This observation was likely due to the decomposition of the pyrite and the formation of an intermediate iron oxide product at 400-500°C that is highly soluble.

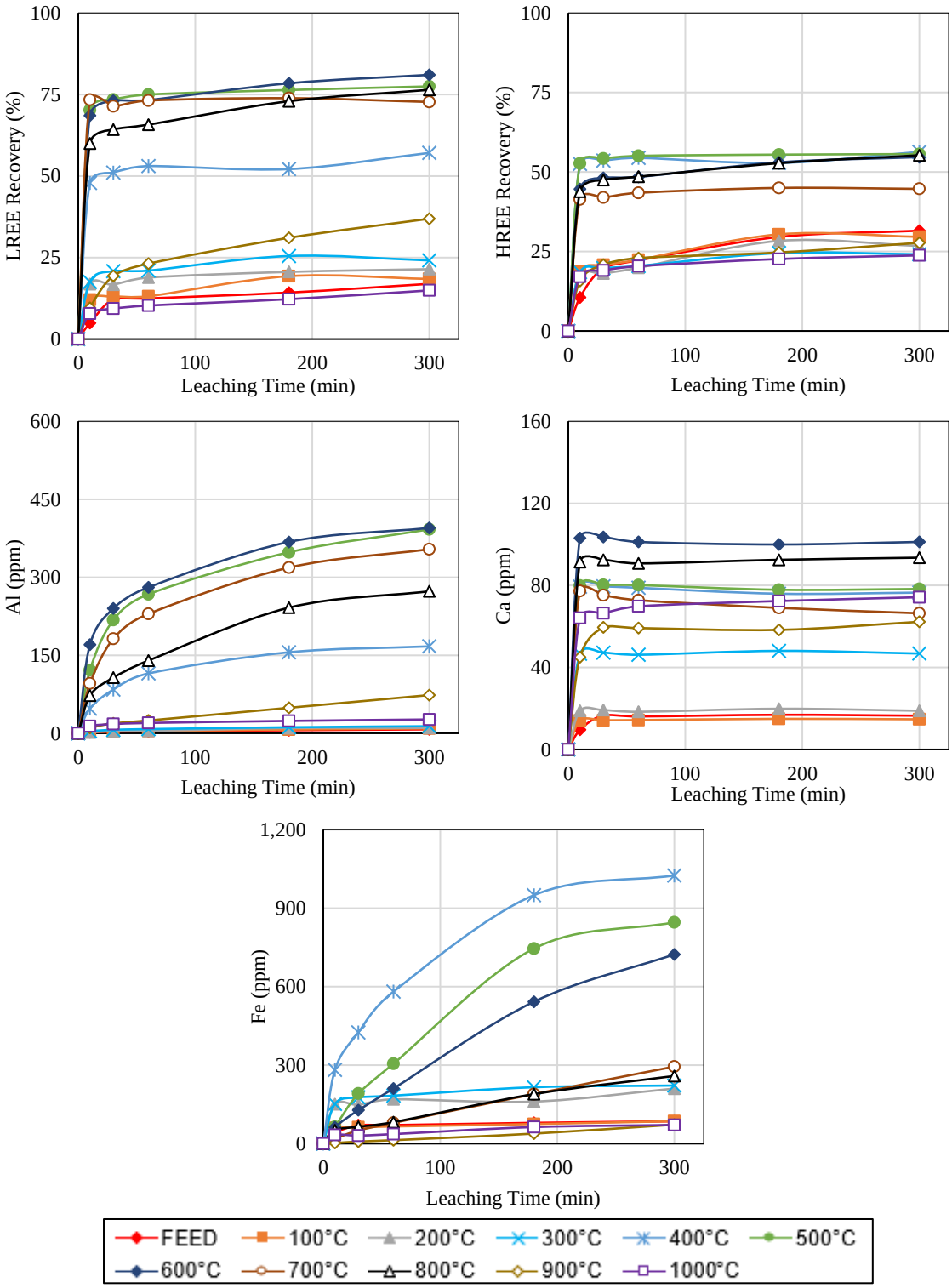



Figure 3.6.15. Acid leaching test results with 1.2M $H_2SO_4$ at 75°C for the 1.60 SG float of West Kentucky No.13 calcined at different temperatures in air.

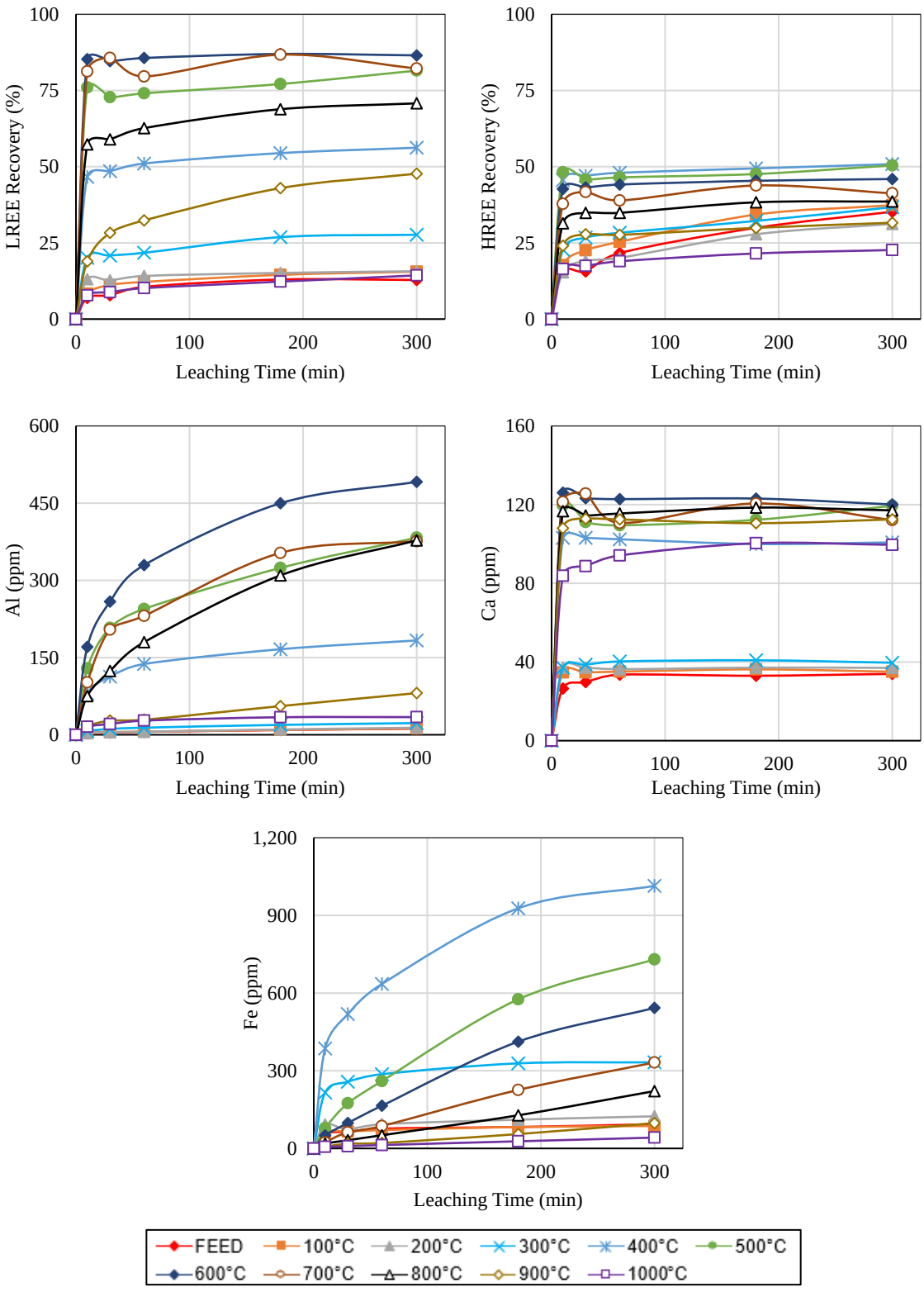



Figure 3.6.16. Acid leaching test results with 1.2M $H_2SO_4$ at 75°C for the 1.60x1.80 SG of West Kentucky No.13 calcined at different temperatures in air.

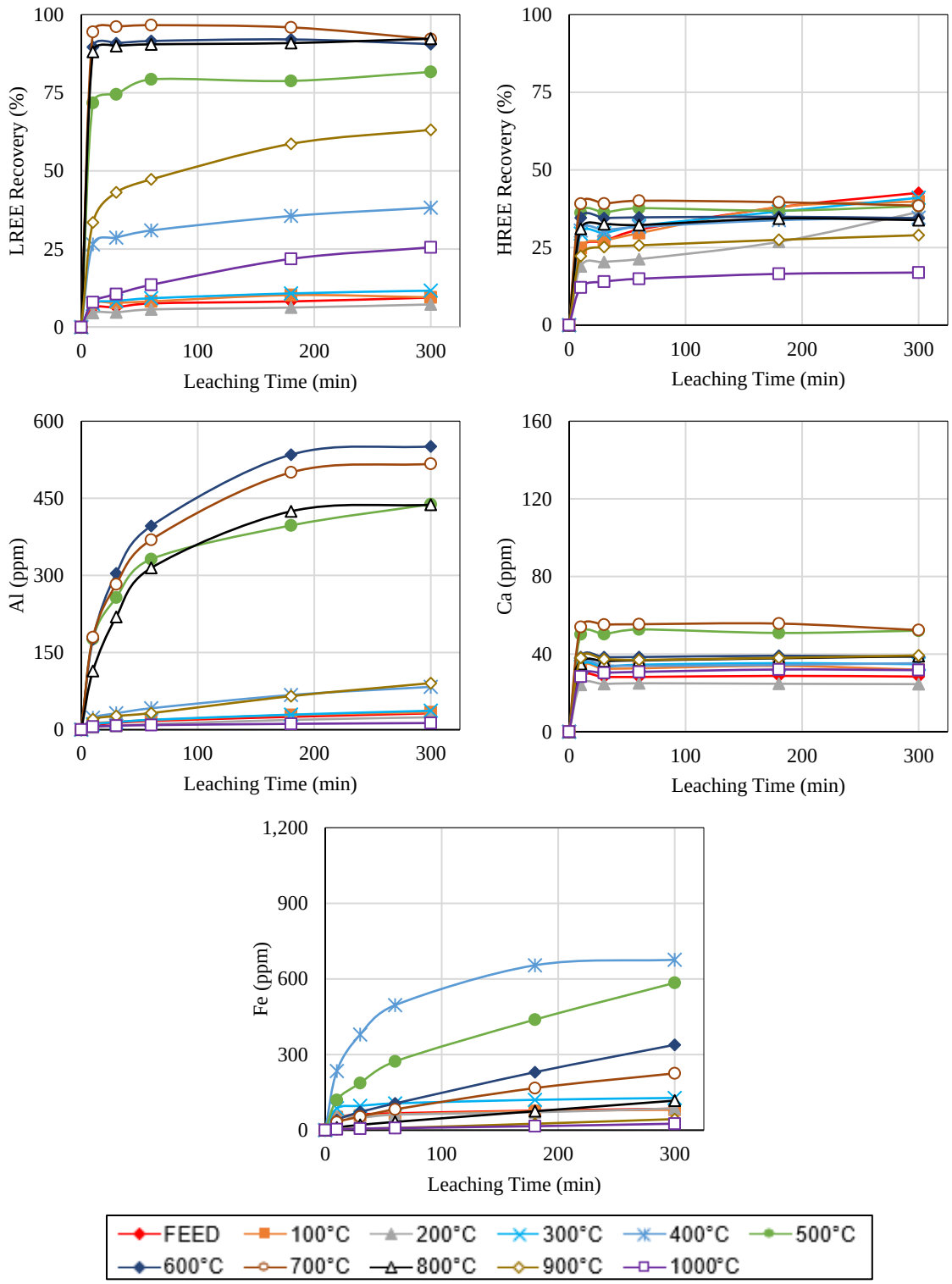



Figure 3.6.17. Acid leaching test results with 1.2M $H_2SO_4$ at 75°C for the 1.80x2.20 SG of West Kentucky No.13 calcined at different temperatures in air.

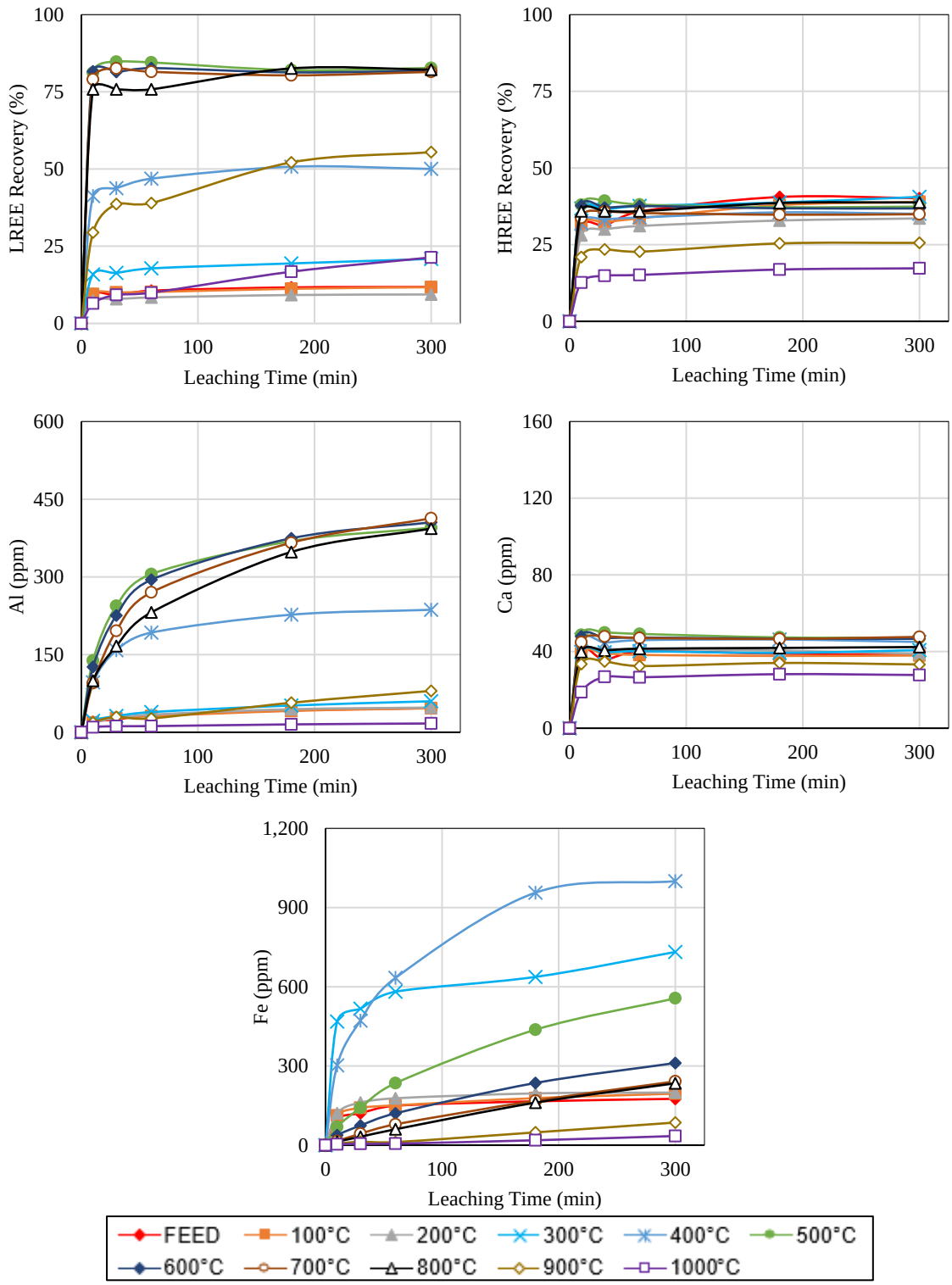



Figure 3.6.18. Acid leaching test results with 1.2M $H_2SO_4$ at 75°C for the 2.20 SG sink of West Kentucky No.13 calcined at different temperatures in air.

Calcination for the Fire Clay density fractionated also had a significant impact on the REE leaching characteristics. The leaching recovery for light and heavy REEs along with the concentration of Al, Ca, and Fe for the calcination products of all the density fractions of Fire Clay at temperatures ranging from 100-1000°C is provided in Figure 3.6.19-Figure 3.6.22. The LREE recovery improved significantly for all the density fractionated samples, especially the 1.60 SG float (from 34% to 77%) after high-temperature treatment at 500°C for 2 hours. However, for the other three density fractions, calcination at 600-700°C for 2 hours provided the highest leach recovery. The leaching kinetics data suggests different mineralogy and/or association characteristics of REEs in the Fire Clay from the Baker seam material owing to the better leaching kinetics of the former for an untreated sample. Calcination of the Fire Clay material at 600-800°C provides a general increase in the recovery of the REEs. However, the improvement in REE recovery was not as pronounced as the West Kentucky No.13 seam source. Ji et al. determined that LREEs in Fire Clay coal were predominantly associated with monazite, compared to Baker seam coal where REEs were associated with crandallite group minerals. Since monazite is thermally stable, and crandallite group minerals are easily decomposed at elevated temperatures, this is likely the reason for lower LREE recovery improvement in the Fire Clay material relative to Baker seam coal. The leaching behavior of the HREEs associated with the heavy density fractions of Fire Clay was found to be unaffected by the high-temperature calcination process. As the HREEs are associated with thermally stable minerals such as xenotime and xenotime, lower recovery was expected (Ji, Li, & Zhang, 2022b).

Based on the calcination results, high-temperature blank calcination significantly improved the recovery of REEs, particularly the light REEs. However, concentration of contaminants (Al, Ca, and Fe) also increased in the subsequent leaching unit operation. The benefits in REE leachability for both the coal seams were realized in the 400-800°C temperature range. Significant changes in Fe and Al leaching behavior were also observed within the same temperature values. Therefore, a subsequent test program was designed to optimize the input parameters to maximize REE recovery and minimize the concentration of contaminants in the leachate by keeping the lower and upper-temperature values at 400°C and 800°C, respectively.

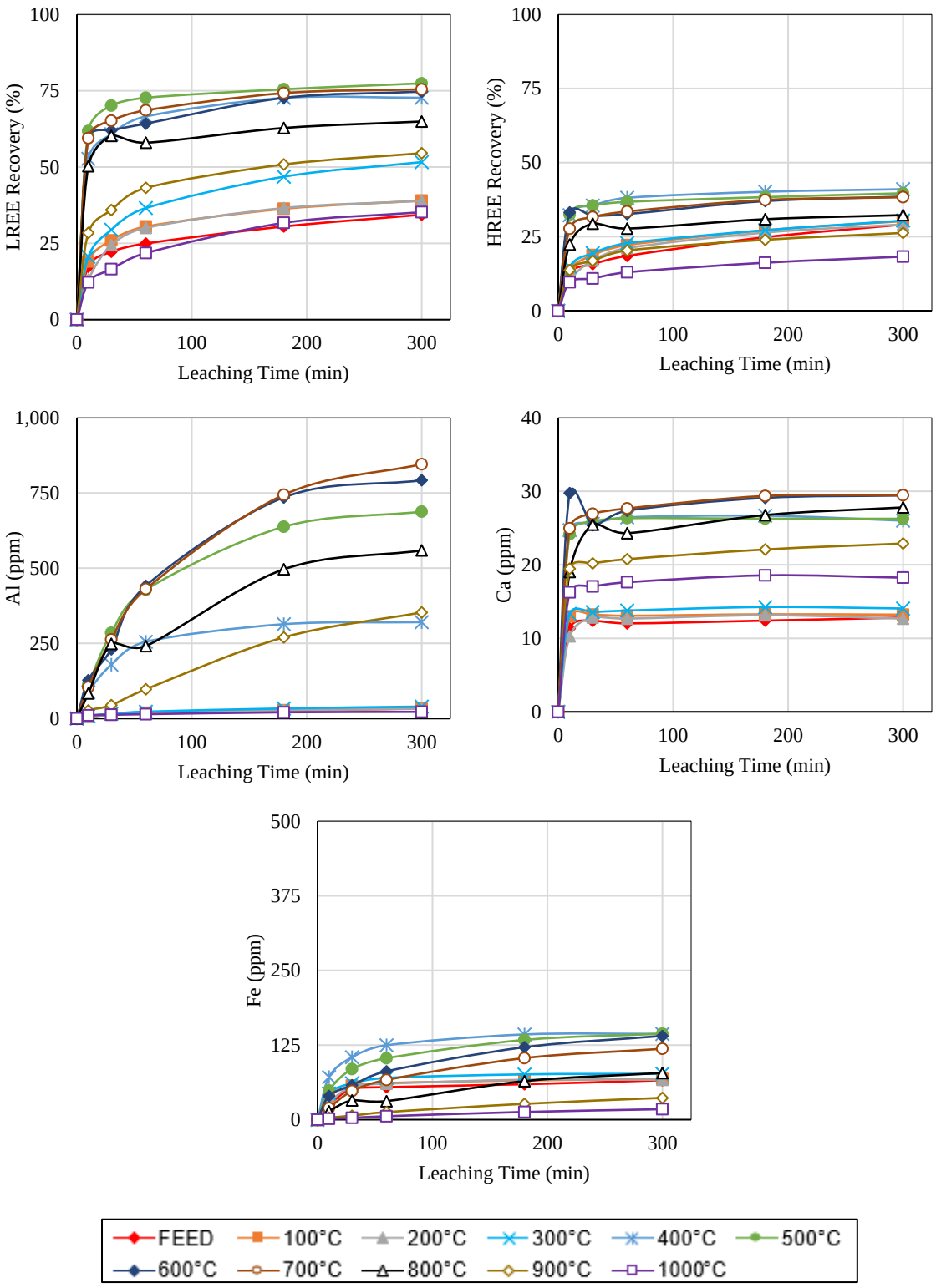



Figure 3.6.19. Acid leaching test results with 1.2M $H_2SO_4$ at 75°C for the 1.60 SG float of Fire Clay calcined at different temperatures in air.

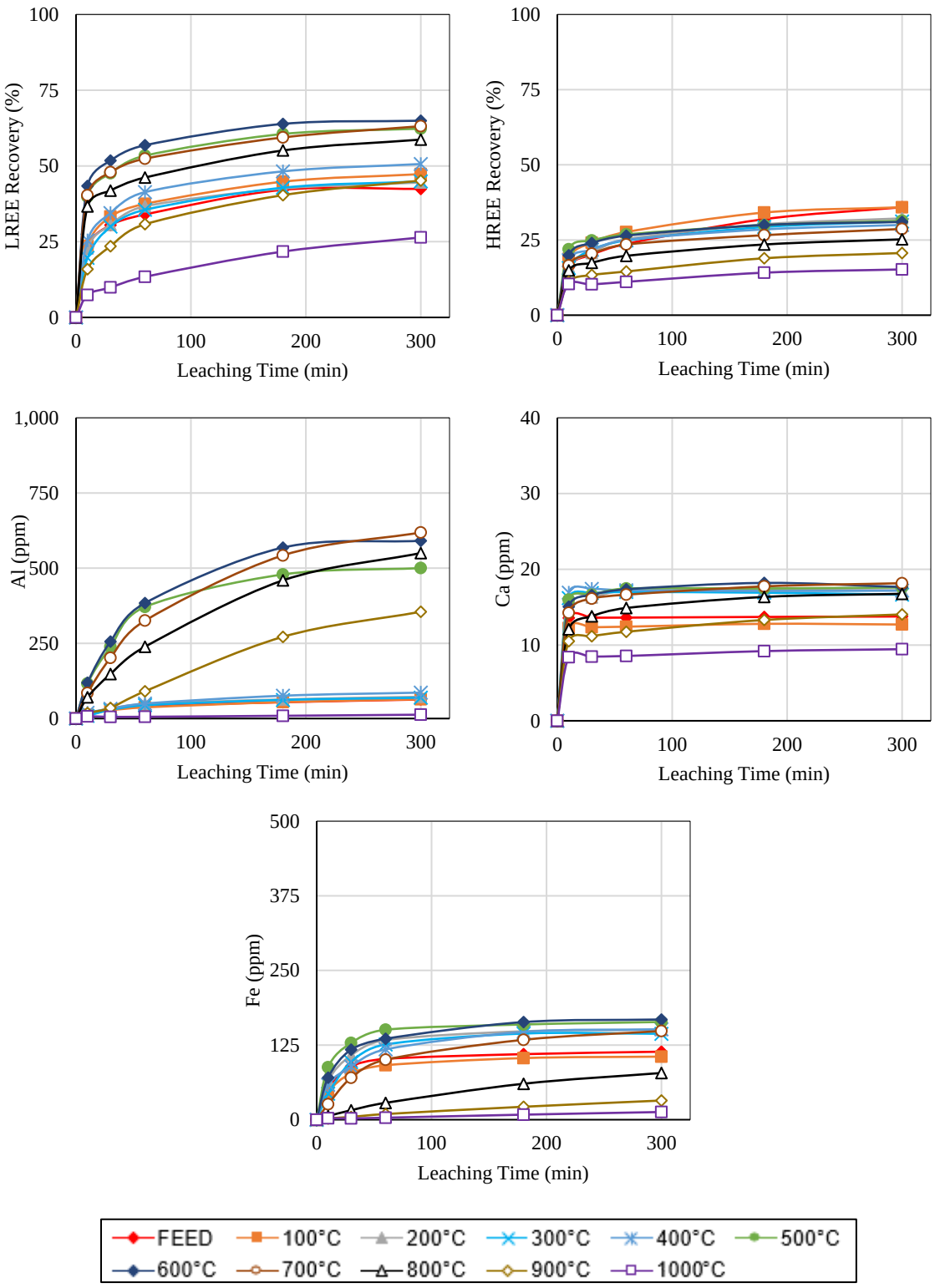



Figure 3.6.20. Acid leaching test results with 1.2M $H_2SO_4$ at 75°C for the 1.60x1.80 SG of Fire Clay calcined at different temperatures in air.

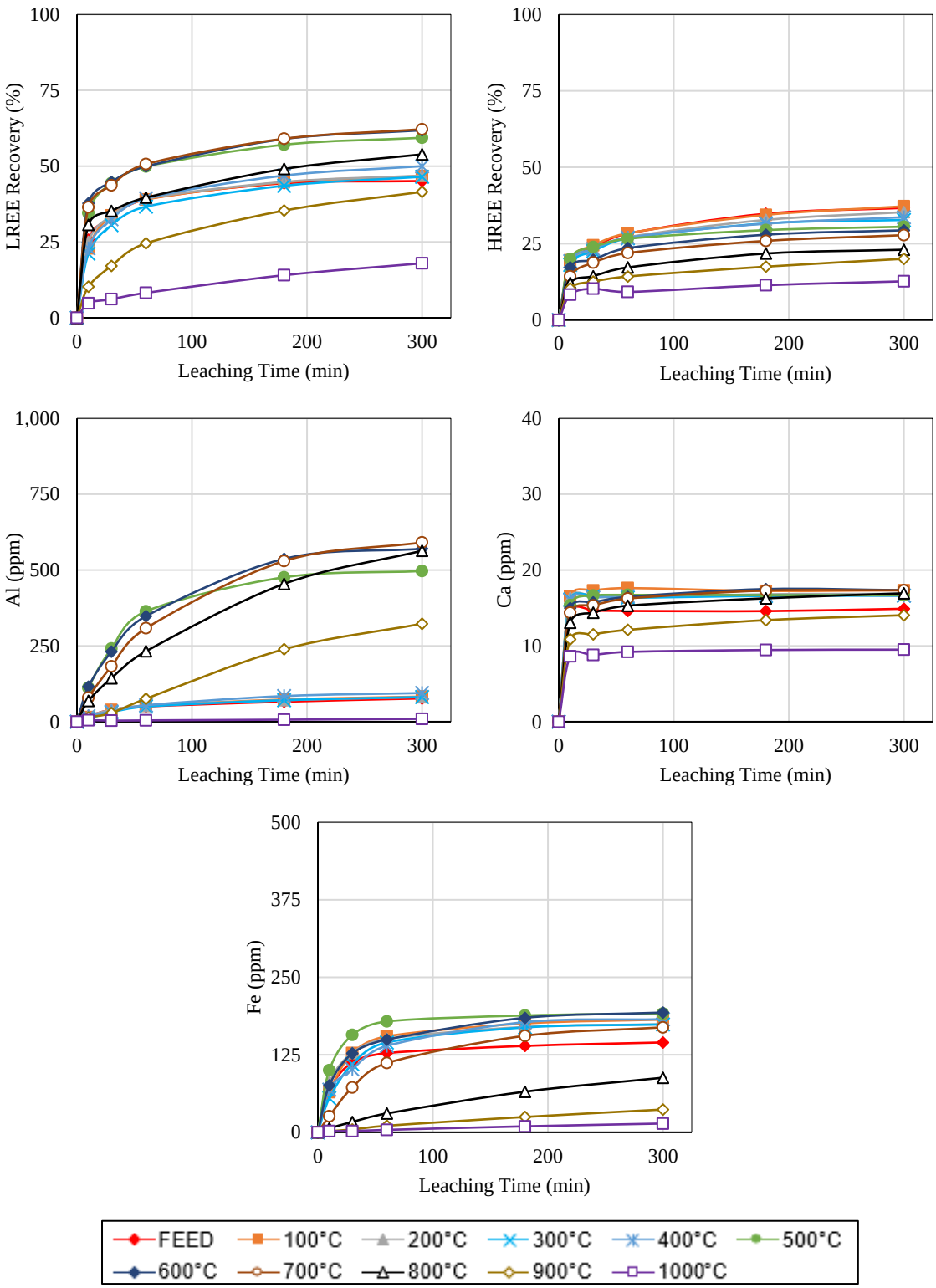



Figure 3.6.21. Acid leaching test results with 1.2M $H_2SO_4$ at 75°C for the 1.80x2.20 SG of Fire Clay calcined at different temperatures in air.

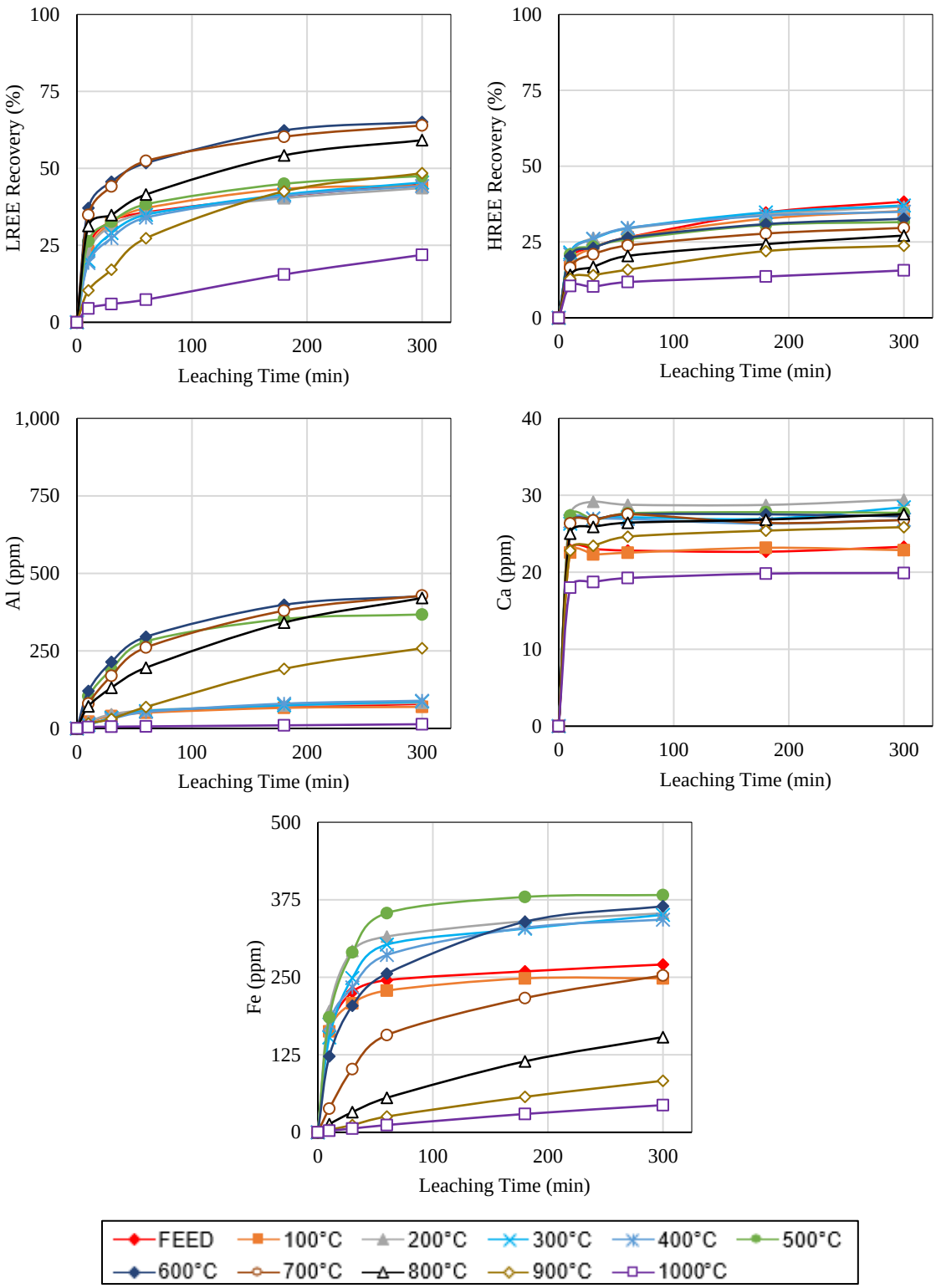



Figure 3.6.22. Acid leaching test results with 1.2M $H_2SO_4$ at 75°C for the 2.20 SG sink of Fire Clay calcined at different temperatures in air.

### *3.6.6 Parametric Optimization Study*

**West Kentucky No.13 Seam:** The details of the calcination test conditions performed on the 1.6 SG float and 2.2 SG sink fractions of the West Kentucky No.13 seam coarse refuse material are provided in Figure 3.6.6**.** Significant benefits in the recovery of REEs were achieved for the 1.60 SG float fraction as compared to the 2.20 SG sink material at 400°C. The partial removal of organic matter associated with the lighter density fraction allowed for higher LREE and HREE recoveries at 400°C. However, exceptionally high LREE recoveries (>80%) were achieved at high calcination temperatures (600-800°C) for the 2.20 SG sink fraction. The difference in optimum calcination temperatures between the two density fractions could be due to the different modes of occurrence of the REEs. Since the majority of REEs in the heavier density fractions are associated with clays. A higher optimum temperature for is needed to dehydroxylate the clays and subsequently liberate/decompose some associated REE-bearing minerals.

Empirical models that describe REE recovery as a function of the process parameter values were developed. The 3-dimensional surface plots for the main and interaction effects are provided in Figure 3.6.23 and Figure 3.6.24. LREE showed a quadratic relationship with temperature (Figure 3.6.23.a-i & a-ii). Significant improvement in the LREE recovery of 2.2 SG fraction with an increase in temperature is likely due to liberation/decomposition of REE-bearing minerals through clay dehydroxylation. Higher HREE recovery in the 1.6 SG float fraction (Figure 3.6.23.b-i) as compared to the 2.2 SG sink fraction (Figure 3.6.23.b-ii) corresponds to an increase in the ash content by calcination and suggests an organic association. The recovery of HREE from calcined products of 2.20 SG sink fraction was marginally higher relative to the feed but was unaffected by a further increase in temperature. The HREE recovery was within the range of experimental error in the test program (Figure 3.6.23.b-ii). Thus, about 60% of heavy REEs remained locked in the WK-13 ash material and were difficult to leach despite strong acid-leaching conditions.

Table 3.6.6. Input factors and the results of the Box- Behnken experimental design for 1.60 SG float and 2.20 SG sink of West Kentucky No.13.

| Run | Temp. (°C) | Heating Rate (°C/min) | Hold Time (min) | TREE (%) | LREE (%) | HREE (%) | Al (ppm) | Ca (ppm) | Fe (ppm) | TREE (%) | LREE (%) | HREE (%) | Al (ppm) | Ca (ppm) | Fe (ppm) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | 1.60 SG float | | | | | | 2.20 SG sink | | | | | |
| Untreated Feed | | | | 20 | 17 | 32 | 7 | 16 | 85 | 16 | 12 | 40 | 47 | 38 | 176 |
| 1 | 600 | 2 | 0 | 75 | 79 | 61 | 348 | 70 | 1265 | 79 | 85 | 41 | 356 | 42 | 781 |
| 2 | 600 | 6 | 60 | 67 | 71 | 52 | 357 | 71 | 731 | 77 | 84 | 37 | 408 | 45 | 564 |
| 3 | 600 | 6 | 60 | 66 | 71 | 51 | 379 | 74 | 731 | 77 | 83 | 38 | 404 | 45 | 581 |
| 4 | 600 | 6 | 60 | 73 | 77 | 58 | 410 | 74 | 972 | 77 | 83 | 38 | 401 | 44 | 560 |
| 5 | 400 | 6 | 120 | 56 | 55 | 57 | 113 | 50 | 1109 | 34 | 33 | 37 | 91 | 46 | 844 |
| 6 | 800 | 6 | 0 | 65 | 70 | 49 | 410 | 76 | 368 | 77 | 83 | 39 | 436 | 51 | 321 |
| 7 | 400 | 2 | 60 | 57 | 56 | 59 | 82 | 43 | 1039 | 31 | 30 | 36 | 85 | 45 | 995 |
| 8 | 400 | 6 | 0 | 53 | 54 | 52 | 51 | 26 | 797 | 28 | 27 | 35 | 71 | 43 | 861 |
| 9 | 600 | 6 | 60 | 66 | 69 | 53 | 385 | 81 | 694 | 75 | 80 | 40 | 381 | 48 | 798 |
| 10 | 600 | 2 | 120 | 67 | 71 | 54 | 383 | 78 | 817 | 74 | 80 | 37 | 431 | 52 | 869 |
| 11 | 800 | 2 | 60 | 58 | 62 | 43 | 373 | 69 | 363 | 72 | 78 | 37 | 464 | 53 | 283 |
| 12 | 800 | 6 | 120 | 57 | 60 | 45 | 352 | 70 | 225 | 72 | 77 | 35 | 429 | 49 | 218 |
| 13 | 400 | 10 | 60 | 50 | 49 | 52 | 89 | 40 | 775 | 34 | 33 | 36 | 84 | 42 | 841 |
| 14 | 800 | 10 | 60 | 50 | 53 | 37 | 280 | 73 | 129 | 75 | 81 | 39 | 422 | 50 | 219 |
| 15 | 600 | 10 | 0 | 64 | 68 | 51 | 346 | 68 | 913 | 72 | 77 | 37 | 382 | 49 | 905 |
| 16 | 600 | 6 | 60 | 61 | 66 | 48 | 369 | 71 | 935 | 75 | 81 | 39 | 435 | 51 | 682 |
| 17 | 600 | 10 | 120 | 61 | 64 | 48 | 349 | 76 | 379 | 61 | 64 | 48 | 349 | 76 | 379 |

The Pearson linear correlation values for the selected response and input variables are summarized in Table 3.6.7. Aluminum was found to have a strong positive relationship with temperature for both density fractions. This can be attributed to the dehydroxylation of clay minerals (kaolinite and illite), which are the primary source of aluminum and are more leachable at higher temperatures. The decline in Al concentration at 800°C may be due to the conversation of metakaolin to mullite or aluminum-silicon spinel and amorphous silica, which has a lower solubility or due to sintering of clay at high temperatures (Figure 3.6.24. a-i) (Nawab et al., 2022b). The presence of calcium in the leachate was due to the trace amounts of calcite present in the Baker seam material. The increase in the concentration of calcium was most likely due to higher calcium content in calcined products (Figure 3.6.24. b-i & b-ii). Calcite decomposes at 800°C to form calcium oxide. Both of these species have high solubility in acidic solutions (Nawab et al., 2022b). Iron in the feed was found to be present in the form of pyrites. The concentration of iron in the

acid leachate shows an inverse relationship with temperature and heating rates. More iron was leached when the sample was treated at lower temperatures for a longer duration of time. The lower temperatures facilitated the formation of more soluble intermediate iron oxides which were converted to α-hematite at higher temperatures which have very low solubility (Figure 3.6.24. c-i & c-ii) (Gupta et al., 2023). The heating rate was found to be an insignificant parameter for final concentrations of Al and Ca, suggesting that their solubility was driven solely by the setpoint temperature (Gupta et al., 2023).

Table 3.6.7. Correlation matrix of the major response variables with the input variables used in the calcination studies of the West Kentucky No.13 sample.

| 1.60 SG float | | | | | | 2.20 SG sink | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | LREE | HREE | Al | Ca | Fe | LREE | HREE | Al | Ca | Fe |
| Temperature | 0.31 | -0.65 | 0.75 | 0.73 | -0.72 | 0.78 | 0.33 | 0.86 | 0.69 | -0.87 |
| Heating Rate | -0.35 | -0.43 | -0.08 | -0.03 | -0.35 | 0.00 | 0.00 | -0.01 | -0.04 | -0.09 |
| Hold Time | -0.20 | -0.15 | 0.03 | 0.20 | -0.22 | 0.00 | -0.37 | 0.08 | 0.28 | -0.08 |

From the results, it can be concluded that temperature is the most important parameter among the process parameters evaluated. The heating rate of the sample has an insignificant effect on the leachability of REEs as well as the concentration of contaminants. However, the hold time or residence time plays a significant role, especially at high setpoint temperatures. A high temperature coupled with long residence times allows for the sintering of particles which subsequently reduces the REE recoveries.

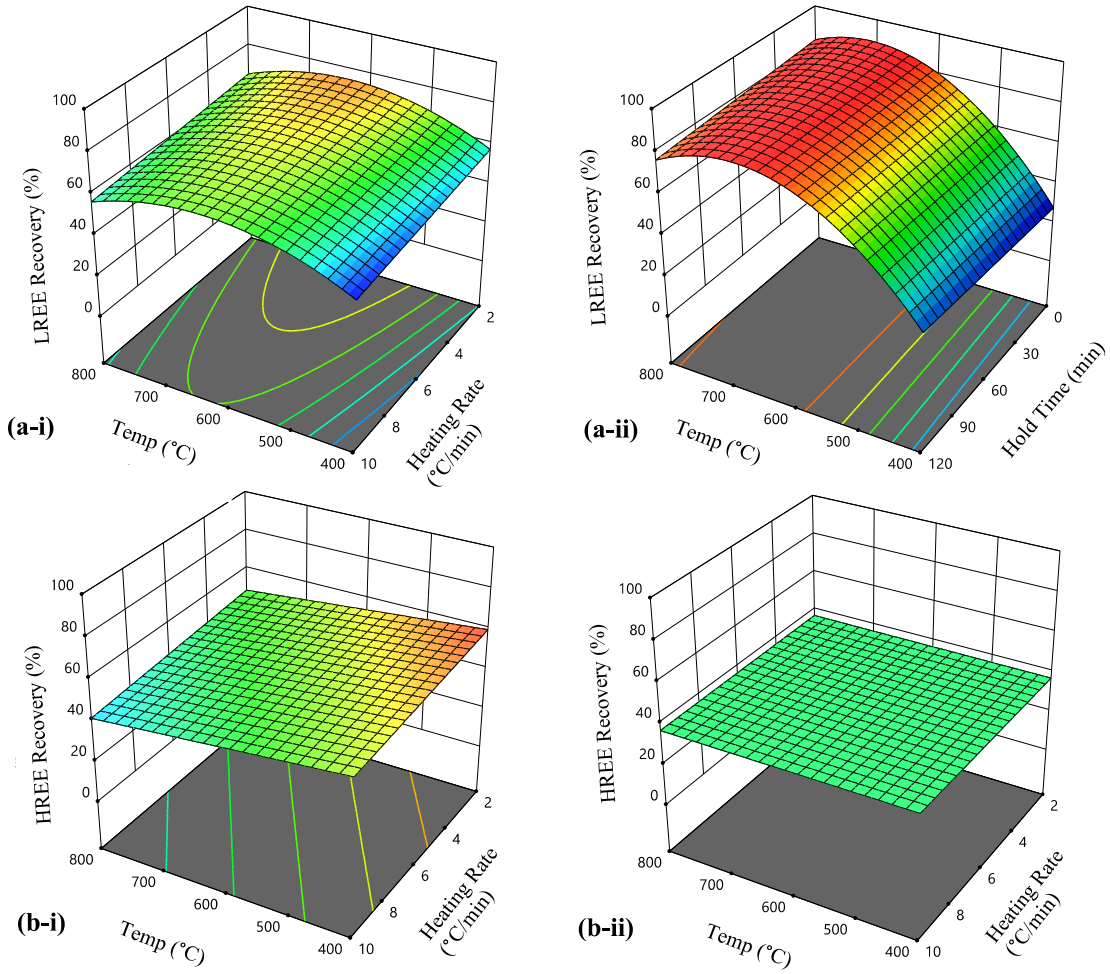



Figure 3.6.23 Response surface for the association of (a) LREE and (b) HREE recovery (%) of the calcined (i) 1.60 SG float and (ii) 2.20 SG sink fraction of West Kentucky No.13 as a function of main and interaction effects of the significant variables

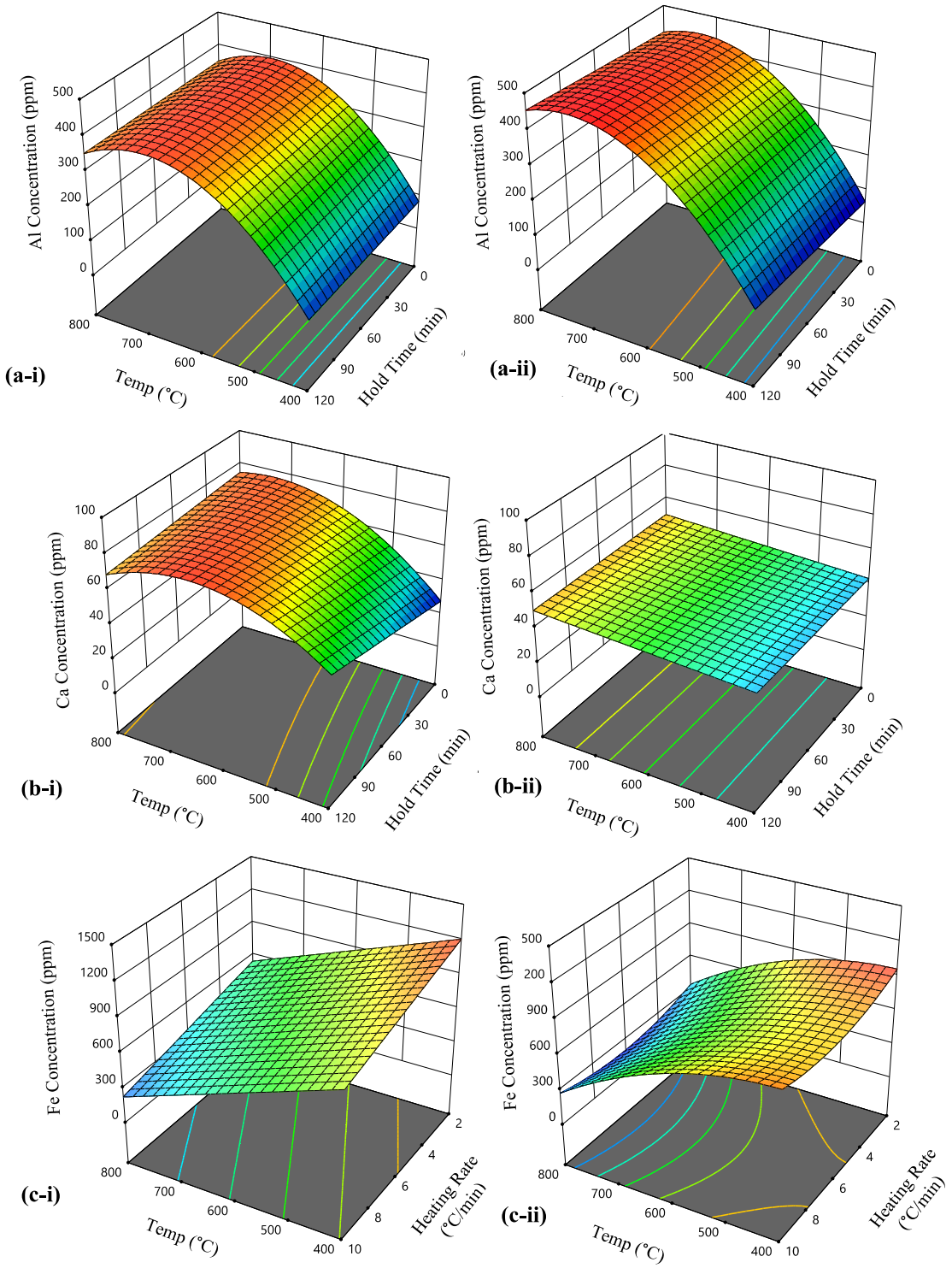



Figure 3.6.24. Response surface for the association of (a) Al, (b) Ca, and (c) Fe concentration of the calcined (i) 1.60 SG float and (ii) 2.20 SG sink fraction of West Kentucky No.13 as a function of main and interaction effects of the significant variables.

The optimization module of the Stat-Ease software was used to search for a combination of levels of inputs that can simultaneously satisfy the criteria placed on each of the response variables. The optimization algorithm uses the empirical models developed during the data analysis of the Box-Behnken design of experiments. The input factors are set to be "in range" and the combination of maximization/minimization criteria are assigned to the responses. Numerical optimization then uses the models to search the factor spaces for the best tradeoffs to achieve the desired goals. Considering the results from Table 3.6.8, the recovery of highly valued HREEs can be maximized at 400°C with a minimum concentration of Al and Ca in the pregnant leach solution. This set of input parameter values, however, allows for maximum contamination of Fe in the system and lower LREE recovery. Additional operational benefits can be realized if the iron in the calcined products can be removed by magnetic separation or by removing the pyrites in the feed using a density separator or froth flotation. The optimum conditions to maximize TREE recovery and minimize the total contaminants can be achieved at 776°C and a hold time of 84 minutes.

Table 3.6.8. Set of optimum parameter values for the desired goals of the response variables from the Box-Behnken Design and the value of other output variables for 1.60 SG float of West Kentucky No.13 (Cont.=Total Contaminants).

| | Temp (°C) | Heating Rate (ºC/min) | Hold Time (min) | TREEs (% Recovery) | LREEs (% Recovery) | HREEs (% Recovery) | Al (ppm) | Ca (ppm) | Fe (ppm) | Total Cont. (ppm) | Probability |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Max LREEs | 629 | 2 | 71 | 71 | 75 | 54 | 386 | 77 | 833 | 1296 | 0.87 |
| Max HREEs | 400 | 2 | 60 | 58 | 58 | 60 | 84 | 40 | 1210 | 1334 | 0.95 |
| Max TREEs | 616 | 2 | 10 | 71 | 75 | 54 | 379 | 72 | 854 | 1305 | 0.91 |
| Max Al | 689 | 2.5 | 110 | 69 | 73 | 52 | 400 | 78 | 710 | 1188 | 0.97 |
| Max Ca | 650 | 9 | 120 | 63 | 67 | 47 | 394 | 79 | 509 | 982 | 0.97 |
| Max Fe | 400 | 2 | 60 | 58 | 58 | 60 | 84 | 40 | 1210 | 1334 | 0.95 |
| Min Al | 400 | 2 | 60 | 58 | 58 | 60 | 84 | 40 | 1210 | 1334 | 0.91 |
| Min Ca | 400 | 6 | 0 | 54 | 54 | 57 | 84 | 28 | 1049 | 1161 | 0.97 |
| Min Fe | 800 | 10 | 60 | 53 | 57 | 42 | 354 | 72 | 230 | 656 | 0.91 |
| Max TREEs Min Cont. | 776 | 2 | 84 | 64 | 68 | 50 | 371 | 73 | 591 | 1035 | 0.57 |

Similarly, optimization studies were conducted for the 2.20 SG sink fraction of the West Kentucky No.13 seam (Table 3.6.9). The concentration of Al and Ca in the leachate was found to be maximum at higher temperatures. LREE recovery maximized in the higher temperature range. A TREE recovery of 79% could be achieved with a temperature set point of 800°C and a contaminant concentration of 983 ppm in solution. The high temperature facilitates low Fe dissolution in the leachate while maximizing the LREE recovery. While the optimum conditions for maximum REE

recovery for both the coals require high temperatures (775-800°C), the 1.60 SG float fraction was found to need a higher hold time (residence). This was required for the complete removal of the combustible content and liberation of the REEs micro dispersed mineral matter within the organic matrix.

Table 3.6.9. Set of optimum parameter values for the desired goals of the response variables from the Box-Behnken Design and the value of other output variables for 2.20 SG sink of West Kentucky No.13 (Cont.=Total Contaminants).

| | Temp (°C) | Heating Rate (°C/min) | Hold Time (min) | TREEs (% Recovery) | LREEs (% Recovery) | HREEs (% Recovery) | Al (ppm) | Ca (ppm) | Fe (ppm) | Total Cont. (ppm) | Probability |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Max LREEs | 626 | 9 | 3.5 | 77 | 85 | 38 | 409 | 48 | 663 | 1120 | 1.00 |
| Max HREEs | 400 | 10 | 60 | 32 | 31 | 38 | 83 | 44 | 904 | 1031 | 0.40 |
| Max TREEs | 624 | 3.2 | 16 | 79 | 84 | 38 | 410 | 48 | 701 | 1159 | 1.00 |
| Max Al | 789 | 2.3 | 55 | 76 | 81 | 38 | 442 | 51 | 355 | 848 | 1.00 |
| Max Ca | 626 | 8.3 | 11.4 | 78 | 85 | 38 | 410 | 48 | 642 | 1100 | 0.39 |
| Max Fe | 400 | 2 | 60 | 32 | 31 | 38 | 83 | 44 | 970 | 1097 | 0.74 |
| Min Al | 400 | 6 | 0 | 29 | 28 | 38 | 66 | 44 | 833 | 943 | 0.95 |
| Min Ca | 400 | 6 | 0 | 29 | 28 | 38 | 66 | 44 | 833 | 943 | 0.68 |
| Min Fe | 800 | 6.6 | 13 | 76 | 82 | 38 | 425 | 51 | 206 | 682 | 0.93 |
| Max TREEs Min Cont. | 800 | 2 | 3 | 79 | 83 | 38 | 423 | 51 | 343 | 817 | 0.95 |

**Fire Clay Seam:** The test conditions and respective output values for the 1.60 SG float and 2.20 SG sink fraction of Fire Clay material are provided in Table 3.6.10. The untreated density fractions (feed) of the Fire Clay had higher LREE and HREE recovery values as compared to their Baker seam counterparts. The effect of calcination on the 1.60 SG float fraction was found to be more profound than the 2.20 SG sink. The higher TREE recoveries result from the combustion of the carbon matter present in the sample which liberates or oxidizes the REEs contained in the organic matrix. On the other hand, a majority of REEs remain insoluble in the 2.20 SG sink of the Fire Clay material despite calcination treatment and strong acid leaching conditions. This difference in the leaching behaviors between the two feed sources suggests a very different mode of occurrence and association of the REEs in the mineral matrix. The LREEs in the 1.60 SG float show an improvement of about 50% points when calcined at 600-800°C with long residence times. Like the West Kentucky No.13 seam material, the HREEs were relatively inert to the thermal treatment as the maximum HREE recovery obtained from the 1.60 SG float fraction was about 45%. Heating rate and hold time were found to be insignificant for all the response variables associated with the 2.20 SG sink material of Fire Clay. The lower recovery values for the Fire Clay material suggest that a large fraction of REEs in the Fire Clay 2.20 SG sink material are difficult to decompose

under the investigated set of conditions and may require rigourous chemical treatment at elevated temperatures.

The Pearson linear correlation values for the selected response and input variables are summarized in Table 3.6.11. Like the West Kentucky No.13 materials, LREEs and Al show a strong positive correlation with temperature for both the density fractions. The HREEs and Fe in the 2.20 fraction are negatively correlated with temperature. In the absence of pyrite, the absolute concentration of Fe in the leachate is about 25% relative to its corresponding fraction in the West Kentucky No.13 seam material. Figure 3.6.25. a-i & Figure 3.6.25. .a-ii show that LREEs in both the density fractions have a strong positive association with the first and second-order terms of temperature.

Table 3.6.10. Input factors and the results of the Box- Behnken experimental design for 1.60 SG float and 2.20 SG sink of Fire Clay.

| Run | Temp. (°C) | Heating Rate (°C/min) | Hold Time (min) | TREE (%) | LREE (%) | HREE (%) | Al (ppm) | Ca (ppm) | Fe (ppm) | TREE (%) | LREE (%) | HREE (%) | Al (ppm) | Ca (ppm) | Fe (ppm) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | 1.60 SG float | | | | | | 2.20 SG sink | | | | | |
| Untreated Feed | | | | 34 | 34 | 29 | 34 | 13 | 66 | 44 | 45 | 38 | 79 | 23 | 271 |
| 1 | 600 | 2 | 0 | 70 | 74 | 41 | 779 | 28 | 188 | 56 | 60 | 32 | 429 | 29 | 338 |
| 2 | 600 | 6 | 60 | 69 | 73 | 40 | 826 | 28 | 149 | 53 | 57 | 28 | 444 | 30 | 339 |
| 3 | 600 | 6 | 60 | 70 | 74 | 41 | 898 | 29 | 158 | 49 | 53 | 27 | 411 | 28 | 313 |
| 4 | 600 | 6 | 60 | 70 | 75 | 41 | 871 | 29 | 148 | 53 | 57 | 28 | 430 | 28 | 320 |
| 5 | 400 | 6 | 120 | 63 | 66 | 42 | 149 | 24 | 156 | 36 | 38 | 29 | 85 | 25 | 289 |
| 6 | 800 | 6 | 0 | 68 | 73 | 39 | 931 | 30 | 110 | 52 | 56 | 26 | 349 | 27 | 190 |
| 7 | 400 | 2 | 60 | 63 | 66 | 41 | 99 | 21 | 120 | 43 | 44 | 38 | 92 | 26 | 284 |
| 8 | 400 | 6 | 0 | 55 | 58 | 35 | 68 | 17 | 96 | 40 | 41 | 34 | 82 | 25 | 262 |
| 9 | 600 | 6 | 60 | 73 | 77 | 43 | 903 | 29 | 165 | 61 | 65 | 37 | 329 | 27 | 283 |
| 10 | 600 | 2 | 120 | 77 | 81 | 47 | 978 | 30 | 170 | 56 | 60 | 29 | 339 | 27 | 282 |
| 11 | 800 | 2 | 60 | 78 | 81 | 49 | 977 | 30 | 94 | 54 | 58 | 26 | 407 | 28 | 162 |
| 12 | 800 | 6 | 120 | 70 | 74 | 40 | 986 | 30 | 90 | 51 | 55 | 26 | 399 | 27 | 153 |
| 13 | 400 | 10 | 60 | 64 | 67 | 43 | 121 | 22 | 131 | 40 | 42 | 32 | 83 | 26 | 302 |
| 14 | 800 | 10 | 60 | 64 | 68 | 35 | 968 | 30 | 87 | 58 | 62 | 29 | 393 | 27 | 159 |
| 15 | 600 | 10 | 0 | 71 | 75 | 41 | 879 | 29 | 157 | 52 | 56 | 30 | 344 | 26 | 335 |
| 16 | 600 | 6 | 60 | 71 | 76 | 41 | 962 | 30 | 151 | 54 | 58 | 29 | 381 | 27 | 329 |
| 17 | 600 | 10 | 120 | 69 | 74 | 38 | 626 | 27 | 132 | 56 | 60 | 30 | 368 | 27 | 298 |

Table 3.6.11. Correlation matrix of the major response variables with the input variables used in the calcination studies of Fire Clay sample.

| | 1.60 SG float | | | | | | 2.20 SG sink | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | LREE | HREE | Al | Ca | Fe | | LREE | HREE | Al | Ca | Fe |
| Temperature | 0.60 | 0.06 | 0.86 | 0.82 | -0.35 | | 0.75 | -0.62 | 0.79 | 0.50 | -0.64 |
| Heating Rate | -0.26 | -0.54 | -0.05 | -0.03 | -0.19 | | -0.03 | -0.13 | -0.05 | -0.28 | 0.04 |
| Hold Time | 0.23 | 0.3 | 0.02 | 0.15 | 0.00 | | 0.00 | -0.20 | 0.00 | -0.12 | -0.13 |

The 3-dimensional response surface of the contaminant ions in Figure 3.6.26. shows a quadratic relationship of Al and Fe with temperature. Calcium concentration in the leachate was within the range of experimental error and variability in the feed.

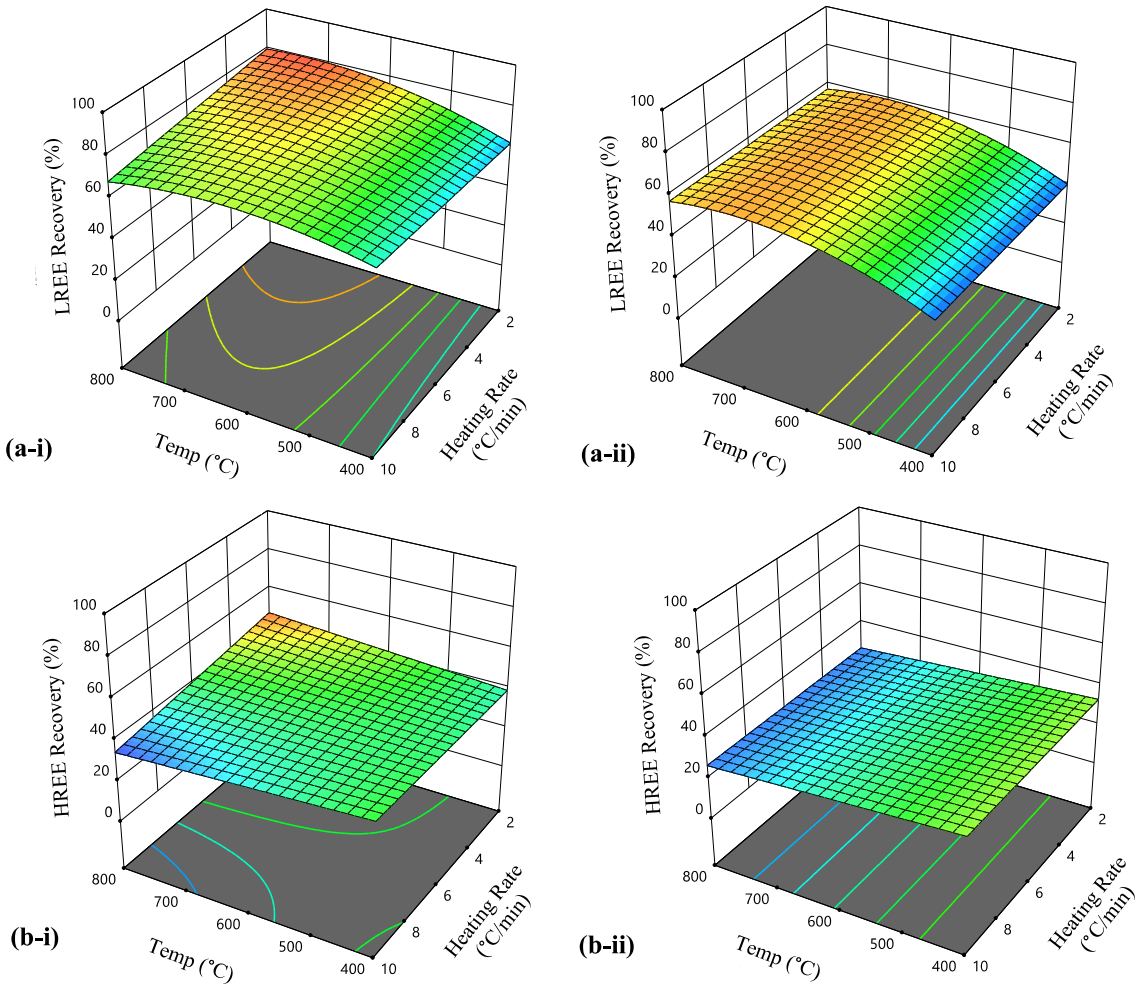



Figure 3.6.25. Response surface for the association of (a) LREE and (b) HREE recovery (%) of the calcined (i) 1.60 SG float and (ii) 2.20 SG sink fraction of Fire Clay as a function of main and interaction effects of the significant variables

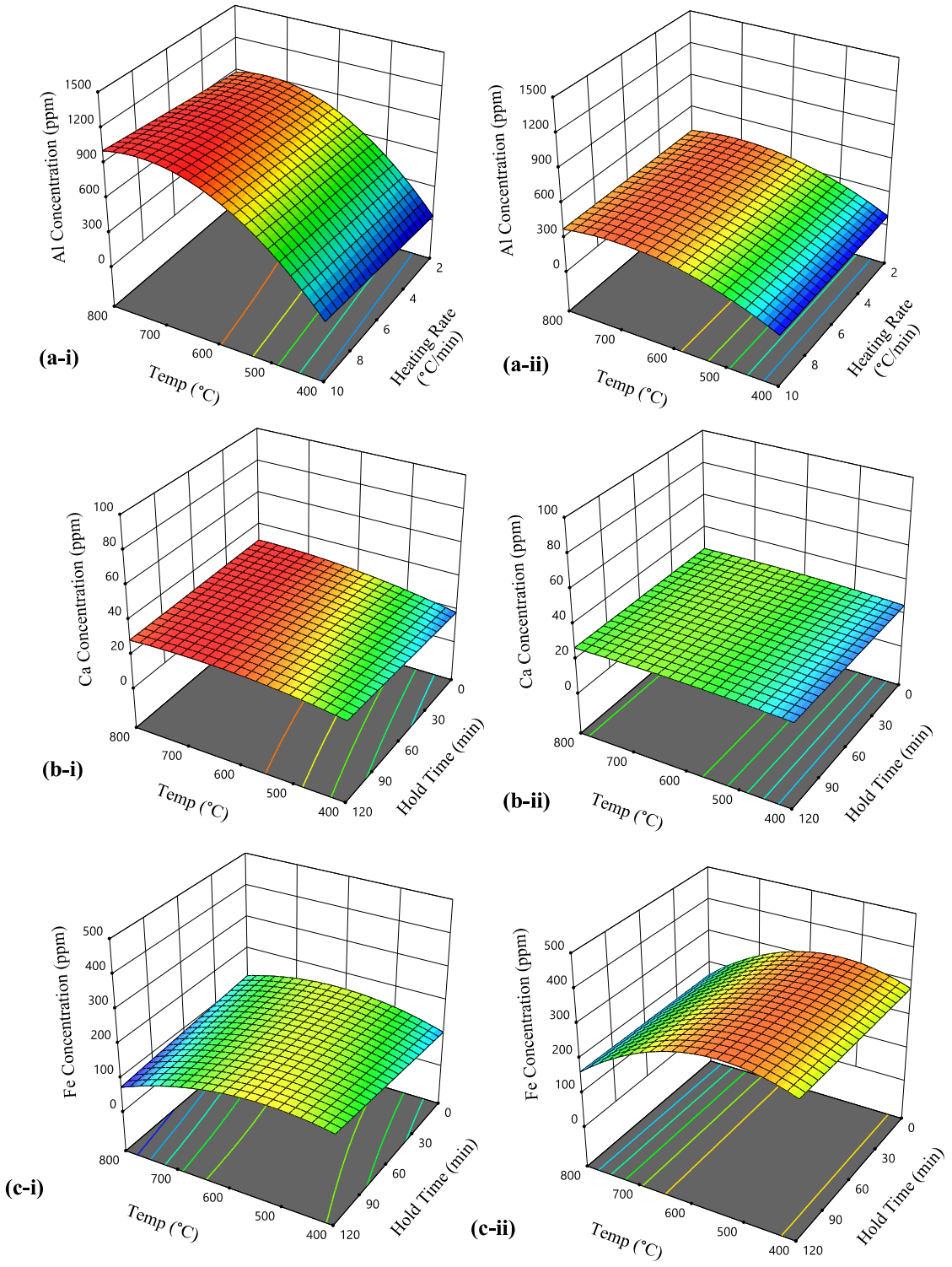



Figure 3.6.26. Response surface for the association of (a) Al, (b) Ca, and (c) Fe concentration of the calcined (i) 1.60 SG float and (ii) 2.20 SG sink fraction of Fire Clay as a function of main and interaction effects of the significant variables.

Due to the low pyrite content in the Fire Clay material, the contamination due to Fe was very small as compared to Baker seam material. Additionally, as the clays were not fully dehydrated, a small amount of Al was leached into the solution. Also, high TREE recoveries can be achieved at relatively lower temperatures keeping the contaminant ions in check. Although to achieve maximum LREE and HREE recovery values temperatures >770°C are required for the 1.60 SG float (Table 3.6.12). Similarly, 700°C allows for high TREE recoveries for the 2.20 SG sink fraction. The association of Fe in the Fire Clay sample is dehydoxylation of illite and therefore maximum Fe in solution occurs at temperatures that decompose clay minerals by dehydration thereby providing access to the interlayers (Figure 3.4.13). Iron in aluminosilicate clays has also been found to be present as impurities of iron oxide(Malden & Meads, 1967).

Table 3.6.12. Set of optimum parameter values for the desired goals of the response variables from the Box-Behnken Design and the value of other output variables for 1.60 SG float of Fire Clay (Cont.=Total Contaminants).

| | Temp (°C) | Heating Rate (°C/min) | Hold Time (min) | TREEs (% Recovery) | LREEs (% Recovery) | HREEs (% Recovery) | Al (ppm) | Ca (ppm) | Fe (ppm) | Total Cont. (ppm) | Probability |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Max LREEs | 773 | 2 | 19 | 76 | 80 | 44 | 934 | 31 | 121 | 1086 | 1.00 |
| Max HREEs | 800 | 2 | 120 | 76 | 80 | 52 | 1119 | 29 | 75 | 1223 | 0.59 |
| Max TREEs | 791 | 2.5 | 1 | 75 | 79 | 43 | 894 | 31 | 119 | 1044 | 1.00 |
| Max Al | 727 | 3.88 | 44 | 74 | 78 | 43 | 999 | 30 | 132 | 1161 | 1.00 |
| Max Ca | 737 | 9.3 | 37 | 67 | 72 | 37 | 1009 | 30 | 130 | 1169 | 1.00 |
| Max Fe | 524 | 6.4 | 120 | 69 | 73 | 42 | 646 | 28 | 164 | 838 | 0.76 |
| Min Al | 400 | 10 | 81 | 63 | 66 | 41 | 45 | 22 | 133 | 200 | 1.00 |
| Min Ca | 400 | 6 | 0 | 61 | 64 | 39 | 99 | 19 | 106 | 224 | 0.89 |
| Min Fe | 798 | 8.5 | 90 | 66 | 71 | 37 | 917 | 30 | 86 | 1033 | 1.00 |
| Max TREEs Min Cont. | 407 | 10 | 120 | 63 | 66 | 41 | 15 | 24 | 148 | 187 | 0.76 |

Table 3.6.13. Set of optimum parameter values for the desired goals of the response variables from the Box-Behnken Design and the value of other output variables for 2.20 SG sink of Fire Clay (Cont.=Total Contaminants).

| | Temp (°C) | Heating Rate (°C/min) | Hold Time (min) | TREEs (% Recovery) | LREEs (% Recovery) | HREEs (% Recovery) | Al (ppm) | Ca (ppm) | Fe (ppm) | Total Cont. (ppm) | Probability |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Max LREEs | 694 | 6.9 | 82 | 56 | 60 | 28 | 424 | 28 | 267 | 719 | 0.57 |
| Max HREEs | No solution | | | | | | | | | | |
| Max TREEs | 689 | 5.6 | 30 | 56 | 60 | 29 | 423 | 28 | 271 | 722 | 0.55 |
| Max Al | 700 | 9.4 | 57 | 56 | 60 | 28 | 424 | 28 | 263 | 715 | 0.94 |
| Max Ca | 672 | 4.2 | 47 | 56 | 60 | 29 | 421 | 28 | 282 | 731 | 0.61 |
| Max Fe | 534 | 9.8 | 111 | 51 | 55 | 31 | 321 | 27 | 325 | 673 | 0.92 |
| Min Al | 400 | 10 | 60 | 40 | 41 | 33 | 85 | 26 | 284 | 395 | 0.99 |
| Min Ca | 400 | 6 | 120 | 40 | 41 | 33 | 85 | 26 | 284 | 395 | 0.87 |
| Min Fe | 800 | 6 | 120 | 54 | 58 | 27 | 387 | 27 | 166 | 580 | 0.93 |
| Max TREEs Min Cont. | 800 | 6 | 0 | 54 | 58 | 27 | 387 | 27 | 166 | 580 | 0.51 |

In addition to the recovery of REEs and the concentration of contaminant elements in the calcination products of the two feed sources, an artificial variable called ‘Relative Concentration’ (RC), which is the ratio of the total concentrations of contaminant ions to REEs was created. RC is defined by the following expression:

$$\text{Relative Concentration (RC)} = \text{Total Contaminants}/\ \text{Total REE} \qquad \text{(Eq. 5-1)}$$

Relative concentration is a measure of selectivity for the dissolution of REEs and rejection of the bulk of the contaminants. A lower RC value would result in higher REE ions in the solutions and relatively lower contaminant ions. Empirical models were developed for RC associated with the 1.60 SG and 2.20 SG sink fractions of the two feed sources. The actual and predicted values for the artificial variable are provided in Figure 3.6.27. The linear distribution of the actual and predicted values along with high $R^2$ and adjusted $R^2$ values suggest that the models were an excellent fit for the experimental data. The model was minimized to obtain the highest selectivity for each of the two density fractions for both sources and the results are presented in

Table 3.6.14. The values for maximum selectivity correlate well with the findings from the optimum set of parameter values for WK-13 (Table 3.6.8 and Table 3.6.9) and Fire Clay (Table 3.6.12 and Figure 3.6.13).

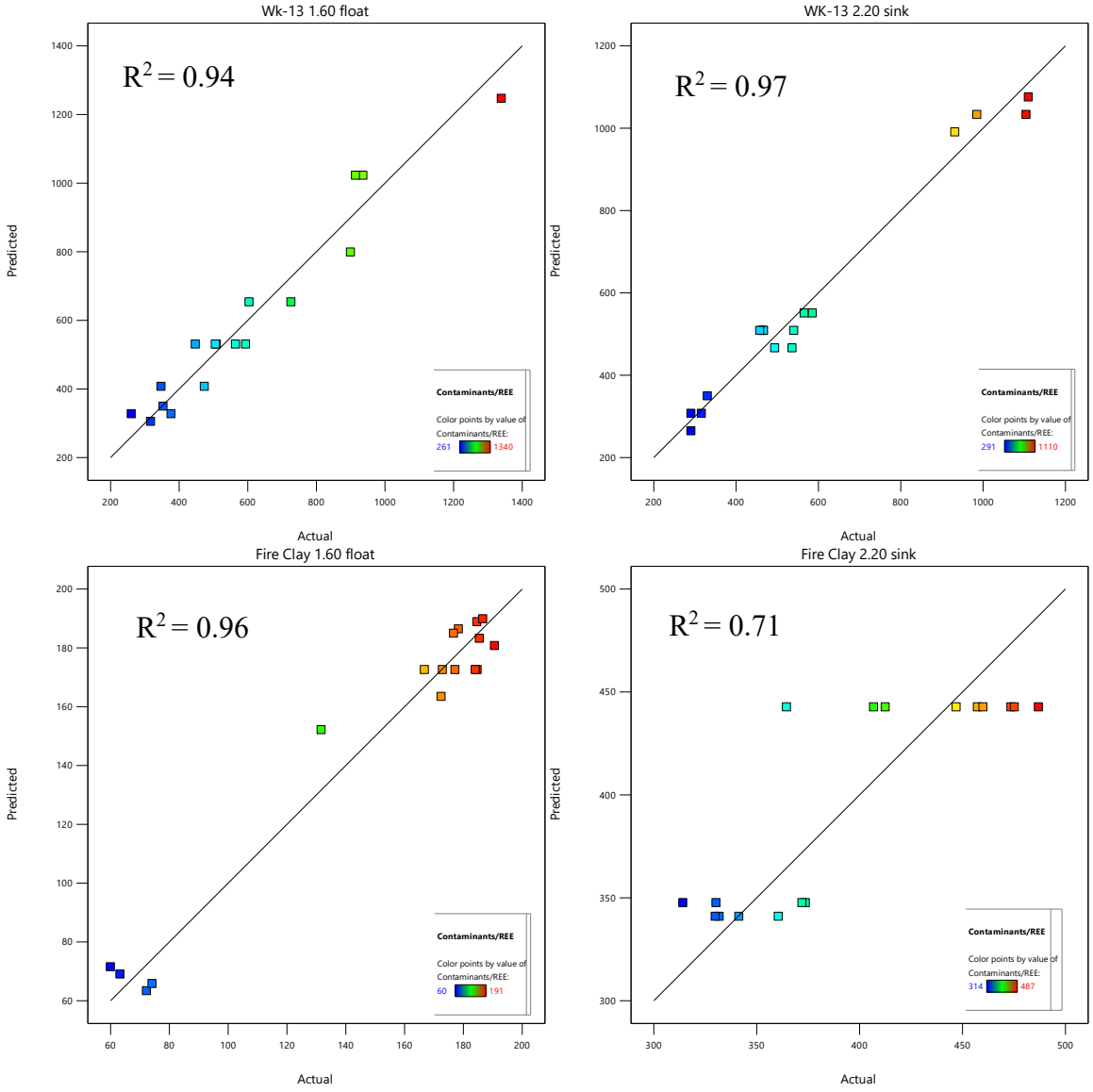



Figure 3.6.27. Actual vs predicted plots for the 1.60 SG float and 2.20 SG sink fractions of West Kentucky No.13 and Fire Clay, respectively.

Table 3.6.14. The minimum values for artificial variable Relative Concentration (RC) for achieving maximum selectivity for REE recovery from the two coal sources.

| | Temp (°C) | Heating Rate (°C/min) | Hold Time (min) | TREEs (% Recovery) | LREEs (% Recovery) | HREEs (% Recovery) | Al (ppm) | Ca (ppm) | Fe (ppm) | Relative Concentration Probability | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| West Kentucky No.13 | | | | | | | | | | | |
| 1.60 SG float | 771 | 5 | 120 | 61 | 65 | 47 | 375 | 73 | 480 | 302 | 0.96 |
| 2.20 SG sink | 799 | 10 | 86 | 74 | 78 | 38 | 445 | 51 | 261 | 289 | 1.00 |
| Fire Clay | | | | | | | | | | | |
| 1.60 SG float | 404 | 8.5 | 88 | 62 | 65 | 41 | 63 | 22 | 135 | 60 | 1.00 |
| 2.20 SG sink | 800 | 2 | 60 | 54 | 58 | 27 | 387 | 27 | 166 | 341 | 0.84 |

### *3.6.7 Blank Roasting*

The leaching recoveries of total, light, and heavy REEs along with major contaminants are shown in Figure 3.6.28 when treating the 2.2 SG float fraction of the Fire Clay coarse refuse. The light REEs (LREE) refer to Sc, La, Ce, Pr, Nd, Sm, and Gd, and the heavy REEs (HREE) refer to Y, Tb, Dy, Ho, Er, Tm, Yb, and Lu. The results indicate a positive impact of calcination on the recovery of rare earth elements. The overall recovery increased from 18.6% to 36.3% when the temperature was increased from 100-600°C. This was probably due to the oxidation of REEs into a more soluble rare earth oxide (REO) at the current leaching condition (Gupta, 2021). The recovery of individual elements shown in Figure 3.6.29 indicates that LREEs respond better to thermal treatment than HREEs. The recovery of several heavy rare earth elements, namely Y, Dy, Er, Yb, slightly increased in the 100-600°C temperature range, but Gd and Ho were unaffected by thermal treatment. This indicates that HREEs are likely associated with minerals that are either very difficult to leach and/or not liberated by simple thermal treatment. Therefore, more rigorous conditions are required to extract the HREEs. Interestingly, the increase and decrease of REE recovery correlate well with the Al recoveries over a range of thermal treatment temperatures, especially for the HREEs, which indicated a possible association between HREEs and clay minerals. A significant rise in the scandium recovery was also noted with an increase in the Al recovery. Zhang et. al. has reported the presence of scandium in the Al-O octahedrons of kaolinite (P. Zhang et al., 2017). Additionally, Arbuzov et al. found that the amounts of scandium associated with the minerals resistant to an acid attack were comparable to the minerals decomposed in the acids (Arbuzov et al., 2015). Therefore, it is probable that a portion of Sc along with HREEs are entrained within the clay structure, which are released through the dehydroxylation of clays such as kaolinite.

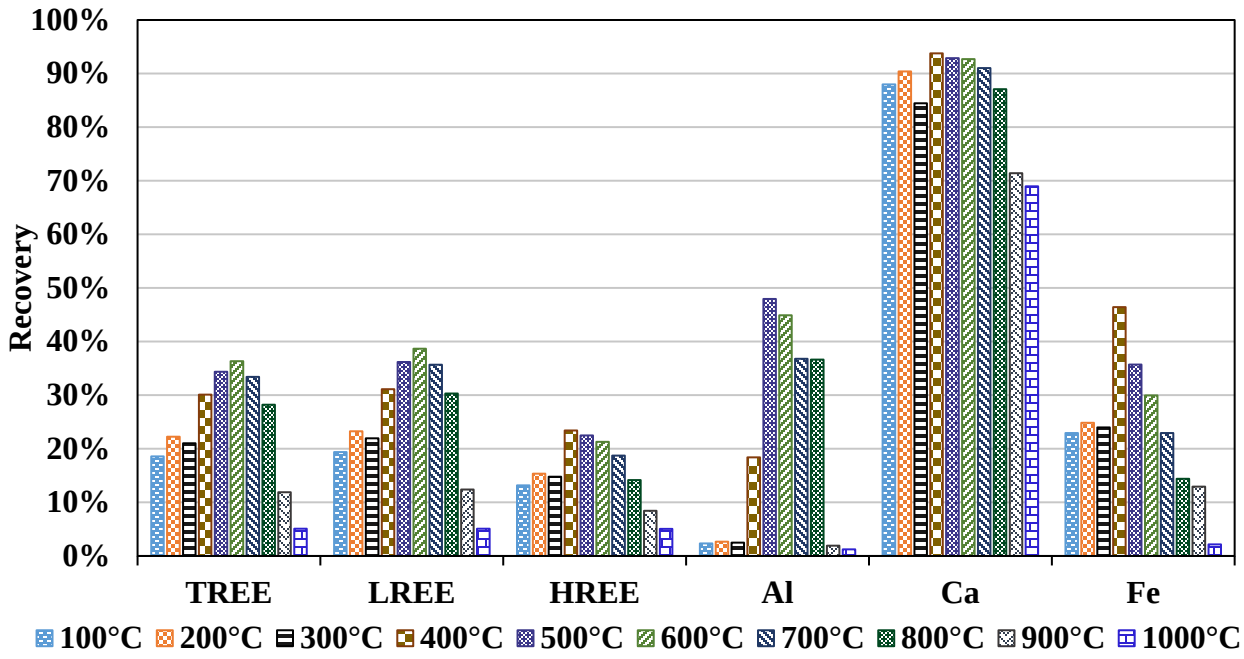



Figure 3.6.28. Leaching recovery of REEs and major contaminants obtained from the thermal treatment of Fire Clay S.G. 2.2 float material from 100-1000°C followed by leaching with 50g/L $H_2SO_4$ at S/L of 1/20 and 75°C for 2hr.

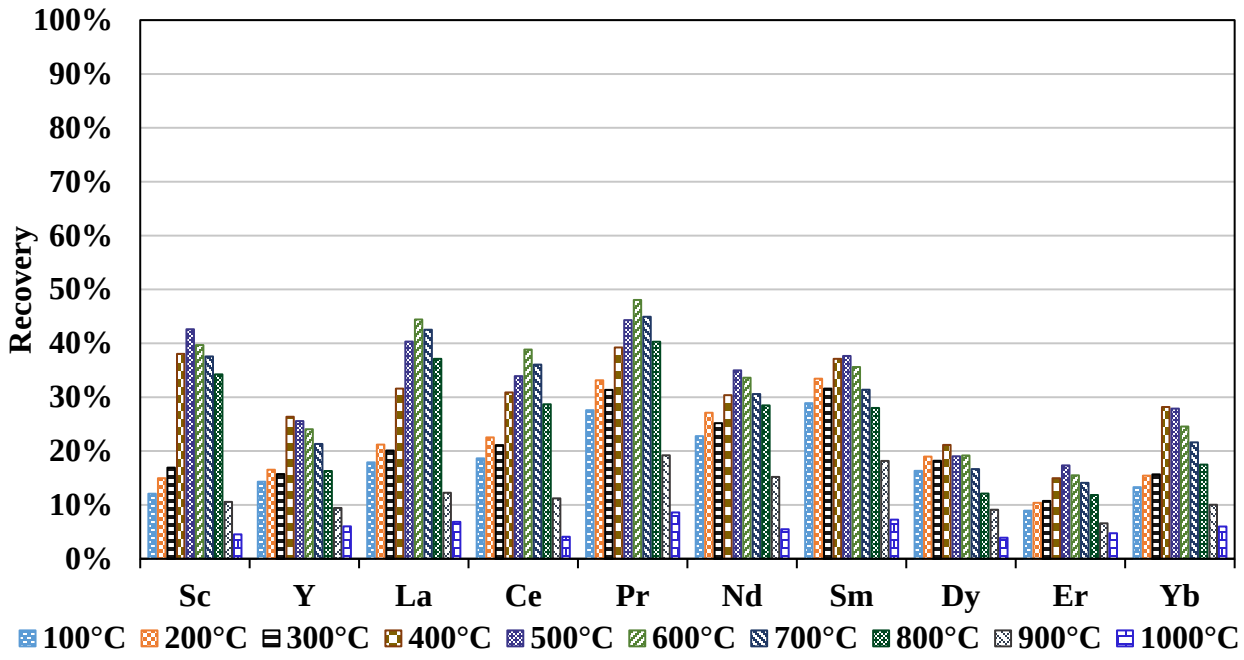



Figure 3.6.29. Leaching recovery of individual REEs from the thermal treatment of Fire Clay S.G. 2.2F material from 100-1000°C followed by leaching with 50g/L $H_2SO_4$ at S/L of 1/20 and 75°C for 2hr (Gd and Ho were excluded because of insignificant recovery)

The thermal treatment of coal up to 300°C did not impact either REEs or Al recovery, likely owing to the high stability of kaolinite towards acids. The recovery of REEs gradually increased from 21% to 36% while increasing the roasting temperature from 400°C to 600°C. Kaolinite typically converts to its meta kaolinite form at a temperature of 500-600°C, which has been found to dissolve relatively easily compared to its parent kaolinite (Gao et al., 2016). The results indicate that approximately 21% of the REEs are likely not associated with clay minerals, whereas about 16% of REEs are released due to clay decomposition after calcination.

A sharp decrease in the Al recovery was observed above 900°C which was likely due to the sintering of clays through the conversion of meta-kaolinite to mullite. Edomwonyi et al. proposed the following series of reactions for conversion of kaolinite to meta-kaolinite and eventually mullite(EDOMWONYI-OTU et al., 2013):

$$\boldsymbol{2Al_2Si_2O_5(OH)_4 \rightarrow 2Al_2Si_2O_7 + 4H_2O}$$

$$2Al_2Si2O_7 \rightarrow Si_3Al_4O_{12} + SiO_2$$

$$Si_3Al_4O_{12} \rightarrow 2Si_2Al_6O_{13} + 5SiO_2$$

Mullite is highly stable in strong acid solutions ((Gupta, 2021),(Liu et al., 2015)). Calcium recovery was high and remained unaffected by an elevation in calcination temperature up to 800°C. The primary source of Ca ions in the pregnant leachate solution was calcite which was naturally present in the coal and readily soluble in acid ((Dash et al., 2000; Sanders & Gallagher, 2002)). The calcite has been reported to decompose between 600-800°C into calcium oxide and carbon dioxide through following reaction(Rodriguez-Navarro et al., 2009):

$$\boldsymbol{CaCO_{3(s)} \rightarrow CaO_{(s)} + CO_{2(g)}}$$

Various researchers have reported formation of new meta-stable phases through the interaction between the calcium oxide resulting from the decomposition of the calcite and clays. The new phases were reported to be wollastonite, anorthite and gehlenite ((Allegretta et al., 2016; Ghosh, n.d.; Rodriguez-Navarro et al., 2009; Traoré et al., 2003)). The formation of these new aluminosilicate phases might be the reason for a drastic decrease in the Ca recovery observed above 800°C. Traore reported following reactions for the formation of anorthite and gehlenite (Traoré et al., 2003):

$$\boldsymbol{Al_2Si_2O_7 + (2+n)CaO \rightarrow Ca_2Al_2SiO_7 + n(CaOSiO_2)}$$

$$Ca_2Al_2SiO_7 \rightarrow 3SiO_2 + Al_2O_3 \rightarrow 2CaAl_2Si_2O_8$$

The Fe contamination in the pregnant leach solution (PLS) increased in the 400-500°C range due to the decomposition of pyrite into an intermediate iron oxide product that is highly soluble. At higher temperatures, the intermediate iron oxide transforms to α-hematite, which has lower solubility under the current leaching condition, thereby resulting in a decrease in recovery ((Gupta, 2021),(Mitchell, 2002)). The Spearmen's correlation coefficient values for the REEs and major contaminants with temperature are shown in Table 3.6.15. The data has been divided into two parts based on the impact of calcination temperature on the recovery of REEs. The correlation

coefficient with temperature in the 100-600°C range shows a strong positive correlation. A strong positive correlation coefficient of LREEs with temperature compared to HREEs reiterates the conclusion drawn earlier that thermal treatment primarily impacts the LREEs. A strong negative correlation observed for both REE groups and the contaminants shows the adverse impact of clay sintering in this temperature range.

Table 3.6.15 Correlation coefficients for the blank roasted samples with temperature

| | LREE | HREE | Al | Ca | Fe |
|---|---|---|---|---|---|
| Temperature (100-600°C) | 0.953 | 0.790 | 0.901 | 0.628 | 0.742 |
| Temperature (700-1000°C) | -0.976 | -0.975 | -0.955 | -0.975 | -0.982 |

### *3.6.8 Roasting with additives:*

The thermal treatment of REE-bearing minerals is popular for the efficient extraction of REEs. However, some REE-bearing minerals such as monazite and xenotime are not significantly impacted by the simple thermal treatment. Therefore, different additives are used in the thermal treatment step to facilitate the decomposition of these REE-bearing minerals. Similarly, the studies performed on the REE extraction from coal byproducts, especially Fire Clay coal seam, reported low HREE recoveries by simple thermal treatment. Therefore, the acid baking technique was applied to the coal samples to analyze its impact on REE recoveries, specifically HREEs.

**Direct Acid Baking:** The direct acid baking involves thermally treating the Fire Clay 2.2 SG float fraction in the coarse refuse sample with an acid under different temperatures. The acid-baked samples are then leached using 100ml DI water at 5% S/L, 75°C, and for 2 hrs. The coal-to-acid ratio of 1:1 was used in these tests. The results of the study are shown in Figure 3.6.30. The tests showed a significant increase in the recovery of both LREE and HREE recovery at only 100°C. The TREE recovery was 60.18%, which is significantly higher than those reported previously. The HREE recovery raised to 52.42% at only 100°C using significantly lower acid concentrations than reported previously. The overall REE recovery further increased at 150°C to 64.45%. The increase in the LREE recovery from 100-150°C was not significant. However, the HREE recoveries increased approximately 12 absolute points by the increase in temperature. This shows that acid baking significantly impacts the HREE recovery. Further increase in the temperature from 150-300°C resulted in a significant decrease in REE recovery, possibly due to the evaporation of sulfuric acid with an increase in the roasting temperature. The recovery of individual REEs is shown in Figure 3.6.31. It shows that the increase in the temperature adversely impacts the recovery of all the REEs. Interestingly, the Sc recovery is not significantly affected by the rise in temperature. The Al recovery at initial temperatures is considerably lower, which shows that the REEs extracted at 100°C are probably not associated with clays. An increase in the roasting temperature probably impacted the clay decomposition, thereby increasing the overall REE recovery. This is also evident in the XRD data shown in Figure 3.6.34, where a new potassium aluminum phosphate peak was detected. As this species is highly soluble in water, therefore the Al recovery increased with a rise in roasting temperature. The Ca recovery seemed unaffected, whereas the Fe recovery improved with the increase in the temperature.

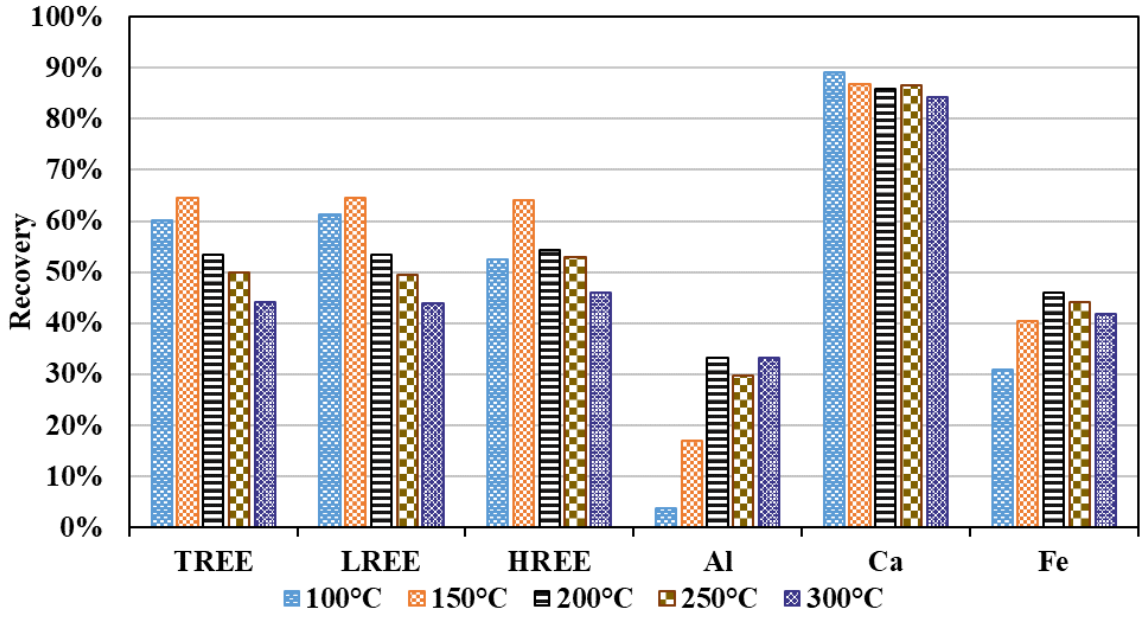



Figure 3.6.30 Recovery of REEs and major contaminants obtained by direct acid baking of raw coal feed from 100-300°C at 1:1 coal to acid ratio followed by leaching with 100ml DI water at 5% S/L, 75°C for 2hrs; Fire Clay 2.2 SG float material.

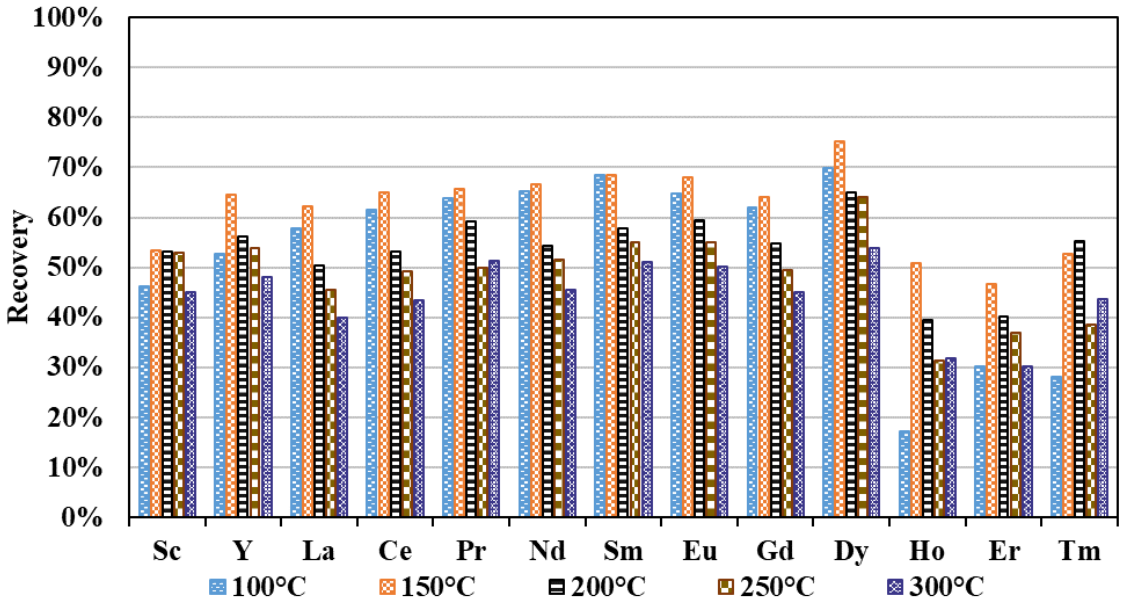



Figure 3.6.31 Recovery of individual REEs obtained by direct acid baking of raw coal feed from 100-300°C at 1:1 coal to acid ratio followed by leaching with 100ml DI water at 5% S/L, 75°C for 2hrs; Fire Clay 2.2 SG float material.

**2nd Stage Acid Baking:** The 2nd stage acid baking refers to the acid baking of pre-calcinated coal samples. In this investigation, the coal was pre-roasted at 600°C and then treated with acid at different temperatures. The acid treatment conditions were kept the same as direct acid baking.

The results shown in Figure 3.6.32 and Figure 3.6.33 indicate a significant increase in the recovery of rare earth elements over direct acid baking. The TREE recovery of approximately 80% was obtained at only 100°C. This is an increase of approximately 30 absolute points over the direct acid baking under the same test concentrations. Further increase in the temperature does not significantly impact the TREE recovery. This is because the LREEs, which represent a significant portion of the REEs in the coal, are not significantly impacted by an increase in the temperature. The LREE seems to reach the maximum recovery of 85.46% at 150°C and decrease slightly with an increase in the temperature beyond this temperature. The HREE recovery is impacted significantly with an increase in the temperature and rises from approximately 45% to 78% with a rise in temperature from 100-250°C. Interestingly, the rise in HREE recovery correlates well with a corresponding increase in the Al recovery. This gives an indication that some of the REEs liberated through acid baking is probably associated with the clays. The XRD data shown in Figure 3.6.34 shows a new aluminum sulfate peak which is highly soluble in water. Therefore, it is possible that the increase in the Al recovery might be due to the decomposition of Al from clays into a more soluble form. The recovery of Ca is again not impacted by the increase in temperature, possibly due to the high solubility of calcite in acidic conditions. The overall increase in the REE and contaminant recovery observed in the 2nd stage acid baking might be because of the liberation of REEs from REE-bearing minerals in the pre-calcination step.

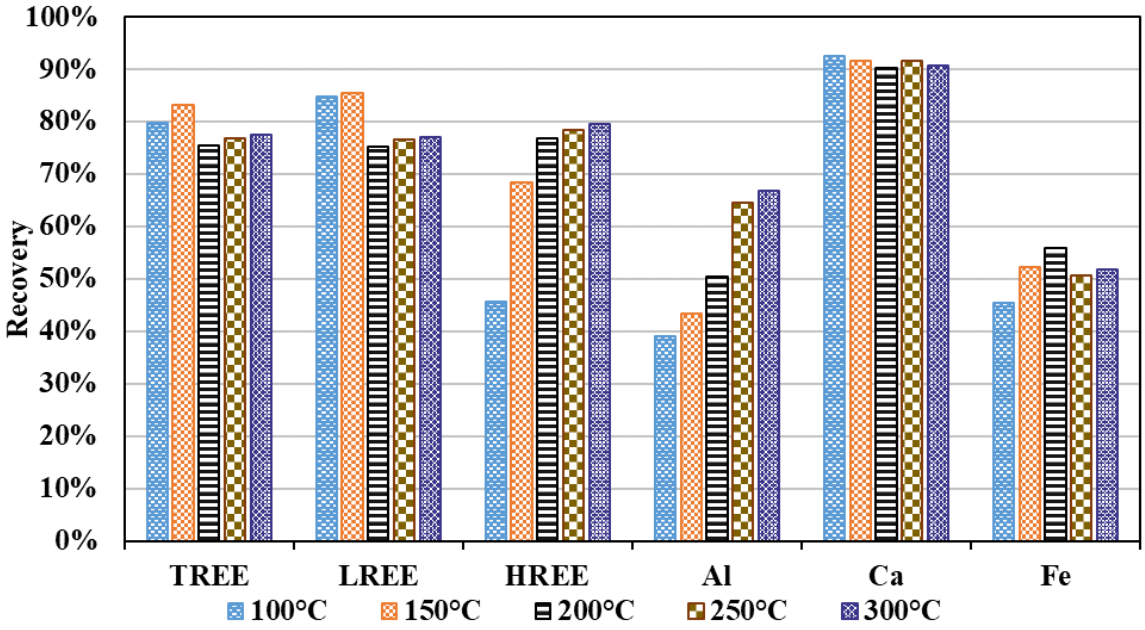



Figure 3.6.32 Impact of 2nd stage acid baking temperature on leaching recovery of REE and major contaminants (2nd stage acid baking: Calcining at 600°C followed by acid baking at 250°C with 1:1 solid to acid ratio and leaching with 100ml DI water at 5% S/L, 75°C for 2hrs); Fire Clay 2.2 SG float material.

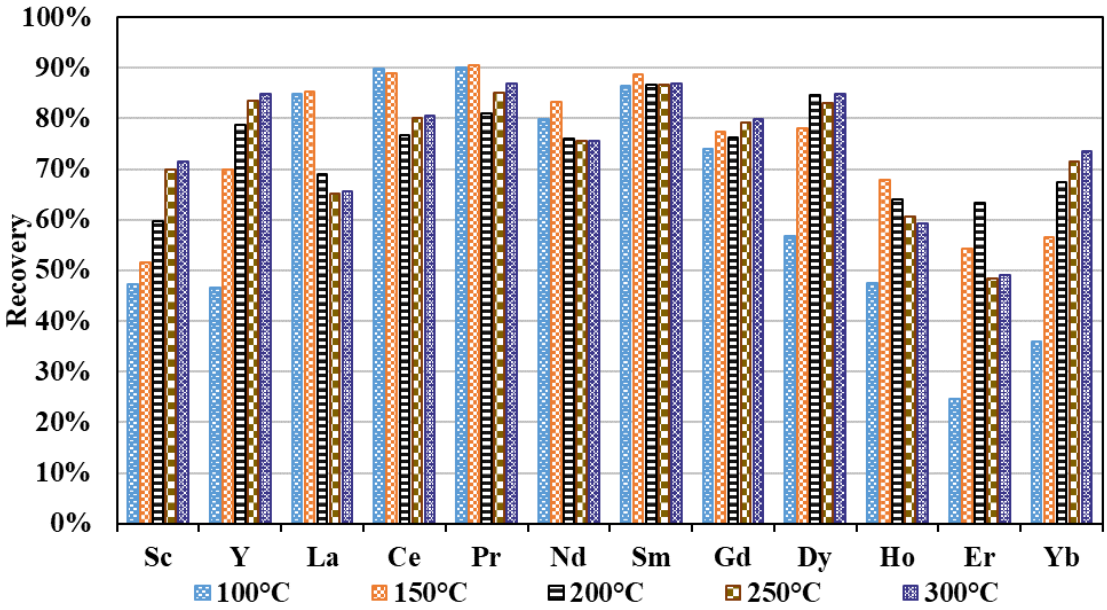



Figure 3.6.33 Impact of 2nd stage acid baking temperature on leaching recovery of individual REEs (2nd stage acid baking: Calcining at 600°C followed by acid baking at 250°C with 1:1 solid to acid ratio and leaching with 100ml DI water at 5% S/L, 75°C for 2hrs); Fire Clay 2.2 SG float material.

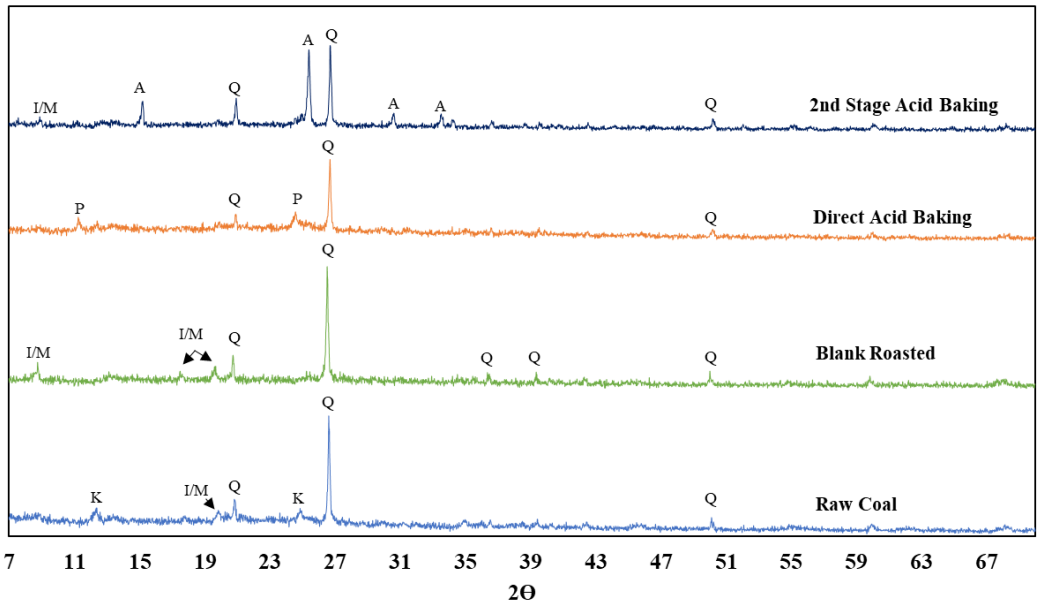



Figure 3.6.34 XRD patterns of Raw Coal, Blank Roasted: Raw coal calcined at 600°C, Direct Acid Baking: Raw coal acid baked at 250°C with 1:1 solid to acid ratio for 2hrs, 2nd Stage Acid Baking: Raw coal calcined at 600°C followed by acid baking at 250°C with 1:1 solid to acid ratio for 2hrs (Q=Quartz, A=Aluminum Sulfate I/M=Illite/Muscovite, K=Kaolinite, P=Potassium Aluminum Sulfate); Fire Clay 2.2 SG float material.

**Application on West Kentucky #13 coal:** The novel direct and 2nd stage acid techniques were applied on the West Kentucky no. 13 material to evaluate their ability to enhance the HREE recovery on a different coal source. The results shown in Figure 3.6.35 indicate that the REE recovery from raw WK#13 coal samples is also significantly lower similar to the recovery obtained from raw FC-CR samples. However, simple thermal treatment of WK#13 material significantly increases the LREE recovery. Even though an increase in the TREE recovery is observed with just blank roasting, the HREE recovery seemed unaffected. This shows that the HREEs in the WK#13 coal samples are also associated with the clays which are more resistant to acid attack. Therefore, acid treatment is essential to decompose the clays and consequently increase the HREE recovery. The direct acid baking results revealed a significant increase in the HREE recovery at only 250°C, however, the LREE recovery was considerably lower than blank roasting. This might be due to high pH of the solution which has been shown to impact the solubility of REE-bearing minerals. The 2nd stage acid baking demonstrated a significant increase in both HREE and LREE recovery. This is possibly also due to the liberation of REEs from REE-bearing minerals as a result of thermal treatment. The overall improvement in HREE recovery from both direct and 2nd stage acid baking shows that this novel technique can be applied to different coal samples to enhance the HREE recovery.

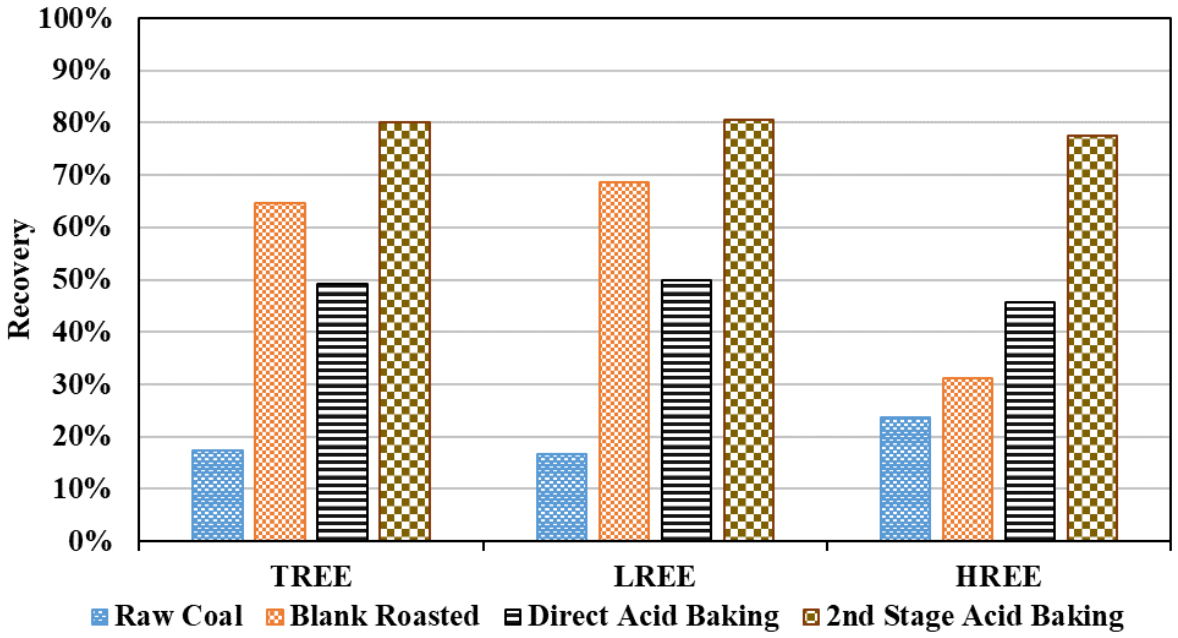



Figure 3.6.35 REE recovery from WK#13 samples treated under different conditions (Raw Sample: Direct leaching with 100ml of 50g/L H2SO4 at 5% S/L, 75°C for 2hrs, Blank Roasted: Calcining at 600°C and leaching with 100ml 50g/L H2SO4 at 5% S/L, 75°C for 2hrs, Direct Acid Baking: Acid baking at 250°C with 1:1 solid to acid ratio and leaching with 100ml DI water at 5% S/L, 75°C for 2hrs, 2nd Stage Acid Baking: Calcining at 600°C followed by acid baking at 250°C with 1:1 solid to acid ratio and leaching with 100ml DI water at 5% S/L, 75°C for 2hrs).

**Clay Mineral Testing:** The results discussed in the previous report showed that the REE recovery rises with a subsequent increase in the Al recovery, indicating that a strong association between Al and REEs might exist, especially HREEs. However, as observed in direct acid baking, the increase

in the REE recovery might not be completely associated with a corresponding rise in the recovery of Al. Therefore, it is essential to understand the impact of thermal treatment and acid baking on the clay samples to understand the association between clays and REEs. As such, tests were performed on two major clays present in coal, i.e., kaolinite and illite.

The REE recoveries of REEs from both kaolinite and illite raw samples were insignificant, as shown in Figure 3.6.36 and Figure 3.6.37, indicating that naturally, the REEs in both clay minerals are not in a soluble form. When the clay samples were calcined at 600°C, the TREE recovery for kaolinite increased from 2.81 to 84.93%, and 12.63 to 53.96% for illite samples. The increase in the recovery might be due to the de-hydroxylation of clays. The XRD data for 600°C calcined kaolinite shown in Figure 3.6.38 does not show a kaolinite peak which is indirect evidence of its de-hydroxylation into metakaolinite. Comparatively, the XRD data for calcined illite samples shown in Figure 3.6.39 still shows that the illite mineral crystal structure is not destroyed after 600°C roasting, which aligns with the findings in other literature (He et al., 1995a). Interestingly, the thermal treatment only enhanced the HREE recovery from kaolinite, the HREEs in illite samples were unaffected by the calcination. This indicates that illite is probably more resistant to acid attack compared to kaolinite.

The thermal behavior of both kaolinite and illite clays is shown in Figure 3.6.40 and Figure 3.6.41, respectively. There are two characteristic exothermic peaks observed at 545°C and 992°C from the calcination of the kaolinite sample. The first endothermic peak might be due to the de-hydroxylation of kaolinite to its metakaolin form, which has been reported to occur between 450-650°C((Belver et al., 2002; Gupta, 2021; Ilić et al., 2010; Meinhold et al., 1992)) and can be represented by the following reaction(Murat & Driouche, 1988);

$$Al_2O_3.2SiO_2.2H_2O \rightarrow Al_2O_3.2SiO_2 + 2H_2O$$

The second exothermic peak noticed at 992°C corresponds to the conversion of metakaolinite to mullite ((Meinhold et al., 1992),(Belver et al., 2002)). In the case of illite, TGA-DSC results revealed two reaction peaks at 580°C and 815°C. Earnest, in their investigation on illite clay, found that the first peak noticed around 580°C corresponds to the de-hydroxylation(Earnest, 1991). According to Gupta, the second peak observed at 820°C was probably due to the destruction of the illite structure(Gupta, 2021). He et al. also indicated in their work that illite is more stable towards thermal treatment compared to kaolinite(He et al., 1995b). The XRD analysis performed on the 600°C calcined coal and illite samples show the illite/muscovite peaks, which implies that de-hydroxylation did not destroy the crystal structure of illite, reaffirming the findings of previous researchers((Gupta, 2021),(He et al., 1995b)). Interestingly, the de-hydroxylation temperature ranges of kaolinite and illite samples correspond to an increase in the recovery of REEs and Al in the same temperature range from fireclay coal samples. Since both of these clays have been detected in Fire Clay coal by the XRD analysis, this points towards a potential association between REEs and Al.

The comparison of direct and 2$^{nd}$ stage acid baking of both clays with raw and blank roasting clay results indicate that acid baking treatment primarily impacts the HREE recovery. As mentioned previously, this increase might be due to the acid activation of clays, which increases the surface area and pore sizes. In their investigation, Scott et al. have reported the presence of REE phosphate minerals in the interparticle pore space of the underclays(Scott et al., 2017), and REEs from these phosphate-based minerals are typically extracted through sulfuric acid baking((Demol et al.,

2018a; Kumari et al., 2015)). Therefore, it is possible that HREEs in kaolinite and illite samples exist as a phosphate-based mineral such as xenotime, which requires rigorous acid treatment like acid baking for their extraction. It is possible that part of the HREEs bearing mineral in kaolinite sample is exposed during pre-calcination then attacked by sulfuric acid in 2nd stage acid baking.

The impact of 2nd stage acid baking on HREE seems more profound on kaolinite samples compared to illite. This might be due to the transformation of kaolinite into metakaolinite, which is more susceptible to acid attack, resulting in an increase observed in both HREE and Al recoveries. The Al recovery in the 2nd stage acid baking of illite samples decreases significantly compared to direct acid baking. The XRD analyses for both kaolinite and illite samples showed the formation of aluminum sulfate compounds after acid baking, which indicates that the majority of the clay minerals converted to aluminum sulfate. Most of the aluminum sulfate is dissolved in solution, which contributed to the elevated Al recovery, and a portion of it is remained in solids probably due to its saturation in liquid under current leaching conditions. Comparing the XRD data for direct and 2nd stage acid baking of illite samples, it is evident that the direct acid baking of illite has aluminum sulfate and potassium aluminum sulfate peaks, whereas 2nd stage acid baking only shows aluminum sulfate peak. Since both potassium aluminum sulfate and aluminum sulfate are highly soluble at elevated temperatures, this might be the reason for relatively higher Al recovery observed in the direct acid baking of illite compared to 2nd stage acid baking. Based on this discussion, it can be concluded that an increase in REE recovery, notably HREE, in coal samples is, in fact, partially due to the decomposition of clays observed in both types of acid baking.

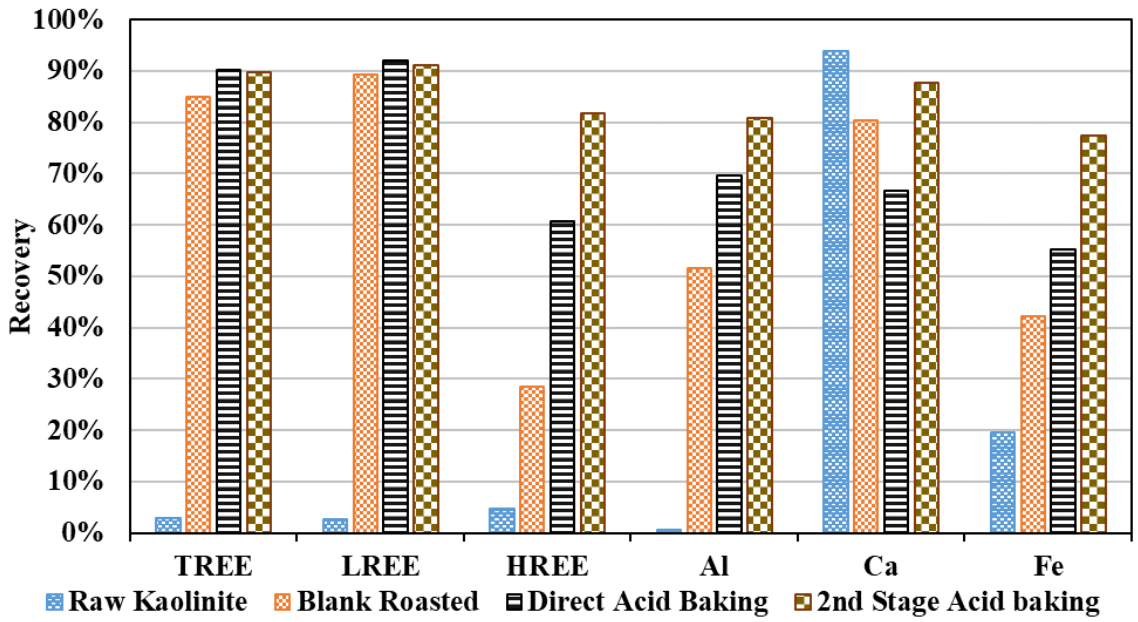



Figure 3.6.36 REE and contaminants recovery from Kaolinite samples treated under different conditions (Raw Sample: Direct leaching with 100ml of 50g/L $H_2SO_4$ at 1/20 S/L, 75°C for 2hrs, Blank Roasted: Calcining at 600°C and leaching with 100ml of 50g/L $H_2SO_4$ at 1/20 S/L, 75°C for 2hrs, Direct Acid Baking: Acid baking at 250°C with 1:1 solid to acid ratio and leaching with 100ml DI water at 1/20 S/L, 75°C for 2hrs, 2nd Stage Acid Baking: Calcining at 600°C followed by acid baking at 250°C with 1:1 solid to acid ratio and leaching with 100ml DI water at 1/20 S/L, 75°C for 2hrs).

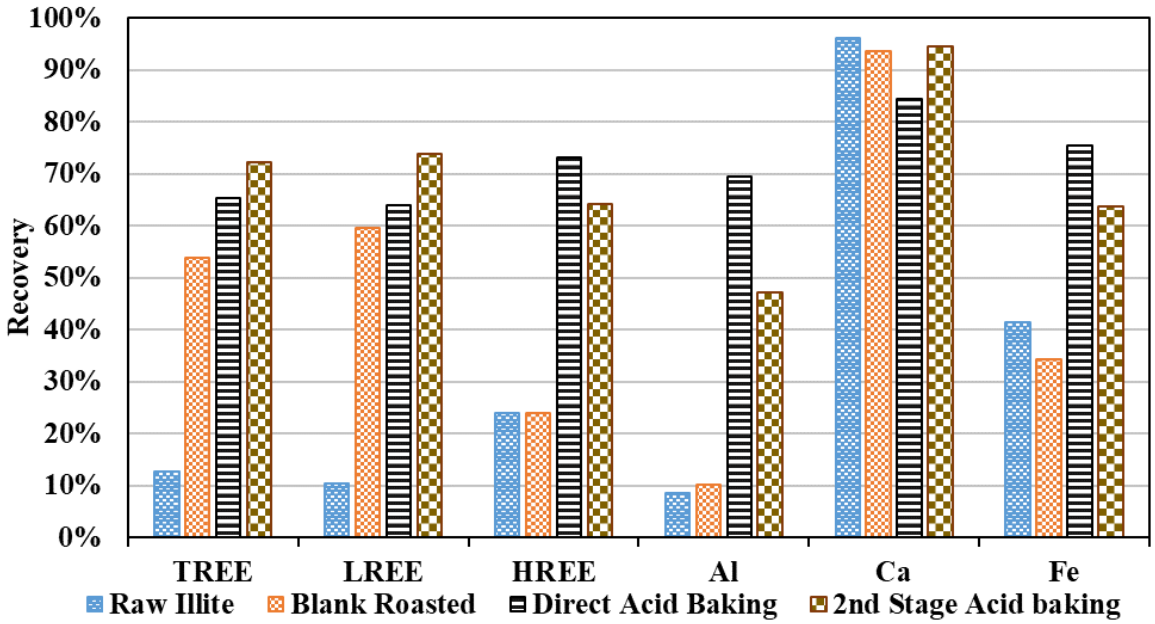



Figure 3.6.37 REE and contaminant recovery from illite samples treated under different conditions (Raw Sample: Direct leaching of raw sample with 100ml of 50g/L $H_2SO_4$ at 1/20 S/L, 75°C for 2hrs, Blank Roasted: Calcining at 600°C and leaching with 100ml of 50g/L $H_2SO_4$ at 1/20 S/L, 75°C for 2hrs, Direct Acid Baking: Acid baking at 250°C with 1:1 solid to acid ratio and leaching with 100ml DI water at 1/20 S/L, 75°C for 2hrs, 2$^{nd}$ Stage Acid Baking: Calcining at 600°C followed by acid baking at 250°C with 1:1 solid to acid ratio and leaching with 100ml DI water at 1/20 S/L, 75°C for 2hrs)

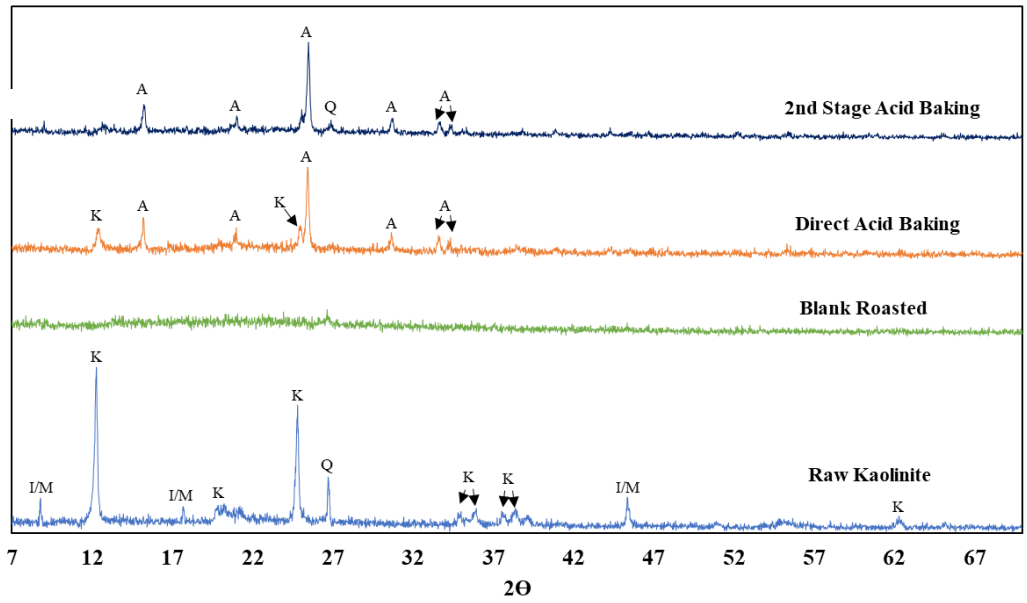



Figure 3.6.38 XRD patterns of Raw Kaolinite, Blank Roasted: Raw kaolinite calcined at 600°C, Direct Acid Baking: Raw kaolinite acid baked at 250°C with 1:1 solid to acid ratio for 2hrs, 2nd Stage Acid Baking: Raw kaolinite calcined at 600°C followed by acid baking at 250°C with 1:1 solid to acid ratio for 2hrs (Q=Quartz, A=Aluminum Sulfate I/M=Illite/Muscovite, K=Kaolinite).

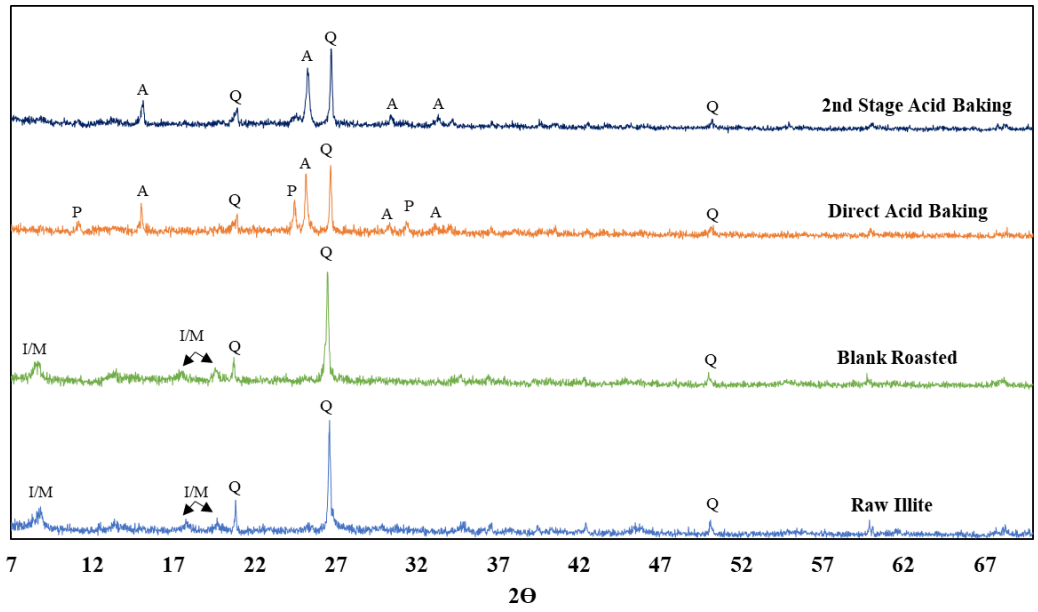



Figure 3.6.39 XRD patterns of Raw Illite, Blank Roasted: Raw illite calcined at 600°C, Direct Acid Baking: Raw illite acid baked at 250°C with 1:1 solid to acid ratio for 2hrs, 2nd Stage Acid Baking: Raw illite calcined at 600°C followed by acid baking at 250°C with 1:1 solid to acid ratio

for 2hrs (Q=Quartz, A=Aluminum Sulfate I/M=Illite/Muscovite, K=Kaolinite, P=Potassium Aluminum Sulfate).

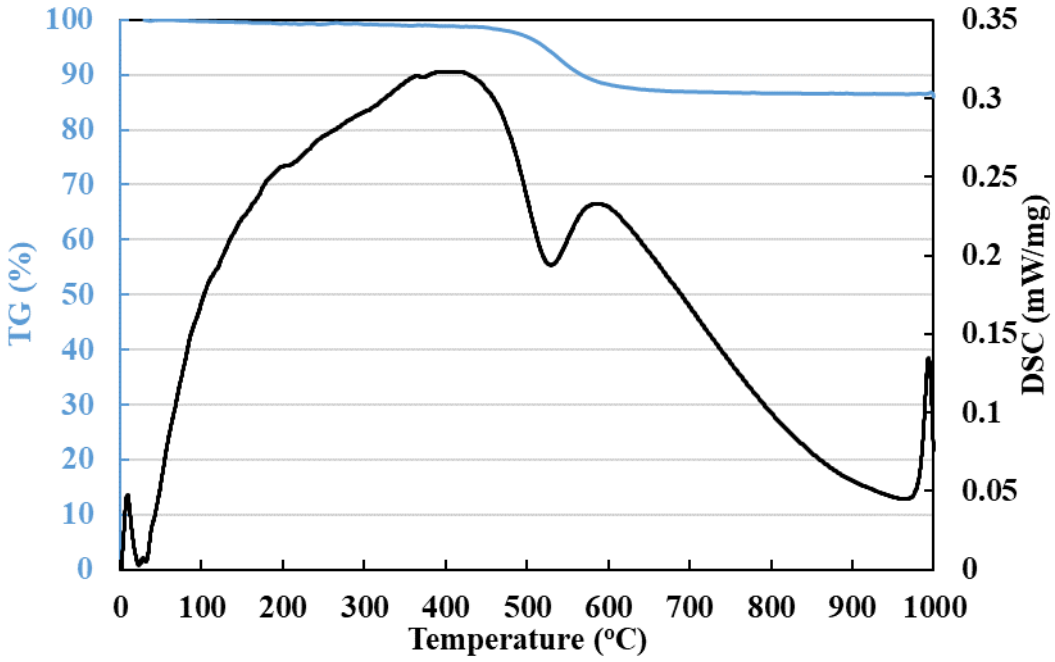



Figure 3.6.40 TGA-DSC curves of mineral kaolinite obtained at the heating rate of 10°C/min and oxidizing condition

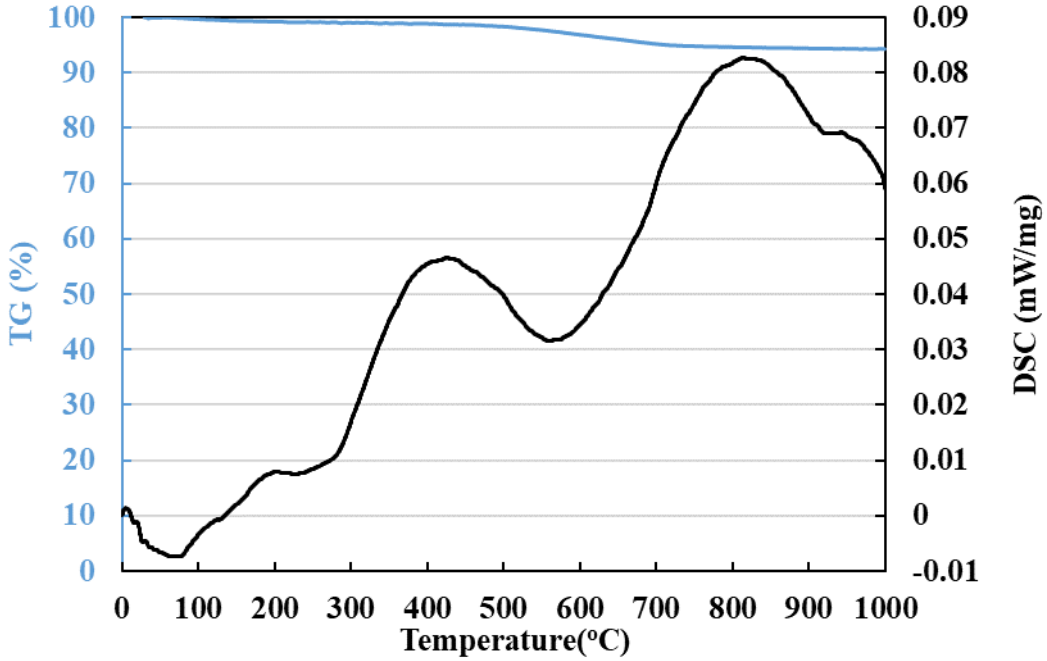



Figure 3.6.41 TGA-DSC curves of mineral illite obtained at the heating rate of 10C/min under oxidizing conditions.

### *3.6.9 Acid Baking optimization to enhance the REE recovery*

The results reported previously indicated that the acid baking of coal has a positive impact on REE recovery. Two different process routes were proposed to enhance the REE recovery. The first was direct acid baking, involving the treatment of raw coal with sulfuric acid at a certain temperature. The REE recovery achieved in this process was significantly higher than the recovery received from simple thermal treatment at the same reagent quantity. The second process involved pre-calcining the coal prior to thermal treatment with the sulfuric acid. Interestingly, the REE recoveries obtained from this 2nd stage acid baking process were significantly higher than direct acid baking. Considering the favorable results achieved in the process, further experiments were performed to optimize the recovery obtained in the 2nd stage acid baking process.

**Impact of time:** Initially, all the 2nd stage acid baking experiments were performed at 2 hrs duration. In order to find the optimum treatment time, the tests were performed at times varying from 0.5 hr to 3 hrs. The experiments were performed at a 1:1 acid to coal ratio using trace metal sulfuric acid at 250 C. The leaching was performed with 100 ml DI water heated to 75 C for 2hrs at 5% S/L. It is evident from Figure 3.6.42 that a maximum TREE recovery of 80% can be achieved only in the first half-hour of the reaction. The recovery of both LREE and HREEs is also approximately 80 percent. All the contaminants also reported significantly high recoveries. As the time of the reaction increases, the recovery of both REEs and contaminants begins decreasing. After three hours, the TREE recoveries drop to approximately 76%, along with the recoveries of major contaminants in the solution. This reduction in REE recovery is possibly due to the continued loss of sulfuric acid, resulting in a higher pH of the lixiviant.

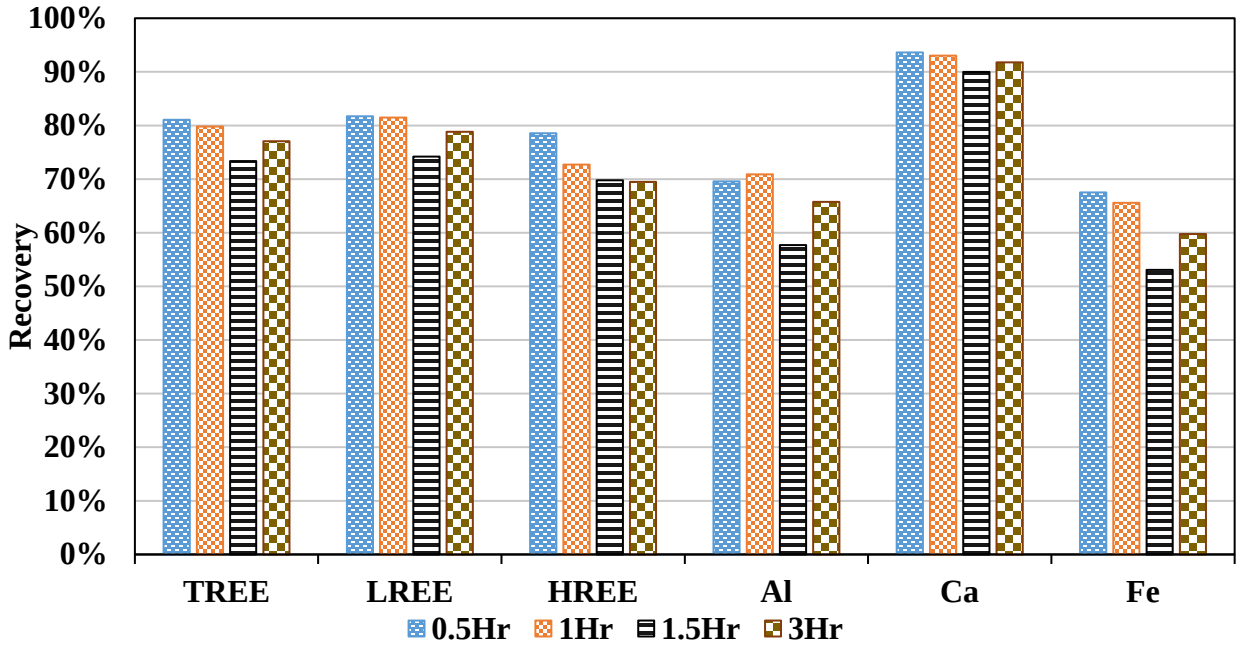



Figure 3.6.42 Impact of time on REE and major contaminant recovery in the 2nd stage acid baking using 1:1 coal to acid ratio.

**Acid Baking Parametric Study:** The impact of various parameters on the REE recovery was investigated by a Box-Behnken statistical design devised using Design-Expert software. The factors used in the experimental design were baking time, acid concentration, baking temperature, and acid: solid ratio (Table 3.6.16). These factors were selected in accordance with their importance reported in a typical acid baking process of monazite and xenotime(Demol et al., 2018b). The acid baking time range was selected based on an initial scoping study conducted at 30-, 60-, 90- and 180-minute intervals. The leaching results indicated that maximum recovery occurred at 30 minutes of baking time. Longer baking time likely resulted in an increase of sulfuric acid evaporation, which increased the solution pH during leaching. Therefore, 30 minutes interval was used as a center point for the study. The acid concentration value was changed from 50-100% (weight basis). This range was selected to test the hypothesis that diluting the sulfuric acid with a constant acid: solids ratio would likely provide better dispersion of the acid in the system, thereby increasing the REE decomposition. The acid baking temperature and acid: solids ranges were selected based on the initial studies performed on the FC-CR SG 2.2 Float material and typical decomposition temperature for acid baking of both monazite and xenotime reported in the literature (Demol et al., 2019).

Table 3.6.16 Parameters and their corresponding ranges used in the Box-Behnken design to analyze their impact on REE recoveries

| Factors | Name | Units | Minimum | Maximum | Mean |
|---|---|---|---|---|---|
| A | Baking Time | min | 10 | 50 | 30 |
| B | Acid Concentration | % | 50 | 100 | 75 |
| C | Baking Temperature | °C | 100 | 300 | 200 |
| D | Acid: Solids | | 0.2 | 1.0 | 0.6 |

**Direct Acid Baking vs. 2nd Stage Acid Baking:** A systematic study previously published by the authors showed a comparison between the direct acid baking and 2nd stage acid baking of the Fire Clay 2.2 float density fraction. It was determined that an additional calcination step is crucial to maximizing REE recovery from the acid baking(Nawab et al., 2022b). The results indicated that HREEs not extractable by simple thermal treatment were easily leached in the 2nd stage acid baking using just DI water. It was concluded that the additional roasting step decarbonized the coal and dehydroxylated the clays, which released REEs bearing minerals, making them much more accessible to an acid attack [(Nawab et al., 2022b)]. Considering the positive impact of acid treatment after thermal treatment on the 2.2F material, some preliminary tests were also conducted to investigate the impact of two different acid baking techniques on REE recovery from the 2.2 sink density fraction (Figure 3.6.43).

The results indicated that simple thermal treatment without additives significantly enhanced the LREE recovery, however, the HREE recovery remained unaffected. This indicates that HREEs present in the 2.2 sink material are associated with difficult to leach minerals (xenotime, zircon), and further treatments are required for their efficient extraction. The increase in LREE recovery is likely due to the thermal decomposition of crandallite group minerals, which have been identified to contain significant concentrations of REEs ((Ji, Li, & Zhang, 2022a; Ji, Li, Tang, et al., 2022)). Compared to simple thermal treatment, the direct acid baking and roasting followed by acid baking at 250°C improved the recovery of both light and heavy REEs. As evident in the results shown in Figure 3.6.43, simple roasting prior to acid baking is not crucial for the extraction of the majority

of REEs from 2.2S density material. Considering the ash content for 2.2F material is approximately 60%, whereas 2.2S material is approximately 90% ash, it is possible that decarbonization is essential for the liberation of REE-bearing minerals in the 2.2F density fraction. Comparatively, since the addition of a roasting step only improved the LREE recovery in the 2.2S material, it is likely that most of the LREEs were either completely liberated or at least partially liberated and were decomposed through contact with sulfuric acid at elevated temperatures. Since the majority of acid baking benefits can be observed simply through direct acid baking, a parametric study was conducted to identify the most significant parameters impacting the recovery of rare earth elements and other contaminants.

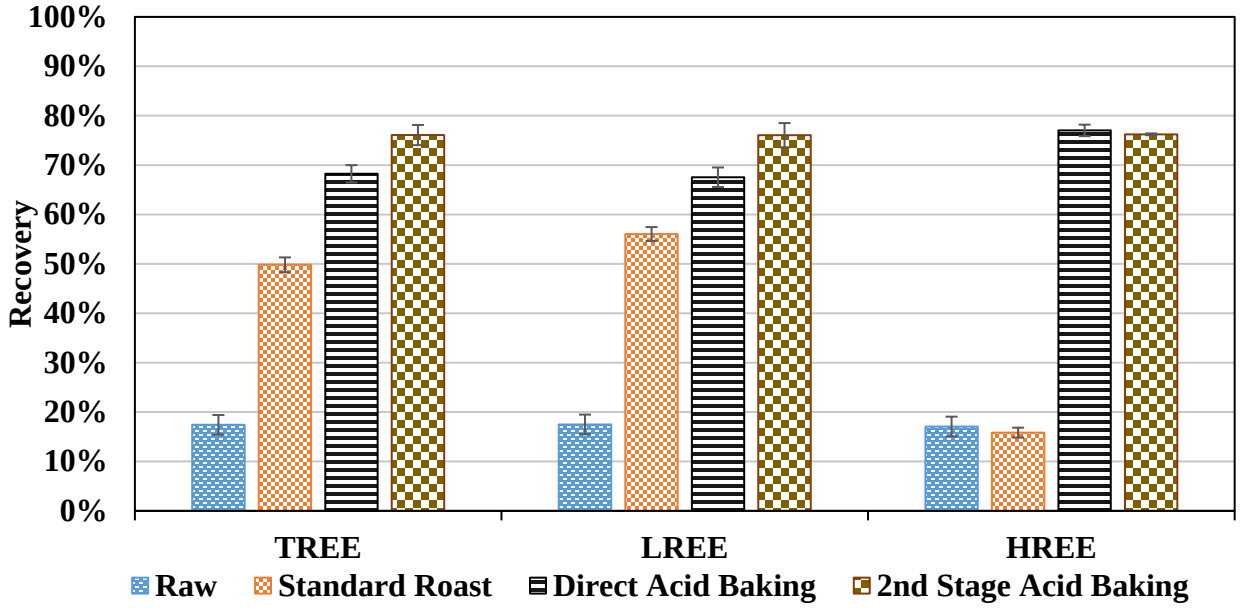



Figure 3.6.43 Impact of different treatment techniques on REE recovery [Leaching:5%S/L, 75°C and 2hrs] (Raw Sample: Direct leaching with 0.5M $H_2SO_4$, Blank Roasted: Calcining at 600°C and leaching with 0.5M $H_2SO_4$, Direct Acid Baking: Acid baking at 250°C with 1:1 solid to acid ratio and leaching with DI water, 2nd Stage Acid Baking: Calcining at 600°C followed by acid baking at 250°C with 1:1 solid to acid ratio and leaching with DI water)

**Experimental Results:** The Box-Behnken design containing the factors as well as subsequent levels used for the optimization of REE recovery is shown in Table 3.6.17. The test program involved 29 experiments, including 5 identical tests. The replicate tests were run numbers 9,11,12,16, and 23 in the experimental design. The standard deviation in all the responses of repeat tests was ≤ 2.0%, indicating excellent repeatability. The primary contaminants considered in this investigation were Al, Ca, and Fe owing to their high concentrations in the feedstock. The results demonstrated that all the factors studied in the design significantly influenced the LREE, HREE and Al recovery. Contrarily, acid concentration and acid: coal ratio substantially impacted the Fe recovery whereas only acid: coal ratio affected the Ca recovery. The highest TREE recovery of 67.19% was observed in the test plan from run 6 where the LREE and HREE recoveries were 66.02% and 74.02%, respectively. The Al recovery ranged from 5.39% from test 19 to 51% from test 2, whereas the Fe recovery ranged from 39% from test 22 to 78% from test 2. Comparatively, the Ca recovery was above 85% in all the experiments.

Table 3.6.17 The Box-Behnken test program used to optimize the REE recovery obtained from the acid baking of the 2.2S density fraction

| Run | Acid Baking Time (min) | Acid concentration (%) | Baking Temperature | Acid: Coal | LREE Recovery (%) | HREE Recovery (%) | Al Recovery (%) | Ca recovery (%) | Fe Recovery (%) |
|---|---|---|---|---|---|---|---|---|---|
| 1 | 50 | 75 | 100 | 0.6 | 19.45 | 31.85 | 8.39 | 92.23 | 60.80 |
| 2 | 50 | 75 | 200 | 1 | 55.87 | 62.18 | 51.33 | 94.19 | 78.18 |
| 3 | 30 | 50 | 100 | 0.6 | 8.75 | 18.96 | 6.57 | 91.06 | 57.74 |
| 4 | 30 | 50 | 300 | 0.6 | 30.31 | 35.46 | 18.47 | 92.75 | 44.33 |
| 5 | 30 | 100 | 200 | 0.2 | 27.45 | 29.47 | 9.57 | 91.07 | 49.71 |
| 6 | 30 | 100 | 200 | 1 | 66.02 | 74.02 | 40.43 | 95.11 | 74.10 |
| 7 | 50 | 75 | 300 | 0.6 | 37.80 | 43.33 | 26.79 | 89.79 | 54.38 |
| 8 | 30 | 75 | 300 | 1 | 51.97 | 61.11 | 47.85 | 94.88 | 73.45 |
| 9 | 30 | 75 | 200 | 0.6 | 44.82 | 47.57 | 26.99 | 91.79 | 68.06 |
| 10 | 50 | 100 | 200 | 0.6 | 59.16 | 62.68 | 35.20 | 93.46 | 70.33 |
| 11 | 30 | 75 | 200 | 0.6 | 46.64 | 49.85 | 29.27 | 94.01 | 69.76 |
| 12 | 30 | 75 | 200 | 0.6 | 47.69 | 50.13 | 29.04 | 94.16 | 69.59 |
| 13 | 10 | 75 | 200 | 0.2 | 11.40 | 20.55 | 8.17 | 86.28 | 53.53 |
| 14 | 30 | 75 | 100 | 1 | 13.5 | 25.35 | 7.34 | 92.06 | 61.11 |
| 15 | 10 | 75 | 300 | 0.6 | 36.96 | 41.54 | 28.81 | 90.64 | 66.73 |
| 16 | 30 | 75 | 200 | 0.6 | 48.57 | 50.03 | 28.76 | 92.31 | 72.65 |
| 17 | 30 | 100 | 100 | 0.6 | 31.02 | 31.85 | 6.81 | 88.85 | 57.98 |
| 18 | 50 | 50 | 200 | 0.6 | 27.04 | 34.00 | 19.96 | 86.72 | 66.41 |
| 19 | 10 | 75 | 100 | 0.6 | 8.61 | 17.26 | 5.39 | 89.48 | 54.11 |
| 20 | 50 | 75 | 200 | 0.2 | 9.62 | 17.60 | 9.05 | 79.59 | 51.23 |
| 21 | 30 | 50 | 200 | 1 | 34.47 | 43.96 | 35.73 | 95.01 | 76.17 |
| 22 | 30 | 75 | 300 | 0.2 | 12.80 | 20.36 | 9.89 | 81.87 | 39.39 |
| 23 | 30 | 75 | 200 | 0.6 | 45.13 | 47.09 | 29.99 | 91.96 | 72.51 |
| 24 | 10 | 50 | 200 | 0.6 | 19.28 | 29.26 | 15.24 | 94.45 | 64.93 |
| 25 | 30 | 100 | 300 | 0.6 | 50.96 | 55.54 | 38.93 | 92.97 | 66.11 |
| 26 | 30 | 50 | 200 | 0.2 | 4.19 | 12.68 | 5.57 | 86.44 | 47.92 |
| 27 | 30 | 75 | 100 | 0.2 | 5.99 | 16.42 | 6.40 | 92.85 | 53.68 |
| 28 | 10 | 100 | 200 | 0.6 | 56.37 | 56.28 | 33.44 | 94.94 | 68.83 |
| 29 | 10 | 75 | 200 | 1 | 41.64 | 46.99 | 25.32 | 94.56 | 71.54 |

**Quadratic Regression Models:** Five distinct regression models were employed to investigate the impact of acid baking time, acid concentration, baking temperature, and acid: coal ratio on the responses under consideration. The significance of individual regression models, their parameters,

and the subsequent interactions between them was tested against the null hypothesis that the coefficient for the variable is null. The parameters with a p-value < 0.05 were deemed significant, and the forward elimination method was employed to remove insignificant parameters. Additionally, the model accuracy was determined through higher adjusted and predicted $R^2$ values.

The model fit statistics shown in Table 3.6.18 indicate that all the models used for analysis were highly significant with <0.0001 p-values. Furthermore, all the parameters studied in the investigation substantially impacted the LREE, HREE, and Al recovery. Whereas only acid concentration and acid: coal ratio seemed to impact Fe recovery, and only acid: coal ratio influenced the Ca recovery. Similarly, the model suggested multiple interactions between the parameters under consideration. Higher $R^2$ and adjusted $R^2$ values coupled with a minimal difference between predicted $R^2$ and adjusted $R^2$ values indicated that the models could accurately predict the responses. Hence, the models were used to investigate the impact of individual parameters on each response.

Table 3.6.18 Model fit statistics, significant parameters, and interactions in each recovery model used in the investigation

| Model | Model p-value | Significant Parameters | Significant Interactions | $R^2$ | Adjusted $R^2$ | Predicted $R^2$ |
|---|---|---|---|---|---|---|
| LREE Recovery | <0.0001 | A, B, C, D | CD | 0.96 | 0.94 | 0.89 |
| HREE Recovery | <0.0001 | A, B, C, D | AD, CD | 0.96 | 0.94 | 0.88 |
| Al Recovery | <0.0001 | A, B, C, D | AD, BC, CD | 0.93 | 0.90 | 0.83 |
| Fe Recovery | <0.0001 | B, D | AC, BC, CD | 0.93 | 0.90 | 0.78 |
| Ca Recovery | <0.0001 | D | - | 0.44 | 0.42 | 0.33 |

**Impact of Temperature:** Thermal Behavior of sulfuric acid: Thermogravimetric-differential scanning calorimetric (TGA-DSC) experiments were conducted to understand the thermal behavior of sulfuric acid at different temperatures with the objective to correlate phase changes to the results obtained from the experimental design. The tests were performed under oxidizing atmosphere using the same concentrations as used in the statistical design, i.e., trace metal grade, 75 and 50% (w/w) sulfuric acid (Figure 3.6.44). It is evident that the trace metal grade sulfuric acid evaporation starts at 150°C, accelerates at 200°C, and completely evaporates at 250°C. The experimental data is in accordance with the simulated data using HSC chemistry software, where sulfuric acid decomposes completely at 260°C. It should be noted here that a wide range of temperature values (200-400°C) have been reported for sulfuric acid decomposition, which is likely due to the type of system and atmosphere under consideration in a study ((Ariffin et al., 2013; Corgnale et al., 2020; Schwartz et al., 2000; Yavors'kyi & Helesh, 2016)).

Interestingly, the equilibrium concentrations data for the sulfuric acid system presented in Figure 3.6.45 indicate that sulfuric acid occurs in a gas phase between 250-600°C. Since a decrease in the $H_2SO_{4(l)}$ concentration is followed by a subsequent increase in the $H_2SO_{4(g)}$ content, it is possible that the $H_2SO_{4(l)}$ converts into $H_2SO_{4(g)}$ prior to its eventual conversion into $SO_3$, $SO_2$, $H_2O$, and $O_2$. This conclusion is supported by the enthalpy of reaction calculated by integrating the area under the curve of the endothermic peak observed in DSC of trace metal grade sulfuric acid. The theoretical reaction enthalpy for the decomposition of sulfuric acid into $SO_3$ and $H_2O$ is

approximately -10.07 kJ/g. In contrast, the actual enthalpy of reaction estimated by the DSC for trace metal grade sulfuric acid was approximately 0.84 kJ/g. This suggests that the direct decomposition of sulfuric acid into $SO_3$ and $H_2O$ is likely not a dominant reaction in the current system. However, since the theoretical enthalpy of reaction for the conversion of $H_2SO_{4(l)}$ to $H_2SO_{4(g)}$ [0.80 kJ/g] matches closely with the calculated enthalpy via DSC, it indicates that the conversion of sulfuric acid into a vapor is, in fact, the dominant reaction in the present system, providing support for the modeled data. The presence of water vapors and $SO_3$ in 250-300℃ implies that some portion of sulfuric acid is directly decomposed into $SO_3$ and $H_2O$ following eq.1((Corgnale et al., 2020; Schwartz et al., 2000)), suggesting multiple reaction mechanisms for sulfuric acid. Both Schwartz et al. and Soltani et al. have shown similar behavior of sulfuric acid ((Schwartz et al., 2000; Soltani et al., 2018)).

$$H_2SO_{4(l)} \rightarrow H_2O_{(g)} + SO_{3(g)} \quad (1)$$

Interestingly, the decomposition temperature of the dilute sulfuric acid is different compared to the trace metal grade sulfuric acid. The total mass loss observed at 200℃ for trace metal grade sulfuric acid was 17%, whereas the thermal analysis of 75 and 50% sulfuric acid solutions revealed as much as 60 and 75% mass loss, respectively, at only 200℃. Yavors'kyi and Helesh et al. studied the concentration of sulfuric acid through evaporation and showed that the concentration process starts with the evaporation of water vapors into the gaseous phase. This process continues till the sulfuric acid concentration reaches approximately 80%. Further heating of the solution results in the loss of sulfuric acid, most likely following eq.(1). Since significant sulfuric acid evaporation has been reported to occur at >150℃, it can be concluded that the initial mass loss observed in the TGA analysis of 75 and 50% sulfuric acid was due to water evaporation. The authors of the study identified the boiling temperature of 85% sulfuric acid to be 226℃, which agrees well with the evaporation temperature of 215℃ (endothermic event for both 75 and 50% sulfuric acid) observed in this study(Yavors'kyi & Helesh, 2016).

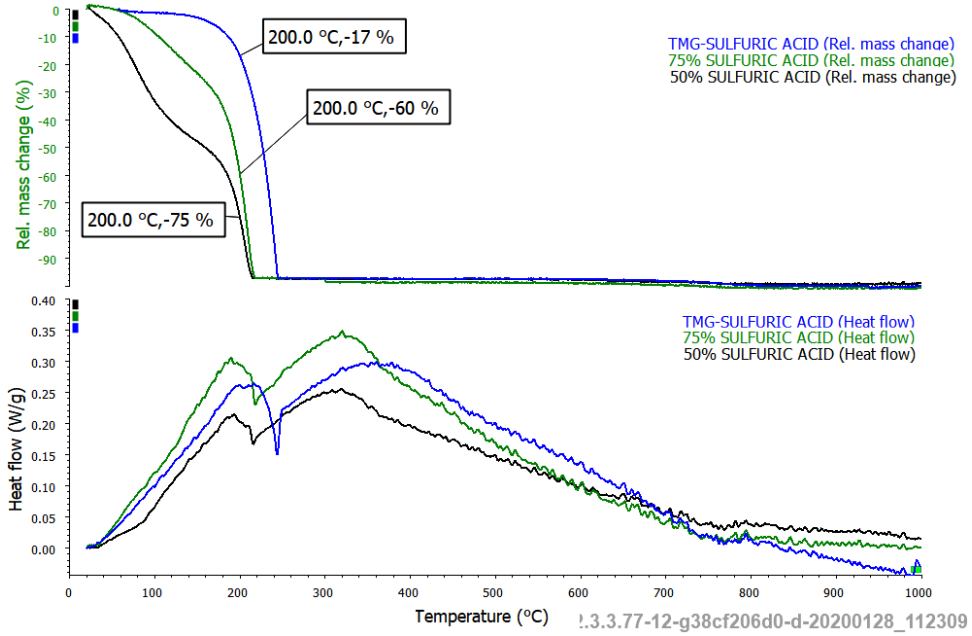



Figure 3.6.44 TGA-DSC analysis for acids of varying strength under oxidizing atmosphere(5°C/min)

As discussed previously, the conversion of sulfuric acid into $SO_3$ is typically followed by the decomposition of $SO_3$ into $SO_2$ between 750-900°C following eq(2)(Corgnale et al., 2020). This conversion from $SO_3$ and $SO_2$ is an endothermic event. However, as evident in the DSC analysis depicted in Figure 3.6.44, no such endothermic event was observed after the conversion of sulfuric acid to $SO_3$. Since the experiments in this study were performed in an open atmosphere, the conversion is unlikely because any $SO_3$ produced will escape the reaction chamber through the exit valve open to the atmosphere. Furthermore, the conversion of $SO_3$ to $SO_2$ requires a catalyst such as $V_2O_5$, therefore, in the absence of a catalyst, even if there was $SO_3$ present in the system, the reaction is improbable(Corgnale et al., 2020).

$$SO_3 \rightarrow SO_2 + \frac{1}{2}O_2 \qquad (2)$$

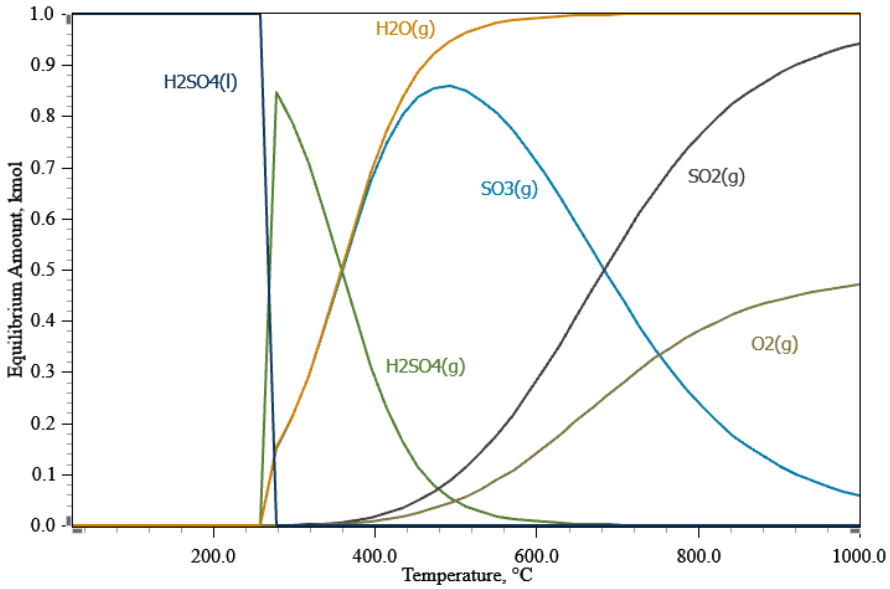



Figure 3.6.45 Equilibrium concentration of different species as a function of temperature during the thermal treatment of sulfuric acid (Calculated with HSC Chemistry 10)

**Impact of temperature on REE recovery:** This investigation revealed an overall positive correlation between the temperature and the recovery of light and heavy rare earth elements (Figure 3.6.46). The LREE recovery improved from 30% at 100°C to 71% at 250°C, whereas HREE recovery from 40% at 100°C to 77% at 250°C. Further increase in the temperature from 250 to 300°C did not significantly impact the light and heavy rare earth recovery. The LREEs have been shown to be associated with crandallite group minerals as well as monazite, whereas HREEs were determined to be associated with xenotime and zircon in the FC-2.2 sink density fraction ((Ji, Li, & Zhang, 2022a; Ji, Li, Tang, et al., 2022)). The study concluded that the primary mode of occurrence for light and heavy REEs was monazite and xenotime, respectively, in the 2.2 sink density material. Therefore, the reaction between these REE-bearing minerals and sulfuric acid likely occurs following eq. (3).

$$\mathbf{2REE\text{-}PO_{4(s)} + 3H_2SO_{4(aq.)} \rightarrow REE_2(SO_4)_{3(s)} + 6H^+_{(aq.)} + 2H_3PO_{4(aq.)}} \quad \mathbf{(3)}$$

Even though both monazite and xenotime are REE-phosphates undergoing the same reaction (3), optimal acid baking conditions for both minerals vary significantly. For instance, a wide range of decomposition temperatures (160-260°C) have been reported to ensure the complete decomposition of monazite(Demol et al., 2019). Similarly, it has been mentioned in the literature that xenotime is more refractory towards an acid attack compared to monazite, and therefore, more stringent conditions (>250°C) are required for its complete decomposition ((Banks et al., 1959; Hadley & Catovic, 2014; Vijayalakshmi et al., 2001)). Based on these previous findings, it is interesting to note that at any given temperature, HREE recovery is seemingly higher than LREE recovery.

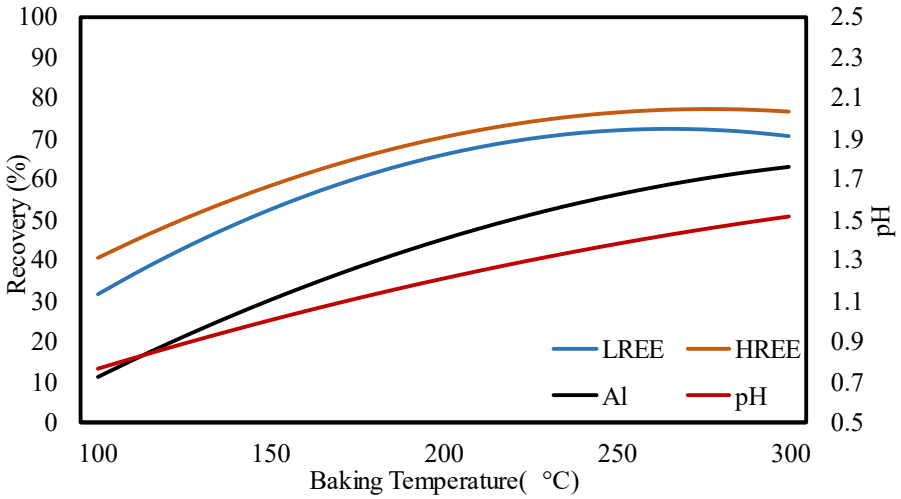



Figure 3.6.46 Influence of temperature on LREE, HREE, and Al recovery estimated by the model (A: 30 min, B:100%, D: 1:1)

An improvement in the light and heavy REE recovery as a function of increasing temperature is likely due to the increased decomposition of completely or partially liberated rare earth-bearing minerals and the released REE-containing minerals through the dehydroxylation of clays. A strong positive correlation of light and heavy REE recoveries with the Al recovery supports this conclusion. A similar strong association of heavy REEs with the clays has been demonstrated in the previous investigation by the authors (Nawab et al., 2022b). As the temperature for the acid baking increases from 250 to 300℃, it is evident that there is not a significant change in the REE recovery, which is likely a result of sulfuric acid evaporation occurring at 250℃ (Figure 3.6.44). A significant increase in the solution pH during leaching of acid-baked product at various temperatures also suggested decomposition of sulfuric acid (Figure 3.6.46).

As depicted in Figure 3.6.43, it is possible that some LREE-bearing minerals are either associated with the carbon matter or are still entrapped within a dominant clay mineral. Considering the tendency of sulfuric acid to decompose the clays, the decomposition of rare earth-bearing minerals associated with clays can likely be accomplished at temperatures under consideration. However, since the decarbonization requires treatment at 400℃, this may be a reason for a relatively lower LREE recovery compared to HREEs (Figure 3.6.43 and Figure 3.6.46).

Table 3.6.19 Correlation coefficients between the responses estimated by Design-Expert

| (%) | LREE Recovery | HREE Recovery | Al Recovery | Ca Recovery | Fe Recovery |
|---|---|---|---|---|---|
| LREE Recovery | 1 | | | | |
| HREE Recovery | 0.986 | 1 | | | |
| Al Recovery | 0.903 | 0.931 | 1 | | |
| Ca Recovery | 0.619 | 0.657 | 0.583 | 1 | |
| Fe Recovery | 0.754 | 0.79 | 0.786 | 0.686 | 1 |

**Impact of temperature on contaminant recovery:** Kaolinite and illite are refractory clay minerals and therefore require dehydroxylation at approximately 600℃ following eq.(4) before their complete decomposition following eq.(5) using high acid concentrations ((Belver et al., 2002; Demortier et al., 1999; Luo et al., 2017; Murat & Driouche, 1988; Okada et al., n.d.)). However, recent investigations have indicated that the decomposition of the clays can be accomplished in the presence of a strong acid at much lower temperatures ((Colina et al., 2001, 2002, 2006)). In that regard, Colina et al. performed a systematic study to investigate the impact of different acids on the decomposition of kaolinite and concluded that sulfuric acid provided the maximum Al yield due to its high boiling point. Additionally, it was also determined that the high dehydrating properties of the acid could significantly decrease the temperature range for thermal dehydroxylation (Colina et al., 2001).

$$Al_2O_3.2SiO_2.2H_2O \rightarrow Al_2O_3.2SiO_2 + 2H_2O \quad (4)$$

$$Al_2O_3.2SiO_2 + 3H_2SO_4 \rightarrow Al_2(SO_4)_3 + 2SiO_2 + 3H_2O \quad (5)$$

The results depicted in Figure 3.6.46 indicate that change in the temperature significantly impacts the Al recovery, which increases from 11% at 100℃ to as much as 62% at 300℃. The increase in recovery is likely a result of the higher decomposition of Al-bearing clays such as kaolinite into a water-soluble species as a function of an increase in temperature following eq. (6) ((Makó et al., 2006),(Gao et al., 2016)). The SEM images and EDS spectra shown in Figure 3.6.48 demonstrate the conversion of clay particles from predominantly alumino-silicate to an aluminum sulfate complex. Furthermore, since the crandallite group minerals are also a source of Al, therefore, it is expected that their decomposition will also contribute to the observed increase in the Al recovery in the current system ((Ji, Li, & Zhang, 2022a; Ji, Li, Tang, et al., 2022)). This conclusion is also supported by the strong correlation between the LREE and Al recovery shown in Table 3.6.19.

$$Al_2O_3.2SiO_2.2H_2O + 3H_2SO_4 \rightarrow Al_2(SO_4)_3 + 2SiO_2 + 5H_2O \quad (6)$$

Since the clays such as kaolinite and illite are the primary source of Al in the coal material, their thermal behavior was investigated through a series of acid baking tests on the pure clay minerals. The results shown in Figure 3.6.46 indicate that the untreated material is predominantly kaolinite, which decomposes as the acid baking temperature is increased from 200 to 300℃. It is evident that kaolinite converts to oxonium aluminum sulfate at 200℃ before its complete conversion to aluminum sulfate at 250℃. The data shown in Table 3.6.20 supports the earlier conclusion that increased temperature favors the decomposition of kaolinite.

It can be concluded that a strong positive association of Al recovery with an increase in temperature agrees with the previous studies conducted on the activation of clays with sulfuric acid at elevated temperatures ((Colina et al., 2001, 2002; Gao et al., 2016)). Interestingly, one author working on the sulfuric acid baking of coal tailings reported no statistically significant change in the Al recovery when the temperature was increased from 200 to 300℃ (Kuppusamy & Holuszko, 2022). Since raising the temperature above 200℃ drastically improves the evaporation rate of sulfuric acid (Figure 3.6.44), with complete evaporation occurring at 250℃, the lack of sulfuric acid availability was one possible reason for the noted anomaly. Regarding Fe and Ca, it was determined that the acid baking temperature does not significantly impact their recovery. This is

likely a result of increased solubility of resulting species from the acid baking treatment. Additionally, at least some Fe and Ca recovered is likely a result of the decomposition of illite/muscovite.

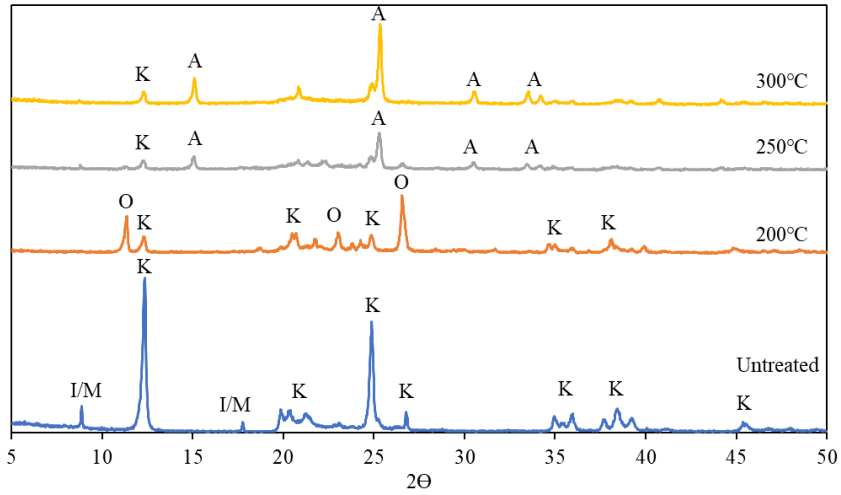



Figure 3.6.47 XRD analysis of untreated and acid baked samples at various temperatures with 1:1 solid to acid ratio and 30 min (K=Kaolinite, A=Aluminum Sulfate, I/M=Illite/Muscovite, O=Oxonium Aluminum Sulfate).

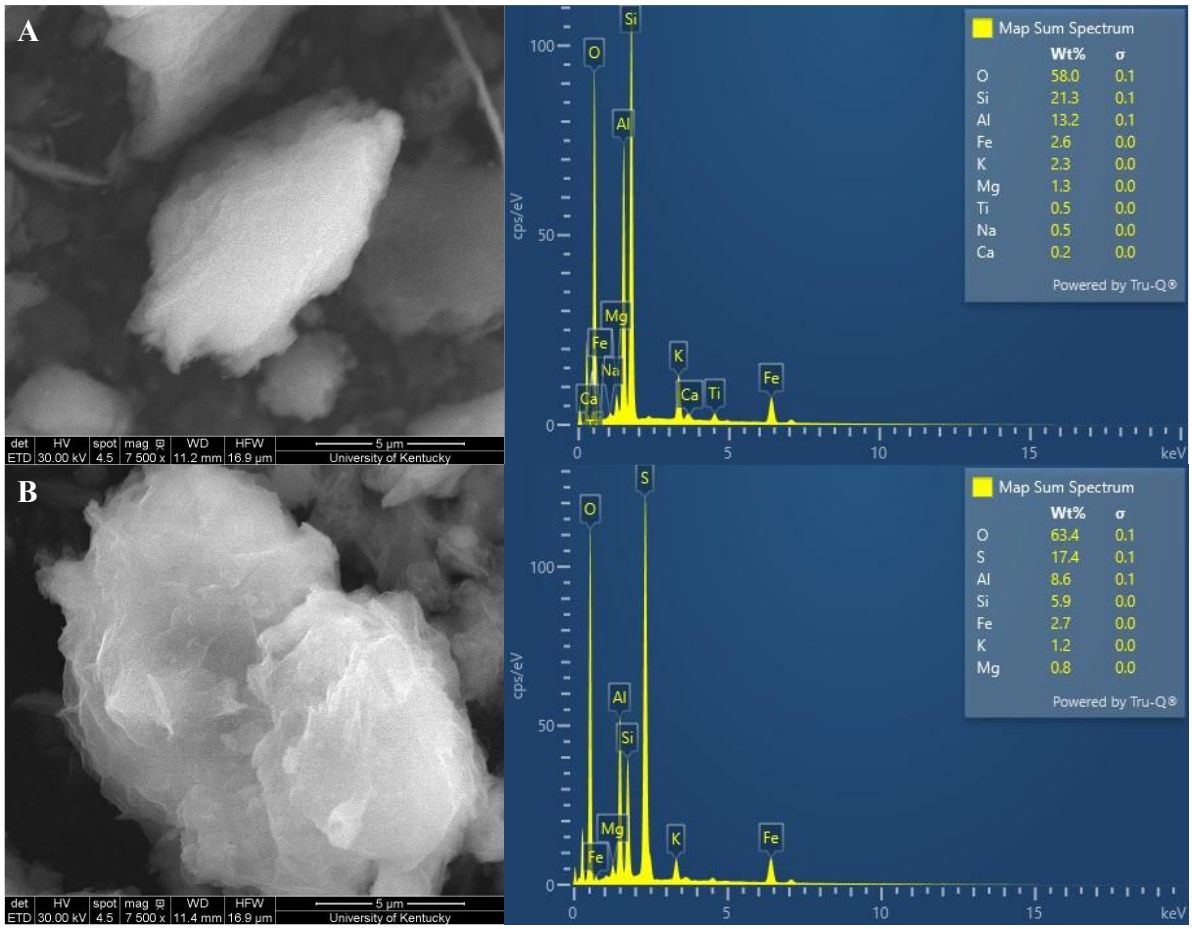



Figure 3.6.48 SEM image and EDS spectrum of: A) Sample roasted at 600°C B) Sample acid baked at 250°C at 1:1 acid to coal ratio for 1 hr.

Table 3.6.20 Mineral distribution estimated through XRD analysis for raw and acid baked kaolinite samples

| Composition | Kaolinite | Illite | Oxonium aluminum sulfate | Aluminum sulfate |
|---|---|---|---|---|
| Untreated | 92.16% | 7.84% | - | - |
| 200°C | 58.29% | - | 41.71% | |
| 250°C | 23.28% | - | 2.90% | 73.82% |
| 300°C | 17.06% | - | - | 82.94% |

**TGA-DSC Analysis:** The impact of temperature on the acid baking of coal refuse was also investigated using TGA-DSC (Figure 3.6.49). Similar to the results observed in Figure 3.6.44, there is a significant increase in mass loss when the temperature increases from 150 to 250°C. However, while the mass loss depicted in Figure 3.6.44 was solely because of the evaporation of sulfuric acid, the mass loss at 250°C in Figure 3.6.49 is likely a combination of sulfuric acid evaporation and structural water loss due to the dehydroxylation of the clays. Correlating the endothermic event observed at 250°C in Figure 3.6.49 with Figure 3.6.44 suggests that sulfuric

acid evaporation is the dominant reaction at this temperature. As the temperature increases from 250℃ to 650℃, a gradual mass loss is likely due to the decarbonization of coal and the increased conversion of kaolinite into aluminum sulfate (Colina et al., 2002). A sharp increase in the mass loss from 650 to 725℃ was possibly due to the decomposition of aluminum sulfate into aluminum oxide through eq. (7) (Truex et al., 1977).

$$Al_2(SO_4)_3 \rightarrow Al_2O_3 + 3SO_3 \quad (7)$$

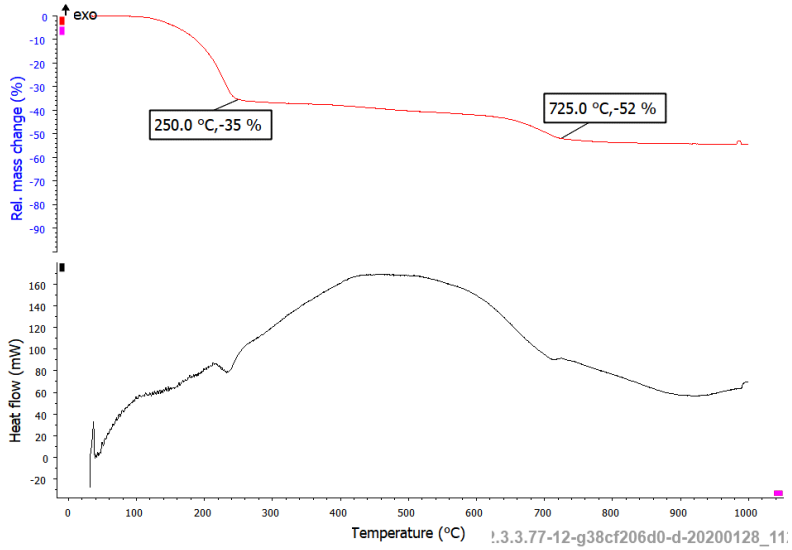



Figure 3.6.49 TGA-DSC analysis performed on the acid baking of FC-CR 2.2S material under oxidizing atmosphere

**Impact of Time:** The reaction between monazite and sulfuric acid at elevated temperatures has been shown to have fast kinetics during the initial 15 minutes of the reaction ([30], [33], [34]). The results depicted in Figure 3.6.50 suggest a similar mechanism for the REE recovery in the current system. Approximately 63% LREEs were recovered in the initial 10 minutes of the reactions. Further increasing the retention time to 30 minutes enhanced the LREE recovery to 72% and remained consistent at this level until 40 minutes and decreased to 69% at 50 minutes. Contrarily, the HREE recovery at 10 minutes was approximated to be 65% and progressively improved to 82% at 50 minutes. Similar to HREE recovery, the Al recovery also enhanced steadily from 47% at 10 minutes to 66% at 50 minutes. The model suggested minimal change in the pH with an increase in the

The initial increase in the LREE recovery was likely due to the decomposition of crandallite group minerals and monazite particles in the presence of excess sulfuric acid availability in the system. As the residence time increases to 30 minutes, more sulfuric acid reacts with the monazite, resulting in an increase in the LREE recovery. Further increase in the residence time to 40 minutes didn't change the LREE recovery. It has been previously reported that increasing the acid baking time also improves acid consumption by gangue minerals, resulting in their decomposition (Berry et al., 2018). Considering the relatively much higher contaminant concentrations in the present system, this may be a reason for the lack of improvement in the REE recovery from 30-40 minutes (Figure 3.6.50). Further increasing the acid baking time reduced the LREE recovery. Since the

evaporation of sulfuric acid also occurs at approximately the same temperature (Figure 3.6.44), prolonged retention time (>30 min) would likely evaporate some sulfuric acid. Hence, the unavailability of sulfuric acid due to increased reaction with contaminants and evaporation may cause a reduced conversion of REE-bearing minerals to water-soluble sulfates.

As aforementioned, the recovery of both heavy rare earth elements and aluminum steadily improved as a function of acid baking time possibly due to the enhanced decomposition of clays. Similar to the increased decomposition of clays observed with an increase in temperature (Figure 3.6.47), the decomposition of kaolinite has also been shown to improve with an increase in residence time at a same temperature (Gao et al., 2016). Then, the dehydroxylation of clays liberated the HREE-bearing minerals, which were subsequently decomposed through the reaction with sulfuric acid. The model for Fe and Ca recovery depicted that baking time was an insignificant parameter, which is likely due to the fast reaction kinetics of the Fe and Ca-bearing minerals.

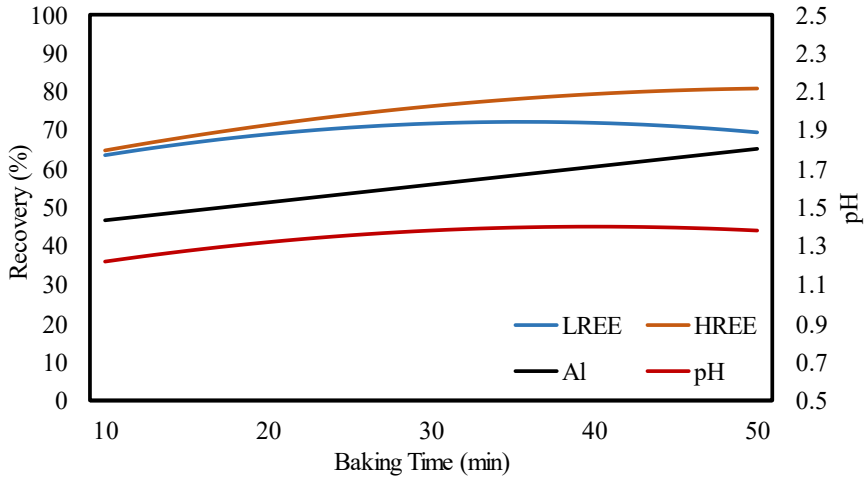



Figure 3.6.50 Impact of acid baking time on the recovery of LREE, HREE, Al recovery and pH (B:100%, C:250°C, D:1:1)

**Impact of Acid Concentration:** This investigation revealed a linear correlation between the REE recovery and the acid concentration (Figure 3.6.51). The results depicted that an increase in the acid concentration is better for the decomposition of REE-bearing minerals such as crandallite group minerals, monazite, xenotime, and zircon. The LREE recovery increased from approximately 43% at 50% acid concentration to the maximum of 72% at 100% sulfuric acid concentration. Comparatively, the HREE recovery improved from 52% to 76% under the same conditions. Blickwedel discovered that decreasing the acid concentration of sulfuric acid substantially decreases the reaction rate in acid baking (Blickwedel, 1949). Contrarily, Shaw et al. reported that a slight decrease in the acid concentration to 93 wt.% improved the reaction kinetics by allowing better mass transfer of the acid (Shaw, 1953). However, as reported in this study, the REE recovery seems to improve with an increase in the acid concentration, indicating that the mass transfer of acid is not a limitation under the current reaction setup. Furthermore, diluting the sulfuric acid reduces its boiling point resulting in its evaporation at a lower temperature (Figure 3.6.44). This accompanied by the lower reaction rate, would explain the lower REE recovery obtained from the acid baking at lower acid concentrations.

In regard to Al recovery, the clays have been shown to decompose through a reaction with sulfuric acid due to its high dehydrating power and boiling point ((Colina et al., 2001), (Criado et al., 1984)). Colina et al. concluded that the kaolinite decomposition was primarily contingent upon the proton to aluminum ratio as well as the reaction temperature and time. The addition of water at a constant $H^+$ ion concentration may improve the contact of acid with the clay surface, but it did not impact the reaction yield (Colina et al., 2002). Since decreasing the acid concentration also impacts the proton concentration in the solution, along with the boiling point of the solution, this may be the reason for a decrease in Al recovery. Similarly, since some of the Fe contamination also exists in clays such as illite, an improvement in the Fe recovery is expected. The initial high recovery of Fe is likely due to easier decomposition Fe-bearing minerals and higher solubility of resulting Fe-sulfates [11 mol/L] ((Berry et al., 2018),(Lide, 2004)).

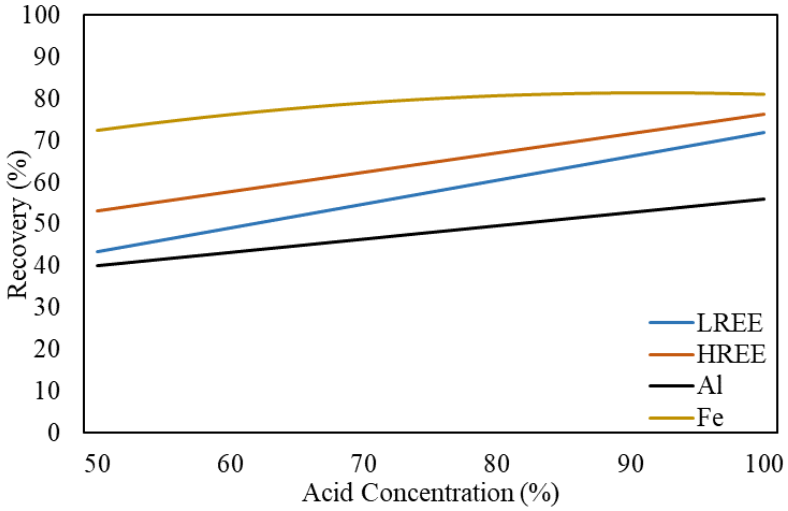



Figure 3.6.51 Impact of acid concentration on the LREE, HREE, Al and Fe recovery (A:30 min, C: 250°C, D:1:1)

**Impact of Acid: Coal ratio:** The effect of acid: coal ratio on REE recovery was determined at 5 different levels (Figure 3.6.52). As anticipated, increasing the amount of acid in the system benefits rare earth element recovery. The LREE recovery increased from 30% at 0.2:1 to 72% at 1:1, whereas HREE recovery improved from 34% to 76.5% at the same acid: coal ratios, respectively. Similarly, the Al recovery also increased from 19 to 56.2%, and the iron recovery raised from 52 to 80% under the aforementioned treatment conditions. The improvement in the LREE recovery as a function of increasing acid to coal ratio is likely a result of improved decomposition of LREE-bearing minerals such as monazite and crandallite group minerals. Interestingly, significant improvement in the LREE recovery was observed when the acid to coal ratio is increased from 0.2:1 to 0.8:1. A further increase in the acid content did not impact the LREE recovery. Comparatively, the HREE recovery continued to improve with an increase in the acid to coal ratio. As explained in previous sections, this is likely a result of increased decomposition of clays, which is also evident through an improvement in the Al recovery (Figure 3.6.52). Furthermore, as evident in the results shown in Figure 3.6.52, increasing the acid: coal ratio also decreased the solution pH, which is an expected change. It is likely that at lower acid concentrations, some of the decomposed rare earth element may have precipitated due to the lower solution acidity. A

significant Ca recovery even at such a low acid: coal ratio is due to the easy dissolution of Ca-bearing minerals. As explained in the previous section with respect to iron, elevated Fe recovery at such a low acid quantity is likely a result of high solubility of Fe-sulfates.

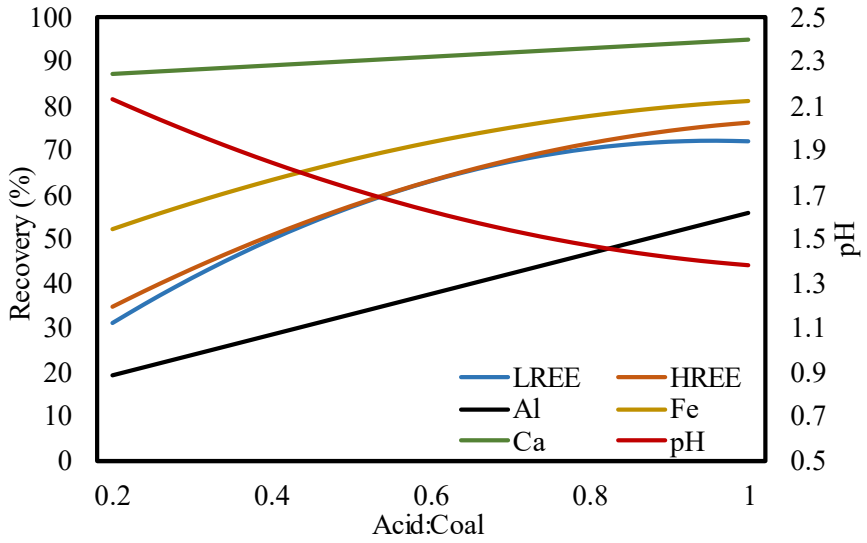



Figure 3.6.52 Influence of acid: coal concentration on REE and contaminant recovery (A:30 min, B:100%, C: 250°C)

**Response Surface:** The experimental design performed in this study revealed various interactions between different parameters (Figure 3.6.53). The response surface on the relation between temperature and acid: coal ratio indicated that using high reaction temperatures at a low acid: coal ratio slightly improved the recoveries of LREE, HREE, and Al, likely due to the incomplete decomposition of the host minerals (Figure 3.6.53(i), (iii), (vi)). Contrarily, high acid: coal ratios at lower temperatures benefits LREE and HREE recoveries more compared to Al recovery. This is likely due to the relatively higher temperature requirement for the decomposition of clays. Additionally, since there are various modes of occurrence of light and heavy REEs, some more amenable to an acid attack than others, an improvement in the recovery is expected. A concurrent increase in both temperature and acid: coal ratio substantially improved the recoveries. However, the baking temperature becomes insignificant when the acid: coal ratio is high (1:1). When the acid : coal ratio is low (0.2:1), the optimal baking temperature occurs at 200-250°C, which is likely a result of sulfuric acid evaporation.

Similarly, the combination of acid: coal ratio and baking times were also determined to influence HREE and Al recovery (Figure 3.6.53(ii),(iv)). It was found that in the presence of high sulfuric acid content, a substantial concentration of HREE and Al-bearing minerals can be decomposed in the first 10 minutes of the reaction. A strong correlation in the increase in Al and HREE recovery reiterates the earlier conclusion that decomposition of clays provides sulfuric acid access to HREE-bearing minerals. It is interesting to note that at a lower acid content, even increasing the treatment time did not significantly impact the recoveries likely due to the insufficient concentration of reactant. Finally, an interactive effect of temperature and acid concentration on Al recovery was likely due to two reasons: i) Reducing the acid concentration also reduces the $H^+$ ion concentration, which, as discussed previously, is crucial for clay decomposition. ii) A decrease in acid

concentration also decreases the decomposition temperature of sulfuric acid (Figure 3.6.44), which is crucial for the decomposition of clays.

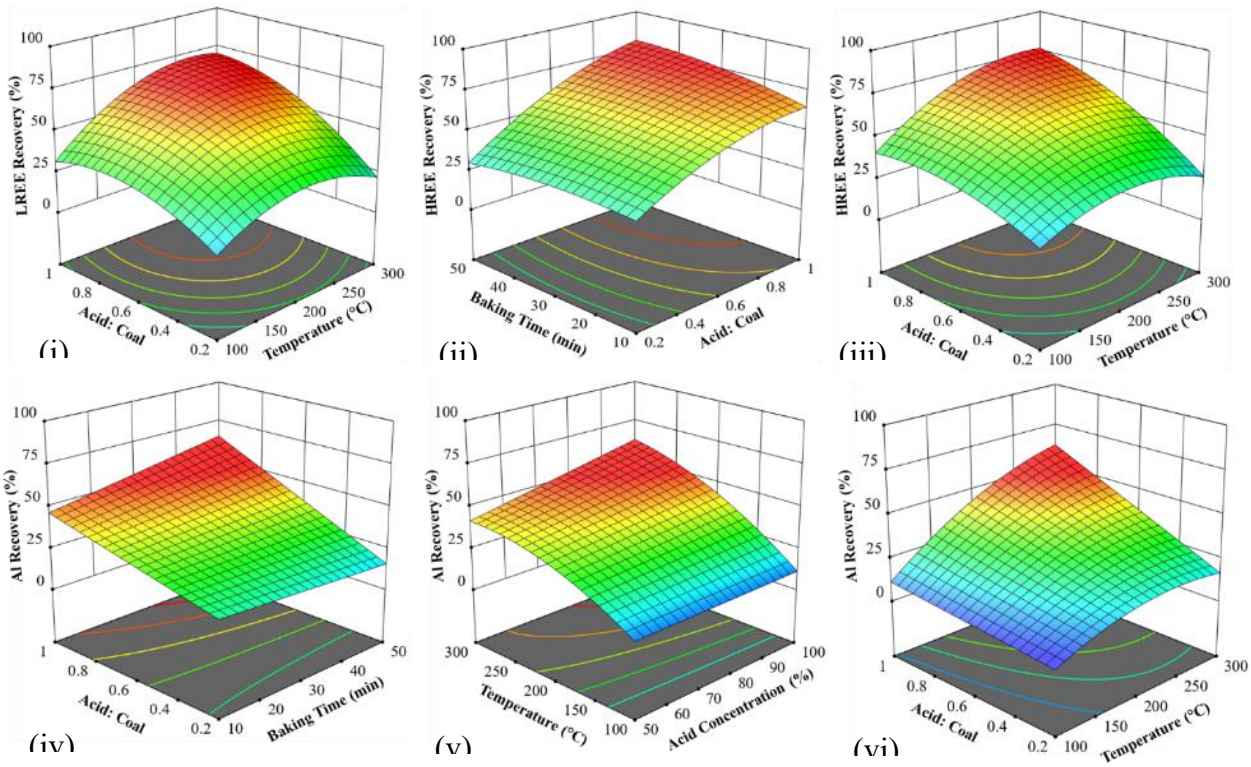



Figure 3.6.53 Response surface plots for LREE, HREE and Al recovery as a function of main and interaction effects of the significant variables

**Impact of Leaching on REE recovery:** All the leaching experiments discussed so far in the parametric study was conducted at 75°C. This was done to directly compare the recoveries obtained from acid-baked samples and be consistent with the extensive work previously completed on the same feedstock. Interestingly, the leaching operations of acid-baked products are typically carried out at room temperature due to the inverse relationship between REE-sulfates solubility and temperature(Pilkington & Wylie, 1947). Therefore, a hypothesis was developed that some of the REEs not recovered during this study may be due to the insolubility of potentially produced rare earth sulfates at elevated temperatures. This hypothesis was tested by running run 6 in Table 3.6.17 at different leaching temperatures. The results depicted in Figure 3.6.54 suggested that a decrease in temperature does, in fact, improve the REE recovery. This leaching behavior is in agreement with the previously reported trend of REE-sulfate recovery using DI-water(Demol et al., 2019).

Similarly, the solution pH has also been shown to impact the REE recovery. This is due to the stability region of REE-sulfates. Several researchers have shown previously that REE-sulfates in the solution can precipitate as REE-phosphates between 1 and 3 solution pH ((Han, 2019, 2020; Han & Kim, 2021; E. Kim & Osseo-Asare, 2012)). In the current system, the primary reason for a change in the solution pH was due to the evaporation of sulfuric acid and consumption by gangue minerals. Essentially, while an increase in the temperature improves the decomposition of REE-

bearing minerals, the resulting increase in the pH can precipitate at least some of the REEs. Furthermore, the reaction of calcite with sulfuric acid produces gypsum, which has been shown to decrease REE recovery due to isomorphous substitution and surface adsorption ((Dutrizac, 2017; R. Kim et al., 2020; Sadri, n.d.)). Therefore, this may also be one of the contributing factors to <80% REE recoveries reported in this investigation. In essence, a systematic study on the leaching aspect of acid-baked is required to maximize the recovery of rare earth elements.

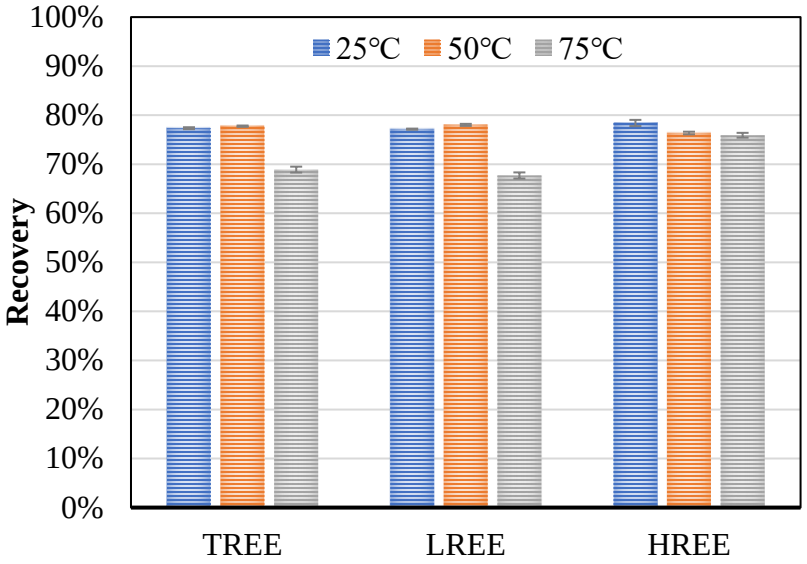



Figure 3.6.54 Impact of leaching temperature on the REE recovery using 5% S/L ratio at 75°C for 2hrs (A:30 min, B:100%, C:200°C, D:1:1)

**BET analysis:** The Brauner-Emmer-Teller (BET) analysis was performed on acid-baked solids from run 6 and 26 in the statistical design as well as the solid residue obtained from its leaching operation. These tests were selected due to their highest and lowest REE recovery in the design, respectively. Since the recoveries discussed in this investigation were the highest REE-recovery ever obtained from this density fraction of coal at such a low acid concentration (0.5M $H_2SO_4$), the BET data was also compared to raw coal, 600°C roasted coal, and the solid residue obtained from leaching. The coal roasted at 600°C was selected because previous studies revealed it to be the optimum temperature to maximize the REE recovery ((Gupta, 2021; W. Zhang et al., 2020)).

The specific surface area, pore volume, and average pore diameter data shown in Table 3.6.21 suggest that acid-baked solids undergo significant structural alterations compared to other feeds under consideration. Interestingly, the raw samples have a higher surface area and average pore diameter compared to 600°C roasted coal, even though much higher REE recoveries are obtained from the latter (Figure 3.6.43). This may be due to the segregation of clays such as kaolinite and illite during dehydroxylation at these temperature ranges(Balek & Murat, 1996). A significant improvement in the surface area of acid-baked products is likely due to the acid activation of clays in the coal refuse. Multiple researchers have reported the delamination of clay particles, mineral impurity removal, and destruction of the external layer during acid activation ((Lenarda et al., 2007; Tang et al., 2017; Yang et al., 2017)). Two essential reactions happen during the activation of clays. First, the acid partially dissolves aluminum, as well as magnesium and calcium oxide, opening the crystal lattice and increasing the internal surface area. The second reaction is the

progressive substitution of $Ca^{2+}$, $Mg^{2+}$, and $Fe^{3+}$ ions at the crystal surface with $H^{+}$ ions from the mineral acid [45]. As a result, they reported an increase in the surface area, pore volume, and surface activity.

The adsorption-desorption behavior depicted in Figure 3.6.55 seems to follow the H4 hysteresis loop according to the IUPAC classification ((Cychosz & Thommes, 2018; Thommes et al., 2015b)). It is interesting to note that desorption branch for all the solids investigated in this study closes around 0.4-0.45 relative pressures. This sudden disappearance has been found to be due to the hemispherical meniscus collapse during capillary evaporation ((Groen et al., 2003; Kuila & Prasad, 2013)). Another interesting observation is a sudden $N_2$ at low relative pressures in the acid-baked samples and leaching solid residues of 600°C roasted material. This has been shown to be due to the filling of micropores(Thommes et al., 2015a). A much higher surface area in leachate solid residues compared to the feeds is likely due to the removal of clays and other impurities, which seems to improve significantly because of acid baking (Table 3.6.17).

Table 3.6.21 BET analysis of coal treated under different conditions as well as solid residues (SR) obtained from leaching

| | Surface area | Pore Volume | Average Pore Diameter |
|---|---|---|---|
| | $m^2.g^{-1}$ | $cm^3.g^{-1}$ | Å |
| Untreated Feed | 12.740 | 0.028 | 81.131 |
| Roasted at 600°C | 6.499 | 0.027 | 159.411 |
| Roasted at 600°C SR | 52.511 | 0.065 | 47.629 |
| Acid baking Test #6 | 39.847 | 0.054 | 51.910 |
| Acid baking Test #6 SR | 105.389 | 0.110 | 40.291 |
| Acid baking Test run #26 | 4.132 | 0.010 | 94.922 |
| Acid baking Test run #26 SR | 28.447 | 0.046 | 62.901 |

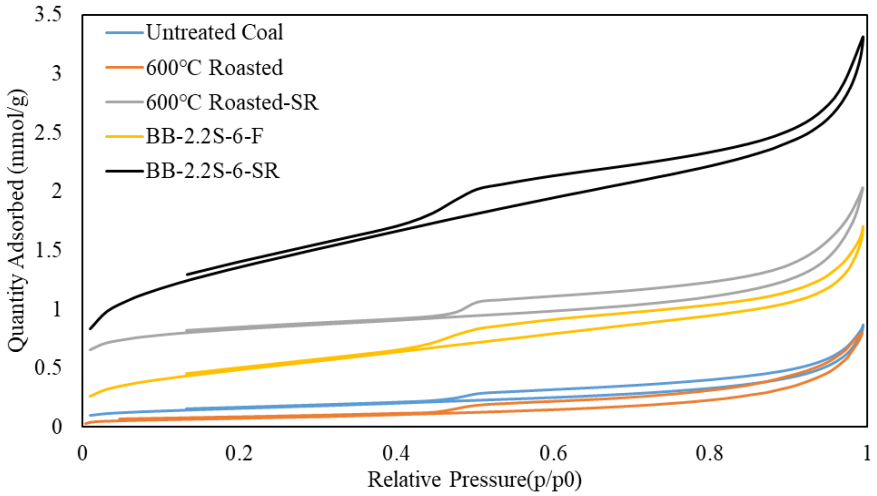



Figure 3.6.55 Adsorption and desorption isotherms of various solids investigated in the study (F: Feed, SR: Solid Residue).

It is evident from the results discussed in this section that sulfuric acid baking significantly enhances the structural characteristics of the solids. A substantial increase in the surface area, pore-volume, and the presence of micropores is expected to provide the sulfuric acid much better access to the REE-bearing minerals, resulting in their efficient decomposition.

*3.6.10 Bio-oxidation Solution Leaching Parametric Effects*

Leaching experiments were carried out using the acid solution generated from the bioleach reactor received from University of Utah. The solution pH was in a range of 1.1-1.2 and the Eh was in a range of 900-910 RmV. The bioleach acid solution contained about 10000 ppm of Fe (majority in ferric ion form) and 2-2.5 ppm of REEs. The composition of the major elements in bioleach solution is as shown in Table 3.6.22.

Table 3.6.22. Composition of major elements in bioleach acid solution.

| Labs | TREE | Al | Ca | Co | Fe | Mg | Mn |
|---|---|---|---|---|---|---|---|
| ICP-MS Utah | 2.1 | 60 | - | 1.3 | 10950 | 303 | - |
| ICP-MS VT | 2.5 | 31 | 650 | 1.4 | 10198 | 208 | 39 |
| ICP-OES UK | 2.2 | 25 | 645 | 1.2 | 11276 | 239 | 44 |
| ICP-OES KGS | 2.6 | 27 | 451 | - | 12430 | - | - |

**Materials**: Leaching experiments were initiated with the West Kentucky No. 13 coarse refuse material. The material was screened at 1-1/2 inch using a vibratory dry screen. The undersize material was collected and ground to a top size of 2 mm using a hammer mill. The material was then roasted at 600 °C for 20 minutes using a muffle furnace to mimic the feed preparation condition employed at the pilot plant. To examine the size and roasting condition effect on leaching performance, the minus 1-1/2 inch material was also pulverized to a top particle size of 80 mesh (177 microns) and roasted at 600 °C for 2 hours using a muffle furnace.

**Apparatus**: The leaching experiment was carried out using a 1-Liter triple neck round bottom flask. A total reflux condenser was connected to the middle neck to reflux the water vapor to maintain the total leaching volume throughout the leaching experiments. Water bath was heated at 75 °C using an programable immersion heater to provide a constant temperature of the leaching reactor throughout the test. Agitation of the leaching reactor was provided using a magnetic stir at a rotation speed of 850 rpm with a 1-inch long egg shape stir.

**Procedure**: A 200 ml of bioleach acid was first introduced to the reactor and heated to 75°C. A thermometer was immersed in the aqueous to monitor the temperature of the slurry. 2-ml of slurry sample was collected at different time intervals during the leaching experiment, the sample was immediately centrifuged at 4000 rpm for 5 minutes to separate the liquid and solid. Then, 1-ml of the effluent was removed from the top of the liquid without removing the solid in the bottom and mixed with 9-ml of 5% nitric acid to stabilize and pre-dilute for ICP analysis. To compare and validate the ICP analytical results, sample was split and further diluted using proper dilution factor to be analyzed in the ICP-OES at the University of Kentucky pilot plant, the ICP-OES at the Kentucky Geological Survey, and the ICP-MS at the Virginia Tech. The results from the three labs were compared. The solid residue sample was collected at the end of the experiments and filtered using a vacuum filter. The pregnant leachate solution (PLS) filtrate was collected for further precipitation test using calcite. The solid residue was washed with DI water to rinse off the residual PLS entrained in the cake then dried in the oven for 24 hours at 65°C. The Eh, pH, and temperature during the leaching experiment was measured using a Thermoscientific Orion Star multi meter and the value was recorded at different time interval while the sample was taken.

Results and Discussion:

**Test XBL-BA2001:** The leaching experiment was conducted using the 2 mm top size material and roasted at 600 °C for 20 minutes. The Eh and pH value were recorded throughout the leaching test as shown in Figure 3.6.56. The pH increased from 1.233 to 2.348 over a period of 5 hours leaching time. The pH increased rapidly within the first hour of leaching which indicated that majority of the material dissolution occurred at the beginning of the leaching. Then the pH slowly increased from 1 hour to 5 hour without reaching a plateau which indicated acid consumption through dissolution of material. The Eh decreased from 858.1 RmV to 693.9 RmV over 5 hours of leaching time due to the precipitation of ferric ion with the increase of pH over time.

The concentration of total rare earth elements (REEs) and the major contaminant ions Al, Ca, and Fe are analyzed and compared among three analytical labs as plotted in Figure 3.6.57. The TREEs increased rapidly at the beginning of the leaching process and reached peak concentration of about 18 ppm after 60 minutes of leaching. The TREEs concentration in PLS then reduced slowly and dropped to about 12 ppm at the end of the leaching at 5 hours. The reduction of TREEs is likely due to the co-precipitation of light REEs with the formation of jarosite. Sodium is one of the key components for jarosite formation. The concentration of sodium started to decrease after 60 minutes of leaching which showed similar trend as the TREEs as shown in Figure 3.6.59.

Lanthanum and cerium are the two major elements of which concentrations reduced after 1 hour of leaching time as shown in Figure 3.6.58. The cerium concentration increased to 6.3 ppm at 1 hour and then reduced to 2.8 ppm at the end of leaching. The peak lanthanum concentration of 2.8 ppm occurred at 30 minutes of leaching then reduced to 0.5 ppm at the end of leaching. The loss of Ce and La during leaching process alone is approximately 56% and 83%, respectively. Other than Ce and La, the concentration of thorium (Th) also decreased significantly with prolonging the leaching time. The Th concentration reached 1 ppm after 30 minutes of leaching then reduced to 0.16 ppm at 5 hours, which was lower than the starting point of 0.215 ppm from the bioleach acid solution. It indicated that 100% of the Th extracted during the leaching process was removed, with extra from the bioleach solution. This will benefit the downstream process on product refining and purification.

Praseodymium (Pr) and neodymium (Nd) concentration showed a slightly head down after 90 minutes of leaching. The loss of Pr and Nd during leaching process alone is 27% and 9%, respectively. The reduction curve aligned well with the Ca reduction of concentration in solution as shown in Figure 3.6.57. The Ca reduction was due to the formation of gypsum ($CaSO_4$) with the excessive amount of sulfate present in the solution. The rest of the REEs does not show any reduction of recovery during leaching process. Most of heavy REs and the critical elements concentration reached plateau within 1 hour of leaching time. Sc, Y, Co, Mn, and all the heavy REEs were not affected by Fe precipitation as shown in Figure 3.6.58 and Figure 3.6.59. Therefore, the recovered value of REEs and critical elements in leachate increased with prolonged leaching time.

The Al concentration increased steadily from 0 to about 4000 ppm over the period of 5 hours. The main source of Al is the dissolution of alumina-silicate material due to decomposition of clays during roasting. The high concentration of Al may affect the chemical consumption in the

downstream process. By shortening the leaching retention time, the Al concentration in PLS can be reduced, however, the concentration of Fe increases. Therefore, there is a balance between the Al leaching and Fe precipitation. The Al dissolution consumed $H^+$ ion and released $OH^-$, then the Fe consumed $OH^-$ ion and formed jarosite.

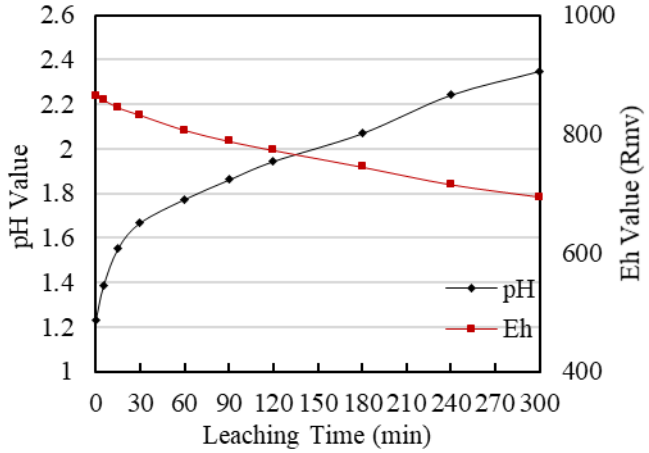



Figure 3.6.56. Eh and pH value of leaching experiment XBL-BA2001. (Lixiviant: Bioleach acid; Feed solid: 2 mm top size roasted at 600 °C for 20 minutes; S/L:1/5 w/v; Leaching temperature: 75 ℃).

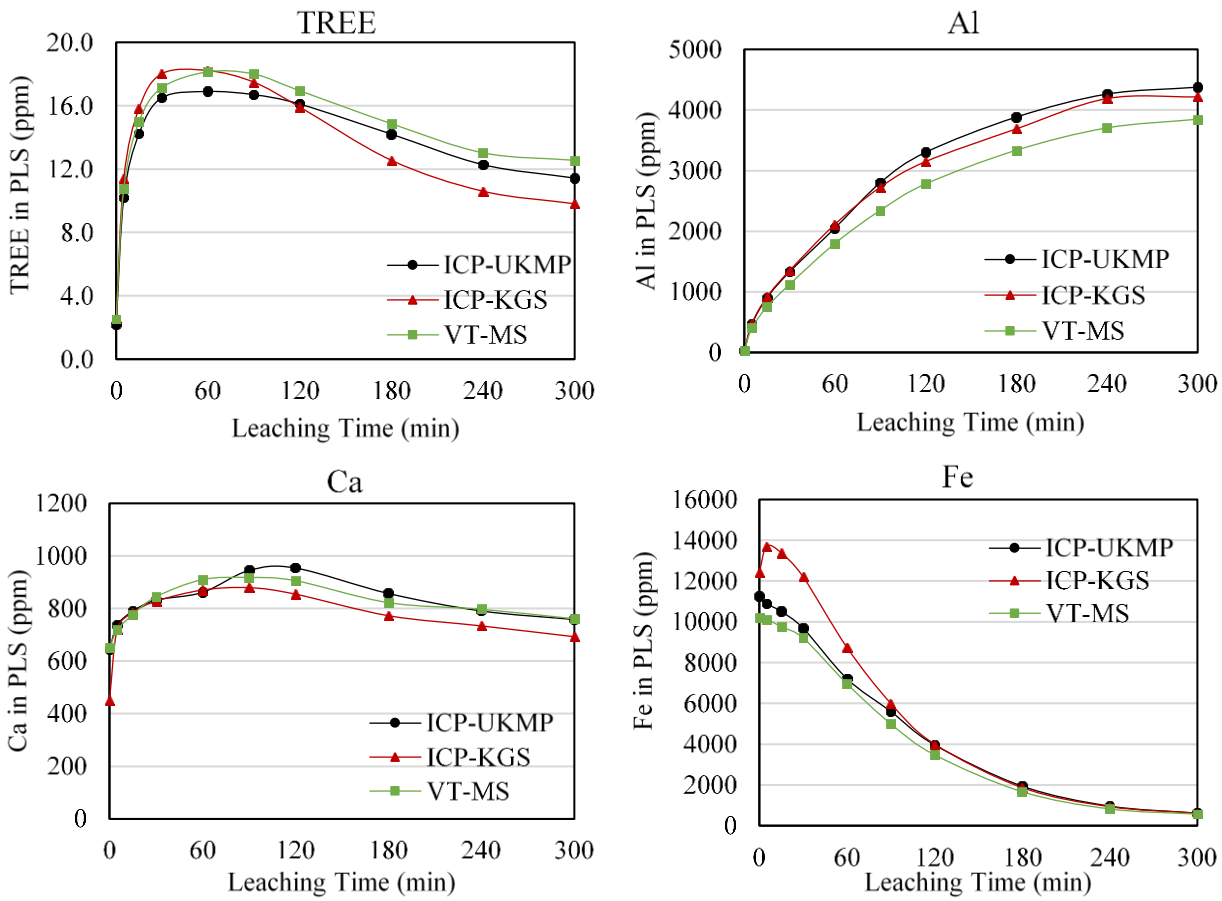



Figure 3.6.57. Concentration of TREEs, Al, Ca, and Fe in pregnant leachate solution (PLS). (Test XBL-BA2001. Lixiviant: Bioleach acid; Feed solid: 2 mm top size roasted at 600 °C for 20 minutes; S/L:1/5 w/v; Leaching temperature: 75 °C.)

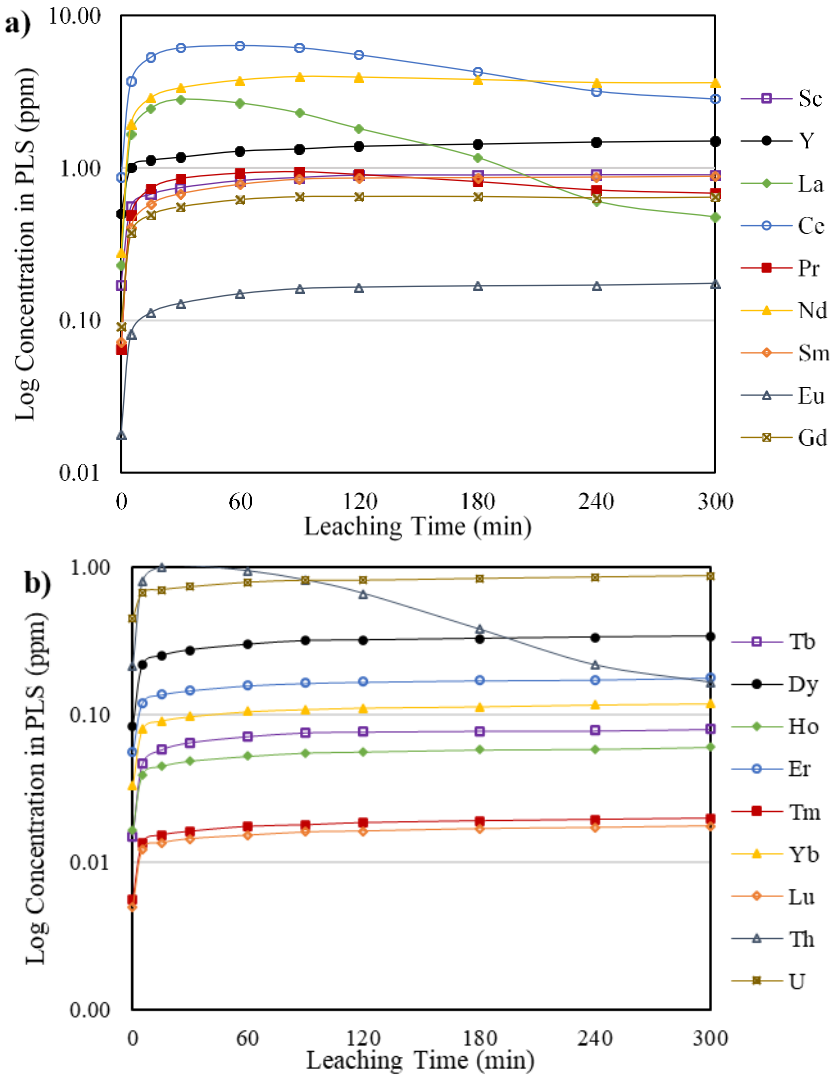



Figure 3.6.58. Concentration of REEs in PLS a) scandium to gadolinium; b) terbium to uranium. (Analyzed by ICP-MS at Virginia Tech)

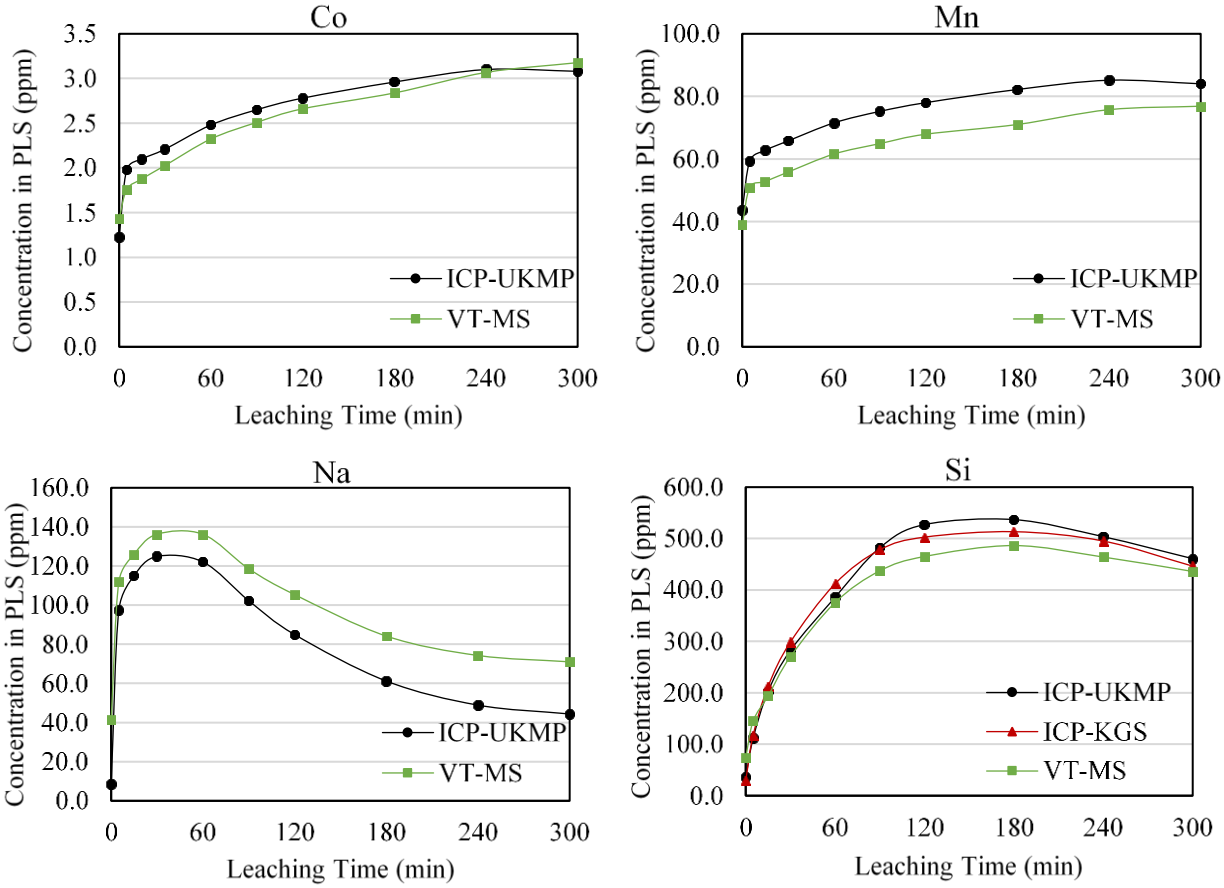



Figure 3.6.59. Concentration of selective elements in PLS.

**Test XBL-BA2002:** To examine the effect of particle size and roasting condition, leaching experiment was conducted on material pulverized to 80 mesh (177 microns) top size and roasted at 600 °C for 2 hours in a muffle furnace. Compared to test XBL-BA2001, the end pH is slightly higher due to the leaching kinetics. The Eh and pH value were recorded throughout the leaching test as shown in Figure 3.6.60. The pH increased from 1.248 to 2.499 over a period of 5 hours leaching time, whereas the Eh decreased from 892.2 RmV to 721.7 RmV. The end pH was slightly higher than that of the previous test. The ending Eh was higher as well, however, the concentration of Fe was much lower (Fe~200 ppm) than that of the previous test (Fe~500 ppm) which indicated that majority of the Fe was in ferric state. This is possibly because in the test BA2001, the pyrite was not completely roasted due to the coarser particle size and shorter roasting time. The remaining pyrite reacted with the ferric ion in the solution and convert part of the Fe into ferrous state which would precipitate at higher pH than ferric ions.

The elements Ce, La, and Th were still showing a decrease with prolonging the leaching time. The peak recovery for these elements occurred at 15 minutes of leaching time, which indicated that the finer particle size and advanced roasting provided fast leaching kinetics (Figure 3.6.62). The absolute concentration of light REEs are slightly higher than those of the previous tests, whereas the heavy REEs and critical elements (i.e. Sc, Co, and Mn) are not much improved (Figure 3.6.62 and Figure 3.6.63).

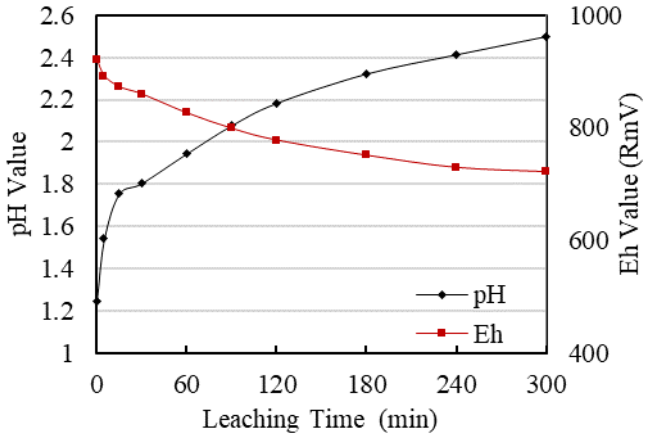



Figure 3.6.60. Eh and pH value of leaching experiment XBL-BA2002. (Lixiviant: Bioleach acid; Feed solid: 177 microns top size roasted at 600 °C for 2 hours; S/L:1/5 w/v; Leaching temperature: 75 ℃.)

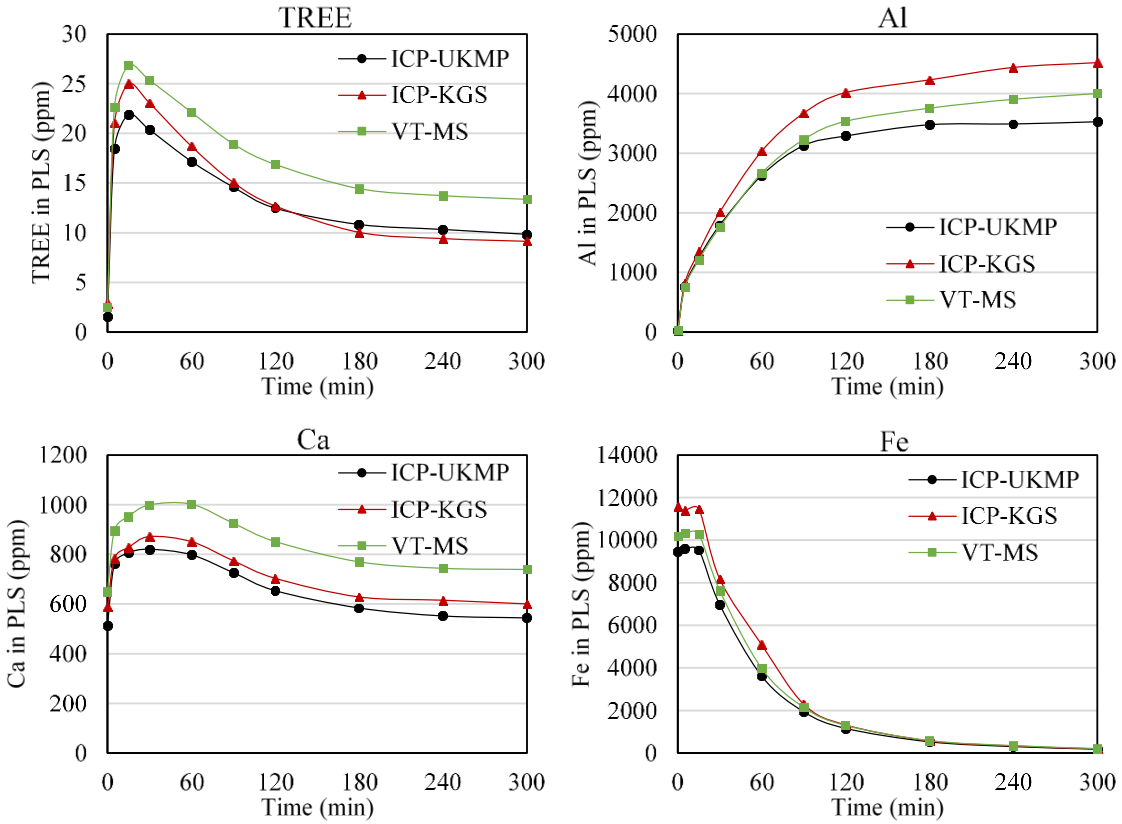



Figure 3.6.61. Concentration of TREEs, Al, Ca, and Fe in pregnant leachate solution (PLS). (Test XBL-BA2002. Lixiviant: Bioleach acid; Feed solid: 177 microns top size roasted at 600 °C for 2 hours; S/L:1/5 w/v; Leaching temperature: 75 ℃.)

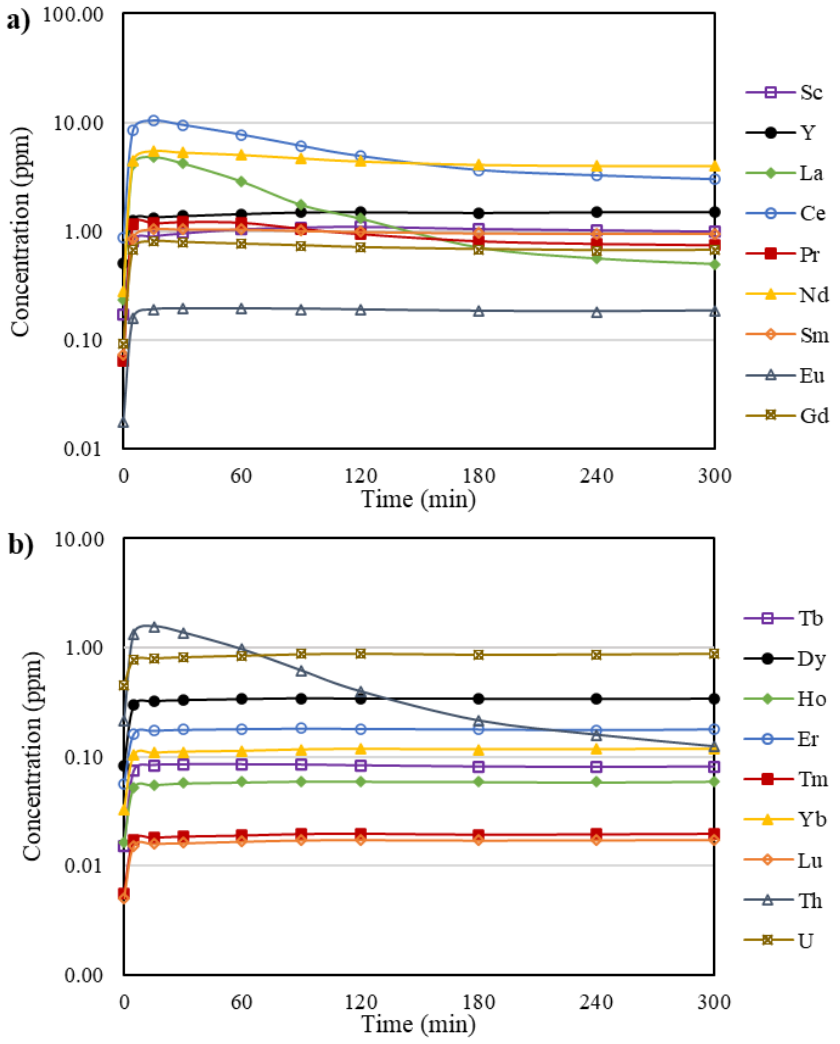



Figure 3.6.62. Concentration of REEs in PLS a) scandium to gadolinium; b) terbium to uranium. (Analyzed by ICP-MS at Virginia Tech).

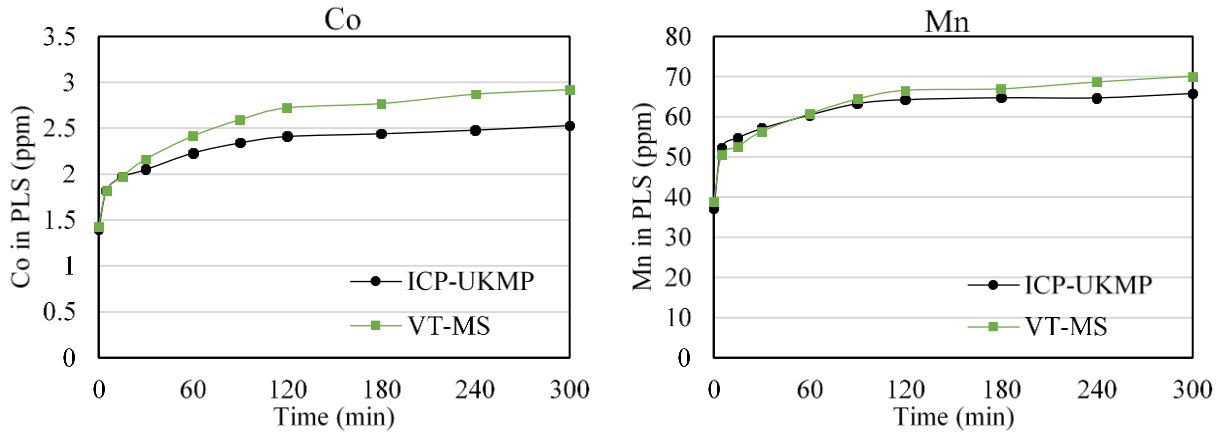



Figure 3.6.63. Concentration of Co, and Mn in PLS.

**Mass Balancing:** Since the bioleach acid contains trace amount of REEs and critical elements, a mass balancing calculation sheet was developed to calculate the leaching recovery through balancing the total mass of each element in feed solid and bioleach acid (lixiviant) with the total mass of the element in solid residue and pregnant leachate solution, with minimize the change of assays in feed solid, bioleach acid, leachate and solid residue streams.

The recovery of each element was calculated using the formula below:

$$\text{Recovery (\%)} = \frac{c_{PLS} * V_{PLS} - c_{BA} * V_{BA}}{m_{FD}}$$

where, $c_{PLS}$ is the concentration of element in PLS, $V_{PLS}$ is the volume of PLS, $c_{BA}$ is the concentration of element in bioleach acid, $V_{BA}$ is the volume of bioleach acid. The constraint to be fulfilled in the mass balancing data adjustment is:

$$m_{FD} = m_{SR} + m_{PLS} - m_{BA}$$

where, $m_{FD}$ is the mass of element in feed solid, $m_{SR}$ is the mass of element in solid residue, $m_{PLS}$ is the mass of element in leachate, $m_{BA}$ is the mass of element in bioleach acid. By minimizing the weighted sum of square (WSSQ) of the acceptable relative error for all the assays:

$$\text{WSSQ} = \sum_{i=1} \sum_{k=1} \frac{(c_{i,k}{}^{*} - c_{i,k})^2}{(e\% * c_{i,k})^2}$$

where, $c_{i,k}$ is the concentration of element in stream *i* for element *k*, $c_{i,k}{}^{*}$ is the concentration of element in stream *i* for element *k* after adjustment. The acceptable relative error *e%* used in this calculation is 5%.

The recovery of each elements was calculated using the mass balancing data adjustment sheet for the two leaching experiments mentioned above. The percentage recovery of each REE by time is plotted for test BA2001 and BA 2002 as shown in Figure 3.6.64 a) and b). The peak recovery of leaching 2 mm top size material roasted at 600 ℃ for 20 minutes reached 24.6% at 1 hour of leaching time. With decreasing the particle size to 80 mesh (177 microns) and 2 hours roasting, the peak recovery of TREEs achieved 38.3% after 15 minutes of leaching time. The improvement of leaching recovery was contributed by light REEs (Ce, La, and Nd, etc.). The ending leaching recovery at 5 hours for test BA2001 and BA2002 was 15.8% and 17.1%, which is not very significantly different. By reducing the particle size and advancing the roasting time, the leaching recovery of heavy REEs improved, especially for the valuable elements such as Gd (33.42% vs. 37.58%), and Dy (29.12% vs. 37.12%).

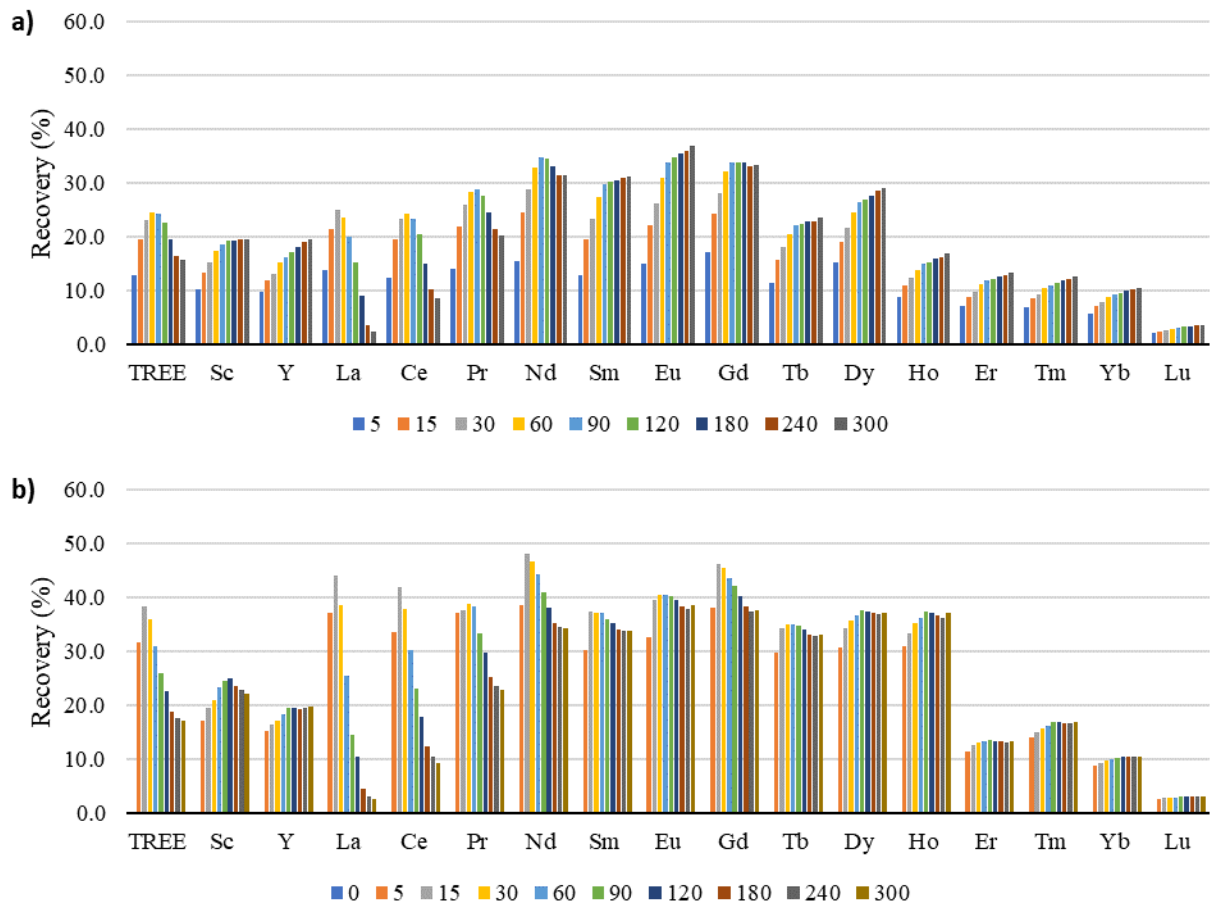



Figure 3.6.64. Recovery of each REE for a) test XBL-BA2001, and b) test XBL-BA2002.

**Solid Residue:** The weight of leaching solid residue increased by 12% and 13% in both leaching experiments after 5 hours. The solid was dried and analyzed using XRD to identify the composition of the material. The XRD detected strong jarosite peaks in both solid residue material after leaching (Figure 3.6.65). The quantitative analysis showed jarosite content in BA2001 and BA2002 tests solid residue is 34% and 36%, respectively.

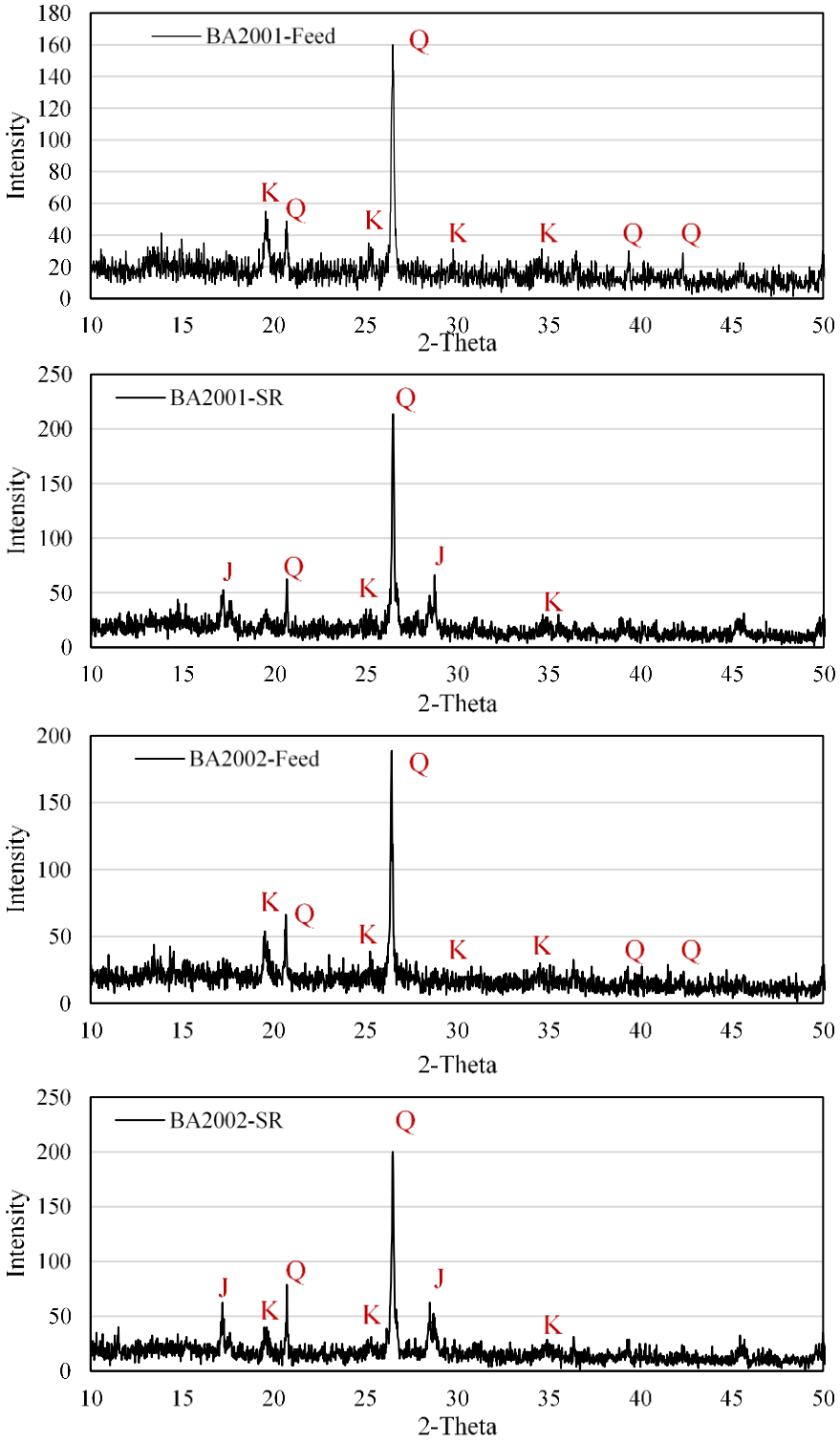



Figure 3.6.65. XRD analysis of solid material before and after leaching. (Q-quartz, K-potassium aluminum silicate, J-jarosite)

### *3.6.11 Counter-Current Leaching*

The results of the bioleach acid leaching tests indicated that prolonged leaching time caused formation of jarosite and RE loss due to Fe precipitation. The concept of counter current leaching was proposed to examine the potential of extracting RE from spent solid residue using fresh bioleach acid. The peak RE recovery occurred after around 30 minutes of leaching, therefore, the residence time within each individual stage of leaching was set to be 30 minutes. The bench scale experiments were designed to simulate leaching process in 4 scenarios:

- L1: Fresh solid with Fresh Acid
- L2: Leached solid with Fresh Acid
- L3: Fresh Solid with Spent Acid
- L4: Leached Solid with Spent Acid

**Material and Methodology:** The solid feed material used in the counter current leaching test is the hammer milled product (top size 2 mm) from the West Kentucky No. 13 coarse refuse weathered material roasted at 600 °C for 20 minutes in a muffle furnace. The first stage leaching (L1) utilized 200 ml of fresh bioleach acid and 40 gram of fresh solid leaching for 30 minutes at 75°C. The slurry was filtered on a vacuum filter to separate solid and liquid. The pregnant leachate solution (PLS) collected from the L1 stage leaching was measured to be 180 ml, which indicated that approximately 20 ml of PLS was lost in the filter cake. The filtered solid residue cake with some residue leachate in the cake was transferred to the next leaching stage (L2) using 180 ml of fresh bioleach acid solution preheated at 75°C and reacted for 30 minutes. The L2 leachate collected from filtration was 165 ml which indicated that 15 ml was lost in the solid residue. The L2 PLS was saved for next stage leaching. The solid residue was then washed using DI water, incremental volume of wash water was collected for elemental analysis. In the third leaching stage (L3), 40 grams of fresh solid feed was added into the 180 ml of L1 PLS solution and leached at 75°C for 30 minutes. The last leaching stage used the PLS solution collected from L2 and the solid residue cake from L3 at 75°C for 30 minutes. A schematic of the counter current leaching test was summarized in Figure 3.6.66. Sample collected from each leachate and solid stream were subject to elemental analysis using ICP-OES. The feed solid and leached solid sample were digested using modified ASTM D6357-11 method. Due to the high concentration of Fe in the leachate solution, the solution sample was diluted 10, 100, and 1000 times to match with the ICP-OES calibration ranges.

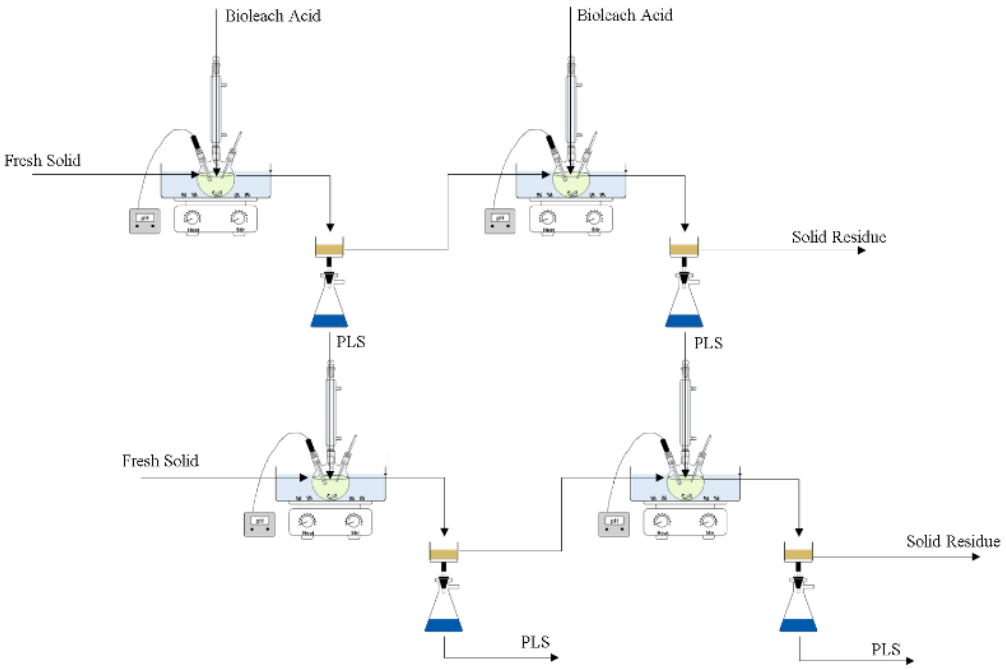



Figure 3.6.66. Schematic of the Lab scale counter current leaching test.

**Results and Discussion:** The total rare earth content and the concentration of Al, Ca, and Fe measured in each stream in the counter current leaching test are listed in Figure 3.6.67. The mass of TREE, Al, Ca, and Fe in each stream was balanced using the minimization of weighted sum-of-square method. Since the filter cakes obtained from first stage and third stage leaching were directly feed into the next stage leaching, the elemental concentration of the cakes were not measured. The two midpoint between L1 to L2 and L3 to L4 were calculated based on the mass balance.

In the first stage leaching (L1), the pH and Eh value after 30 minutes leaching were 1.644 and 728.8 RmV, respectively. The L1 leaching stage picked up about 13 ppm of TREE, 1500 ppm of Al, 230 ppm of Ca, without losing much Fe due to precipitation. The solid residue was then re-leached using fresh bioleach acid in the second stage leaching (L2). Another 5 ppm of TREEs was extracted from the leached solid with similar amount of Al and Ca. The ending pH of L2 was lower than that of the L1 which was 1.548. The filtered solid residue was rinsed using DI water for 3 times until the wash water contains no measurable amount of elements. 86 ml of wash water was consumed which contained about 2.2 ppm of TREEs, 450 ppm of Al, 460 ppm of Ca, and 1900 ppm of Fe. The washed solid residue weight was not significantly changed, however, the content of Fe increased from 4.3% in the feed to about 5.9% in the solid residue indicating that small amount of Fe was transferred from the bioleach acid solution to be solid precipitate form.

The leachate filtered from the L1 leaching stage contains 15 ppm of TREEs, which was then used to leach fresh solid. The pH increased from 1.64 to 1.87 after leaching resulting in about 4000 ppm Fe concentration reduction. The L3 leachate picked up another 10 ppm of TREE and 1000 ppm of Al, and 50 ppm of Ca. The low recovery of Ca in this stage is likely indicating that the Ca concentration in solution is close to the saturation point. The leached solid residue was then transferred to the next stage leaching (L4) using the spent bioleach acid collected from L2 stage.

The end pH of L4 is lower than L3 at 1.74. The concentration of TREEs in L2 PLS increased from 7.4 ppm to 12.8 ppm with 1200 ppm Al and 20 ppm of Ca elevated in L4 PLS. The Fe concentration reduced in a similar pattern as in L3. The solid residue after rinsing and drying contains about 8.0% Fe which is 3.7% higher than the original feed solid.

The leaching recovery of each stage was calculated using the mass balanced data. The leaching recovery of each stage was plotted in Figure 3.6.68. The positive sign on the y-axis indicates the leaching of elements, whereas, the negative sign on the y-axis shows the precipitation of the elements when the element concentration in the leachate is less than the feed lixiviant and the element concentration in the solid residue is higher than the feed solid. The leaching recovery of TREEs is about 20% while leaching using fresh solid and fresh bioleach acid. The second stage leaching using fresh bioleach acid can extract about 9.8% more TREE from the leached solid. Using the spent acid to leach fresh solid, about 15.6% of TREEs can be extracted in 30 minutes. And 10% more of TREEs can be extracted while using spent acid and leached solid which represents the middle stage of counter current leaching circuit. Compared to one stage leaching, extra 30 minutes of leaching time can only increase about 1.6% of recovery. In the meanwhile, the counter current leaching also increased the leaching recovery of Al and Ca. The schematic of 3 stage counter current leaching is shown in Figure 3.6.69, the total recovery of the proposed circuit is estimated to be 31.77% after 3 stage of 30 minutes leaching per stage, whereas the leaching recovery of one stage leaching for 90 minutes is 24.38%.

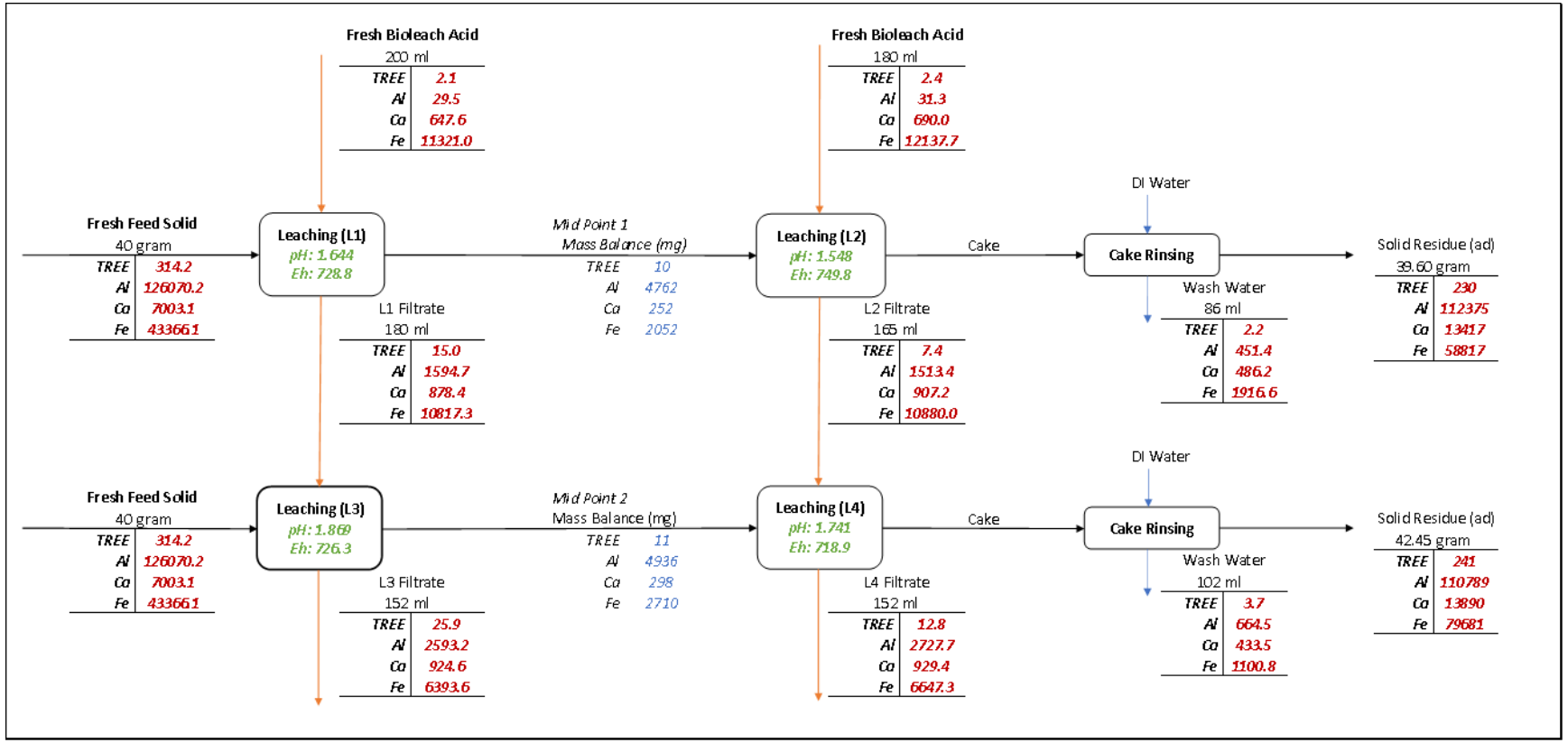



Figure 3.6.67. Mass balance and element al concentration in each stream in the counter current leaching circuit




Figure 3.6.68. Comparison of leaching recovery of major elements of interest in counter current leaching (a) and one stage leaching (b).

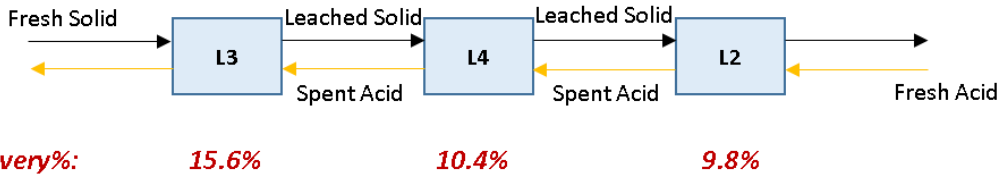



Figure 3.6.69. Schematic and estimated stage recovery of 3 stage counter current leaching (West Kentucky No. 13 roasted material and bioleach acid).

### *3.6.12 Parametric study on leaching performance using bioleach acid*

In the previous study, leaching experiments were carried out using the acid solution generated from the bioleach reactor received from University of Utah. The results showed that the leaching recovery of REEs reached peak in the first hour of leaching reaction then slowly decreased due to co-precipitation of REEs with the formation of iron hydroxide sulfate and calcium sulfate in solution. The formation of precipitates is due to the increasing of pH during leaching while dissolving solid materials. The co-precipitation of REEs can occur by adsorption onto the surface of dominate precipitate. The high concentration of Fe and Ca caused the precipitation pH of REEs to shift to the lower pH range. In addition, the high ionic strength in the leaching solution can deteriorate the REEs leaching performance by decreasing the hydrogen ion activity. To evaluate and optimize the leaching performance, the bioleach acid was diluted with water using different volumetric ratios to reduce the ionic concentration of the lixiviant. The effect of acid concentration, solid concentration, solid to liquid ratio, and residence time in the leaching system are examined.

The test plan designed to investigate the effect of bioleach acid dilution on REEs recovery is shown in Table 3.6.23. The bioleach acid was diluted with deionized water (DI Water) with different volumetric ratio prior to leaching experiments. Previous study showed that the solid concentration in leaching has a significant effect on REE leaching recovery, therefore, the effect of solid-to-liquid ratio using diluted acid with various dilutions was also investigated.

Table 3.6.23. Test plan on bioleach acid dilution effect on REEs recovery.

| Test ID | Bioleach Acid: DI Water (V/V) | S/L (w/V) |
|---|---|---|
| BA2017 | None | 2/10 |
| BA2019 | None | 3/10 |
| BA2020 | None | 1/10 |
| BA2021 | 1:1 | 2/10 |
| BA2022 | 1:1 | 3/10 |
| BA2025 | 3:1 | 2/10 |
| BA2026 | 3:2 | 2/10 |
| BA2027 | 1:1 | 1/10 |

The results on the effect of solid-to-liquid ratio using original bioleach acid is reported in this quarter, the results on the tests conducted on evaluation of acid dilution will be reported in the next quarter as the analytical results become available.

**Materials:** The West Kentucky No. 13 coarse refuse material was used to conduct the leaching experiments. The material was prepared in the pilot plant starting with screening at 2-inch using a vibratory dry screen. The undersize material was collected and ground to a top size of 1 mm using a hammer mill. The material was then roasted in the pilot plant using a rotary tube furnace that provided an inner tube temperature of 600 °C with retention time of 20 minutes.

**Procedure:** The leaching experiment was carried out using a 1-Liter triple neck round bottom flask. A total reflux condenser was connected to the middle neck to reflux the water vapor to maintain the total leaching volume throughout the leaching experiments. Water bath was heated at 75 °C using a programable immersion heater to provide a constant temperature of the leaching reactor throughout the test. Agitation of the leaching reactor was provided using a magnetic stir at a rotation speed of 850 rpm with a 1-inch-long egg shape stir.

Lixiviants (original or diluted bioleach acid solution) of 200 ml in volume were introduced to the reactor and heated to 75°C. A thermometer was immerged in the aqueous to monitor the temperature of the slurry. A measured amount of feed solid was added into the reactor with agitation when the temperature reaches the target setpoint. At different time intervals during the leaching experiment, 3-ml of slurry sample was collected into a 15-ml centrifuge tube and immediately centrifuged at 4000 rpm for 5 minutes to separate the liquid and solid. Then, 1-ml of the effluent was removed from the top of the liquid without removing the solid in the bottom and mixed with 9-ml of 5% nitric acid to stabilize and pre-dilute for ICP analysis. The sample was further diluted using proper dilution factor to be analyzed in the ICP-OES at the University of Kentucky pilot plant. Quality control was performed on a daily basis to ensure the analytical accuracy and the calibration quality of the ICP instrument. The solid residue sample was collected at the end of the experiments and filtered using a vacuum filter. The pregnant leachate solution (PLS) filtrate was collected for further precipitation test using calcite. The solid residue was washed with DI water to rinse off the residual PLS entrained in the cake then dried in the oven for 24 hours at 65°C. The solid sample was digested using a modified ASTM method to ensure complete dissolution of the solids then analyzed using an ICP-OES unit at the Kentucky Geological Survey and the UK pilot plant. The Eh, pH, and temperature during the leaching experiment was measured using a Thermoscientific Orion Star multi meter and the value was recorded at different time interval while the sample was taken.

**Results and Discussion:** The solution pH of the original acid solution obtained from the bioreactor was around 1.1 to 1.2. During leaching reaction, the solution pH slowly increased with time due to the dissolution of solid material as shown in Figure 3.6.70. Using S/L of 1/10 (weight by volume) in leaching system resulted in minor pH variation from 1.1 at the beginning of the leaching test to 1.7 at the end of 5 hours leaching. The ending pH of leaching tests with S/L ration of 2/10 and 3/10 were 2.3 and 2.7, respectively. Similarly, the lower solid concentration resulted in the least change in Eh values throughout the tests. Eh value is an indirect response of the $Fe^{3+}$ concentration in solution. As the leaching reaction proceed, the dissolved solid material consumed $H^{+}$ ions in solution, as a result, the increasing pH caused ferric ion to precipitate as a ferric hydroxide sulfate material. The Eh value (Figure 3.6.70) showed that the Eh value reduced significantly while solid to liquid ratio increased to 2/10 and 3/10. The corresponding Fe concentration at the end of the 5-hour leaching test was 4650 ppm, 522 ppm, and 321 ppm in the pregnant leaching solution obtained from tests S/L ratio of 1/10, 2/10, and 3/10, respectively.

With increasing the solid concentration, the leaching recovery of total REEs (TREEs) decreased due to the change of available $H^+$ in solution. The recovery of TREEs is compared in Figure 3.6.71. The results showed that the peak recovery obtained with 1/10 solid-to-liquid ratio leaching system reached 31% after 1 hour of leaching, whereas the peak recovery for the 2/10 and 3/10 S/L leaching system reached 24% and 18%, respectively, after around 30 minutes of leaching. The pH value is the main key of the leaching performance in the current tested system. The individual REE concentration in the PLS for the three leaching systems with S/L of 1/10, 2/10, and 3/10 is shown in Figure 3.6.72, Figure 3.6.73, and Figure 3.6.74, respectively. The results showed that Ce and La are the two highest concentration elements, which also contributed the most to the REE recovery loss after 1 hour of leaching. The loss of Ce and La in the 1/10 leaching system was about 30% after 5 hours of leaching, whereas the loss was more significant in the higher solid concentration system. The heavy REEs showed not significant loss with prolonged leaching time. Yttrium and scandium recovery increased over time regardless of the solid concentration which indicated that these two elements do not co-precipitate with the dominant precipitate ions, such as Ca and Fe.

Previous study showed that the Fe concentration reduced rapidly and instantaneously after 5 minutes. The leaching recovery of Fe from the solid material is minor and nearly overlapped with the Fe loss due to precipitation. The recovery of major elements of interest are plotted in Figure 3.6.75 to Figure 3.6.77 for the three solid-to-liquid ratios. The recovery of Ca showed similar pattern as TREEs which indicated that the formation of $CaSO_4$ could contribute to the REEs loss during leaching. Moreover, selective co-precipitation is taking place during the leaching and precipitation process. This phenomenon requires further investigation including solution chemistry analysis, surface complexation/adsorption analysis, and the $CaSO_4$ precipitation mechanism. The other major critical element recovery increased over time and showed no potential loss. The increased solid concentration in leaching showed nearly no effect on cobalt leaching recovery, and some minor effect on Mn recovery when the S/L ratio increased to 3/10.

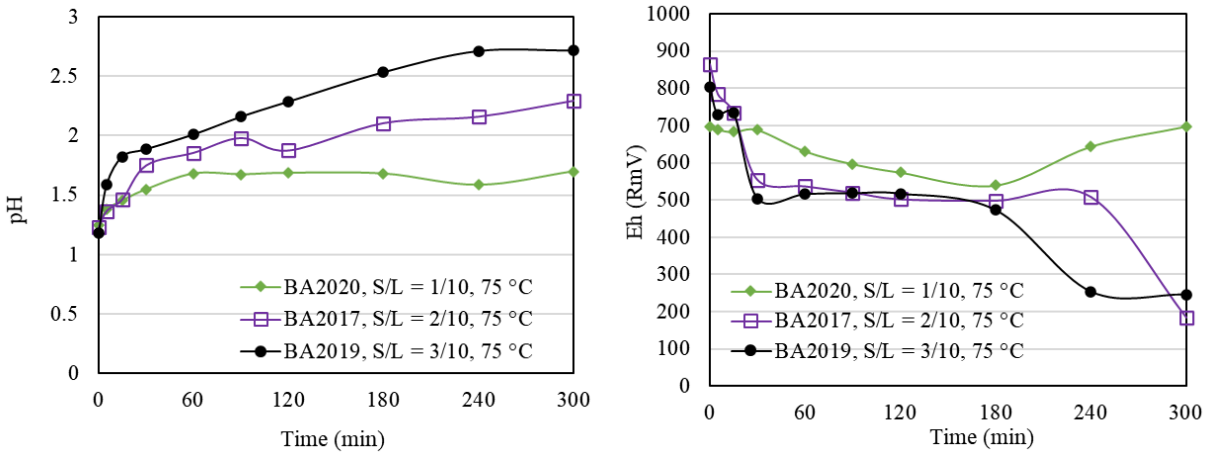



Figure 3.6.70. Recorded value of pH and Eh during leaching test performed over 5 hours.

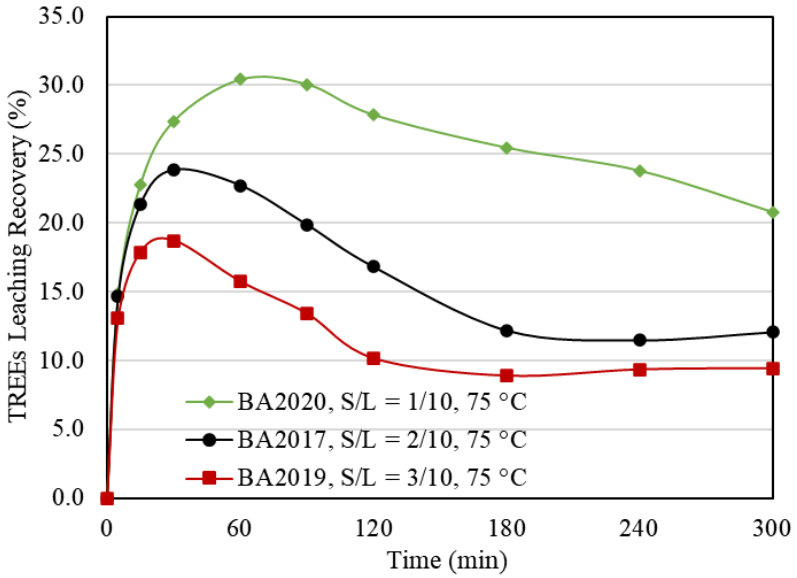



Figure 3.6.71. Leaching recovery of TREEs using bioleach acid under various solid-to-liquid concentration.

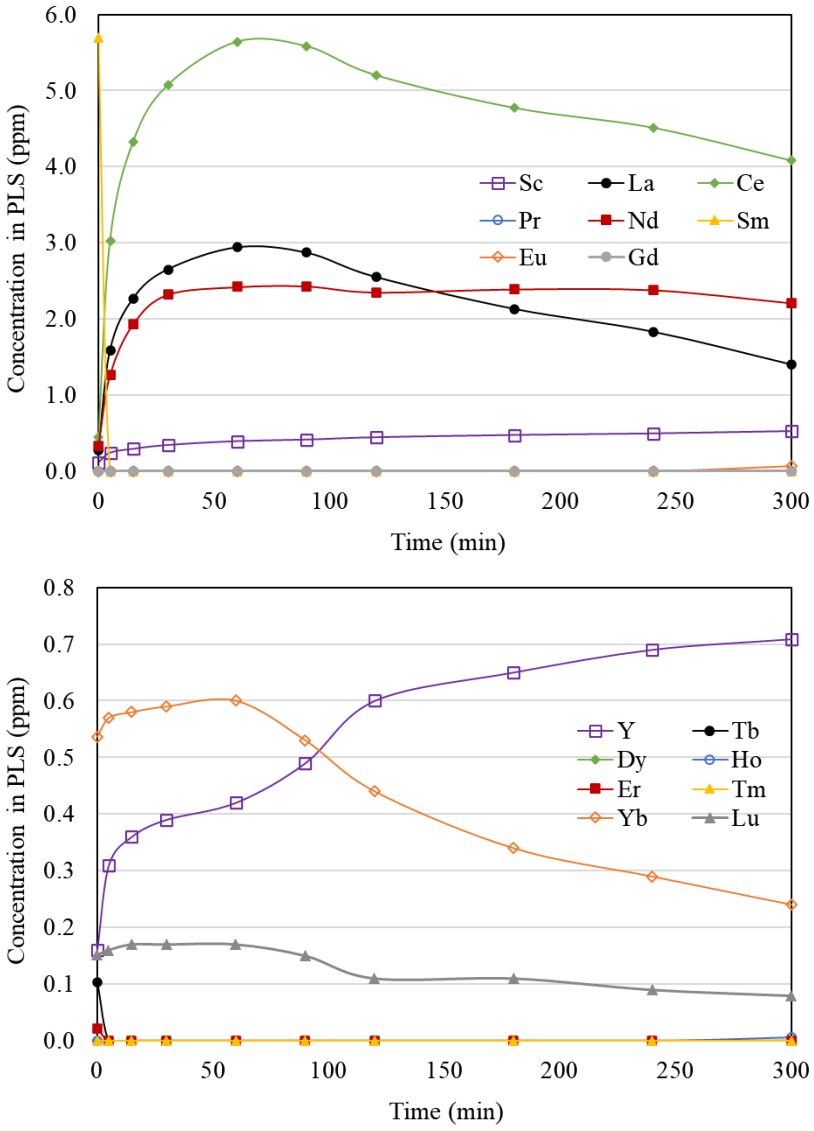



Figure 3.6.72. Concentration of light REEs (top) and heavy REEs (bottom) in PLS solution. (Test BA2020: none diluted acid, S/L=3/10, 75 °C).

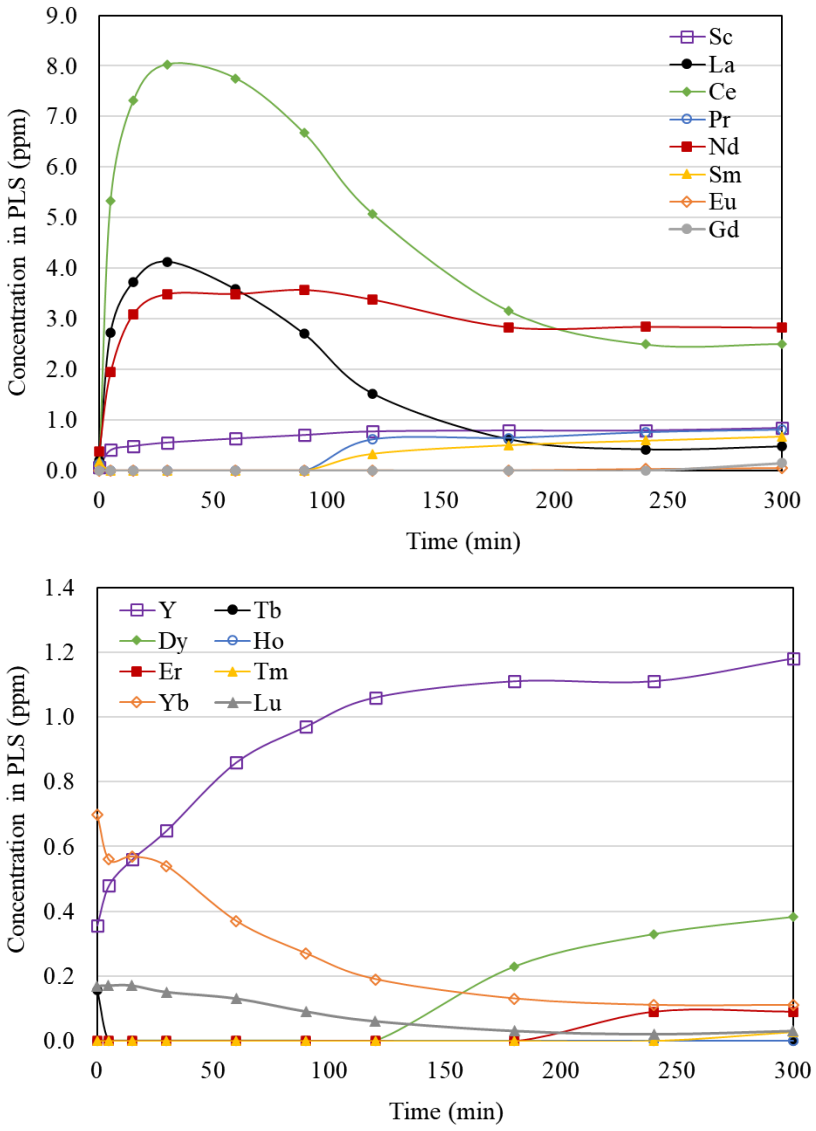



Figure 3.6.73. Concentration of light REEs (top) and heavy REEs (bottom) in PLS solution. (Test BA2017: none diluted acid, S/L=2/10, 75 °C)

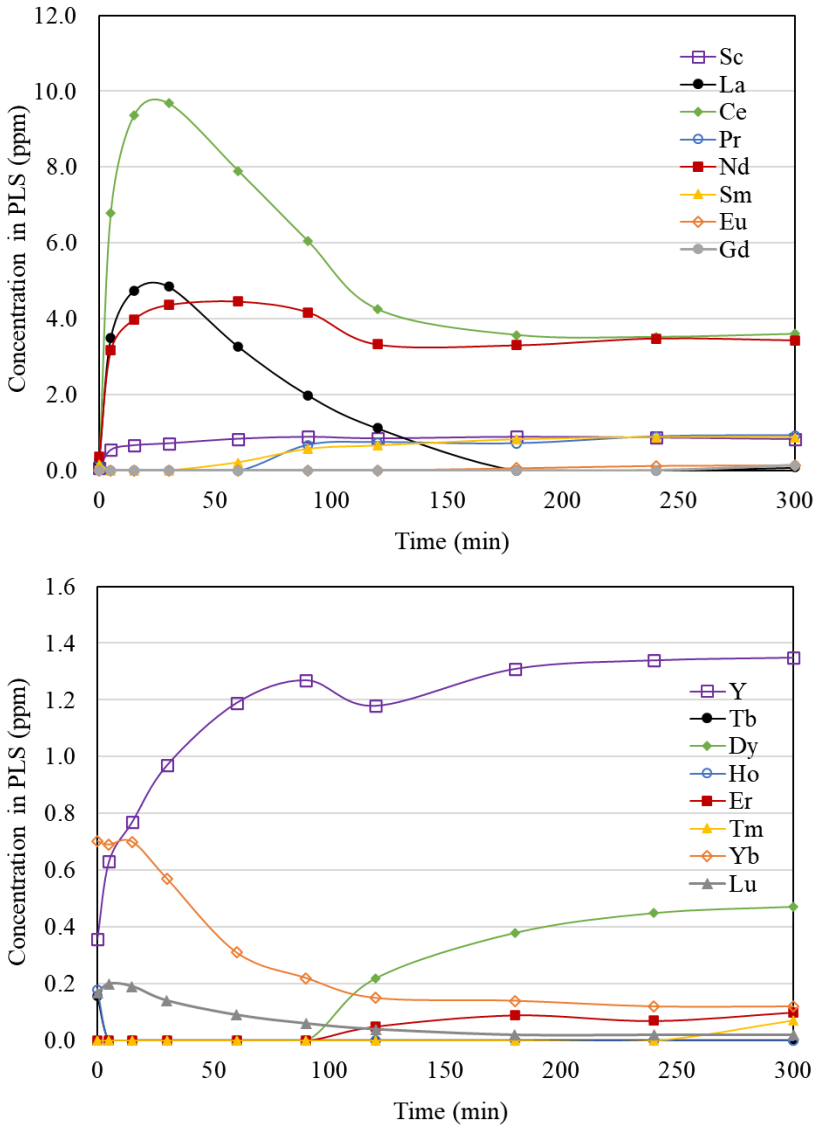



Figure 3.6.74. Concentration of light REEs (top) and heavy REEs (bottom) in PLS solution. (Test BA2019: none diluted acid, S/L=3/10, 75 °C)

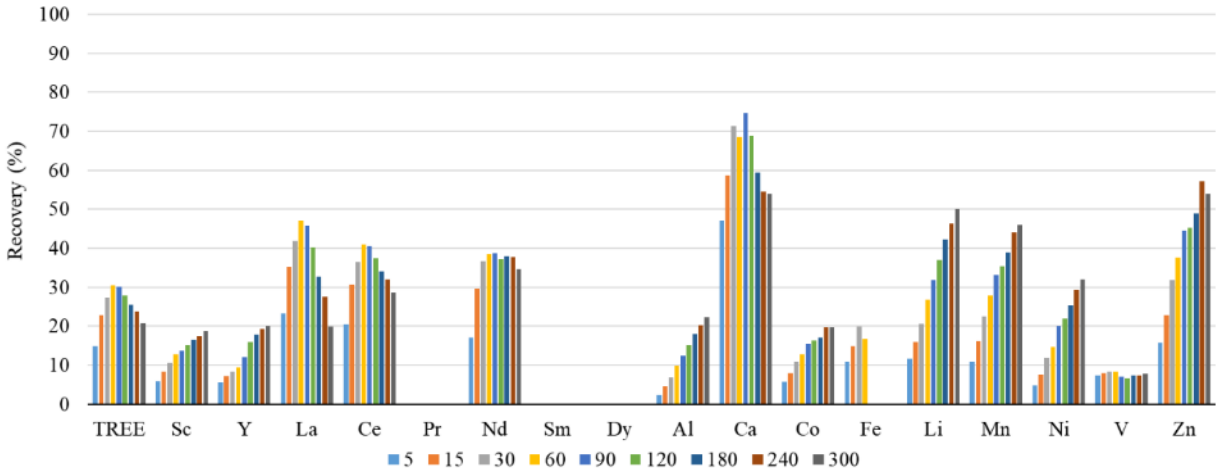



Figure 3.6.75. Leaching recovery of REEs, CMs, and other elements of interest. (Test BA2020: none diluted acid, S/L=1/10, 75 °C)

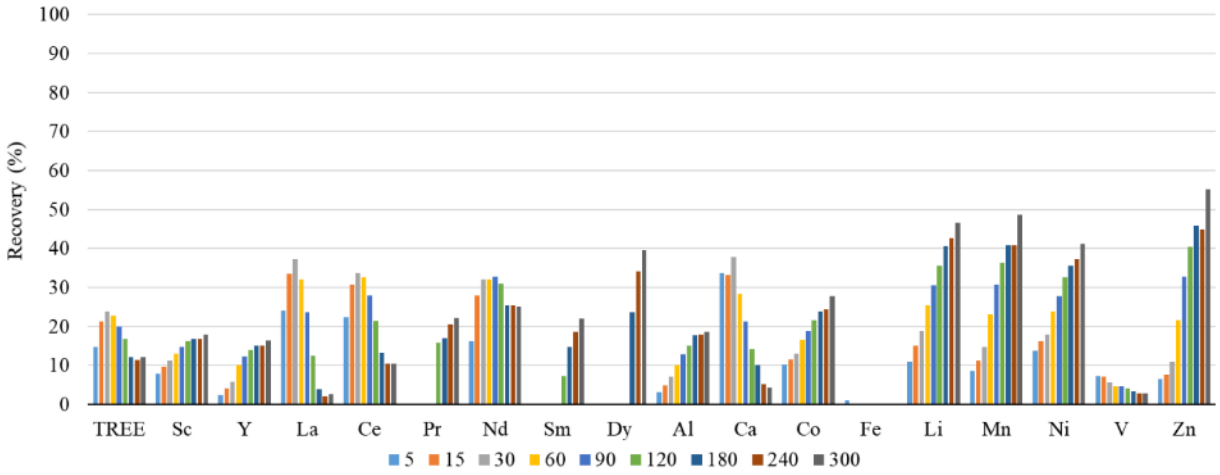



Figure 3.6.76. Leaching recovery of REEs, CMs, and other elements of interest. (Test BA2017: none diluted acid, S/L=2/10, 75 °C).

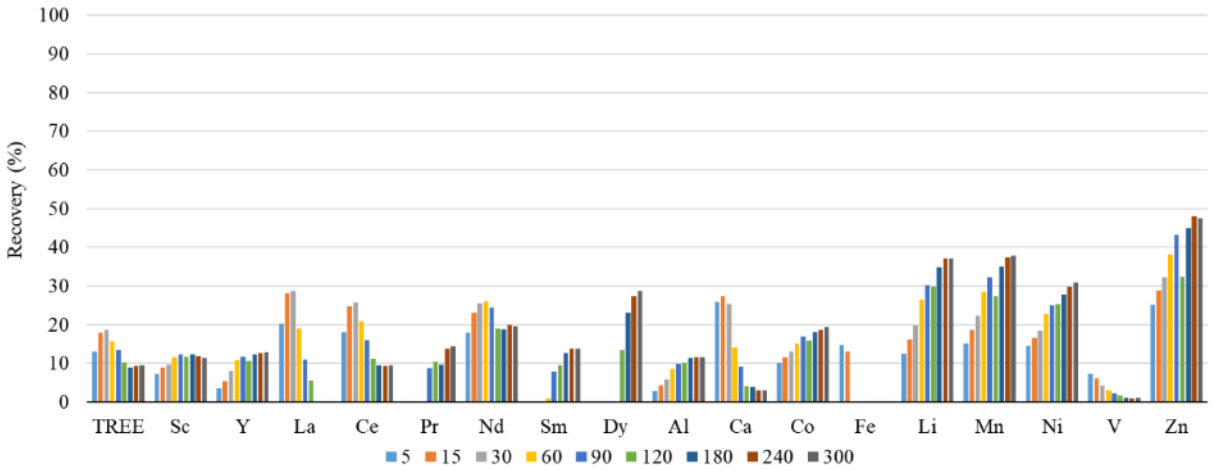



Figure 3.6.77. Leaching recovery of REEs, CMs, and other elements of interest. (Test BA2019: none diluted acid, S/L=3/10, 75 °C).

### *3.6.13 Optimization of leaching temperature using bioleach acid*

Previous study showed that, using bioleach acid, the leaching recovery of REEs increased rapidly at the beginning of leaching then slowly reduced due to the co-precipitation of light REEs (i.e. La and Ce) with the precipitation of jarosite and gypsum. Reducing the solid-to-liquid ratio improved the leaching recovery and postponed and reduced the amount of REEs co-precipitated during leaching. In this section the effect of leaching temperature on REE and critical material recovery was evaluated to optimize the leaching performance and provide a guide to pilot plant operation condition. Leaching experiments were carried out in bench scale using bioleach acid solution. The solid feed material was collected from West Kentucky #13 coarse refuse pile and crushed to a top particle size of 1mm. Then the ground material was roasted in the pilot plant at 600 °C with retention time of 20 minutes. The bioleach acid was received from University of Utah. Leaching experiments were carried out at various temperature ranging from 25°C to 95°C at a solid-to-liquid ratio of 1 to 5 (w/v). Sample was collected as a function of time to evaluate the leaching kinetics. The leachate sample was analyzed at University of Kentucky Center of Applied Energy Research (CAER) using ICP-MS. The solid feed and residue were digested at the Kentucky Geological Survey (KGS) and analyzed using ICP-OES at KGS and the pilot plant.

The results show that leaching at higher temperature did not improve the REE leaching recovery, the peak leaching recovery at 85°C and 95°C was lower than the leaching recovery at 65°C and 75°C, as shown in Figure 3.6.78. The initial reaction rate was nearly identical for leaching reaction at 65°C, 75°C, 85°C, and 95°C. However, the precipitation of REEs occurred at much earlier time with 85°C and 95°C leaching. Interestingly, the co-precipitation of REEs did not occur while leaching at lower temperature, i.e. 25°C and 45°C. The initial leaching kinetic rate at lower temperature was much slower. However, for 25°C leaching, the leaching reaction did not achieve equilibrium even after 5 hours of leaching.

The leaching pH has a significant effect on leaching recovery and is the main drive to the leaching reaction. At lower temperature, such as 25°C, the pH increased merely from 1.2 to 1.4 after 5 hours

of leaching, which provided enough hydrogen ion concentration to drive the reaction forward even after 5 hours. However, higher temperature expedited the dissolution of contaminant minerals which increased the system pH at much faster rate. As shown in Figure 3.6.79, the leaching pH reached 1.75 after 1 hours of leaching at 75°C where the peak recovery was achieved, whereas, at 85°C leaching, the pH reached 1.75 within 15 minutes. The peak recovery for 75°C, 85°C, and 95°C occurred at 60 min, 30 min, and 15 min, respectively, with similar corresponding pH. It indicated that the co-precipitation of REEs is largely controlled by solution pH.

An interesting observation is that the REEs co-precipitation process is more selective with light REEs. The order of co-precipitation efficiency is La> Ce> Pr>Nd as shown in Figure 3.6.80 and Figure 3.6.81. It is possibly due to the affiliation with gypsum formation as these rare earth ions are structurally incorporated in gypsum where three $Ca^{2+}$ ions are replaced by two trivalent rare earth ions and a vacant $Ca^{2+}$ lattice site. Further studies will be carried out to investigate this phenomenon.

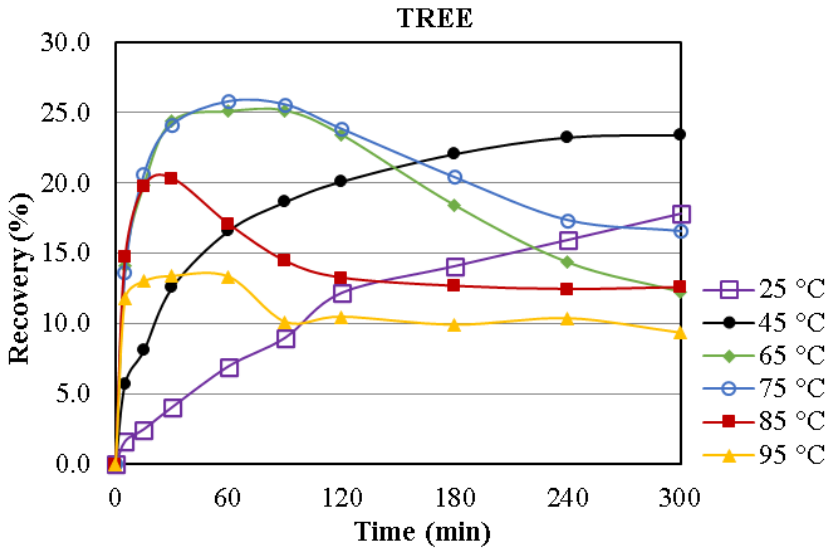



Figure 3.6.78. Leaching recovery of total REEs at various temperature using bioleach acid. (Feed solid: WKY No.13 coarse refuse, S/L=1/5)

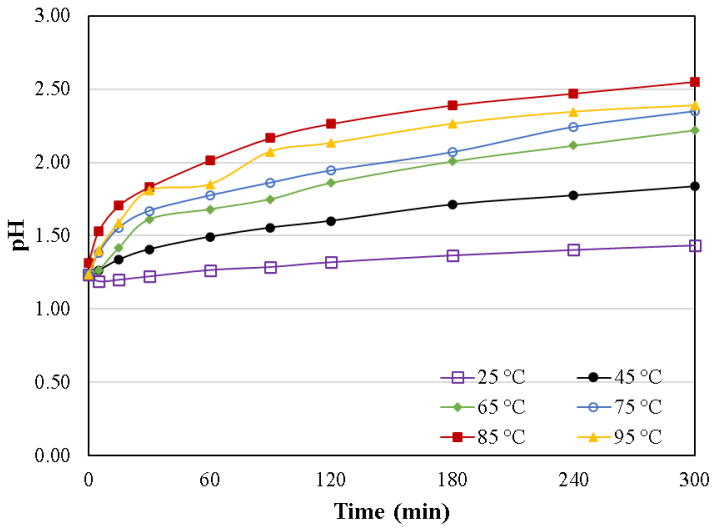
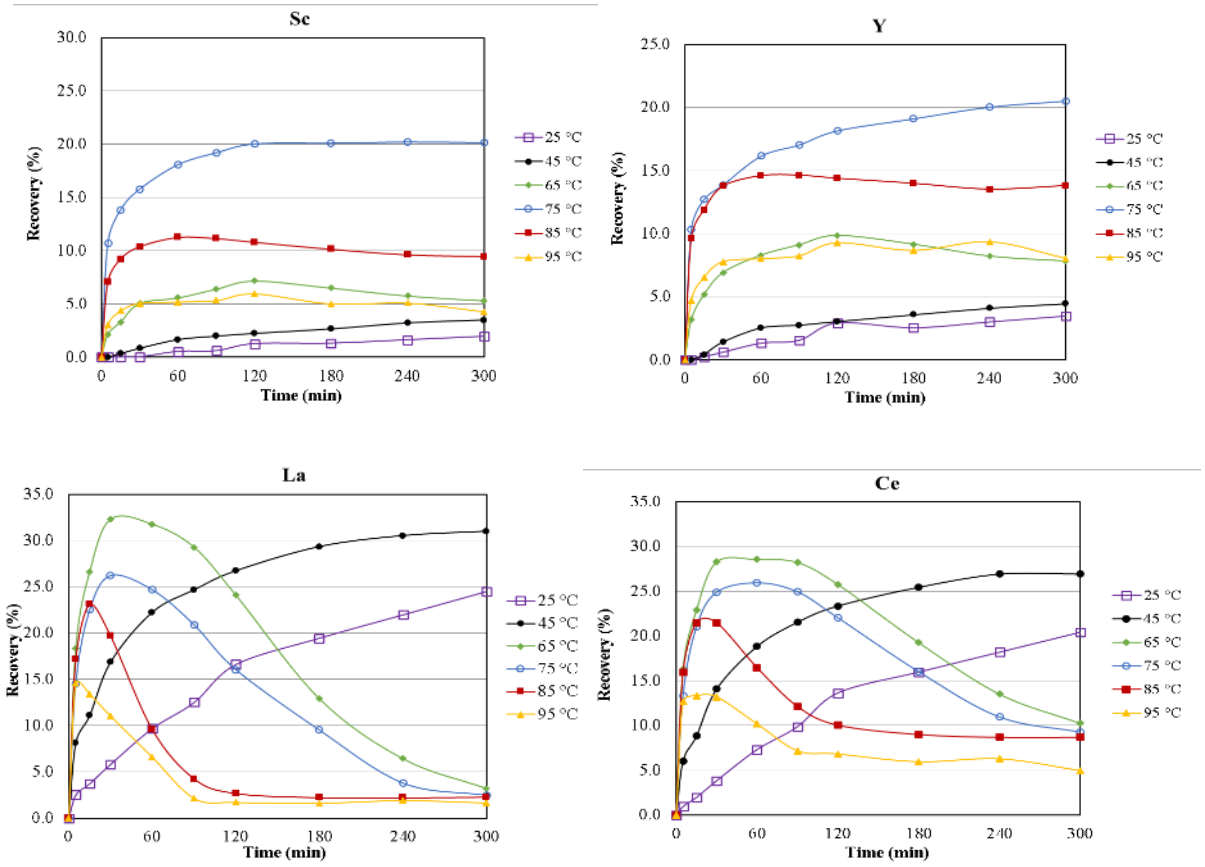



Figure 3.6.79. Measured pH of leaching system at various temperature using bioleach acid. (Feed solid: WKY No.13 coarse refuse, S/L=1/5)



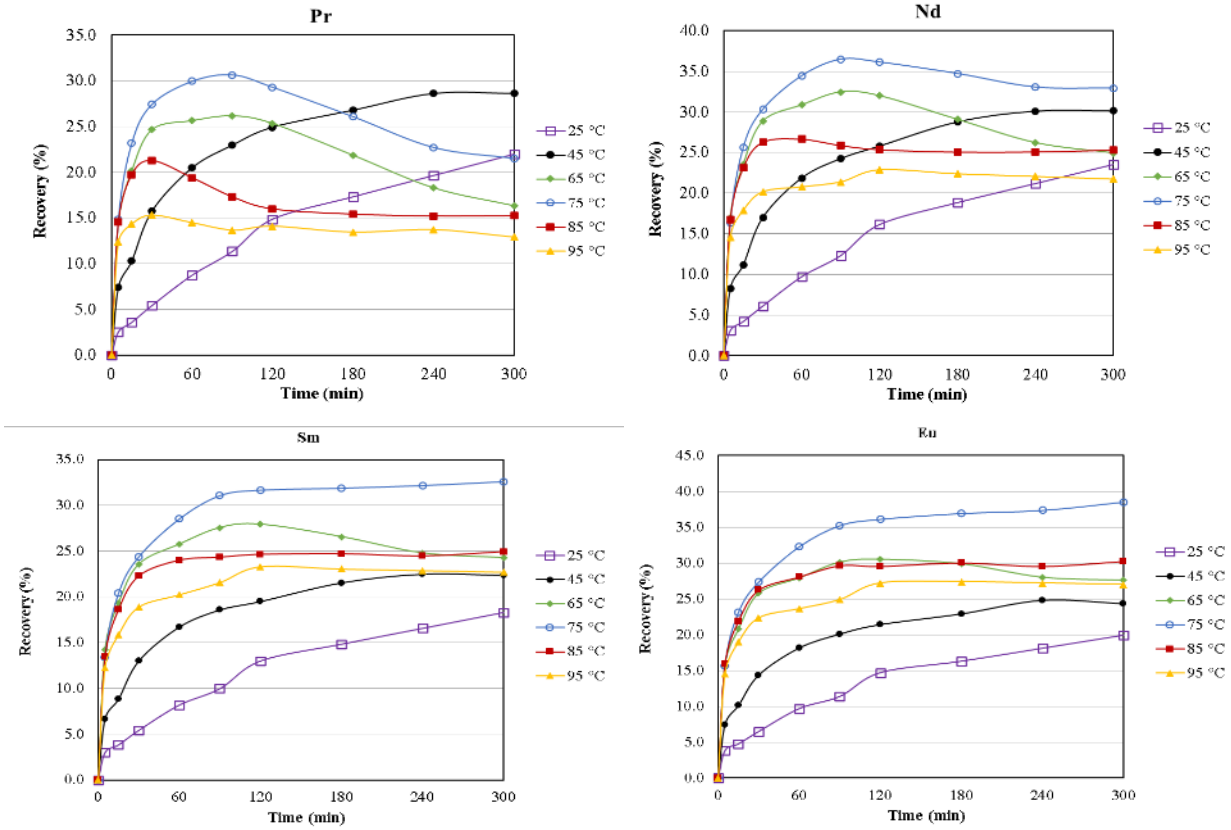
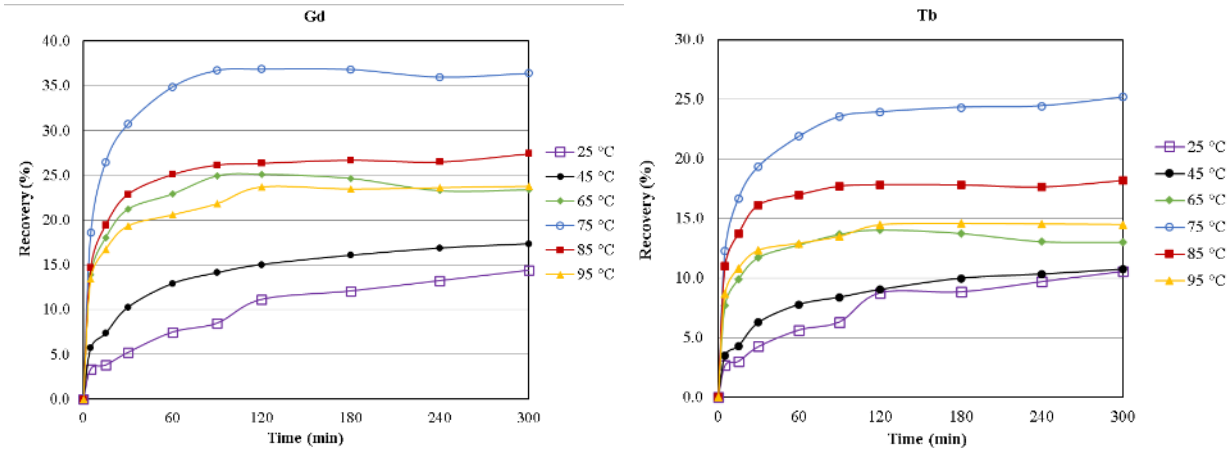



Figure 3.6.80. Effect of temperature on leaching recovery of individual REEs (Sc, Y, and LREEs).



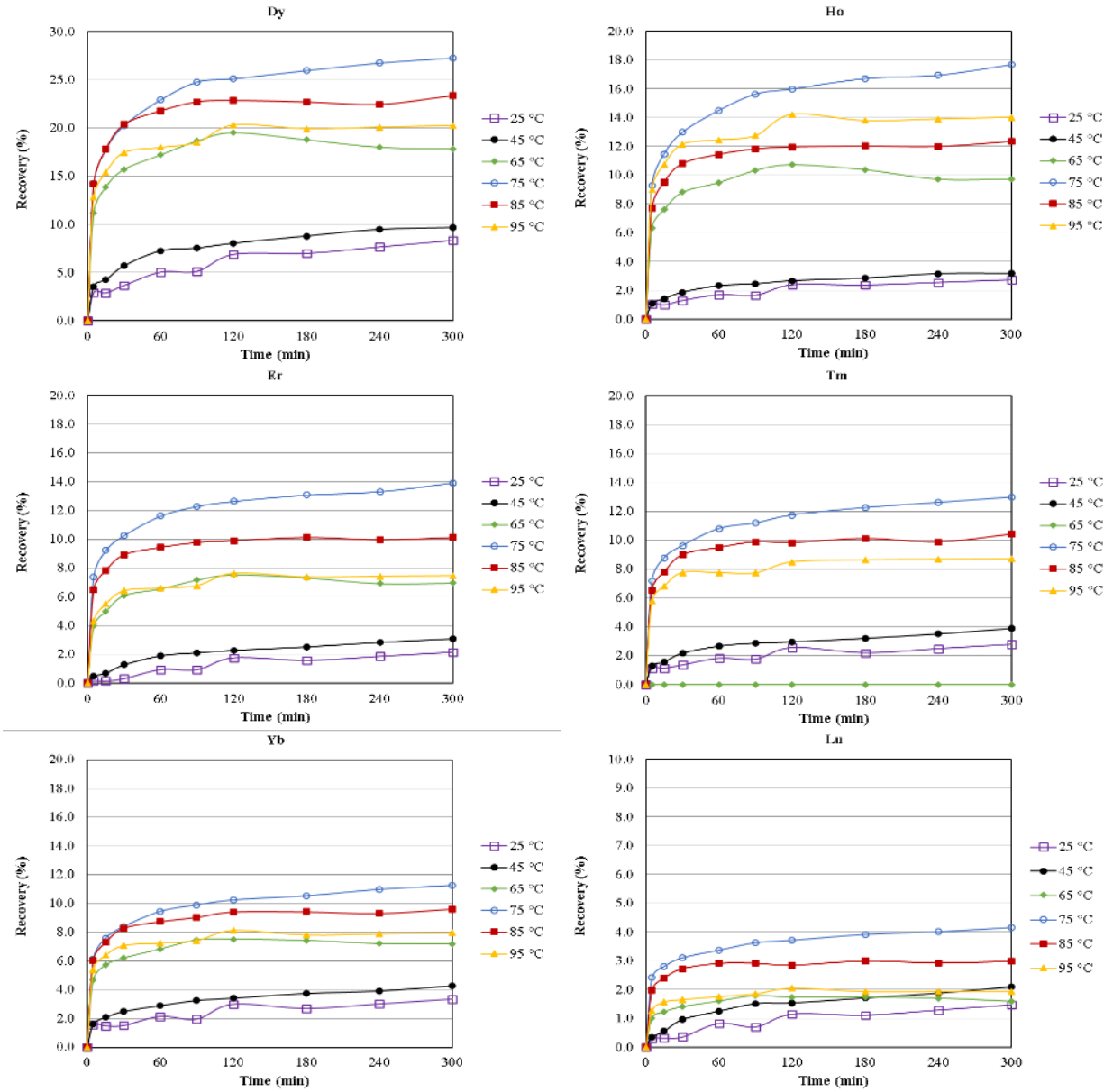



Figure 3.6.81. Effect of temperature on leaching recovery of individual REEs (HREEs).

### *3.6.14 Leaching Characteristics of Shakespeare Material*

A lignite material received great attention due to its high concentration of REEs. The lignite material is a tailing material from a sand operation owned by Shakespeare and Oil Company in Illinois. Several samples received from the sand operation showed that the lignite material could contain as high as 3400 ppm of TREEs on a whole mass basis. Leaching results showed that the REEs in this material was easy to be extracted using sulfuric acid (Figure 3.6.82). The leaching recovery of individual rare earth elements and the major element of interest is shown in Figure 3.6.83. As shown in the results, nearly 100% of Co, Mn, Ni, and Zn can be extracted as well, along

with only about 20% of Al, and 40% of Fe using 0.5M sulfuric acid leaching with 10 g/L solid concentration at 75°C.

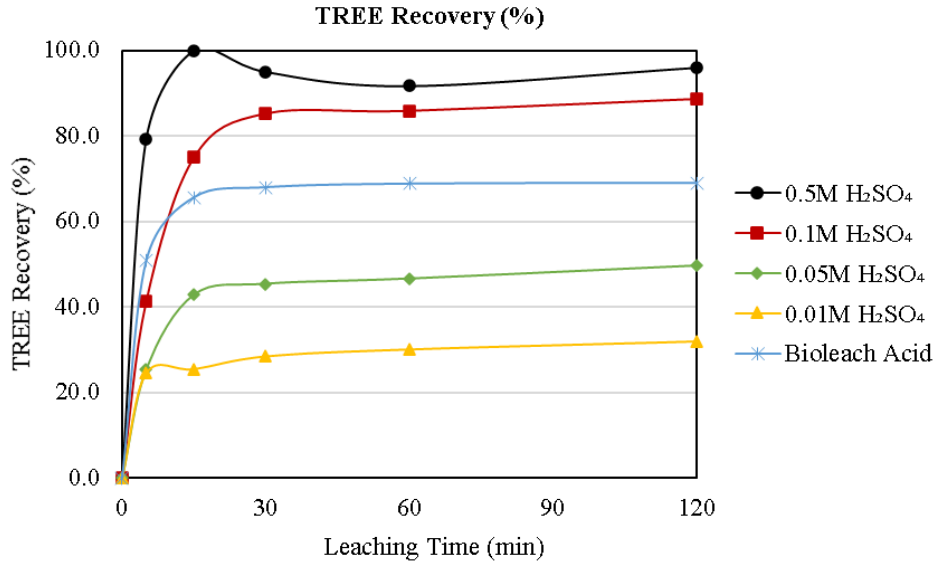



Figure 3.6.82. Leaching characteristic of Shakespeare lignite material. (Solid/Liquid concentration = 10 g/L, temperature = 75 Celsius.)

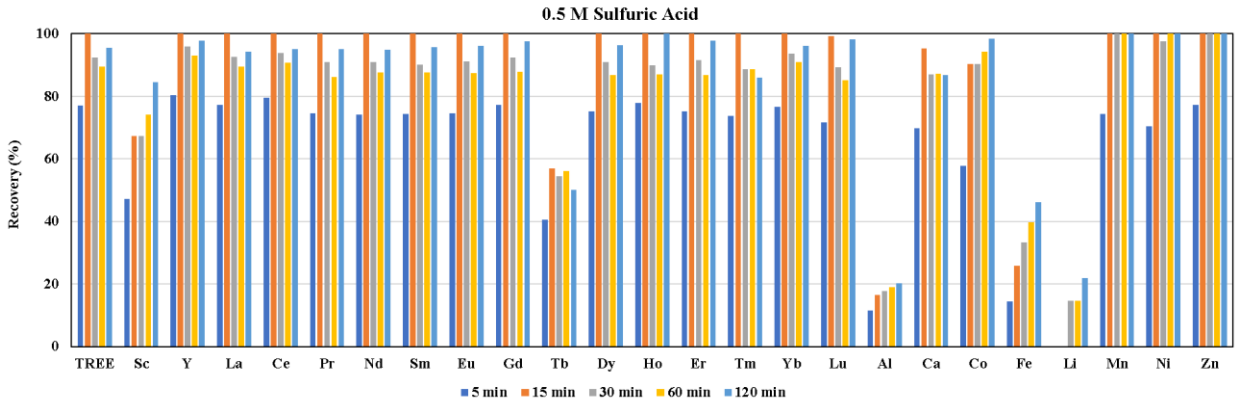



Figure 3.6.83. Leaching recovery of major elements of interest from the Shakespeare lignite material. (Solid/Liquid concentration = 10 g/L, temperature = 75 Celsius.)

In the 4$^{th}$ quarter of 2021, 20 tons of the lignite material was hauled to the pilot plant facility. The material contains 380 ppm of TREEs on a whole mass basis with an ash content of 65%. It was noticed that the majority of the minus 30-mesh material was sand which contains only 73 ppm of TREEs with 98% ash content. Therefore, by screening the material at 30 mesh, the TREEs in the plus 30 mesh fractions upgraded to 844 ppm of TREEs with an ash content of 20.2%. The 20-ton material was processed using a 30-mesh screen and then ground to a top size of 1 mm using a hammer mill. A series of lab tests were carried out to identify the leaching conditions.

Leaching tests were first carried out using 0.5M sulfuric acid at 200 g/L solid to liquid ratio under different temperature, i.e., 25, 50, 75 °C. The tests were carried out using 3-neck round bottom

flask with total reflux condenser. The acid solution was preheated to the designated temperature, then solid was added into the reactor under 700 rpm agitation using a stir bar. The results showed that the effect of leaching temperature on REEs recovery from the Shakespeare material was negligible (Figure 3.6.84). However, with increasing the leaching temperature, the recovery of Ca and Fe increased significantly. It was determined that Shakespeare leaching could be conducted at room temperature to reduce contamination levels and utility costs for heating while still recovering the maximum level of rare earths at 0.5 M sulfuric acid.

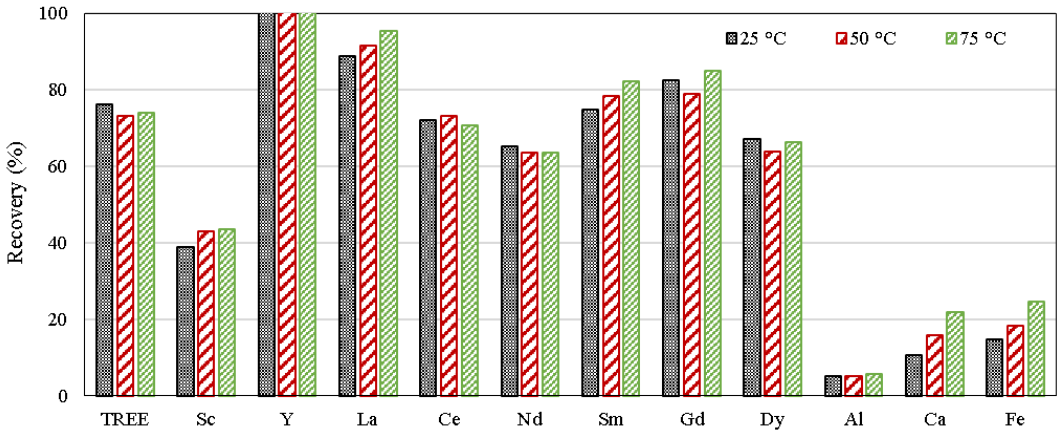



Figure 3.6.84. Effect of leaching temperature on REEs and major impurities leaching recovery from Shakespeare material. (0.5M sulfuric acid at 200 g/L solid to liquid ratio)

In addition to testing the impact of temperature on leaching ability, acid strength was also tested to determine if lower molarity acid could be used in the plant. Sulfuric acid solutions of varying strength were tested at room temperature to test their leaching potential. 0.2 M and 0.3 M acid both performed poorly in recovering rare earths with only 20 ppm and 25 ppm TREE respectively. Leaching with 0.4 M sulfuric acid, however, recovered 55 ppm TREE making it a possible option to test in the plant.

Precipitation test was carried out on the leachate solution generated from the lignite material leaching using 0.5M sulfuric acid. The solution contains 63 ppm of TREEs, 598 ppm of Al, 936 ppm of Ca, and 1000 ppm of Fe. Results showed that the about 72% of Fe was in Fe(III) state which can be removed at pH of 3.5. As shown in Figure 3.6.85 The Al precipitation pH occurs at pH 4.5-5.0 which is slightly higher than that of the heap leach PLS likely due to the lower ionic concentration of the solution. REE precipitation completes at pH 7.0 which is similar to PLS tested before with low contamination ions. The solution also contains about 53 ppm of Co and 103 ppm of Mn which can be precipitated at pH 7.5 and 9, respectively. The precipitation pH of RE determine the fate of Co in the downstream process. Several scenarios will be considered and tested in the pilot plant including: 1) Precipitate majority of Co with REEs at pH 7, then recover Co from the barren solution after REE oxalate precipitation using sulfide precipitation; 2) Precipitate REEs at pH 6-6.5 so that majority of the Co will report to the CM cake and the cake will be redissolved for Co recovery using sulfide precipitation; 3) Precipitate REEs and CMs together at pH 9, then redissolve the REEs and CMs into solution followed by oxalate precipitation and sulfide precipitation to recover RE-oxalate and Co sulfide. 4) Directly process the leachate

solution through oxalate precipitation and sulfide precipitation to recover RE-oxalate and Co sulfide. These four scenarios are tested in the pilot plant under continuous operation.

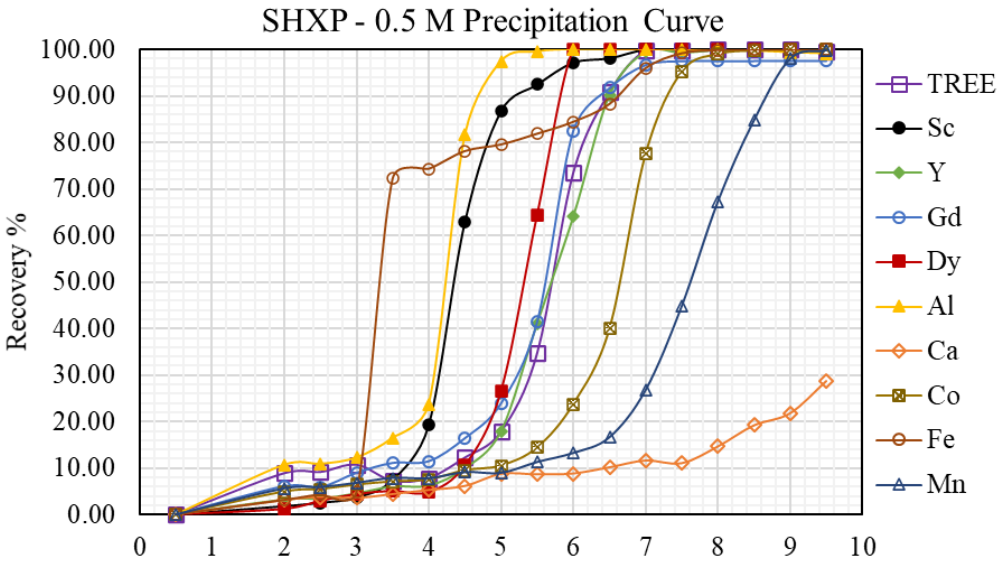



Figure 3.6.85. Precipitation efficiency of elements of interest from the SHXP leachate solution. Initial pH is 0.511. (4M NaOH solution used to adjust pH)

### *3.6.15 In-tank lamella clarifier design*

Owing to the complexities and interaction of the solids with the leaching solution recovery is time based and variable. Because of this characteristic, the residence time of the solids in leaching need to be precisely controlled to allow optimum recovery of the REEs from the roasted solids. To achieve this difficult task a novel reactor named “internal tank clarifier” was considered for use in the leach circuit. This design allows the simultaneous leaching and solid/liquid separation. This reactor will also allow finer solids residence time control and counter current leaching to improve the leaching recoveries. Further, this solution provides considerable cost reduction where the device can be retrofitted into the existing leach tanks. This technology is covered by US patent application 20210023478. The apparatus is shown schematically in Figure 3.6.86 depicting the slurry input, the mixing chamber, the lamella and the solution path for clarification, and the ports to allow counter current operation.

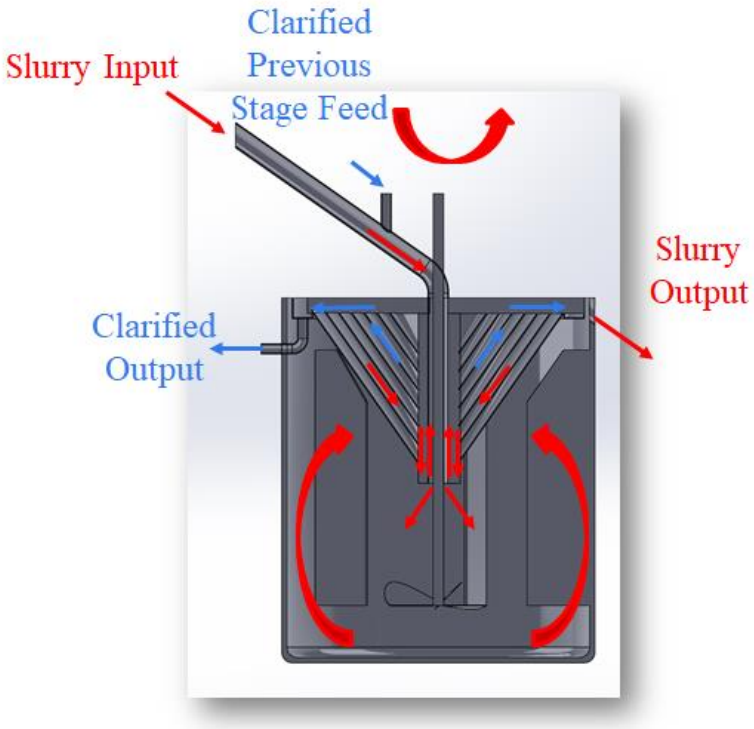



Figure 3.6.86. Schematic representation of an in-tank lamella clarifier.

Shown in Figure 3.6.87 is an isometric drawing depicting the apparatus and the bill of materials. The center well #4 is where the impeller shaft and the inlet liquids are added. The clarified outlet is depicted as piece #5 with a serrated edge to act as an overflow weir patterned after the launderer on a thickener. The lamella (#1) is suspended on the frame (#3) to provide structure and support. Note the cutouts shown by #7 which are provided to allow the passage of tank heaters necessary for leaching. Figure 3.6.88 is provided showing the general layout of top and side view including a cross-section view to show the interior arrangement where clarification occurs. The arrangement of the assembly is shown in Figure 3.6.89 depicting the orientation, spacing and clearances between the clarifier and the tank. For the current embodiment, the outlet is designed to be in the cone bottom of the tank, rather than the side outlet shown in Figure 3.6.87.

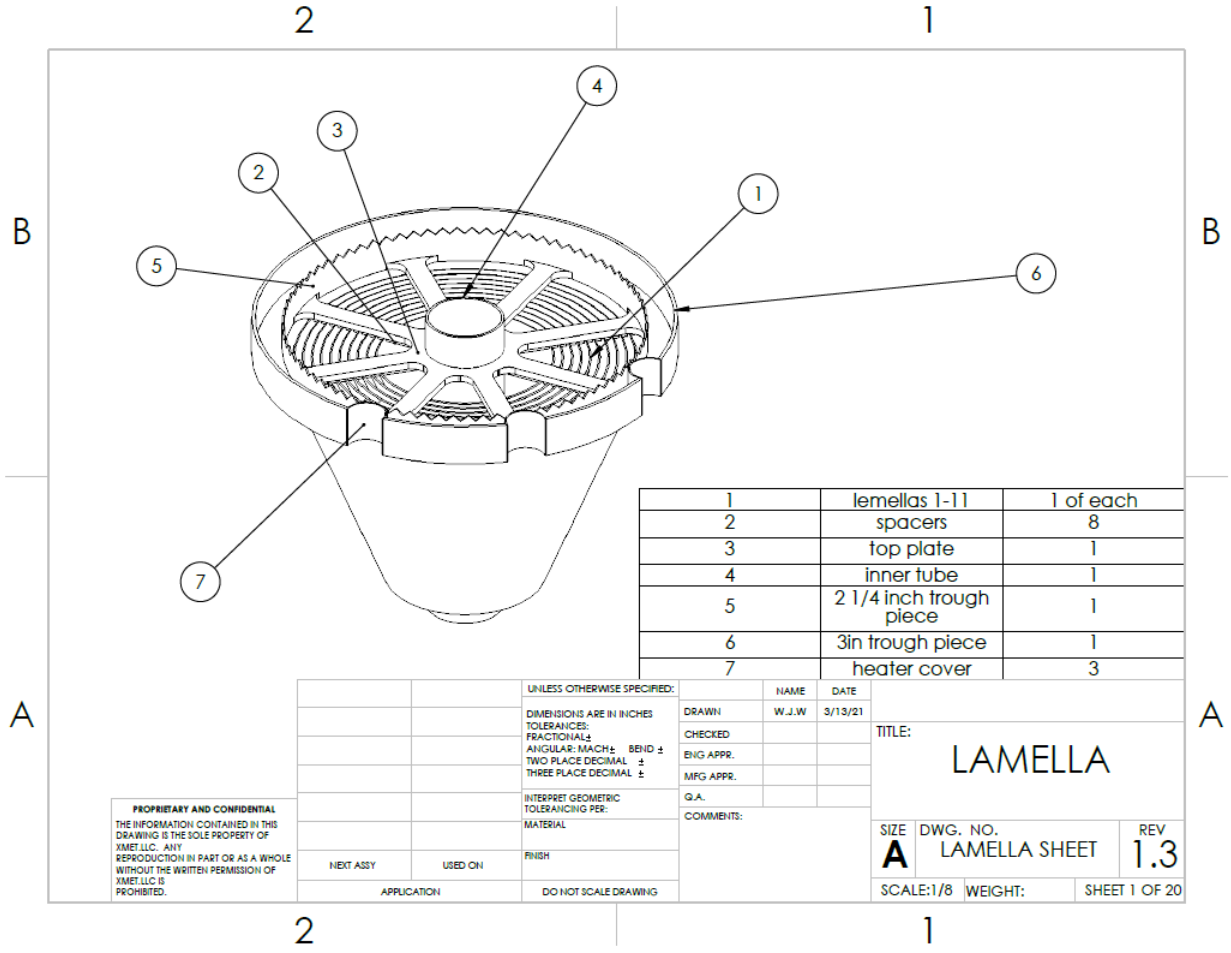



Figure 3.6.87. Isometric representation and bill of materials for the construction of one in-tank lamella clarifier for retrofitting a 100 gal. leach tank.

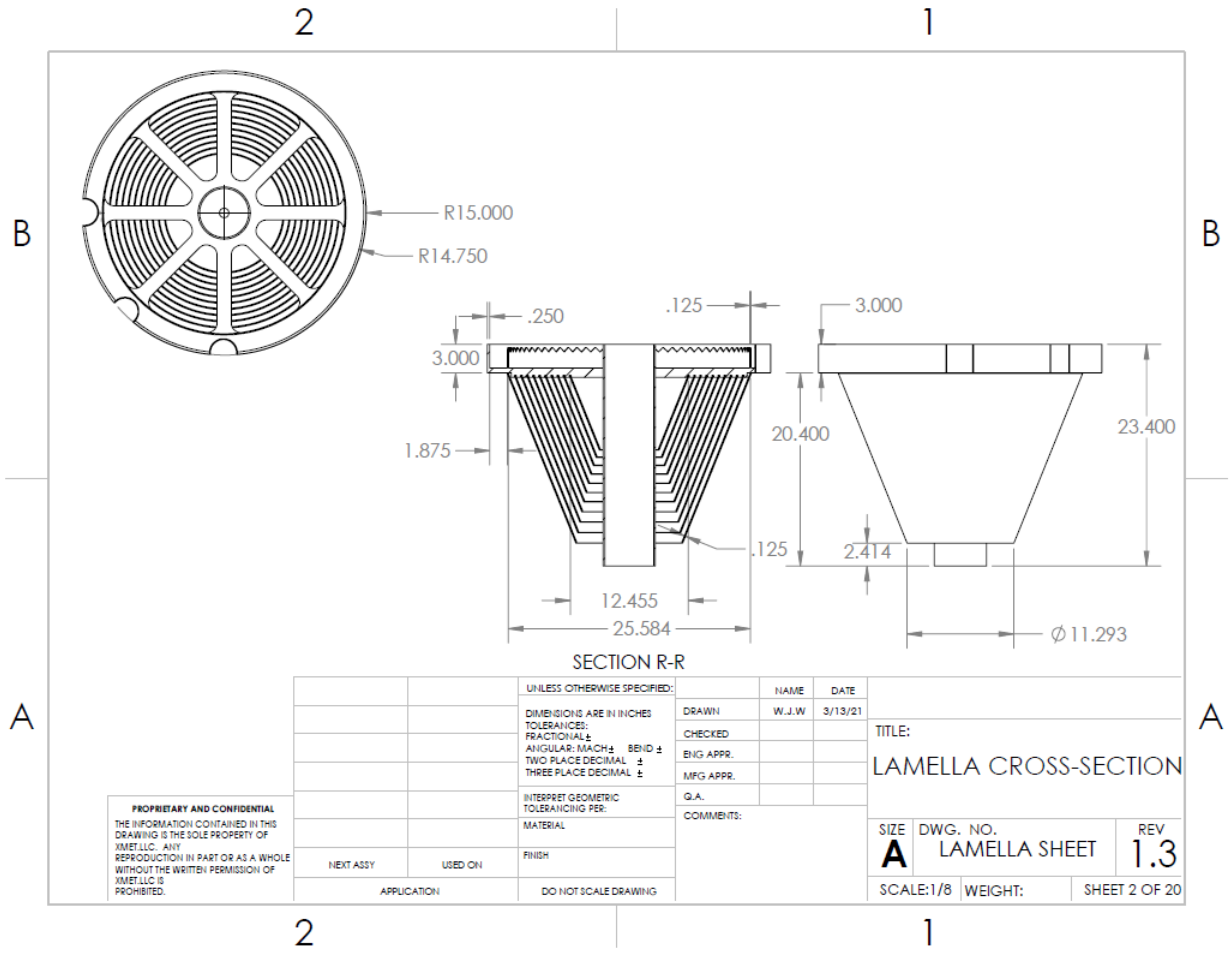



Figure 3.6.88. Cross-sectional drawing showing the internal annular spaces needed for solid/liquid separation.

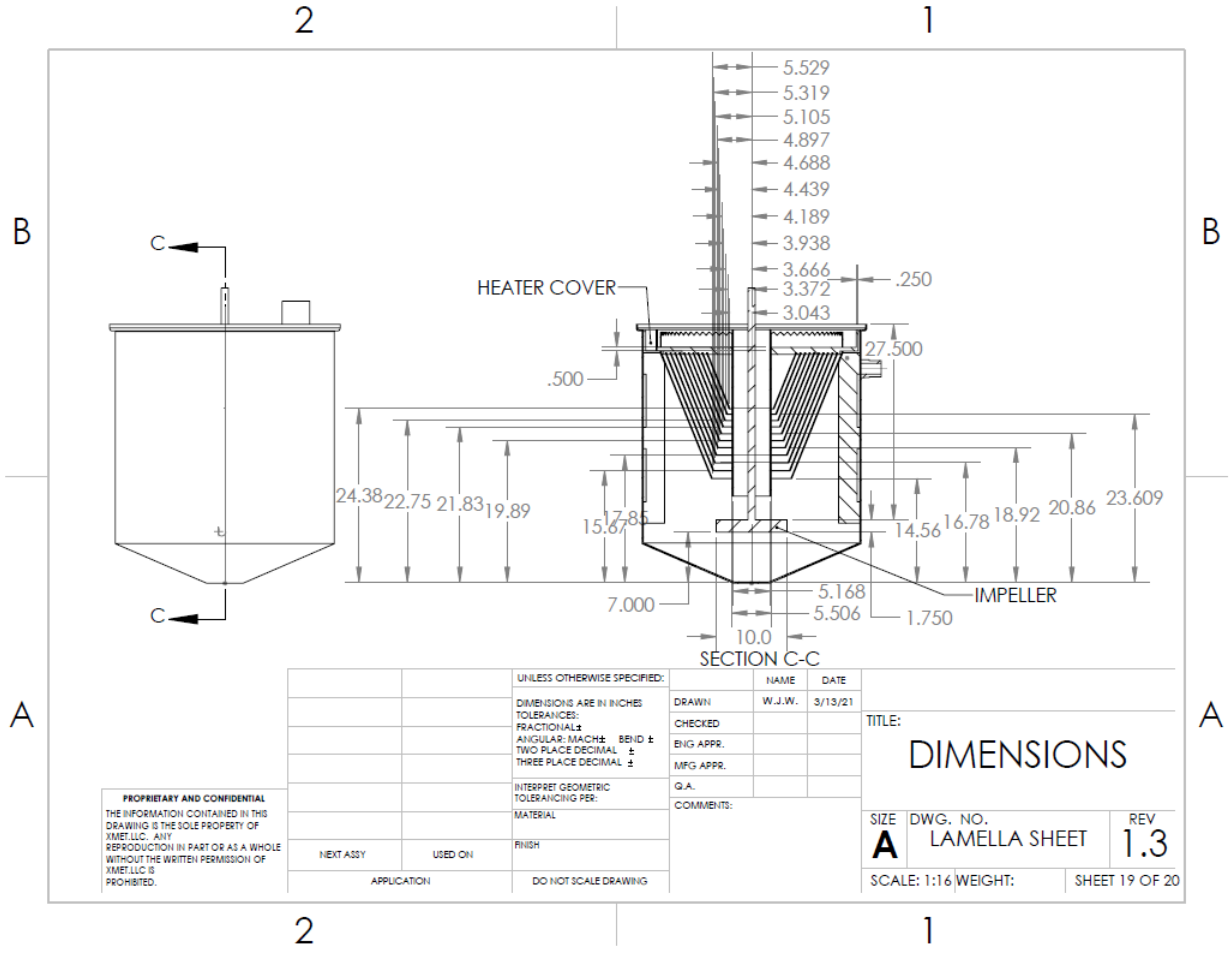



Figure 3.6.89. Tank arrangement of the in-tank lamella clarifier.

To evaluate the design a multiphase CFD simulation was performed to determine the physical design of the reactor. Shown in Figure 3.6.90 is the 2D cross sectional view of the proposed geometry showing the center inlet well, the impeller, the cone bottom tank and the conical circumferential lamella. Even though the current grind size is approximately 1mm, the simulation was conducted with a 0.050 µm particle size to determine the suitability of the design to the smaller particle fractions. From the simulation in Figure 3.6.90, the apparent performance is sufficient to accomplish the solid/liquid separation.

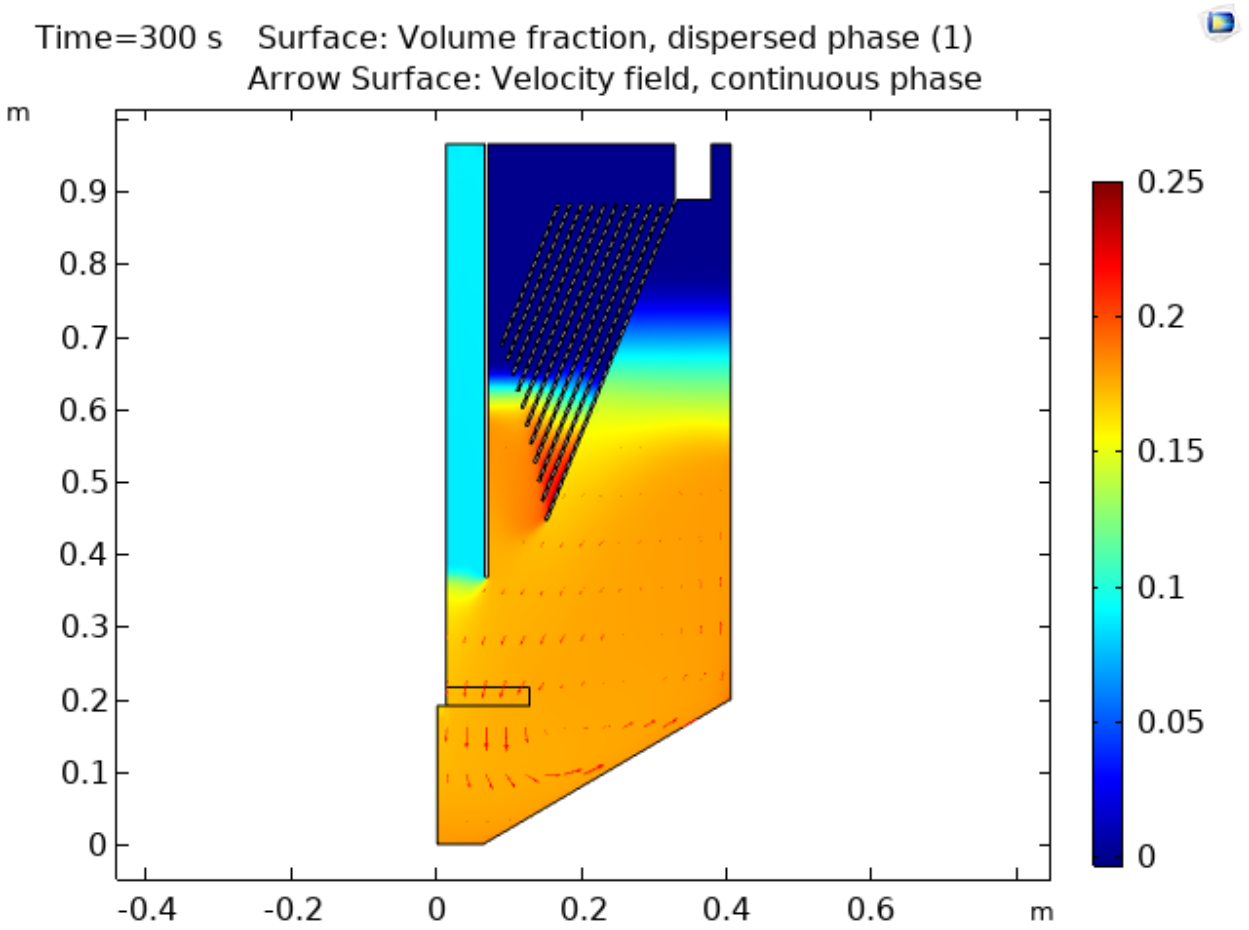



Figure 3.6.90. 2D cross section of the proposed in tank lamella clarifier showing the volume % solids and the flow pattern in the cell from the impeller. The design variables were 0.2 wt % solids at a spherical size of 0.050 µm and a S.G. of 2.6. The impellor was modeled to have an output velocity of

To evaluate the liquid flow pattern in the cone bottom tank with the in-tank lamella a velocity plot (Figure 3.6.91) was added to show the flow magnitude and direction. The impeller was modeled as a volume force to approximate the effects of the impeller. The targeted solution velocity was selected as 0.75 m/s with a PID control utilized to manipulate the volume force sufficient to produce the requisite velocity. An alternate view showing the volume fraction in a 3D representation is depicted in Figure 3.6.92. This shows clearly the center well, the solids holdup in the lamella, and the general mixing patter in the leaching space. It is anticipated that this novel reactor will provide unparalleled flexibility and recovery from the current leaching equipment.

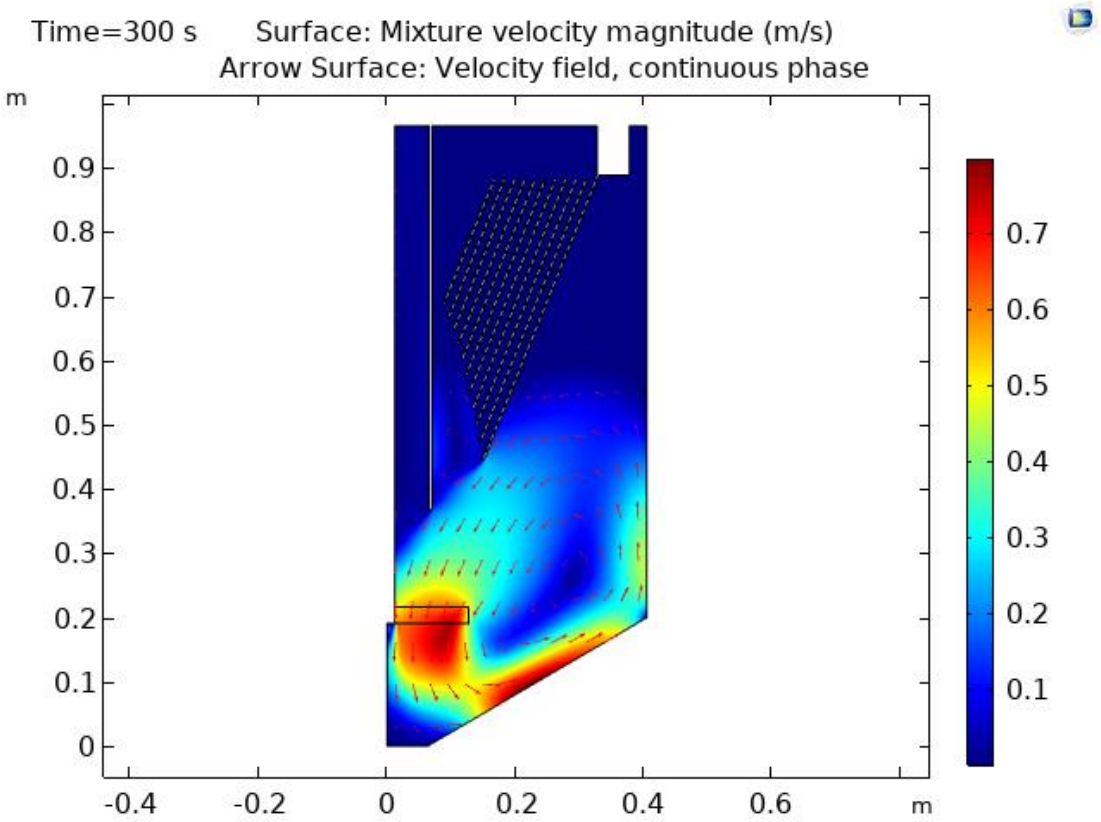



Figure 3.6.91. 2D cross section of the proposed in tank lamella clarifier showing the volume % solids and the flow pattern in the cell from the impeller. The design variables were 0.2 wt % solids at a spherical size of 0.050 μm and a S.G. of 2.6.

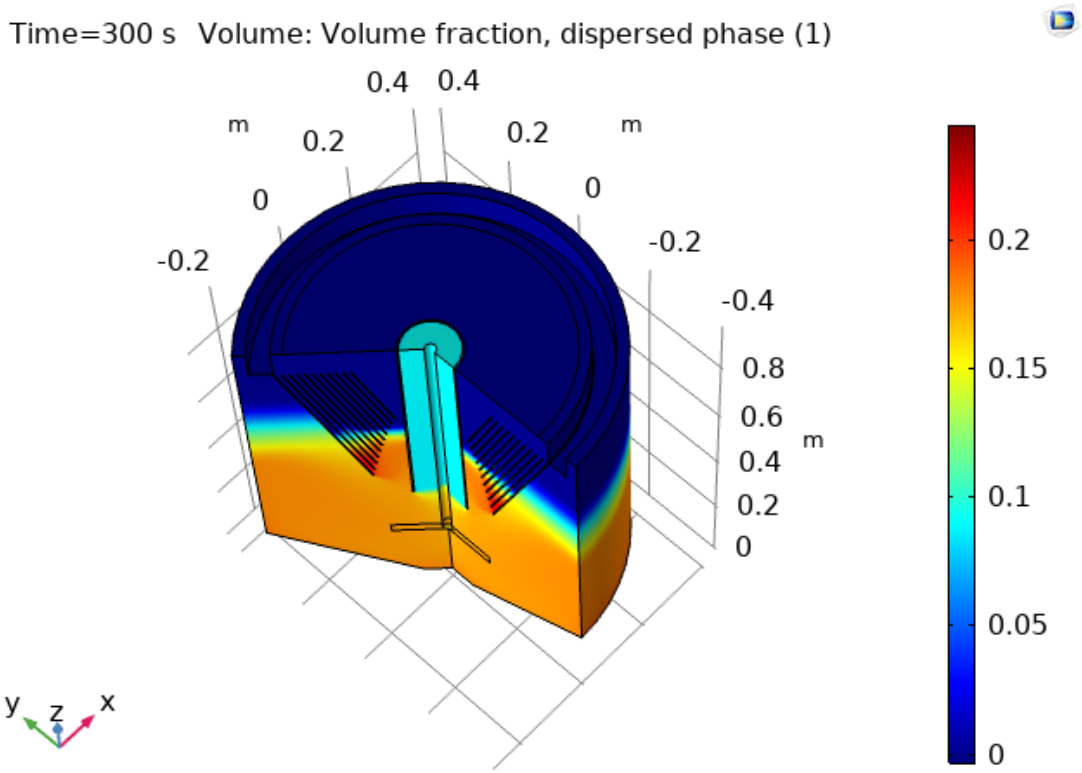



Figure 3.6.92. 3D radial representation of the solids volume

## 3.7 OPTIMIZATION OF RARE EARTH RECOVERY FROM PLS (TASK 9)

The objective of this task was to conduct preliminary laboratory studies to optimize the use of selective precipitation in addition to or in place of solvent extraction for the separation and recovery of rare earth elements from pregnant leach solution (PLS). The bioxidation of coal pyrite to produce sulfuric acid created a significant amount of iron in solution. To remove the iron selectively, a bioxidation reaction was evaluated to covert ferrous to ferric.

Given the unique solution chemistry associated with the bioacid, optimization studies were performed on the staged precipitation of REEs, Co and Mn as well as the contaminant elements. Laboratory experimental studies investigated the precipitation characteristics of both the critical materials and the contaminants (e.g., Al and Mg) using different bases (sodium hydroxide, sodium carbonate, ammonium hydroxide, lime, calcite, etc.). Optimum conditions including the addition method, dosage, temperature, aging time and mixing strength were identified for all the investigated bases. The most efficient and cost-effective base was selected.

The pre-concentrates obtained from staged precipitation were re-dissolved for further extraction and purification purposes. Optimum re-dissolution conditions were identified by performing dissolution tests with different acids, acid dosages, mixing speeds, reaction durations, temperatures, etc. Maximum re-dissolution recovery of the REEs and maximum rejection of the contaminants were used as the performance metrics.

Tests were performed to determine optimum dosage of extractant relative to the pH condition and elemental content of the PLS solution prepared from the re-dissolution of the REEs. Various extractant types were evaluated and the best chosen for more extensive evaluations based on REE recovery, extractant dosage and cost, and acid concentrations needed for stripping.

Finally, optimization of the oxalic acid precipitation process was also performed with the performance target being recovery and purity of the rare earth oxalate product as well as cost. Different precipitation methods in addition to oxalic acid precipitation were assessed including double sulfate precipitation, phosphate precipitation and carbonate precipitation.

### *1.1.1 Staged Precipitation using various precipitating agents*

Rare earth elements and other critical materials were solubilized using sulfuric acid in this project. As the PLS contains dilute REE concentrations relative to other contaminants such as iron, aluminum and calcium, the solution was typically processed through a series of selective precipitation stages to remove the contaminants. Subsequently, an enriched RE and CM was generated which was subsequently processed for the REE and CM precipitation. Since selective staged precipitation for contaminant removal is conducted through pH adjustments, several options of precipitants are available. As such, a systematic study was conducted to identify the optimal precipitant to be employed for contaminant removal in the pilot scale experiments.

For this task, 100-gallon of acid mine drainage was collected and used for bench-scale experimentation. Concentration of the major and trace elements in the sample was measured using ICP-MS. As shown in Table 3.7.1, the leachate contained 6.6 ppm of total REEs (TREE), of which critical REEs, including Nd, Eu, Tb, Dy, and Y, account for 16.6%, 1.0%, 0.7%, 4.2%, and 20.5%, respectively. Therefore, the leachate can be used as a good resource of REEs, especially critical REEs. In addition, the leachate also contains 1.65 ppm Co, 1.39 ppm Li, and 58 ppm Mn, which

have been coded as critical materials by the Department of Interior. Major contaminant metal ions occurring in the sample are Fe (884 ppm), Al (350 ppm), Ca (417 ppm), and Mg (302 ppm).

Table 3.7.1. Elemental concentration (ppm) of the acid mine leachate sample.

| Element | Sc | Y | La | Ce | Pr | Nd | Sm | Eu | Gd | Tb | Dy | Ho |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Concentration | 0.13 | 1.35 | 0.79 | 1.67 | 0.25 | 1.09 | 0.29 | 0.07 | 0.36 | 0.05 | 0.27 | 0.05 |
| % of TREE | 2.0 | 20.5 | 12.0 | 25.4 | 3.70 | 16.6 | 4.5 | 1.0 | 5.4 | 0.7 | 4.2 | 0.7 |
| Element | Er | Tm | Yb | Lu | LREE | | HREE | | TREE | | HREE/LREE | |
| Concentration | 0.12 | 0.01 | 0.08 | 0.01 | 4.23 | | 2.36 | | 6.60 | | 0.56 | |
| % of TREE | 1.8 | 0.2 | 1.2 | 0.2 | 64.2 | | 35.8 | | 100 | | – | |
| Element | Co | Cu | Ni | Zn | Li | Mn | Fe | Al | Ca | Mg | | |
| Concentration | 1.65 | 2.42 | 4.73 | 7.63 | 1.39 | 58 | 884 | 350 | 417 | 302 | | |

Staged precipitation tests were performed on the acid mine leachate aiming to remove the contaminant metal ions and preconcentrate the REEs and other critical metals. Sodium hydroxide (NaOH) and sodium carbonate ($Na_2CO_3$) were used as precipitants for comparison purposes. Furthermore, hydrogen peroxide ($H_2O_2$) was also added into the leachate at the beginning of selected tests, which served as oxidizing agent to ensure all the Fe occurred as ferric ions.

**Results and Discussion:** The precipitation recovery of TREEs and other elements was first analyzed, which will inform the viability of using staged precipitation to preconcentrate REEs and other critical metals. As Figure 3.7.1(a) shows, when 5 M NaOH was used as the precipitant without adding $H_2O_2$, TREEs started to precipitate at pH around 4.0, and complete precipitation was achieved when the pH was elevated to around 7.0. Furthermore, 100% of Al was precipitated at pH 5.1, under which condition, only 25% of TREEs was precipitated. Therefore, Al can be effectively separated from TREEs by staged precipitation without considerable loss of TREEs. The precipitation behavior is different from Al and REEs. As shown, when pH was increased to 4.0, nearly 23% of Fe was precipitate; whereas, as pH increased from 4.0 to 6.0, increases in the recovery were slower than pH < 4.0. Moreover, when pH further increased to above 6.0, the Fe was rapidly precipitated. Therefore, it can be concluded that ferrous and ferric ions occurred simultaneously in the leachate. At lower pH, ferric ions was preferentially precipitated, while ferrous ions tended to precipitate at higher pH. Given the fact that Fe is the most abundant metal ions in the leachate, staged precipitation with 5 M NaOH alone cannot provide satisfactory. For comparison purposes, 3 M $Na_2CO_3$ was also used as a precipitant. As shown in Figure 3.7.1(b), the precipitation characteristics of the elements, except Ca and Mg, are similar to those observed with 5 M NaOH. When using $Na_2CO_3$, nearly 100% of Ca was precipitated at pH 10, which is attributable to the formation of calcite.

To transform the Fe from ferrous to ferric forms, $H_2O_2$ was added at the beginning of the staged precipitation tests. As shown in Figure 3.7.1(c) and (d), nearly 100% of the Fe was precipitated at pH 4.0, regardless whether $H_2O_2$ dosage was 0.5 vol.% or 1 vol.%. Therefore, to achieve a higher-grade preconcentrate of REEs, the ferrous ions need to be oxidized to ferric ions prior to the precipitation.

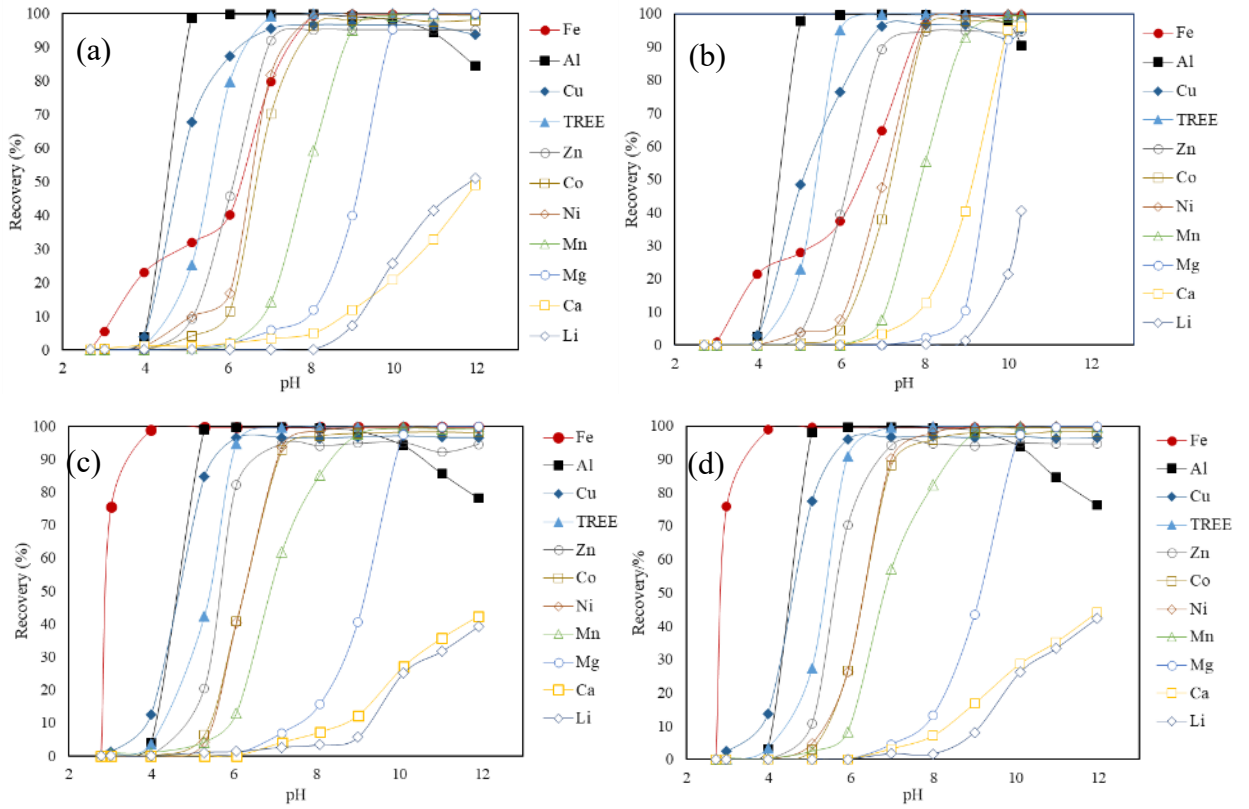



Figure 3.7.1. Staged Precipitation Recovery of TREE and other elements using different reagents: (a) 5 M NaOH; (b) 3 M $Na_2CO_3$; (c) 5 M NaOH plus 0.5 vol.% $H_2O_2$; and (d) 3 M NaOH plus 1 vol.% $H_2O_2$.

The recovery values of the individual element obtained using the different reagents were plotted in the same figure to provide more intuitive comparisons. As shown in Figure 3.7.2, the recovery curve of NaOH plus 0.5 vol.% $H_2O_2$ overlaps the curve of NaOH plus 1 vol.% $H_2O_2$, indicating that $H_2O_2$ dosage of 0.5 vol.% is sufficient to completely oxidize the ferrous ions in the acid mine leachate. Replacing NaOH with $Na_2CO_3$ did not cause considerable differences in the recovery of Al and Fe. However, at pH values greater than 8.0, calcium recovery using $Na_2CO_3$ was much higher than NaOH, which introduced more calcium into the preconcentrate of TREEs, Co, and Mn obtained in the pH range of 5.0 to 9.0. However, based on the titration curves shown in Figure 3.7.3, the molar consumption of $Na_2CO_3$ is less than NaOH, which can be explained by the two-step hydrolysis of $CO_3^{2-}$ of the $Na_2CO_3$. After reaching pH 9.0, adding $Na_2CO_3$ did not considerably change the pH, primarily due to the formation of calcite, which consumed the $CO_3^{2}$.

Based on the titration curve of NaOH, it can be observed that pH gradually elevated with the increase in the amount of precipitant added into the solution. Slower pH increases occurred in four pH ranges: 3.1-3.3, 4.0-4.5, 6.0-7.8, and 8.0-9.4, which correspond to the precipitation of ferric, aluminum, ferrous, and magnesium ions, respectively, per the results shown in Figure 3.7.1 (a). When hydrogen peroxide was added into the leachate, the third range disappeared. This finding corroborates the conclusion that the ferrous ions present in the leachate were completely transformed into ferric ions by adding 0.5 vol.% hydrogen peroxide.

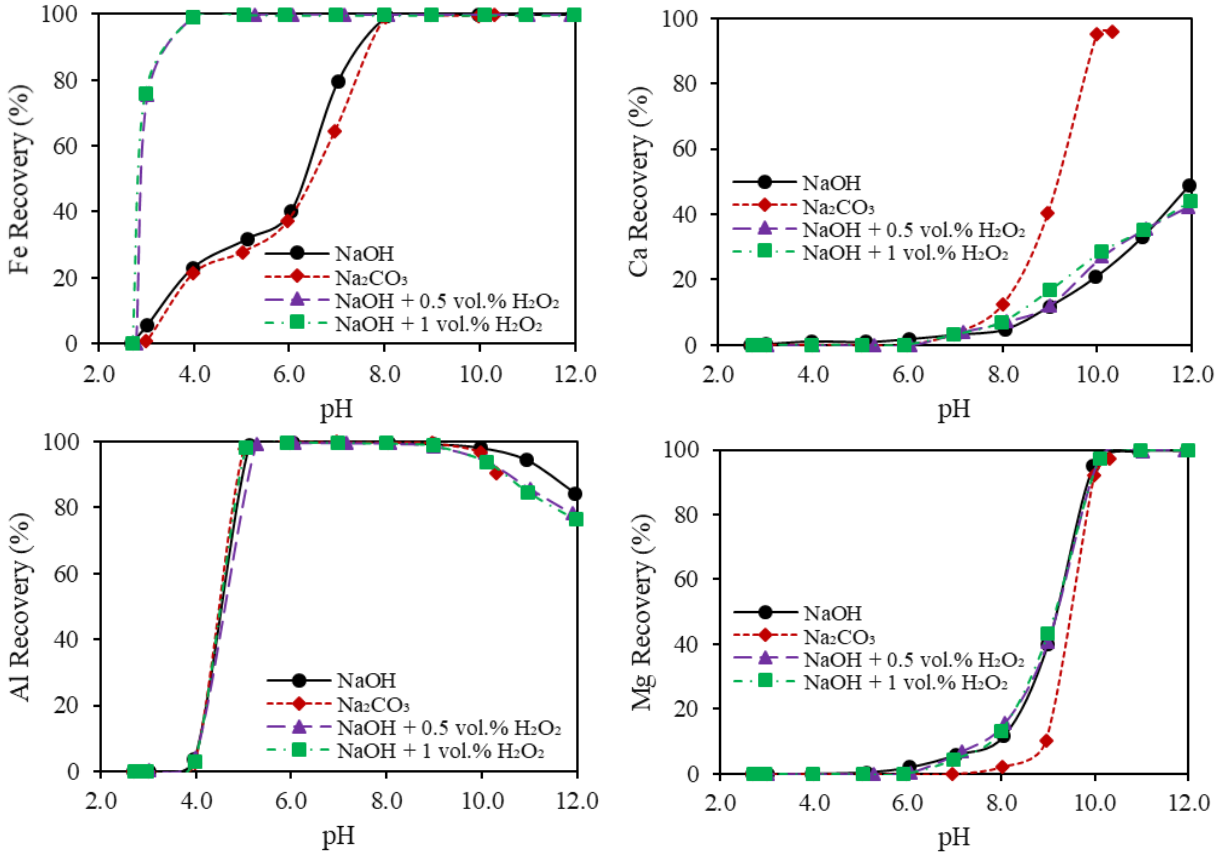



Figure 3.7.2. Staged precipitation recovery of Fe, Al, Ca, and Mg.

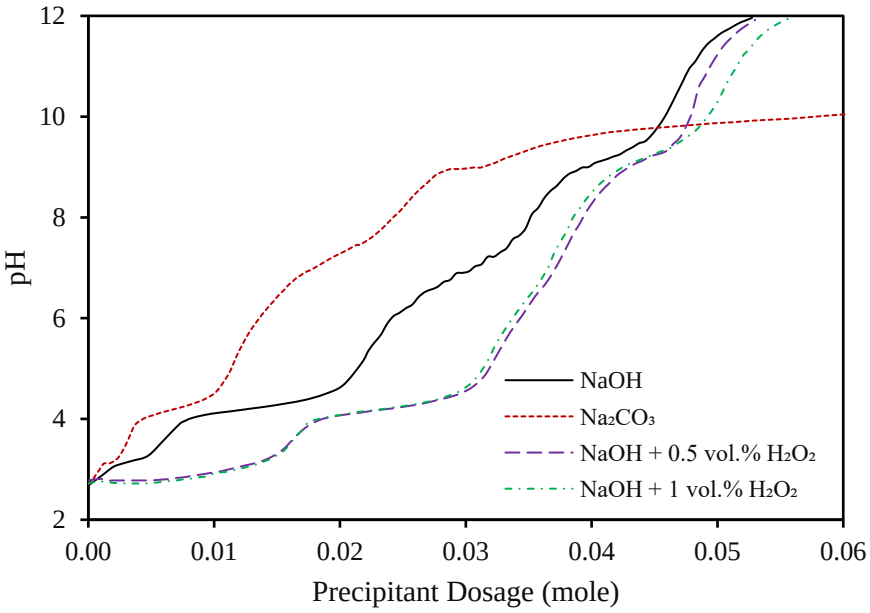
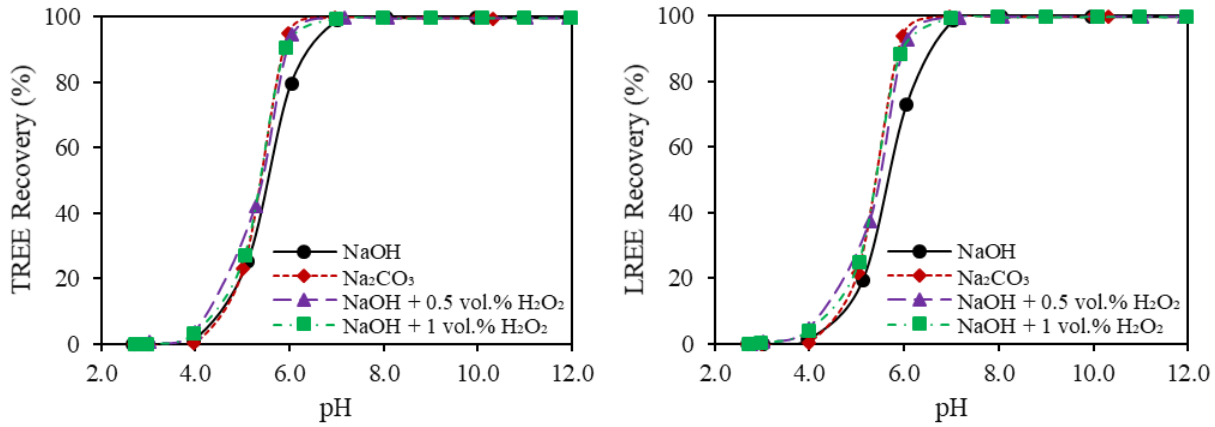


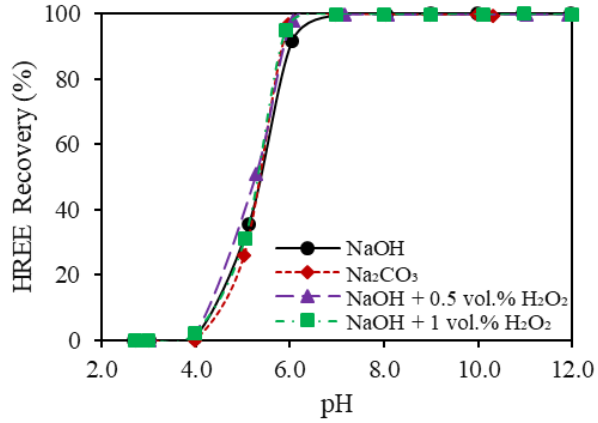

Figure 3.7.3. Titration curves of the acid mine leachate obtained using different reagents.

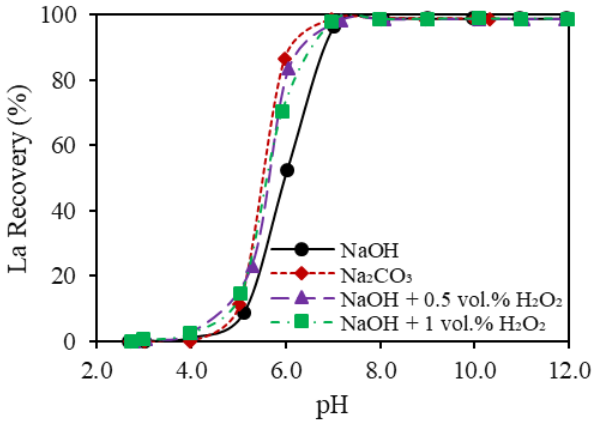
The impact of the different reagent schemes on the precipitation recovery of REEs is shown in Figure 3.7.3. TREE recoveries obtained using $Na_2CO_3$ and NaOH plus $H_2O_2$ are higher than using NaOH. As shown, the former two reagent schemes provided nearly 95% recovery of the TREEs at pH 6.0; whereas, only 80% of the TREEs were precipitated at the same pH when using NaOH. Comparisons between the precipitation behavior of LREEs and HREEs show that the same phenomenon occurred for the LREEs, however no noticeable differences in recovery values of the HREEs were observed. Therefore, lower recoveries of the TREEs are due to the LREEs.

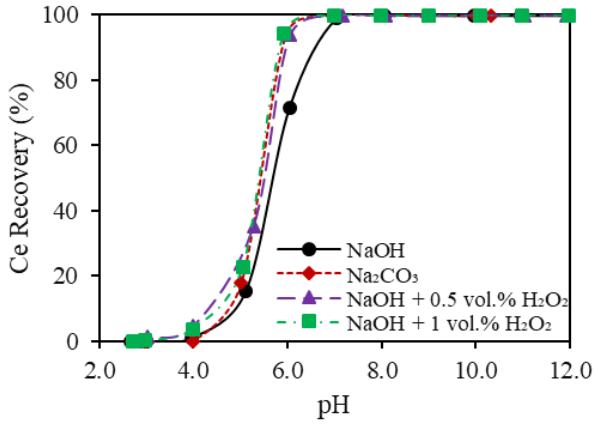
There are two mechanisms that can potentially explain the lower recoveries of the LREEs: (1) the most abundant LREE, Ce, was oxidized from 3+ to 4+ valence by hydrogen peroxide, which is insoluble even under acidic conditions; (2) more ferric precipitate was formed at pH < 4.0 (in advance of the REE precipitation) due to hydrogen peroxide, thereby the LREE recovery was increased due to adsorption. However, the recovery of the individual LREEs shows that not only Ce, but also other LREEs had higher recoveries when hydrogen peroxide was employed. Therefore, rather than the oxidation of Ce, the higher recoveries are more likely due to the adsorption of the LREEs on the surface of ferric precipitate.

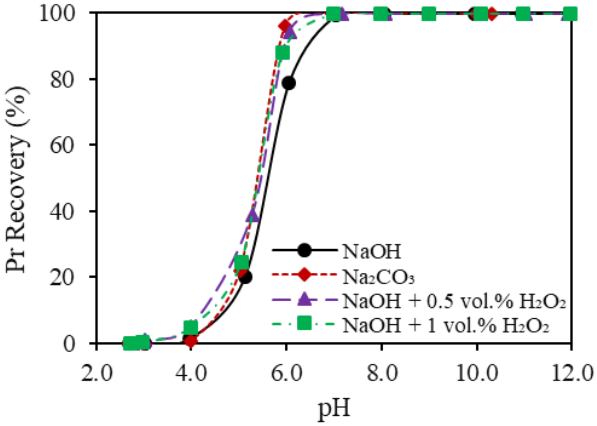
The results shown in Figure 3.7.2a indicated that the precipitation behavior of Fe was not changed when using $Na_2CO_3$ as the precipitant relative to NaOH. However, using $Na_2CO_3$ increased the recovery of LREEs (see Figure 3.7.4). Therefore, the increase in recovery was not due to adsorption on ferric precipitate. Furthermore, the hypothesis of Ce oxidation is also untenable since $Na_2CO_3$ does not have oxidizing capability. Given these facts, the higher recovery of LREEs is likely due to the precipitation of LREEs in the form of rare earth carbonates.

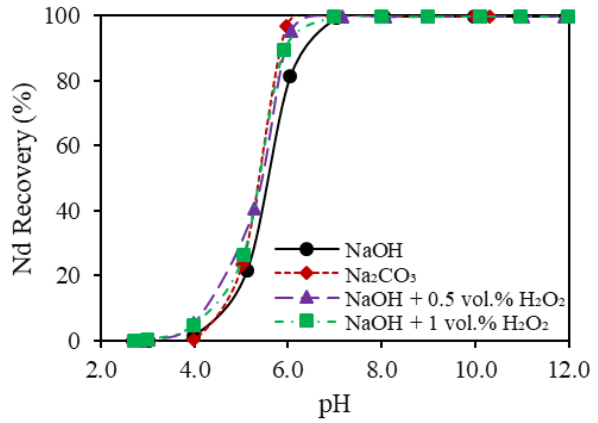


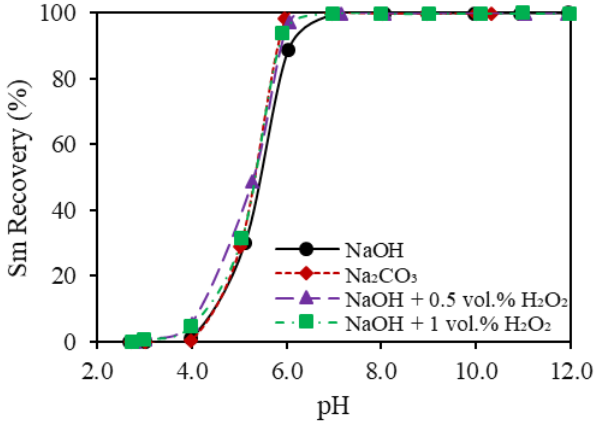

HREE Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2

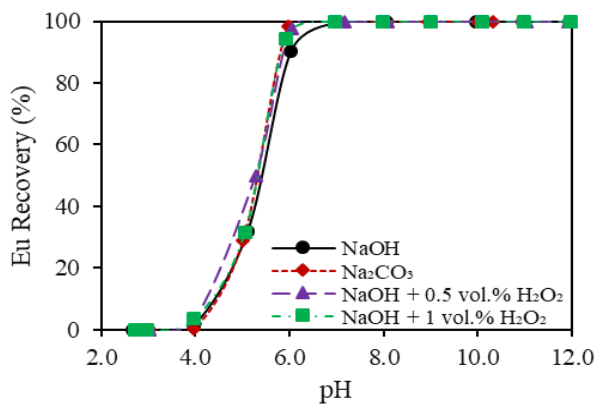

La Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2

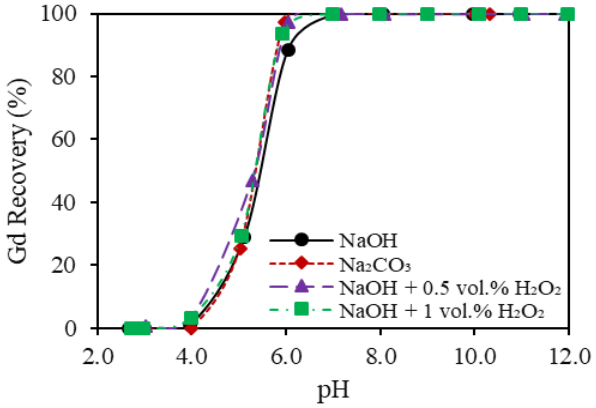

Ce Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2

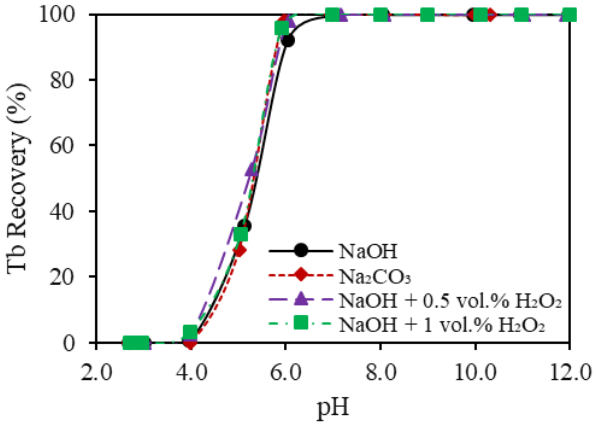

Pr Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2

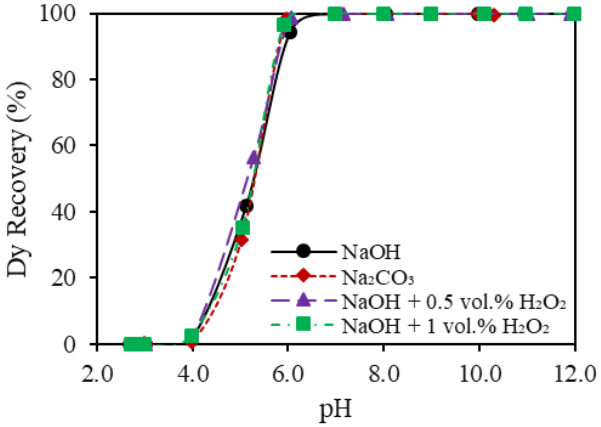

Nd Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2

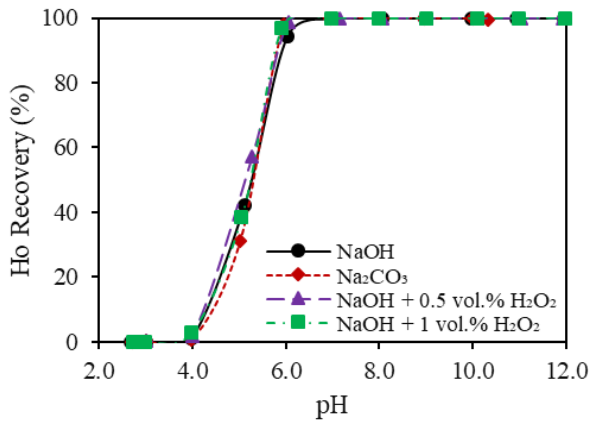

Sm Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2

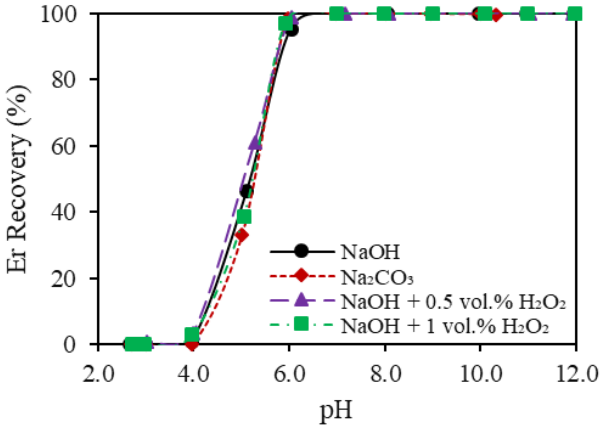

Eu Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2


Gd Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2


Tb Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2


Dy Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2


Ho Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2


Er Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2

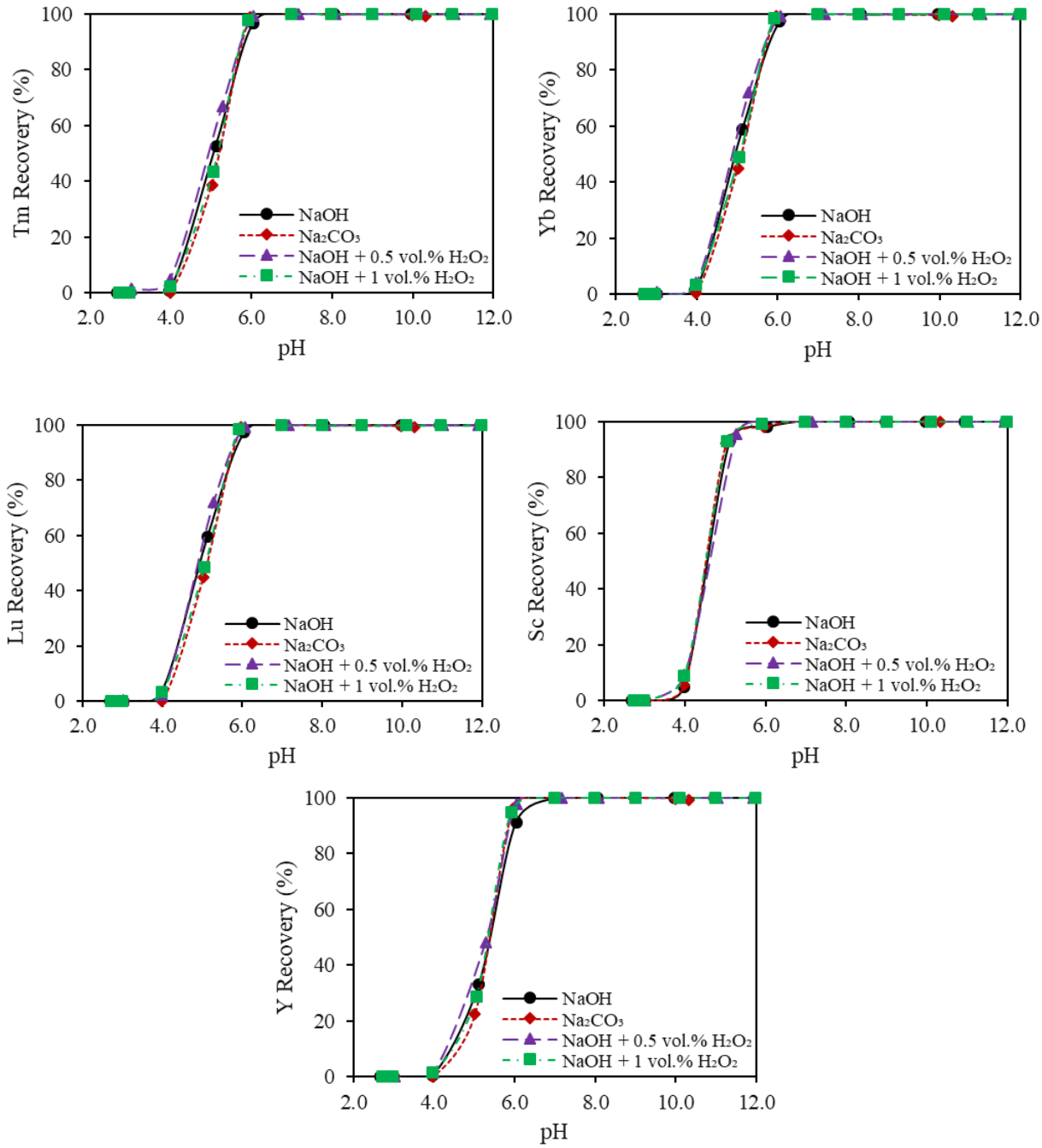



Figure 3.7.4. REE recovery as a function of pH using different reagents.

The recovery of individual REEs using 5 M NaOH plus 0.5 vol.% $H_2O_2$ is shown in Figure 3.7.5. At the same pH value, recovery of LREEs was lower than HREEs. Despite being included as LREEs, Sc was the most preferentially precipitated REEs. As shown, more than 95% of Sc was precipitated at around pH 5, however only 23% of La was precipitated at this pH.

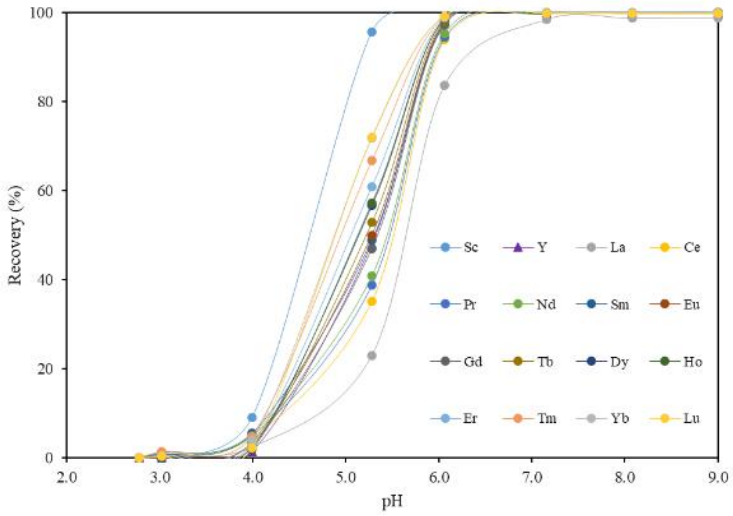



Figure 3.7.5. Recovery of individual REEs as a function of pH using 5 M NaOH plus 0.5 vol.% $H_2O_2$.

### *1.1.2 Model System Precipitation Behavior using NaOH*

In order to gain a deeper understanding of the precipitation behavior of acid mine drainage solutions, a systematic experimental plan was constructed to demonstrate how the presence of certain species effect the precipitation behavior of other species in solution. Solid metal sulfates were dissolved in dilute sulfuric acid to create a sulfate-only system for each test. The pH was raised through the addition of base while pH, Eh, and solid precipitation were monitored. The comparisons between each test result are used to evaluate the effects, if any, of a species on the precipitation behavior of other species in solution.

**Scope of Tests:** Artificial solutions were created by digesting select metal sulfates in sulfuric acid to create a sulfate system similar to the environment of the pilot plant-scale tests. The rare earth elements (REEs) studied were $Gd^{3+}$, $Nd^{3+}$, and $Sc^{3+}$. The other species included were $Al^{3+}$, $Ca^{2+}$, $Fe^{2+}$, $Fe^{3+}$, and $Mn^{2+}$. For each of the three REEs, the following experimental design was planned in order to provide speciation studies on:

$REE^{3+}$: high, average, and low concentrations

$REE^{3+}$, $Al^{3+}$

$REE^{3+}$, $Ca^{2+}$

$REE^{3+}$, $Fe^{2+}$

$REE^{3+}$, $Fe^{3+}$

$REE^{3+}$, $Mn^{2+}$

$REE^{3+}$, $Al^{3+}$, $Fe^{2+}$

$REE^{3+}$, $Al^{3+}$, $Fe^{3+}$

$REE^{3+}$, $Al^{3+}$, $Mn^{2+}$

$REE^{3+}$, $Fe^{2+}$, $Mn^{2+}$

$REE^{3+}$, $Fe^{3+}$, $Mn^{2+}$

$REE^{3+}$, $Al^{3+}$, $Fe^{2+}$, $Fe^{3+}$

$REE^{3+}$, $Al^{3+}$, $Fe^{2+}$, $Fe^{3+}$, $Ca^{2+}$

$REE^{3+}$, $Al^{3+}$, $Fe^{2+}$, $Fe^{3+}$, $Ca^{2+}$, $Mn^{2+}$

In addition to the tests for each of the REEs, single-species precipitation tests were conducted for $Al^{3+}$, $Ca^{2+}$, $Fe^{2+}$, $Fe^{3+}$, and $Mn^{2+}$ to establish the inherent precipitation behavior expected for each species. The concentrations for each species remained consistent across all tests at values similar to that of the leached pilot plant material. For the single-species $REE^{3+}$ tests, concentrations of 100, 10, and 1 µg/mL were targeted to demonstrate any saturation effects that may occur. In all other multi-species tests, the target concentrations for each species are shown in Table 3.7.2.

Table 3.7.2. Target concentrations for each species used in the solution precipitation test plan.

| **Species** | **$REE^{3+}$** | **$Al^{3+}$** | **$Ca^{2+}$** | **$Fe^{2+}$** | **$Fe^{3+}$** | **$Fe^{2+}$ and $Fe^{3+}$** | **$Mn^{2+}$** |
|---|---|---|---|---|---|---|---|
| **Concentration (µg/mL)** | 10 | 100 | 400 | 500 | 500 | 250 and 250 | 40 |

Plots of precipitation versus pH and Eh versus pH were generated to highlight differences in the solution chemistry across all the tests. The goal was to create plots that spanned from pH 2 to pH 10 with intervals of 0.5 pH. For certain tests, additional points at intervals of 0.25 pH were added to better define steep precipitation transition points. There were no time restrictions placed on an interval or an overall test to ensure the test solution was allowed to reach equilibrium at each point along the curves.

**Materials:** The following compounds were purchased from Sigma-Aldrich: ≥ 97% aluminum sulfate octadecahydrate (CAS: 7784-31-8), ≥ 97% calcium sulfate hemihydrate (CAS: 10034-76-1), ≥ 99% ACS reagent iron(II) sulfate heptahydrate (CAS: 7782-63-0), ≥ 97% iron(III) sulfate hexahydrate (CAS: 15244-10-7), and ≥ 99% ReagentPlus managanese(II) sulfate monohydrate (CAS: 10034-96-5). The following compounds were purchased from Alfa Aesar: 99.9% REO gadolinium(III) sulfate octahydrate (CAS: 13450-87-8), 99.9% REO neodymium(III) sulfate octahydrate (CAS: 13477-91-3), and 99.9% REO scandium(III) sulfate hydrate (CAS: 13465-61-7). The following materials were purchased from Fischer Scientific: 600 mL glass beakers; 100 µL, 1000 µL , 5 mL, and 10 mL pipets with their espective pipet tips; 4x4 weighing papers; 50 mL digestion tubes with caps; and 50.8 x 9.5 mm octagon spinbar stirring bars. The 15 mL centrifuge tubes were purchased from Environmental Express.

The pH and Eh probes were connected to Orion Versa Star Pro pH/ISE/Conductivity/Dissolved Oxygen Multiparameter Benchtop Meters purchased from ThermoFischer Scientific. pH was measured using Orion 8104BNUWP ROSS Ultra pH Electrodes filled with Orion ROSS Reference Electrode Filing Solution. Eh was measured using Orion 9180BNMD Triode Refillable Redox/ORP/Temp Electrodes filled with Orion Single Junction Reference Electrode Filing Solution. All aforementioned probes and reference solution were purchased from ThermoFischer Scientific. A miniature vortex mixer was purchased from Benchmark Scientific. An Elmi CM-7S Plus Benchtop Centrifuge was used. For all experiments denoted “A”, a Cole-Palmer Advanced Analog Stirrer 120 VAC was used. For all experiments denoted “B”, a Corning PC-351 Hot Plate

Stirrer 120VAC was used. Elemental analysis was performed by a Spectro ARCOS2 Inductively Coupled Plasma Atomic Emission Spectrophotometer (ICP-OES).

**Experimental Procedure:**

The following procedure was generally followed for each test. For later tests, the design was changed such that there was a duplicate test (denoted "B") performed simultaneously with the base test (denoted "A") to demonstrate the reproducibility of the tests.

The amount of solid compound needed to produce a 500 mL test solution with the desired species concentration was calculated. The solids for each species were weighed on weigh papers and added to a labeled digestion tube. The digestion tube was sealed tightly and stored in a dark container until the day before the experiment was conducted. Approximately 45 mL of 0.01 M $H_2SO_4$ was added to the digestion tube which was then sealed with a cap. The digestion tube was shaken ten times before being placed in a room temperature water-bath sonicator for thirty minutes. Once sonication was complete, 0.01 M $H_2SO_4$ was added to finalize the solution volume to the 50 mL marker on the digestion tube and the digestion tube was shaken ten times. If there were any solid aggreagates remaining in solution, the digestion tube was shaken ten times again and sonicated for an additional thirty minutes. This process was repeated until there were no longer visible solid aggregates in the solution. The digestion tube was sealed tightly and stored in a dark container overnight. A 600 mL beaker and oxagonal magnetic stir bar were washed, bathed in aquaregia, rinsed in deionized water, oven-dried, and then cooled to room temperature overnight.

There were two measurement systems used in the tests, labeled pH Probe #1 and pH probe #2. All experiments denoted "A" were measured using pH Probe #2 and those denoted "B" were measured using pH Probe #1. The pH and Eh probes were cleaned and calibrated prior to the start of each experiment. The cooled 600 mL beaker and stir bar were placed on a stir plate. All experiments denoted "A" were placed on the Corning stir plate and those denoted "B" were placed on the Cole-Palmer stir plate. Approximately 450 mL of 0.01 M $H_2SO_4$ was added to the beaker, with the exact volume of acid added recorded. The pH and Eh probes were inserted into the acid solution contained within the beaker. The probes were positioned towards the back of the beaker such that they would not move or touch the sides of the beaker at any point throughout the test. For tests denoted "A", the Cole-Palmer stir plate was set to a marked position around 3.2 to ensure all tests were conducted at the same speed. For tests denoted "B", the Corning stir plate was set to a marked position that had previously been established as equivalent to the stir speed of the Cole-Palmer stir plate through thorough testing. This experimental set-up for each stir plate is shown in Figure 3.7.6.

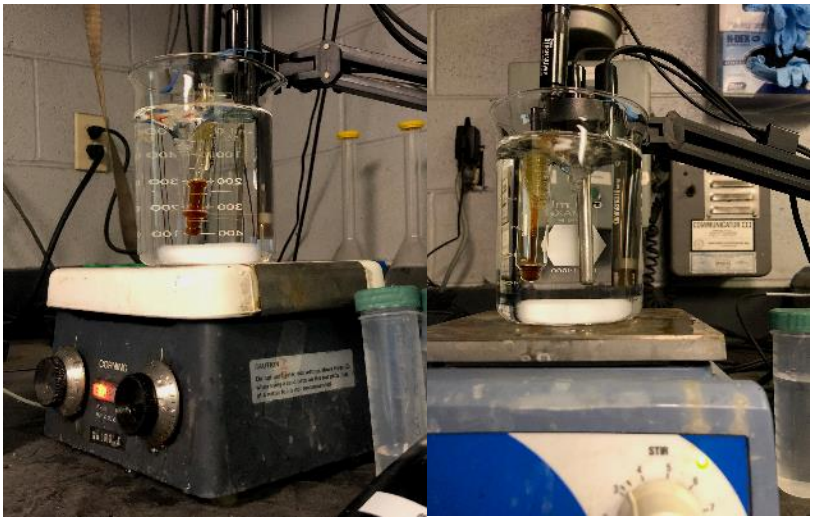


Figure 3.7.6. Appearance of the solution when the feed sample was collected from the single-species Mn2+ precipitation test at pH 1.860 on the Corning stir plate, DD.SPT.20.A.P1 (left). Appearance of the solution when the feed sample was collected from the single-species Mn2+ precipitation test at pH 1.903 on the Cole-Palmer stir plate, DD.SPT.20.B.P1 (right).

The digestion tube containing the digested solids was shaken ten times and then poured into the vortex of the continuously stirred solution contained within the beaker as the test timer was started. Periodic pH, Eh, temperature, and time measurements were recorded. Once the measured pH and Eh values appeared to stabilize for a period of five minutes or longer, a 2 mL aliquot of the test solution in the beaker was removed via a 5 mL pipet and transferred into a 15 mL centrifuge tube. A picture of the test solution was taken to document the appearance of the solution at that pH value. The 2 mL sample in the centrifuge tube was immediately centrifuged at a speed of 3500 rpm for seven minutes. Upon the completion of centrifuging, a 1 mL aliquot of the centifuged sample was carefully removed via a 1 mL pipet from the surface of the sample to avoid the uptake of solid particles deposited on the bottom of the centrifuge tube. The 1 mL aliquot was transferred into another 15 mL centrifuge tube containing 9 mL of 5% (v/v) nitric acid in order to dilute the sample by a factor of ten. To ensure complete transfer of analyte(s) from the pipet tip into the diluted sample, the contents of the diluted sample were drawn up and ejected back into the centrifuge tube five times before the pipet tip was discarded. A cap was used to seal the contents of the diluted sample and the centrifuge tube was vortexed for ten seconds so the contents were mixed thoroughly.

The pH of the solution was raised through the addition of 2 M NaOH. The pH, Eh, temperature, and time were recorded along with the volume and concentration of base added since the last recorded point. Upon the intial addition of base, one or more subsequent measurements of pH, Eh, temperature, and time were recorded, generally at intervals of one minute of greater, to ensure the base was well mixed into the test solution. Once the measurements appeared to indicate the test solution had reached equilibrium, the base addition and monitoring process was repeated until the first pH target was reached. When the pH and Eh measurements indicated the solution had reached equilibrium, the aforementioned sample preparation procedure was repeated and another photo was taken to document the appearance of the solution at that pH value. This process was repeated to obtain the samples for each desired pH target interval for that specific test, aside from minor

deviations from the described procedures that will be explained in the Disscussion section. Elemental analysis of the collected samples was performed via ICP-OES. The samples were analyzed in reverse chronological order in an effort to minimize the effects of analyte crossover contamination.

**Experimental Design Deviations:** In certain pH regions, the pH of the solution is more sensitive to the addition of base due to the logarithmic nature of the pH scale and the consumption of base as precipitation of one or more species occurs. To accommodate this, NaOH was added in concentrations of 2, 0.2, and 0.02 M to ensure the pH of the test solution could accurately match the desired target pH. The volume and concentration of base were recorded upon each addition so the values were converted such that each recorded data point is accompanied by the total volume of 2 M NaOH added in mL. Whenever the the appearance of the solution changed drastically between pH intervals, an additional photo was taken to document said change.

*Calculations* For duplicate tests, values were reported as averages with their respective standard errors. The pH, Eh, total base added, solution volume, and time were recorded for each sample collected. Prior to plotting precipitation values, the results of the elemental analysis were mass balanced to account for the volume changes due to sample removal and base addition throughout the course of the test. After mass balancing, the precipitation values for each element included in a test are calculated by the following equation:

$$Precipitation_i(\%) = \left(1 - \frac{Concentration_{Feed}(ppm) - Concentration_i(ppm)}{Concentration_{Feed}(ppm)}\right) * 100\%$$

**Experimental Precision and Accuracy:** For duplicate tests, all values are reported as averages with its repsective standard error. The pH, Eh, total base added, solution volume, and time were recorded for each sample collected. The recorded values for the mass of the digested solids is shown in Table 3.7.3.

Table 3.7.3. Comparison of the mass needed to produce the targeted concentration in a 500 mL test solution to the actual mass used throughout all tests conducted. The recorded values are reported with their respective standard deviations.

| Compound | Target Concentration | Mass | Actual Mass | Expected Concentration |
|---|---|---|---|---|
| | µg/mL | g | g | µg/mL |
| **$Gd_2(SO_4)_3*8H_2O$** | 10 | 0.0119 | 0.0120 ± 0.0005 | 10.084 ± 0.420 |
| **$Nd_2(SO_4)_3*8H_2O$** | 10 | 0.0125 | 0.0144 ± 0.0021 | 11.520 ± 1.680 |
| **$Sc_2(SO_4)_3*H_2O$** | 10 | 0.0210 | 0.0216 ± 0.0010 | 10.286 ± 0.476 |
| **$Al_2(SO_4)_3*18H_2O$** | 100 | 0.6175 | 0.6179 ± 0.0012 | 100.065 ± 0.194 |
| **$CaSO_4*0.5H_2O$** | 400 | 0.7243 | 0.7248 ± 0.0009 | 400.276 ± 0.497 |
| **$FeSO_4*7H_2O$** | 250 | 0.6223 | 0.6244 ± 0.0016 | 250.844 ± 0.643 |
| **$FeSO_4*7H_2O$** | 500 | 1.2446 | 1.2605 ± 0.0545 | 506.388 ± 21.895 |
| **$Fe_2(SO_4)_3*xH_2O$** | 250 | 0.4475 | 0.4495 ± 0.0037 | 251.117 ± 2.067 |
| **$Fe_2(SO_4)_3*xH_2O$** | 500 | 0.8950 | 0.9079 ± 0.0302 | 507.207 ± 16.872 |
| **$MnSO_4*H_2O$** | 40 | 0.0615 | 0.0622 ± 0.0013 | 40.455 ± 0.846 |

**Results and Discussions:** The test program on Nd precipitation was first completed in various composition of the solution in a modeled system. Without any other contaminants in the solution, 10 ppm of Nd in solution precipitated in 0.01M sulfuric acid system precipitated at around pH 8.5 shown in Figure 3.7.7. With adding 100 ppm of Al in the solution, the precipitation pH of Nd reduced to be around pH 6.5, which indicated that the ionic concentration of solution changes the precipitation pH of Nd. Figure 3.7.8 shows that in the same system (10 ppm Nd and 100 ppm Al), the Al precipitate was completed at pH 4.5, whereas the Nd precipitation at pH 4.5 is about 3%, therefore, co- precipitation or adsorption is not the cause of precipitation pH shift for Nd with presence of Al. With 500 ppm of $Fe^{2+}$ in the solution with Nd, which is a typical ratio of Fe to REEs in a sulfuric acid leaching system, the precipitation pH of Nd shifted lower to pH 6. The state of Fe does not effect the final precipitation completion pH of Nd, however, with presence of $Fe^{3+}$ in the solution, the Nd started to precipitate at pH 3. This is likely due to the $Fe^{3+}$ forms jarosite during precipitation which is a porous and hygroscopic structure which can cause Nd adsorption on the surface and co-precipitation. Intriguingly, with adding Al in the Fe and Nd mixture, the Nd precipitation pH restored to be pH 6-6.5. The results indicated that the presence of Al contributes to the prevention of Nd loss to jarosite by occupying the open sites on the Fe precipitate surface.

The precipitation of Al occurred at pH 4.5 in solution regardless of the recipe of the modelled solution as shown in Figure 3.7.8. The results discovered that Al precipitate, $Al(OH)_3$, formed at pH 4.5 was redissolved when the pH increased to be above pH 9.0. The percentage of redissolution of Al is dependent to the concentration of other elements present in the system. This finding leads to an innovative process modification in the current downstream circuit to further upgrade the RE precipitation cake by redissolve $Al(OH)_3$ using base solution. A detailed discussion on the subject matter will be provided in the upcoming sections.

To examine the precipitation behavior of REEs in the bioleach system with exceptionally high Fe concentration. A modeled system was created with 7000 ppm Fe with 5 ppm Nd, 5ppm Gd, and 1 ppm Sc. The system was modeled based on the bioleach PLS composition obtained in the lab scale leaching tests. Figure 3.7.9 shows the precipitation efficiency of the four abovementioned elements in the combination of solution. The Fe precipitation was completed at slightly lower pH at pH 2.5. In the meanwhile, Nd, Gd, and Sc precipitation pH was also shifted lower to a range of 3.5-4.0. The results support the previous finding that high ionic concentration causes elements precipitate at lower pH. According to the findings discovered in the Fe/RE precipitation system with presence of Al, the precipitation pH of REs can be potentially higher in the real bioleach PLS system due to the presence of Al.

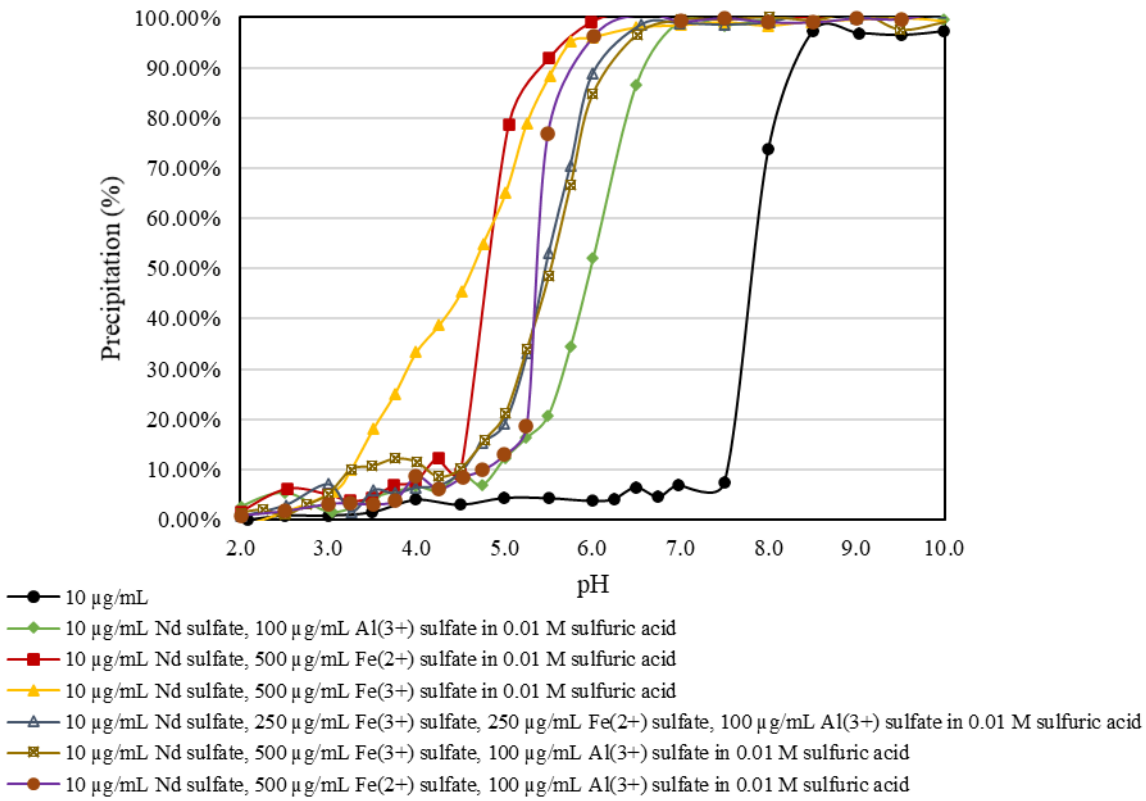



Figure 3.7.7. Nd precipitation efficiency in various modelled solution system.

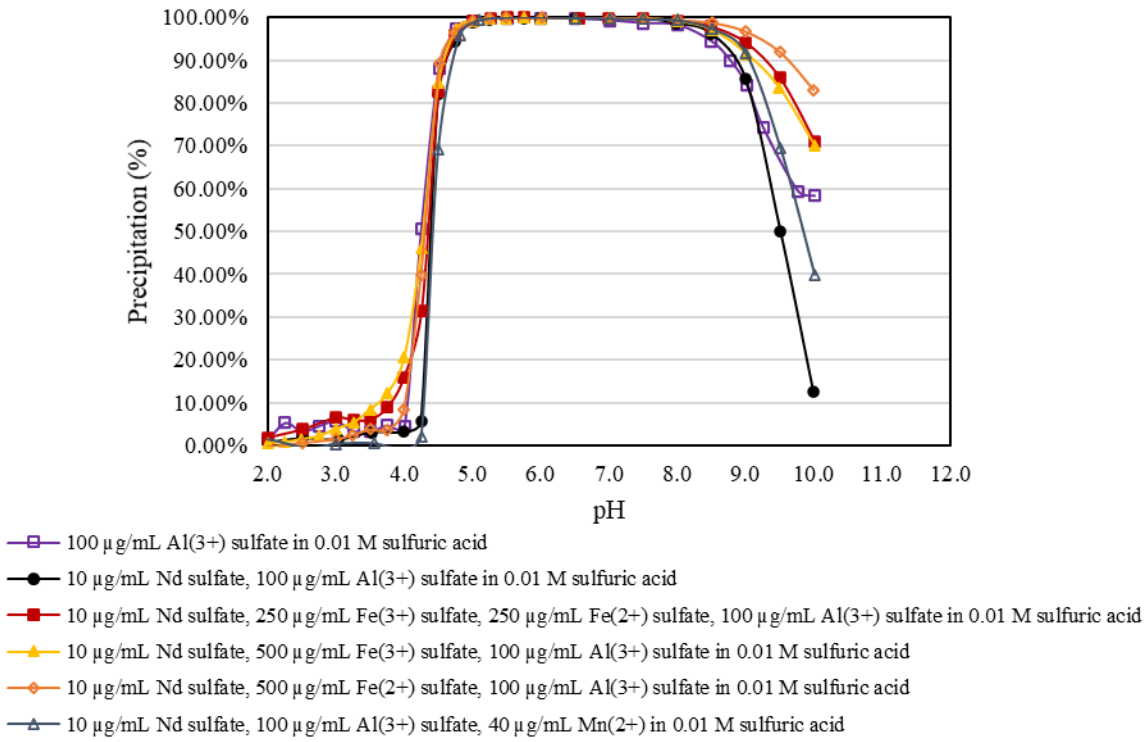



Figure 3.7.8. Al precipitation efficiency in various modelled solution system.

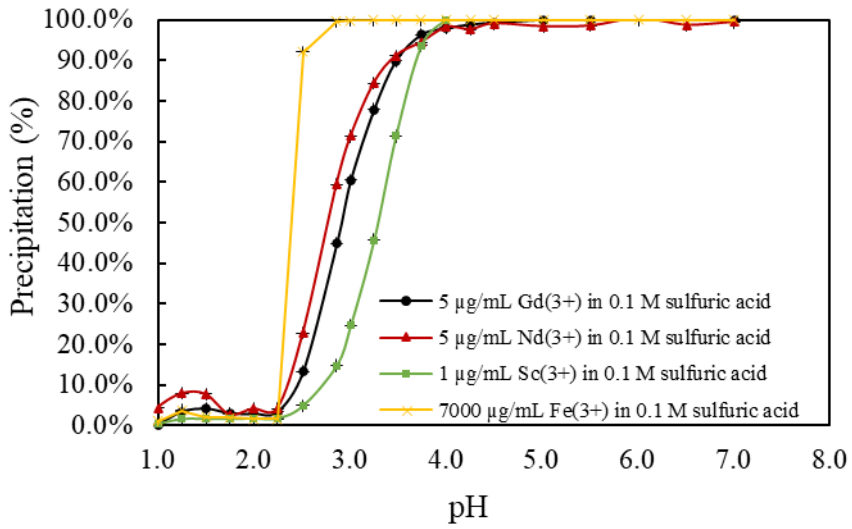



Figure 3.7.9. Precipitation of Sc, Nd, and Gd in a combined model system simulating bioleach PLS with high Fe.

### *1.1.3 PLS Precipitation Behavior using NaOH*

The composition of the leachate solution generated using bioleach acid solution varies significantly after different residence time of leaching as shown in Table 3.7.4. As leaching time increased, a balance between the Al leaching and Fe precipitation occurs in the system. Al dissolution consumed $H^+$ ion and released $OH^-$, then the Fe consumed OH- ion and formed jarosite. To investigate the precipitation behavior of Al and Fe in the bioleach acid leaching system, lab scale experiments were carried out on leachate solution generated at different length of leaching.

**Material and methodology:** To prepare the leachate solution for precipitation test, 200-ml of bioleach acid solution was transferred into a triple neck round bottom flask and preheated to 75°C in a temperature controlled water bath. When the temperature reached the preset level, 40 grams of West Kentucky No. 13 roasted material was added into the lixiviant and leached under 75°C. Agitation was provided using 1.5 inch egg shaped magnetic stir bar at 850 rpm rotation speed. After leaching, the leachate was separated from solid using a vacuum filtration system. The pregnant leachate solution (PLS) was collected without adding any wash water.

The leachate was cooled down to room temperature, then 150 ml of PLS was measured and transferred to a 200-ml beaker for precipitation test. The solution pH, Eh and temperature was measured throughout the precipitation test. 12M and 4M of NaOH solution was prepared to modify the pH for different pH ranges. Since before pH 5, the solution contains significant amount of Al and Fe that requires higher dosage of base, 12M was used for pH adjustment before pH 5. Between pH 5 to pH 8, 4M NaOH solution was used to adjust the pH for a better accuracy and measurement. After pH 8, 12M of NaOH solution was again used due to the total amount of base required to raise pH to 10 is high. A feed sample was taken at the beginning of the precipitation test, then 0.1 ml of prepared NaOH solution was added into solution with pH and Eh monitored continuously. Each increment of base solution was added after the solution pH and Eh reached stability and dosage was recorded along with the mixing time required.

**Results and Discussion:** The leachate solution was darker in color after 50 minutes leaching indicative of the high concentration of $Fe^{3+}$, and, with time, the solution color faded due to Fe precipitation as shown in Figure 3.7.10. The leachate solution Eh reduced from 712 to 412 Rmv with an increase in leaching time, which aligned with the reduced Fe concentration in solution. The PLS pH was measured to be 1.7, 2.1, and 2.3 after 50 min, 3 hours, and 5 hours of leaching, respectively.

The precipitation curve shows that the Fe precipitation occurred at around pH 3.2 to 3.5 which was observed in several acid mine drainage precipitation system studied in the past. The Al and Sc precipitation curves also remained unchanged compared to the acid mine drainage and sulfuric acid leaching system. However, 96% of TREEs precipitated at pH 5.5 when the starting Fe concentration was as high as 7000 ppm in the 50 min PLS as shown in Figure 3.7.11. The TREE precipitation pH was much lower than that of the acid mine drainage (AMD) system as well as the sulfuric acid leaching solution precipitation system. However, it is slightly higher than the precipitation pH point observed in the modelled bioleach solution system where the REs precipitated at pH 4-4.5 discussed in the previous section. This phenomenon can be likely explained with the presence of Al, that Al can be adsorbed onto and/or precipitated with the porous and hygroscopic $Fe(OH)_3$ structure to prevent part of the REs to be co-precipitated with Fe.

Table 3.7.4 Major element concentration in leachate generated from West Kentucky 13 material after different leaching time.

| Leaching Time (min) | Concentration in leachate (ppm) | | | |
|---|---|---|---|---|
| | TREE | Al | Ca | Fe |
| 0 | 2.5 | 31 | 652 | 10359 |
| 5 | 10.8 | 401 | 718 | 10111 |
| 15 | 15.0 | 750 | 776 | 9761 |
| 30 | 17.1 | 1123 | 845 | 9206 |
| 60 | 18.1 | 1799 | 910 | 6961 |
| 90 | 18.0 | 2343 | 918 | 4996 |
| 120 | 16.9 | 2780 | 906 | 3475 |
| 180 | 14.9 | 3336 | 822 | 1664 |
| 240 | 13.0 | 3704 | 797 | 821 |
| 300 | 12.5 | 3854 | 758 | 565 |

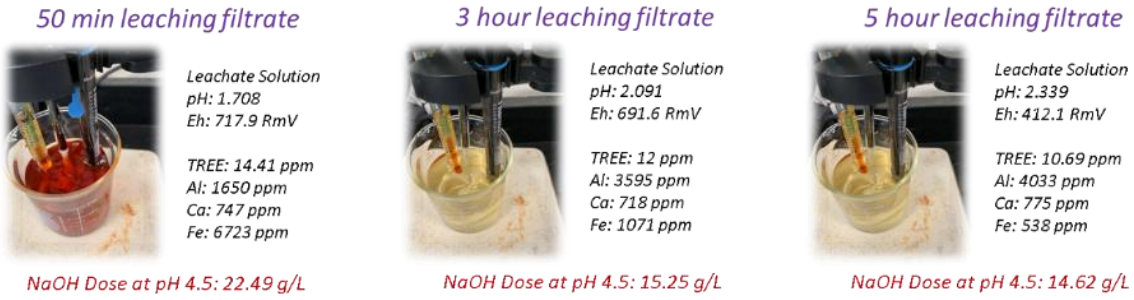



Figure 3.7.10. Composition and characteristic of leachate obtained after different length of leaching time.

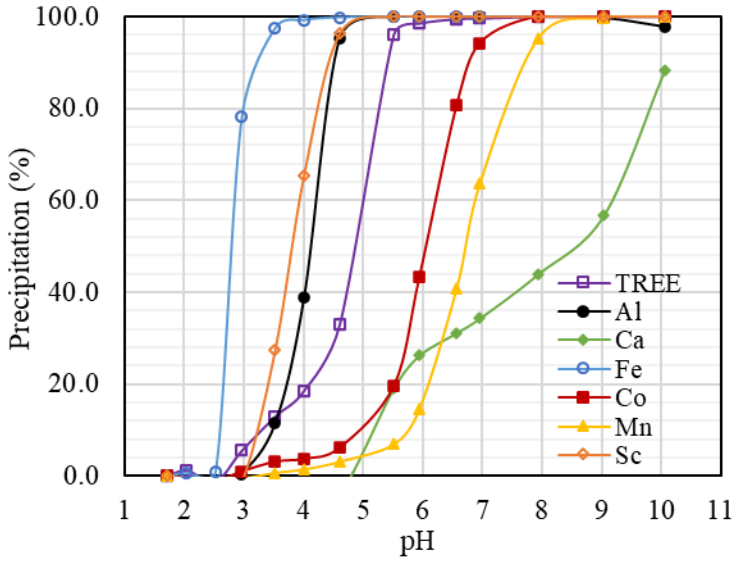


Figure 3.7.11. Precipitation curve of major element of interest in leachate collected after 50 minutes of leaching. (Bioleach acid leaching, West Kentucky No. 13 material, pH modifier: NaOH.)

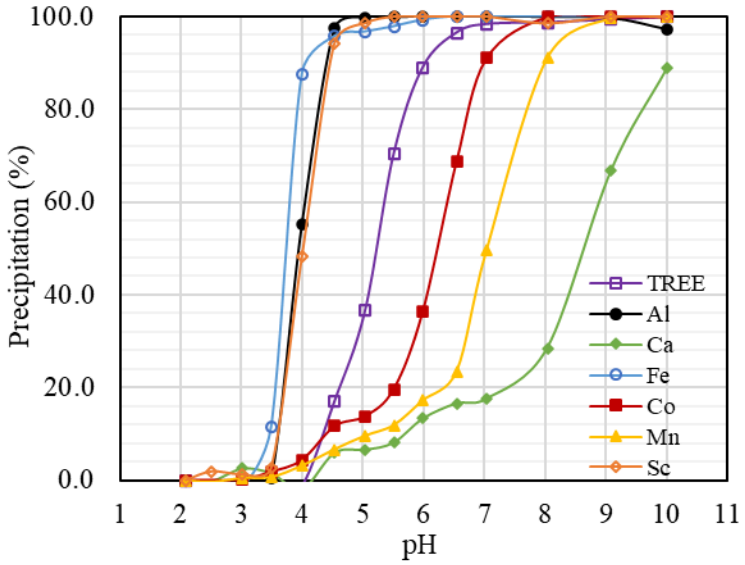



Figure 3.7.12. Precipitation curve of major element of interest in leachate collected after 3 hours of leaching. (Bioleach acid leaching, West Kentucky No. 13 material, pH modifier: NaOH.)

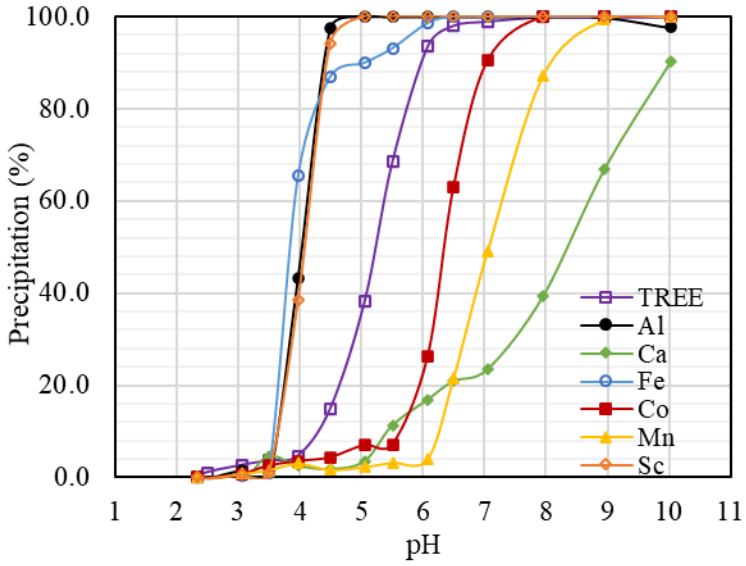



Figure 3.7.13. Precipitation curve of major element of interest in leachate collected after 5 hours of leaching. (Bioleach acid leaching, West Kentucky No. 13 material, pH modifier: NaOH.)

**Base consumption:** The results show that an increasing amount of Al and Fe requires elevating the amount of base solution added to realize precipitation. The major overall reaction in the precipitation of Fe and Al with the presence of sulfate can be expressed as:

Fe Precipitation:

$Fe_2(SO_4)_3 + 6NaOH = 2Fe(OH)_{3\,(s)} + 3Na_2SO_4$

$FeSO_4 + 2NaOH = Fe(OH)_{2\,(s)} + Na_2SO_4$ (small amount of Fe present as $Fe^{2+}$)

$4Fe(OH)_2 + O_2 + 2H_2O = 4Fe(OH)_{3\,(s)}$

Al Precipitation:

$Al_2(SO_4)_3 + 6NaOH = 2Al(OH)_{3\,(s)} + 3Na_2SO_4$

The amount of base needed to precipitate Al and Fe can be calculated. For instance, 1000 ppm (1000mg/L = 1g/L) of Al in solution requires:

$OH^-$ (mol/L) = 3 * $Al^{3+}$ (mol/L) = 3*1 g/L / 27 g/mol = 0.111 (mol/L)

$OH^-$ (mol/L) = 3 * $Fe^{3+}$ (mol/L) = 3*1 g/L / 56 g/mol = 0.054 (mol/L)

Therefore, in terms of absolute element concentration in solution, Al consumes more base to be precipitated than that of $Fe^{3+}$ as shown in Figure 3.7.14. As there exists a balance between the Al leaching and Fe precipitation during leaching using bioleach acid (Al dissolution consumed H+ ion and released OH-, then the Fe consumed OH- ion and precipitate), calculation was performed to determine the base consumption as a function of leaching time.

Using the Al and Fe concentration presented in Table 3.7.4, the amount of base needed to completely precipitate Al and Fe were calculated and plotted in Figure 3.7.15. The plot on the left shows the amount of $OH^-$ needed to react with Al and Fe individually, whereas the plot on the right shows the total $OH^-$ required to remove both Al and Fe as a function of leaching time. The calculation indicated that the PLS generated after 1 hour, 3, hour, and 5 hour of leaching time requires about equivalent to 0.618M, 0.501M, and 0.496M of NaOH in the system to remove Fe and Al, respectively. Convert the molarity of NaOH to mass per liter, the theoretical required NaOH dosage is calculated to be 24.7g/L, 20.0g/L, and 19.84 g/L. These dosages are slightly higher than that of the measured NaOH consumption at pH 4.5 in the experiments shown in Figure 3.7.10 since the Al and Fe in solution was not completely precipitated at pH 4.5, which also indicated that element Fe in the 3-hour and 5-hour leachate is partially present in ferrous state.

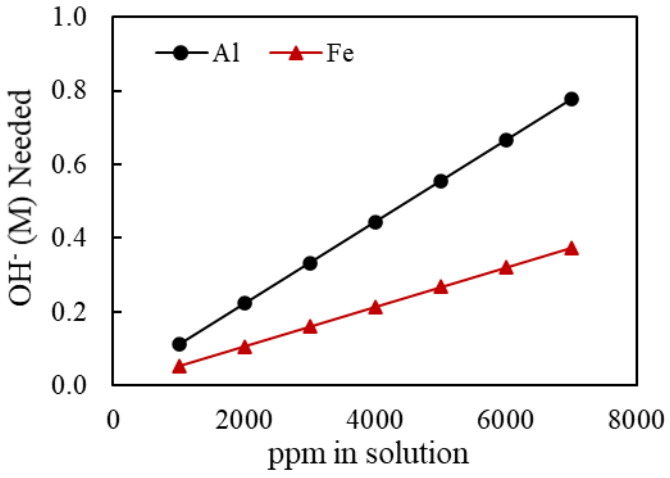



Figure 3.7.14. The amount of $OH^-$ required to precipitate $Al^{3+}$ and $Fe^{3+}$ in leachate.

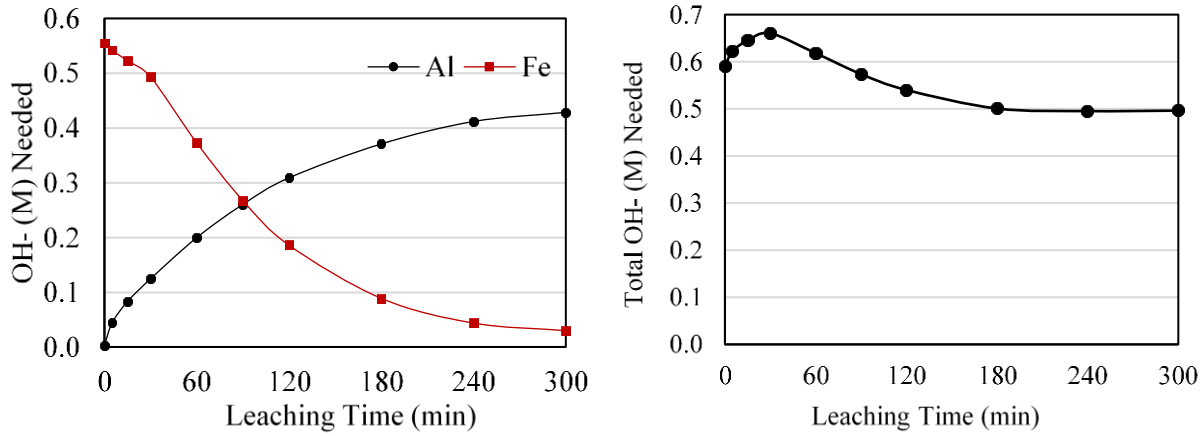



Figure 3.7.15. Theoretical calculated consumption of $OH^-$ for Al and Fe precipitation from bioleach PLS generated at various leaching time.

### *1.1.4 Selective Precipitation using Calcite*

To examine the proposed concept of using calcite as pH modifier to replace NaOH, the following study was conducted on leachate generated from sulfuric acid and bioleach acid. While using calcite as chemical reagent for Al and Fe precipitation with the presence of sulfate ions, the overall reactions include:

$CaCO_3 + H_2SO_4 = CaSO_{4\,(s)} + H_2O + CO_{2\,(g)}$

$3\,CaCO_3 + Al_2(SO_4)_3 + 6H_2O = 3CaSO_{4\,(s)} + Al\,(OH)_{3(s)} + H_2O + CO_{2\,(g)}$

$3\,CaCO_3 + Fe_2(SO_4)_3 + 6H_2O = 3CaSO_{4\,(s)} + Fe\,(OH)_{3(s)} + H_2O + CO_{2\,(g)}$

**Sulfuric Acid PLS:** The precipitation efficiency of Al and Fe using calcite was first carried out on leachate generated from roasted West Kentucky 13 coarse refuse leached with 0.05M sulfuric acid. The leachate used in this study was produced in the pilot scale leaching operation with solid concentration of 15% (w/v) at 75°C. The calcite used for pH adjustment in this section was the plus 100 mesh fraction of the pyrite concentrate collected from river view plant spiral rougher-cleaner-tailing. The material contains 68% of calcite, 6.3% pyrite, 6.4% silica, and 4.9% kaolinite.

Four precipitation tests were carried out with adding different dosage of calcite into 100-ml of leachate solution. As shown in Table 3.7.5, using 1 g/L of calcite, the pH of the leachate solution increased from 3.8 to 4.5 after 30 minutes of residence time. The pH and Eh were also recorded at different time intervals as shown in Figure 3.7.16. The results show that with 1g/L calcite the reaction was approaching equilibrium after 20 minutes of residence time. The solid loss was 36% by weight indicating that even the reaction reached equilibrium, the calcite was not completely dissolved. The calcite dissolution causes formation of $CaSO_{4(s)}$ which created passivation on the particle surface to prevent further reaction.

While increasing the calcite dosage to 5 g/L, the solution pH increased from 3.8 to about 6.0 after 30 minutes, the kinetic data shows that the pH was not reaching equilibrium. However, increasing the calcite dosage to 20g/L and 50 g/L, the resulted ending pH after 30 minutes of residence time appeared to be same as the 5g/L dosage. The ending pH of overdosed calcite was around pH 6

which indicated that at pH 6, the calcite concentration in solution reached the saturation point. The total dissolved solid at 50g/L, 20g/L, and 5g/L dosage was 1.6 g/L, 1.3g/L, and 1.1g/L, respectively.

The precipitation efficiency of REs and major contamination ions using different dosage of calcite is shown in Figure 3.7.17. With 1g/L calcite, 13.4% TREE was precipitated with 40% of Al at ending pH of 4.5. The excessive amount of calcite can precipitate nearly 100% of Al at ending pH of 6, however, the precipitation efficiency of TREEs increased from 21% to 43% with increasing the calcite dosage from 5g/L to 50g/L even though the ending pH was almost the same. The results indicated that the calcium and sulfate reaction contributed to part of the RE precipitation. Since the coal calcite material contains small amount of clay and pyrite, Co, Mn, and Ni was leached out during calcite dissolution.

The precipitation efficiency of REs, Al, and Fe using calcite and NaOH was very comparable as shown in Figure 3.7.18. The Al and Fe precipitation efficiency were slightly different since the coal calcite also contains impurities that can be leached out during pH adjustment. Results show that the to remove Al at pH of 4.5, calcite can be a replacement of NaOH in the sulfuric acid leaching system.

Table 3.7.5. The initial and ending pH and Eh recorded for each precipitation test with various calcite dosage.

| Test ID | Calcite Dose | Initial (Time =0 min) | | | Final (Time =30 min) | | | Wt. Loss (%) |
|---|---|---|---|---|---|---|---|---|
| | | pH | Eh | Temp | pH | Eh | Temp | |
| DTK.C15.A2 | 50 g/L | 3.709 | 365.3 | 28.1 | 6.102 | 140.4 | 25.0 | 3.2 |
| DTK.C15.A3 | 20 g/L | 3.781 | 363.5 | 25.7 | 6.066 | 158.5 | 24.0 | 6.5 |
| DTK.C15.A4 | 5 g/L | 3.769 | 367.0 | 24.4 | 5.969 | 102.1 | 23.9 | 22.7 |
| DTK.C15.A5 | 1 g/L | 3.813 | 376.8 | 23.5 | 4.520 | 305.9 | 23.4 | 36.18 |

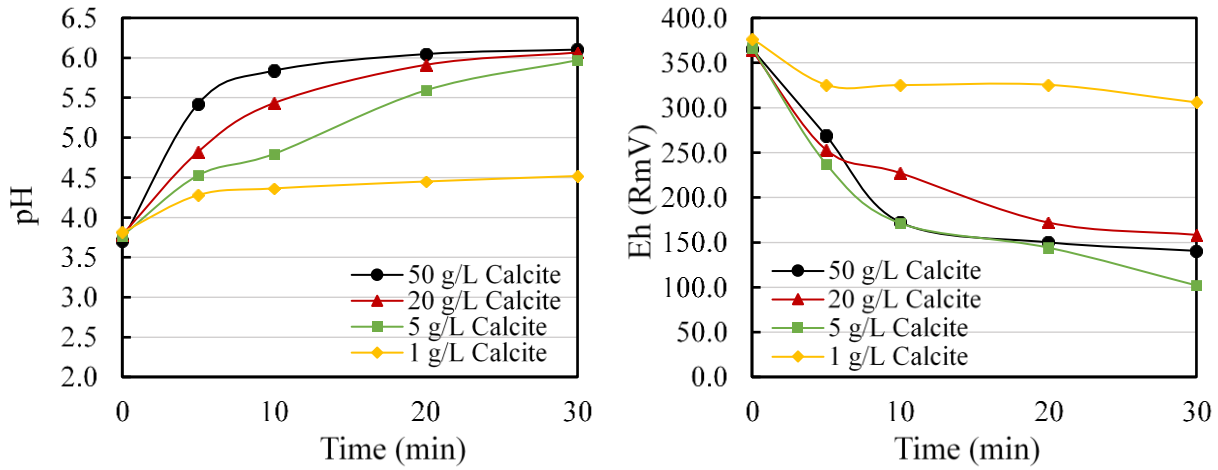



Figure 3.7.16. Recorded pH and Eh value over time using calcite at various dosages.

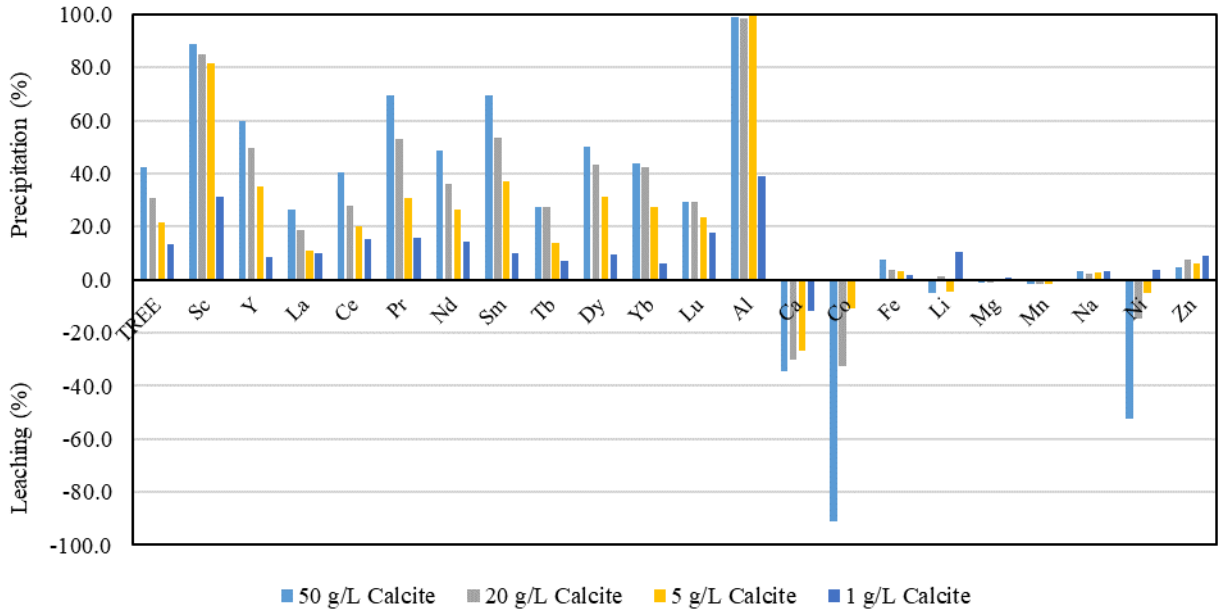



Figure 3.7.17. Precipitation efficiency and leaching recovery of major elements using coal calcite as pH modifier at different dosage in sulfuric acid leaching solution.

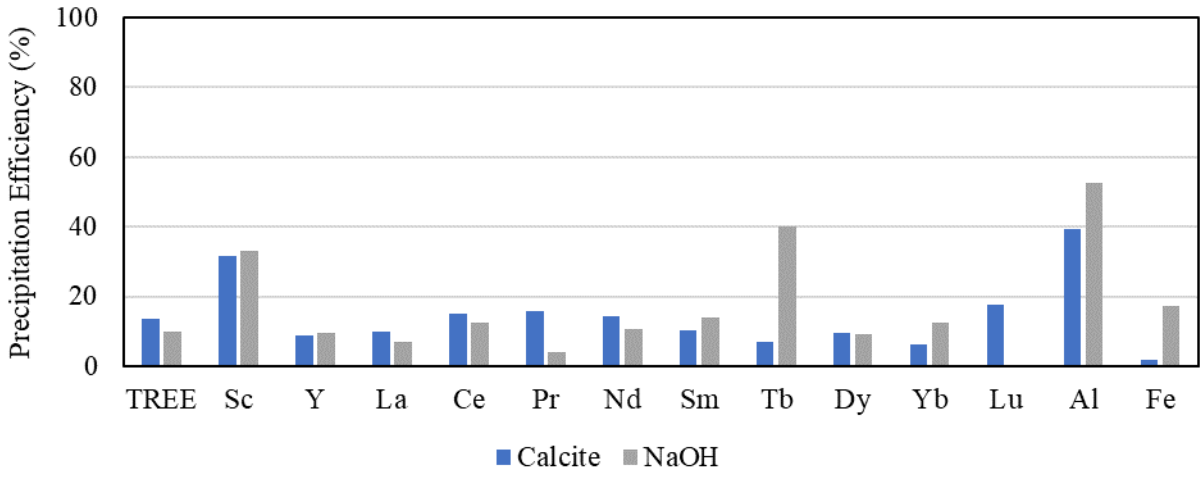



Figure 3.7.18. Comparison on precipitation efficiency of major elements of interest using 1 gram/L calcite versus 0.13 gram/L of NaOH at pH 4.5.

**Bio oxidation PLS:** The precipitation efficiency using coal calcite was then investigated in the bioleach acid leached PLS system. Due to the high concentration of ions present in the bioleach PLS, the calcite precipitation can be complicated. The solution used in this investigation was generated using roasted West Kentucky 13 coarse refuse material leached with bioleach acid with solid-to-liquid ratio of 1/5 (w/v) at 75°C for 50 minutes. Based on the previous section, the calcite dissolution occurs rapidly in solution when the calcite dosage was low. Therefore, the experiment was designed to gradually add 5 grams of calcite into the PLS at every 30 minutes of residence time interval. The pH, Eh and temperature was continuously recorded at 5 minutes time interval,

and 2-ml sample was collected after 5, 15, 30 minutes after each increment amount of calcite was added as shown in table below. The sample was immediately centrifuged and the effluent was diluted 10, 100, and 1000 times for elemental concentration analysis using ICP-OES.

Table 3.7.6. Timeline of actions in precipitation test on 100-ml bioleach PLS using calcite.

| Total Time (min) | Time in Step (min) | Action |
|---|---|---|
| 0 | 0 | Add 5 grams coal calcite |
| 5 | 5 | Sample |
| 15 | 15 | Sample |
| 30 | 30 | Sample |
| 30 | 0 | Add 5 grams coal calcite |
| 35 | 5 | Sample |
| 45 | 15 | Sample |
| 60 | 30 | Sample |
| 60 | 0 | Add 5 grams coal calcite |
| 65 | 5 | Sample |
| 75 | 15 | Sample |
| 90 | 30 | Sample |
| 90 | 0 | Add 5 grams coal calcite |
| 95 | 5 | Sample |
| 105 | 15 | Sample |
| 120 | 30 | Sample |

The pH and Eh of the bioleach PLS after 50 minutes leaching was 1.7 and 790 RmV. The undiluted PLS contains about 17 ppm of TREEs, 2288 ppm of Al, 823 ppm of Ca, and 6542 ppm of Fe. After adding the first 5 gram of calcite, the solution pH increased immediately to 2.6 within 5 minutes, the Eh reduced to 682 RmV as shown in Figure 3.7.19. The solution was extremely bubbly and orange color precipitate was observed within 5 minutes. The pH and Eh did not change significantly from 5 minutes to 30 minutes. From 30 minutes to 60 minutes, another 5 grams of calcite was added. The pH increased gradually in this stage of precipitation from 2.644 to 3.182 over 30 minutes of reaction time, which indicated that complex reactions occurred simultaneously in this stage. The Eh value within this period reduced significantly from 672 to 386 RmV, which indicates that a significant amount of Fe was removed. In the third and fourth stage of calcite precipitation, the pH increased from 3.2 to 4.1, whereas the Eh value was almost unchanged. It indicated that the Al was the dominant precipitation species in these two stages.

The precipitation efficiency of TREE, Al, and Fe is plotted in Figure 3.7.20. The results show that Fe was mostly precipitated within pH 2.6 to 3.2, which is comparable to the results obtained from precipitation test using NaOH. The Al was gradually precipitated over the pH range of 3.2 to 4.1, which is slightly lower range of pH compared to the precipitation system using NaOH. The TREEs precipitation pH shifted to the lower pH range significantly. At pH 4.1, 80% of TREEs were precipitated. This is likely due to the high ionic concentration in the system and the introduction of excessive amount of Ca to form calcium sulfate precipitate. The Ca concentration in solution increased rapidly to 1200 ppm after the first dose of calcite was added, then decreased significantly to 700 ppm which is lower than the feed solution. In the meanwhile, 28% TREE precipitated after

15 minutes of reaction at pH of 2.6, which indicated that the TREE precipitation is correlated to the Ca concentration in solution. The TREE precipitation efficiency in the undiluted bioleach PLS precipitation system at 30, 60, 90, and 120 minutes of reaction was 31%, 57%, 71%, and 80%, respectively.

The precipitation efficiency of TREEs was mostly contributed by the light REs precipitation (i.e. La and Ce) as shown in Figure 3.7.21. Scandium precipitation occurred simultaneously with Al precipitation which was observed in the past study whereas yttrium remained unaffected. The results indicated that the light REEs were likely co-precipitated with the gypsum formed during precipitation due to the excessive amount of Ca in solution.

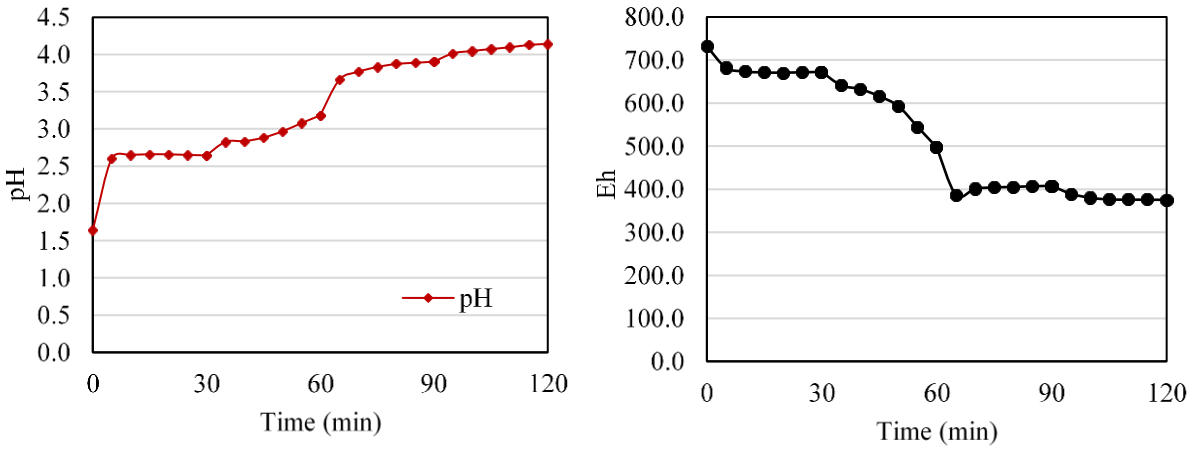



Figure 3.7.19. Solution pH and Eh recorded during precipitation test using calcite in the undiluted bioleach PLS.

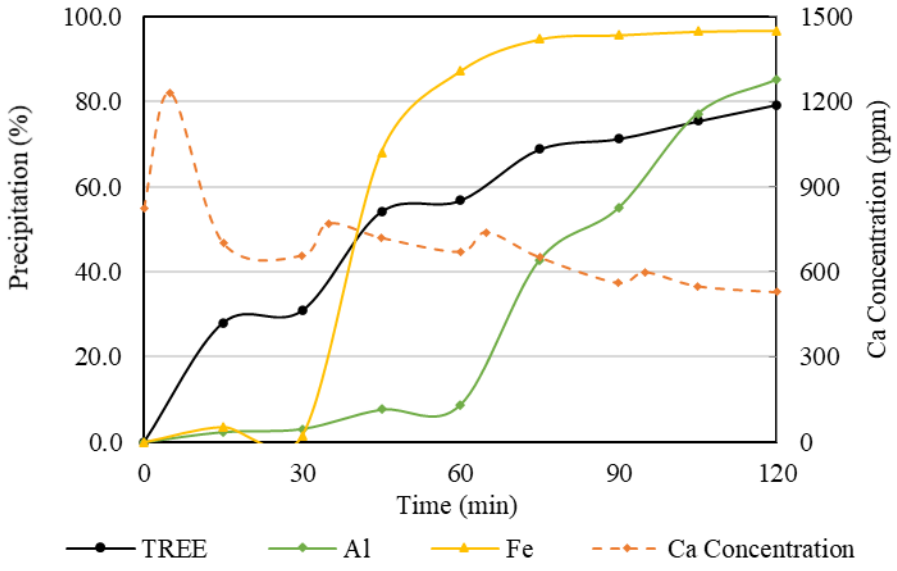



Figure 3.7.20 Precipitation efficiency of TREEs, Al, Fe and Ca concentration as a function of time during precipitation test using calcite in the undiluted bioleach PLS.

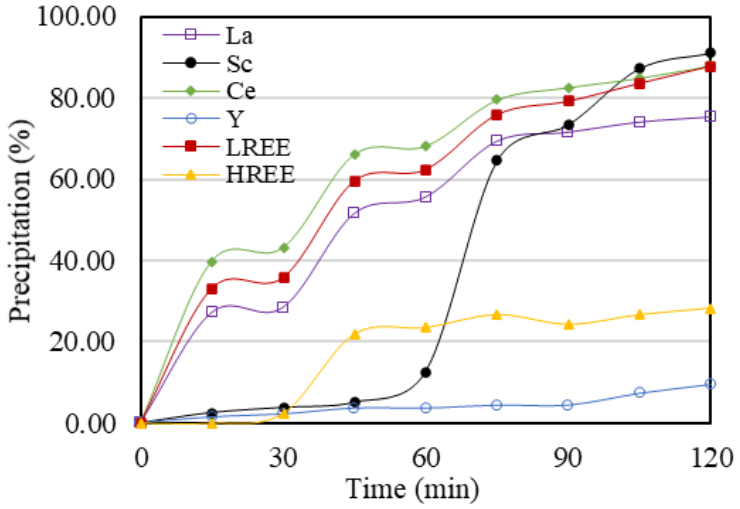



Figure 3.7.21 Precipitation efficiency of major rare earth elements as a function of time during precipitation test using calcite in the undiluted bioleach PLS.

Based on the observation of the precipitation behavior in the undiluted PLS, tests were carried out on diluted system with a hypothesis that lower ionic activity and lower Ca concentration in solution would increase the precipitation pH of REEs. The same precipitation tests were conducted on PLS diluted 2 times and 4 times by volume using DI water. The mass of calcite added to the volume of original undiluted PLS remained the same in the diluted system. The pH and Eh value recorded on the 2x diluted system showed similar trend as that of the undiluted system. The ending pH after 120 minutes was 3.9 which was slightly lower than that of the undiluted system (4.1). However, the end Eh value was significantly lower (100 RmV) due to the diluted concentration of Fe Figure 3.7.22. The precipitation efficiency of Fe in the 2x diluted system was similar to the undiluted system, of which 86% of Fe was removed at pH 3.2. The Al precipitation was restored to about 40% at pH 3.9 instead of 80% at pH 4.1. The TREE precipitation reduced to about 40% after 120 minutes of reaction (Figure 3.7.23). The Ca concentration observed in the 2x diluted system was in the same manner as the undiluted system, however, a higher peak concentration of 1550 ppm was observed after the first dose of calcite added. This is likely due to the lower solid to liquid ratio in calcite dissolution favors the reaction. The major REs contributed to the TREE precipitation were still the light REEs as shown in Figure 3.7.24, however, much lower than that of the undiluted system.

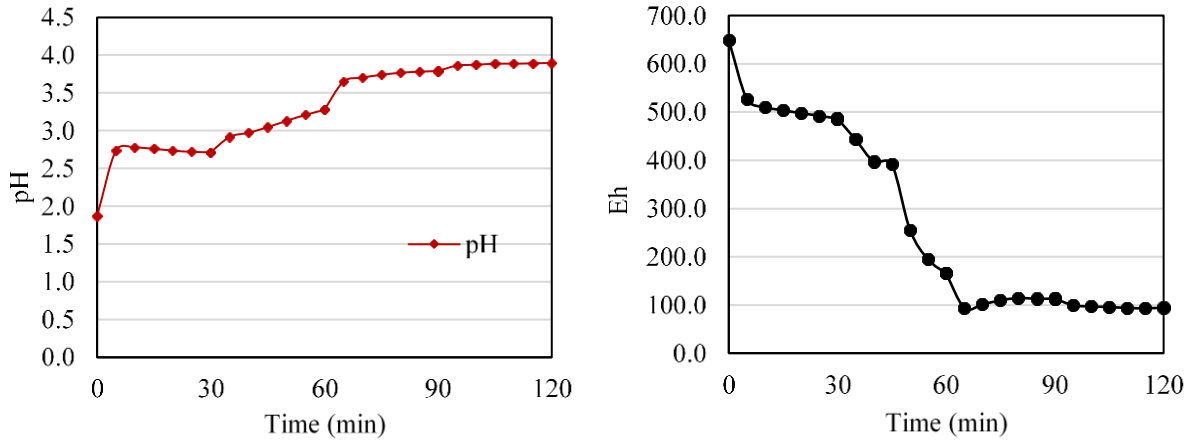



Figure 3.7.22. Solution pH and Eh recorded during precipitation test using calcite in the 2x diluted bioleach PLS.

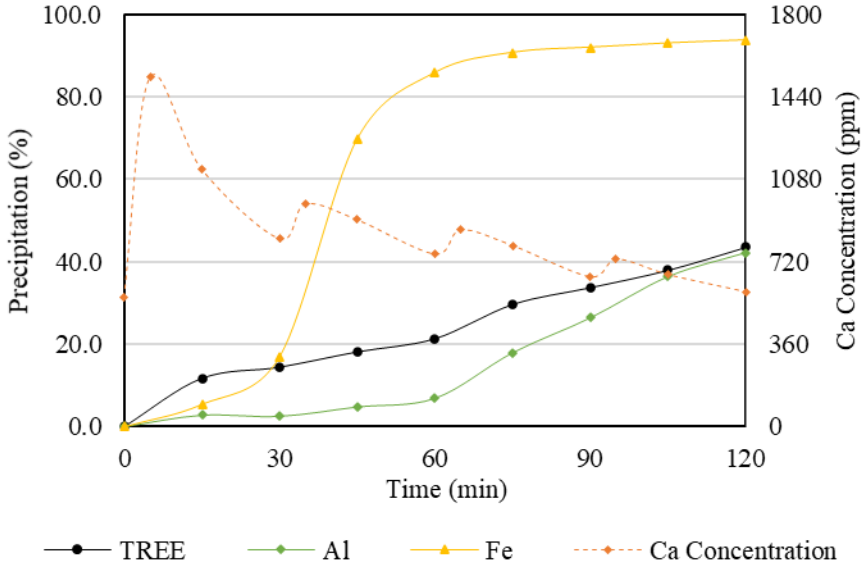



Figure 3.7.23 Precipitation efficiency of TREEs, Al, Fe and Ca concentration as a function of time during precipitation test using calcite in the 2x diluted bioleach PLS.

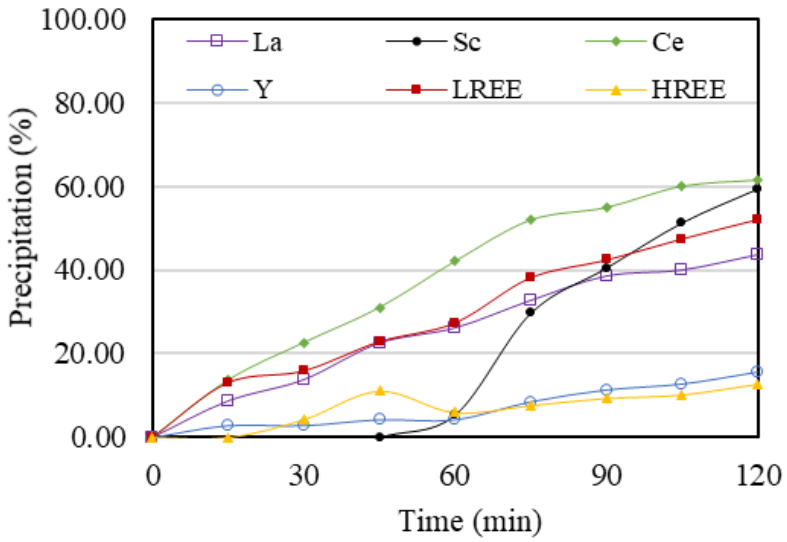



Figure 3.7.24. Precipitation efficiency of major rare earth elements as a function of time during precipitation test using calcite in the 2x diluted bioleach PLS.

In the 4x diluted system, the initial PLS pH was 2.0 due to the large volume of dilution, however, after 4 doses of calcite added, the ending pH reached 4.0 regardless. The Eh value reduced from about 450 RmV to below zero, which indicates that the remaining 7% Fe in solution is mostly ferrous state. The TREE precipitation efficiency after 4 doses of calcite reduced significantly to about 29% with 36% Al removal. The concentration of Ca in solution changed compared to the undiluted and 2x diluted system, likely due to the higher initial pH which hindered the calcite dissolution. The peak concentration of Ca occurred after the third dose of calcite addition at pH 3, where 75% Fe was already precipitated. It indicated that in the 4x diluted system, the first 2 doses of calcite contributed to Fe removal. The results on 4x diluted system are plotted in Figure 3.7.25, Figure 3.7.26, and Figure 3.7.27. Compared to the undiluted and 2x diluted system, the selectivity of RE precipitation restored significantly in 4x dilution system.

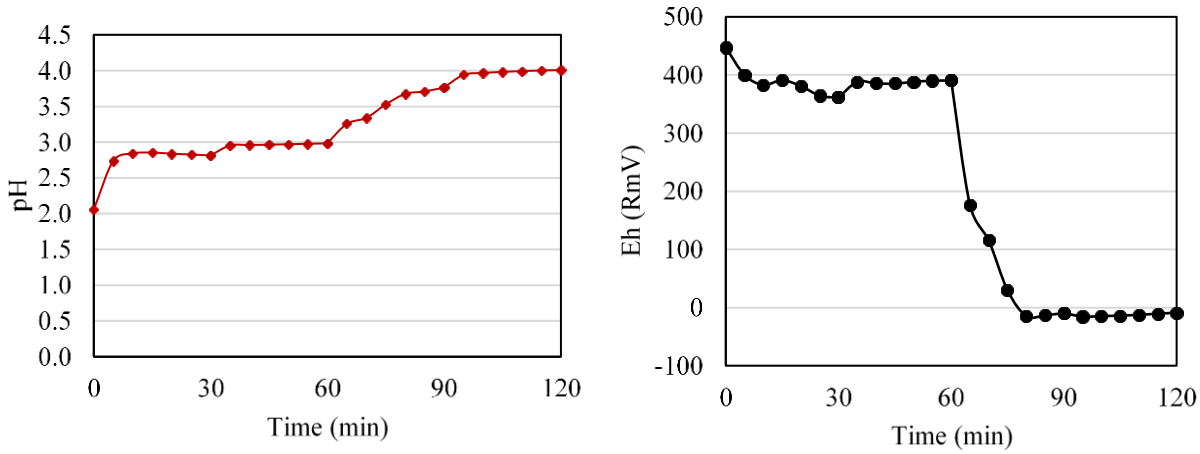



Figure 3.7.25 Solution pH and Eh recorded during precipitation test using calcite in the 4x diluted bioleach PLS.

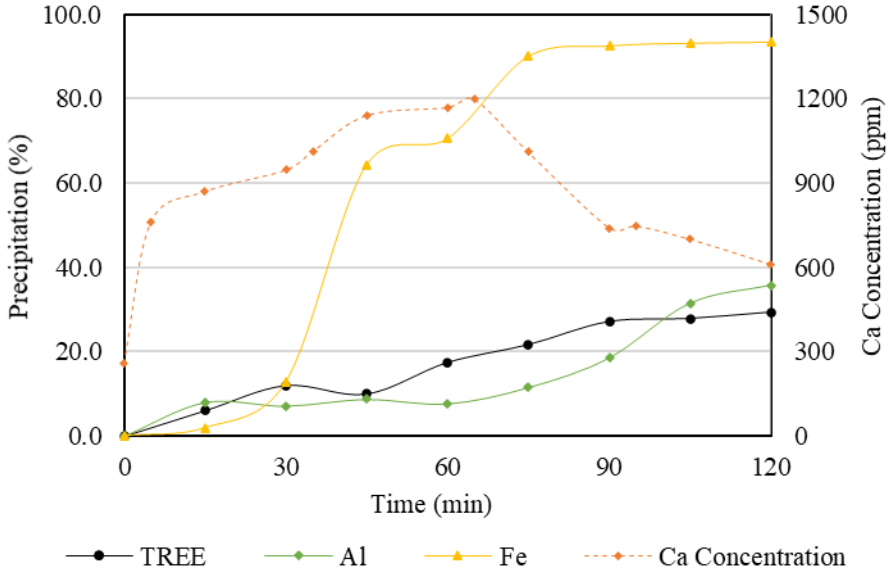



Figure 3.7.26. Precipitation efficiency of TREEs, Al, Fe and Ca concentration as a function of time during precipitation test using calcite in the 4x diluted bioleach PLS.

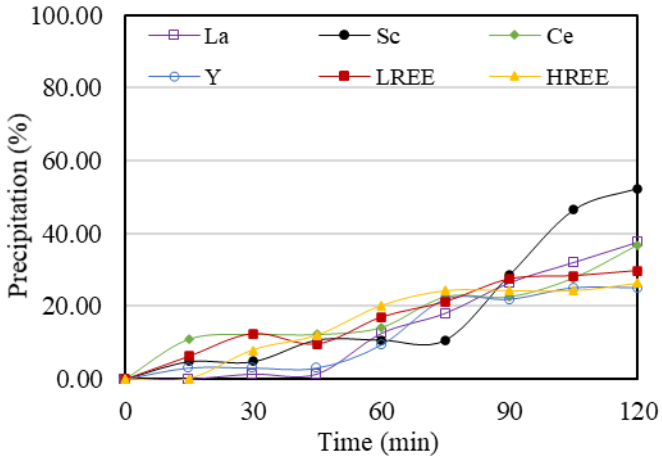



Figure 3.7.27. Precipitation efficiency of major rare earth elements as a function of time during precipitation test using calcite in the 4x diluted bioleach PLS.

### *1.1.5 Leaching Characteristics of WK-13 using bio-acid*

**Sample Collection and Characterization:** Bio-leachate of approximately 10 liters were sent to the Mineral Processing Lab at Virginia Tech from the University of Utah. The leachate was then split and stored in glass jars, which were tightly sealed to reduce oxidation. The appearance of the as-received and split samples is shown in Figure 3.7.28. However, with time going by, the color of the leachate changed in varying degrees (Figure 3.7.29), which means the oxidation might occur

in the bio-leachate. The details regarding the composition of the bio-leachate that marked on the original container is shown in Table 3.7.7.




Figure 3.7.28. Bio-leachate received from the University of Utah: (left) original container; (right) split samples.



6

Figure 3.7.29. Appearance of the bio-leachate after being sealed in glass jars for 30 days.

Table 3.7.7. Information labelled on the original container from the University of Utah.

| Substance | Concentration |
|---|---|
| Ferric sulfate<br>Ferrous sulfate | 20~30 g/L |
| Ammonium sulfate | ~3.0 g/L |
| Dipotassium phosphate | ~0.5 g/L |
| Magnesium sulfate | ~0.25 g/L |
| Potassium chloride | ~0.1 g/L |
| Calcium nitrate | ~0.01 g/L |

A small volume of the bio-leachate was sent to Kentucky Geological Survey to measure the concentration of major anions using Ion chromatography (IC). REEs and other metal ions concentration in the natural leachate was analyzed at the pilot plant facility using an inductively coupled plasma optical emission spectrometry (ICP-OES). The results are showing in Table 3.7.8 and Table 3.7.9, respectively.

Table 3.7.8. Ion-chromatography results of the bio-leachate.

| Anions | Concentration/ppm |
|---|---|
| $SO_4^{2-}$ | 26,210 |
| $NO_3^-$ | 54 |
| $NO_3^-$ | 12.2 |

Table 3.7.9. Concentrations of metal ions in the bio-leachate measured using ICP-OES.

| Elements | TREEs | LREEs | HREEs | Sc | Y | La | Ce | Pr | Nd |
|---|---|---|---|---|---|---|---|---|---|
| Concentration/ppm | 1.21 | 0.75 | 0.46 | 0.098 | 0.11 | 0.047 | 0 | 0.16 | 0.13 |
| Elements | Sm | Eu | Gd | Tb | Dy | Ho | Er | Tm | Yb |
| Concentration/ppm | 0.31 | 0.0036 | 0.084 | 0.044 | 0 | 0 | 0 | 0 | 0.12 |
| Elements | Lu | Al | As | Ca | Co | Cu | K | Li | Mg |
| Concentration/ppm | 0 | 33.25 | 2.599 | 154.2 | 21.6 | 22.93 | 49.6 | 0.013 | 68.67 |
| Elements | Mn | Na | Ni | Zn | | | | | |
| Concentration/ppm | 11.027 | 11.63 | 10.976 | 8.098 | | | | | |

**Experimental Procedure**: Solution pH was measured using a Fisherbrand Accumet AP115 Portable pH Meter with an accuracy of 0.001 pH unit. pH value of the natural Bio-leaching leachate sample measured to be 1.32. According to the pH value, sulfuric acid solutions of different concentrations were prepared to compare the leaching behaviors of REEs the two different types of lixiviant. The 1.8-2.2 SG of western Kentucky No. 13 run-of-mine coal was calcined in a muffle furnace under 600°C. The temperature in the muffle furnace was elevated at a rate of 10°C/min to the target temperature. The sample was calcined under the target temperature for 120 min. After calcination, the temperature in the muffle furnace was naturally cooled to room temperature, which took around 12 h. A static air condition was used during the overall calcination process. After calcination, the products were collected and used for the acid leaching.

Acid leaching tests were conducted in a round-bottom flask, which was submersed in a water bath for temperature control. The flask was connected to a condenser to prevent the liquid loss due to

evaporation. The leaching tests were performed under the uniform conditions of 75°C and 1 wt.% solid concentration. Magnetic stirring at a certain rotation speed was used to ensure sufficient mixing. Solid of two grams were added to lixiviant of 200 mL for each test. Slurry samples (10 mL) were collected 5, 15, 30, 60, 90 and 120 min from the beginning of the leaching process. The slurry samples were centrifuged at 4,000 rpm. Elemental concentrations in the supernatants were measured using ICP-OES. After 120 min of leaching reaction, the residual slurry was centrifuged. The centrifuged cakes were dried at 60°C followed by complete digestion for ICP measurements. To assess the quality of the test data, elemental contents ($E_f$, ppm) in the feed solids were back calculated using the following equation:

$$E_f = 100 \times (C_s \times m_s + C_l \times V_l) \times m_i/(m_i - m_m)$$

where $C_s$ and $C_l$ represent the elemental concentrations (ppm, representing mg/kg for solid and mg/L for liquid) in the solid residue and final leachate; $m_s$ (kg) weight of the leaching solid residue; $V_l$ (liter) volume of the final leachate; $m_i$ (kg) weight of the total leaching slurry; and $m_m$ (kg) weight of all the representative samples collected during the leaching process. The difference between the back-calculated contents and the values measured directly using an ICP-OES was within ± 5%, indicating minor experimental errors.

Leaching recovery ($R_i$, %) was calculated using the following equation:

$$R_i = 100\% \times C_i \times V/(E_f \times 0.002)$$

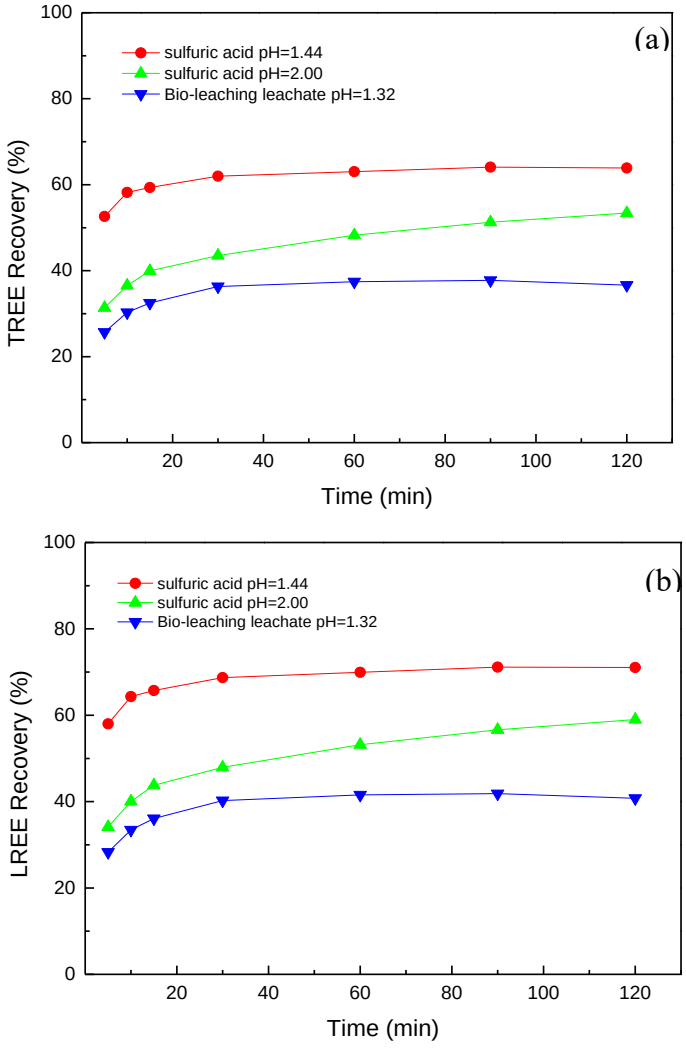
where $C_i$ (ppm) represents elemental concentrations in the filtrates collected during the leaching process, $V$ volume of the feed solution (200 mL), and $E_f$ (ppm) the back-calculated elemental contents in the feed solids. All REE content values in the solids are reported based on a dry whole-sample basis.

**Experimental Results:** Leaching recovery of total, light, and heavy REEs (TREE, LREEs, and HREEs) using the bio-leachate and sulfuric acid solutions of different concentrations is in Figure 3.7.30. It can be seen that sulfuric acid solutions were more favorable than the bio-leachate with higher recovery values being obtained. For example, more than 60% of TREEs were extracted with a sulfuric acid solution with an initial pH of 1.44 after 120 min of reaction; whereas only 36% of TREEs were extracted with the bio-leachate (Initial pH=1.32) under the same conditions. A comparison between Figure 3.7.30(a) and Figure 3.7.30(b) shows that the recovery of TREE and LREE shows similar patterns. However, as shown in Figure 3.7.30(c), the recovery of HREE is much lower than that of TREE and LREE.

Leaching characteristics of selected rare earth elements, including Sc, Ce, Y, La, and Nd, are shown in Figure 3.7.30 . Sulfuric acid solutions of two different initial pH performed better than the bio-leachate in the recovery of Ce, Y, La, and Nd. Moreover, the recovery of Ce, Y, La, and Nd is highest when using the sulfuric acid solution with initial pH of 1.44. However, high Sc recovery values were obtained using the bio-leachate compared to the sulfuric acid solution with an initial pH of 2.00. As shown in Figure 3.7.31(c), only around 10~20% of Y were extracted, which is much lower than the other four selected rare earth elements. For example, 75% of La was recovered when using the sulfuric acid solution with an initial pH of 1.44. By using the bioleachate, 54% of La was leached after reacting for 120 min. In addition, it can be found that the recovery of Nd is higher using the bio-leachate than the sulfuric acid solution (initial pH=2.00) within the first 60

minutes of reaction. However, as the reaction proceeded, higher recovery values were obtained using the sulfuric acid solution (initial pH=2.00).

The leaching experimental data showed that the bio-leachate is less efficient compared with sulfuric acid solutions. However, it has to be notified that the bio-leachate contains tens of thousands ppms of iron. Strong interferences of iron on the REEs analyses were observed when using ICP-OES. Therefore, the accuracy of the bio-leachate leaching data is questionable.



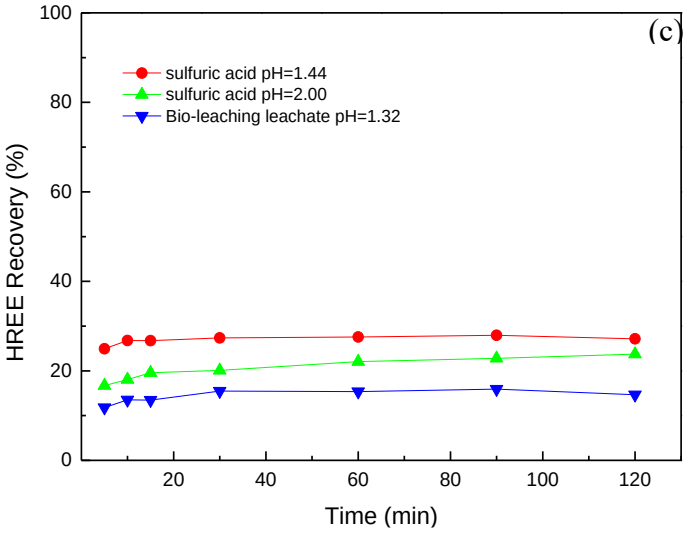
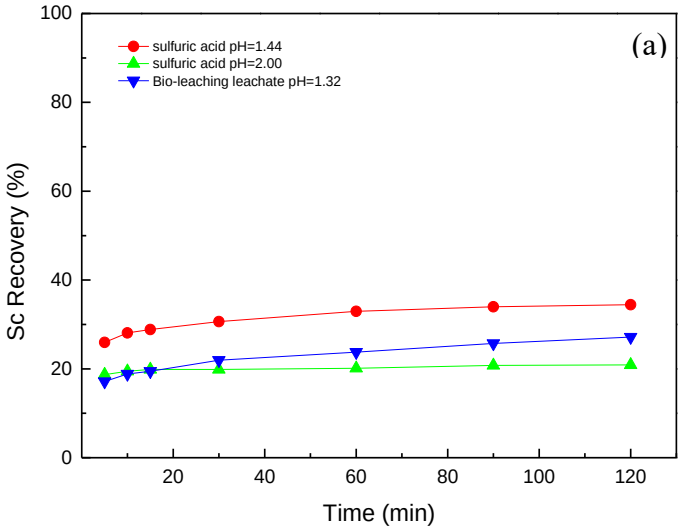
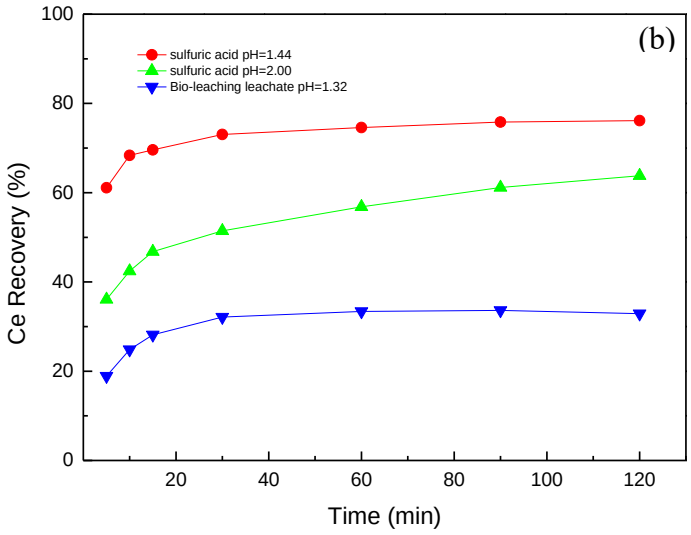
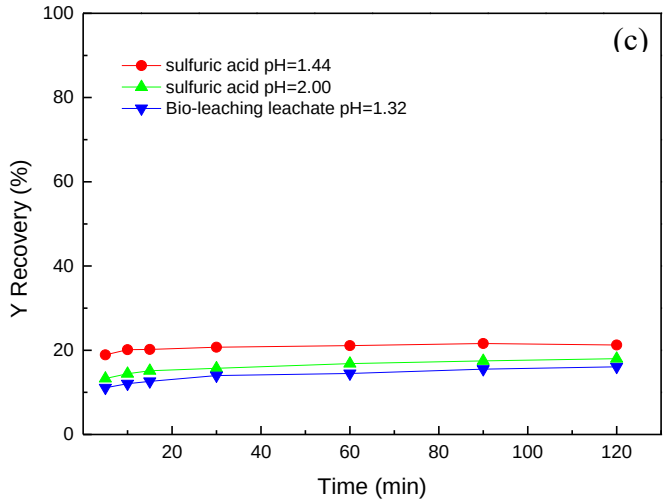
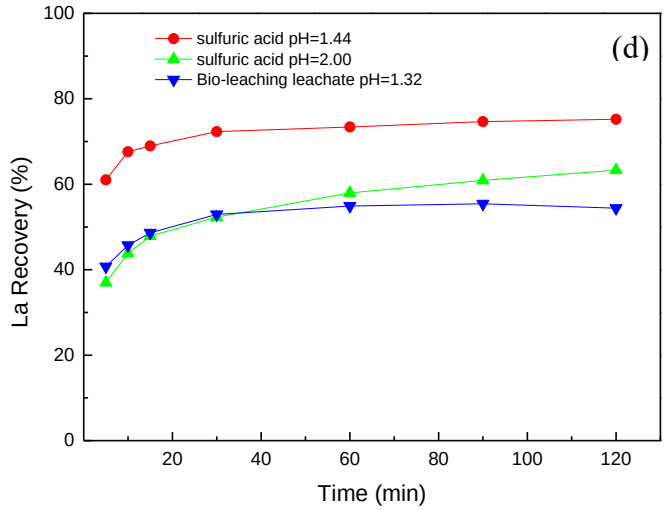



Figure 3.7.30. The effects of different lixiviants on the leaching recovery of (a) total REEs (TREE), (b) light REEs (LREE), and (c) heavy REEs (HREE) from the calcined coal.




(b)
sulfuric acid pH=1.44
sulfuric acid pH=2.00
Bio-leaching leachate pH=1.32
Ce Recovery (%)
Time (min)
0
20
40
60
80
100
120


(c)
sulfuric acid pH=1.44
sulfuric acid pH=2.00
Bio-leaching leachate pH=1.32
Y Recovery (%)
Time (min)
0
20
40
60
80
100
120


(d)
sulfuric acid pH=1.44
sulfuric acid pH=2.00
Bio-leaching leachate pH=1.32
La Recovery (%)
Time (min)
0
20
40
60
80
100
120

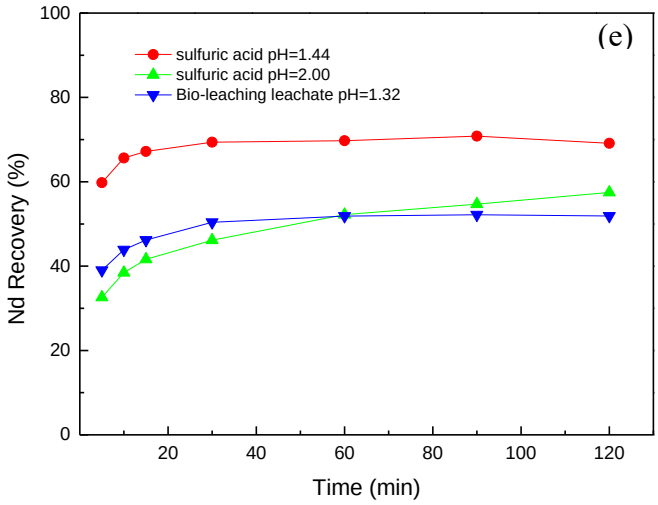



Figure 3.7.31. The effects of different lixiviants on the leaching recovery of (a) Sc, (b) Ce, (c) Y, (d) La, and (e) Nd from the calcined coal.

### *1.1.6 Alkaline Aluminum Stripping from REE Cake*

Roughly 700 gallons of acid mine drainage (AMD) from the Smith Property of the Dotiki Mine Site (Alliance Coal) were sourced and used in a batch processing simulation to assess the feasibility and efficacy of an alkaline redissolution stage to remove aluminum contaminants in the pregnant leachate solution (PLS) circuitry. Based on initial precipitation experiments using sodium hydroxide to incrementally raise the pH of the AMD, two processing scenarios were carried out at the plant scale. The first scenario included an iron precipitation stage (PT0), a rare earth element precipitation stage (PT1-A), two stages of alkaline redissolution on the PT1 cake (RED-1 and RED-2), and a water wash stage to remove excess basic solution (BW-1) from the final rare earth cake product. The second scenario was similar to the first scenario, except for the addition of an aluminum precipitation stage (PT1/2) in between PT0 and PT1. For the second scenario, only one round of alkaline redissolution was performed on the rare earth cake (RED-3), followed by one water wash stage (BW-2). The assessments of these two scenarios follow below.

**Initial Precipitation Testing:** Before processing the AMD through the PLS circuitry, a robust precipitation experiment was carried out on a 500 mL sample to ensure solution chemistry was comparable to that of previous AMD sources tested at the pilot plant facility. Under rigorous stirring, aliquots of 4M sodium hydroxide were added to the solution to hit target pH points after allowing time for the solution to reach equilibrium, ranging from the initial feed pH of 2.4 up to 13.9, which was the limit able to be measured with the pH/Eh module utilized during experimentation. The precipitation curves for total rare earth elements (TREE), aluminum, iron, and scandium, are seen below. For comparison, the precipitation curves for a previous batch of AMD processed in June/July 2020 (AMD20) are shown below in Figure 3.7.32.

From the graphs above, all four of the element precipitation curves for both DTK-SMI and DTK-AMD20 are in good agreement, with iron largely precipitating out at a pH of 3.2, aluminum at 4.5, and rare earths before 7. This is unsurprising, as both sources of AMD were sourced from Dotiki property in Webster Co., KY, the exception being that DTK-SMI was largely runoff from the West

Kentucky 9 refuse piles, while DTK-AMD20 was runoff from the West Kentucky 13 refuse piles, leading to a lower initial TREE concentration in DTK-SMI as had been seen with DTK-AMD20. While lower in rare earth content, DTK-SMI still had an appreciable TREE concentration, ~2 ppm, which made it suitable for processing in the PLS circuitry and in assessing aluminum removal procedures.

From selective precipitation testing presented previously, it was noted that in relatively simple solutions with sulfuric media, aluminum began to redissolve back into solution at a pH of 10 after fully precipitating around a pH between 4.5 and 5. Since aluminum is a primary contaminant and a major source of chemical consumption during the rare earth oxalate precipitation stage, it was desirable to construct precipitation curves over the full pH range to see if this observation held true for our complex DTK-SMI AMD solution. Upon reaching a pH of 13.5, roughly 60% of total aluminum content had been redissolved, while all of the scandium and rare earths remained as precipitate. Since in the acidic pH range, scandium and aluminum precipitation curves are nearly identical, the ability to separate aluminum from scandium and the rare earths in the basic pH range showed promise, and processing in the plant to replicate the results soon followed.

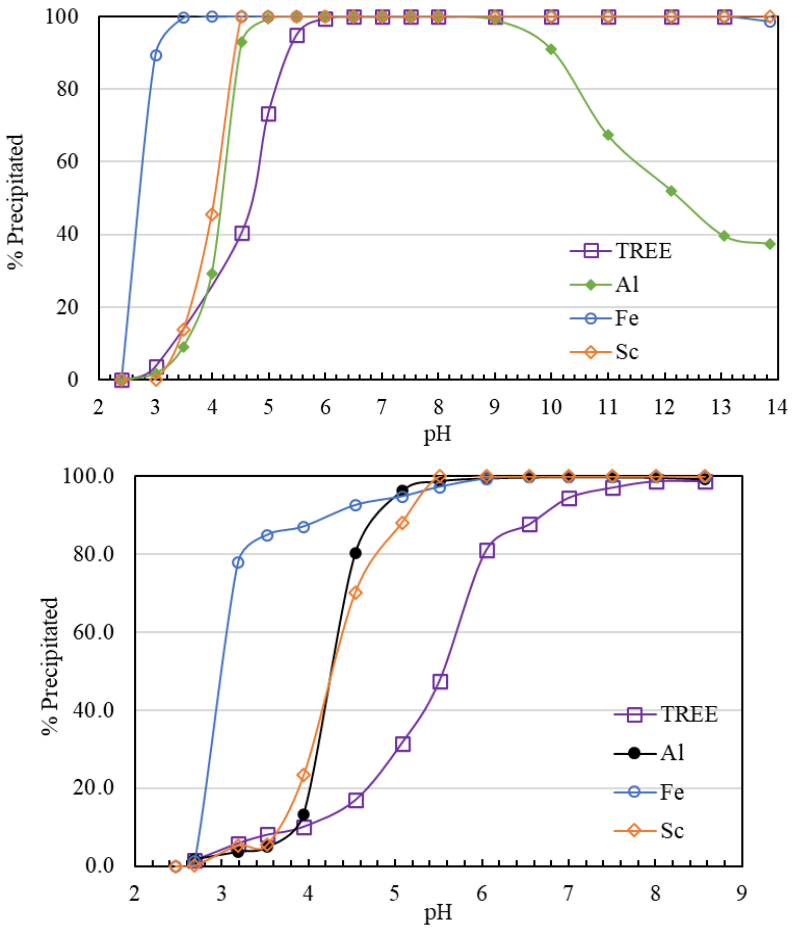



Figure 3.7.32. Precipitation curves of total rare earth elements (TREE), scandium (Sc), aluminum (Al), and iron (Fe) in sulfuric media treated with 4M sodium hydroxide solution. Acid mine drainage obtained from the Dotiki Smith Property (DTK-SMI, top) and from the Dotiki Prep Plant (DTK-AMD20, bottom), show near identical trends among all four element categories.

**Iron Precipitation Processing (PT0):** Previous test work has led to well-defined processing constraints for the PT0 iron precipitation stage. For AMD, the iron is nearly 100% in the ferric oxidation state due to the length of time the AMD is exposed to air during accumulation periods, often several months to years. As a result, it becomes relatively easy to remove almost all of the iron in the AMD during the initial precipitation stage. Based on the precipitation testing discussed above, a pH setpoint of 3.2 was chosen for iron precipitation. All 680 gallons of the AMD were processed through the reaction tank at a flowrate of 2 gal/min. This allowed for 45 minutes of reaction time within the tank. Samples were taken, hourly, of the feed and filtrate and analyzed using inductively coupled plasma – optical emission spectroscopic techniques (ICP-OES). The solution pH and Eh measurements were also recorded for the samples, and percent solids calculations were performed on the filtrate samples, as these were taken directly from the reaction tank and centrifuged in the lab. The average pH attained over the course of the five hours of processing was 3.25 for the filtrate, and 2.41 for the feed. Eh averages were 673 for the filtrate and 767 for the feed. Total base consumption of the 4M NaOH solution was roughly 5 gal. Individual

sample points are shown below in Table 3.7.10. Overall iron removal, as precipitate, from the solution was 95%, cutting down the concentration from 1260 ppm to less than 60 ppm as shown in Figure 3.7.33.

Table 3.7.10. Feed (F) and filtrate (R) samples taken during iron precipitation (PT0) processing.

| # | Time | Hours in Run | % Solids | pH | Eh | 4 M Base (gal) |
|---|---|---|---|---|---|---|
| Start | 11:00 | 0:00:00 | | | | |
| F1 | 11:15 | 0:15:00 | | 2.402 | 764.1 | 36 |
| F2 | 12:15 | 1:15:00 | | 2.401 | 763.6 | 35 |
| F3 | 13:15 | 2:15:00 | | 2.417 | 763.9 | 34.4 |
| F4 | 14:15 | 3:15:00 | | 2.368 | 767.2 | 33.5 |
| F5 | 15:15 | 4:15:00 | | 2.422 | 765.8 | 32 |
| F6 | 16:00 | 5:00:00 | | 2.442 | 779 | 31 |
| Stop | 16:00 | 5:00:00 | | | | |
| Start | 11:00 | 0:00:00 | | | | |
| R1 | 11:15 | 0:15:00 | 0.4173 | 3.267 | 662.7 | |
| R2 | 12:15 | 1:15:00 | 0.3314 | 3.362 | 646.3 | |
| R3 | 13:15 | 2:15:00 | 0.3718 | 3.188 | 703.2 | |
| R4 | 14:15 | 3:15:00 | 0.3862 | 3.197 | 686 | |
| R5 | 15:15 | 4:15:00 | 0.3956 | 3.25 | 664.4 | |
| R6 | 16:00 | 5:00:00 | 0.3927 | 3.257 | 675.8 | |
| Stop | 16:00 | 5:00:00 | | | | |

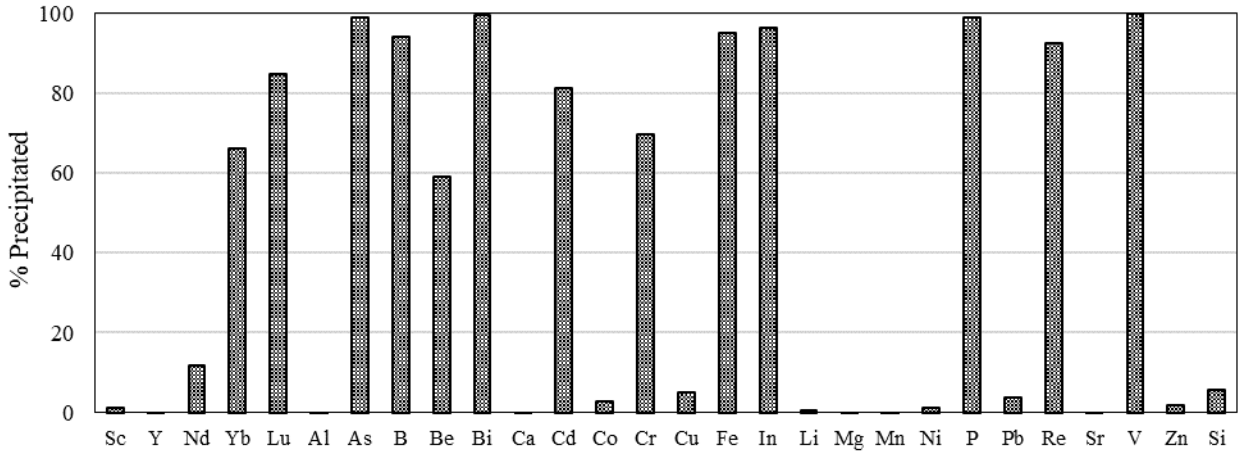



Figure 3.7.33. Elemental recoveries in the filter cake from the iron precipitation stage (PT0).

For further processing and experimentation, the remaining filtrate from PT0 was split into three categories: 300 gal for Scenario 1 and 300 gal for Scenario 2.

**Scenario 1 Processing:** The first 300 gallons of filtrate from PT0 saved for Scenario 1 were sent through another four-stage process to remove aluminum. First, a rare earth precipitation stage was performed on the solution. From precipitation testing, the optimal pH setpoint to remove all of the

rare earths, while still leaving contaminants such as calcium and magnesium in solution, was determined to be 6.8, which agreed with prior batch processing from DTK-AMD20. Referred to as PT1-A for being the first rare earth precipitation stage (PT1), and corresponding to Scenario 1 (A), samples were taken at least once per hour of the feed and filtrate, as was done during PT0. The same measurements of pH, Eh, percent solids, and elemental concentrations were taken as before. These are shown below in Table 3.7.11.

Table 3.7.11. Feed (F) and filtrate (R) samples taken during rare earth precipitation (PT1-A) processing.

| # | Time | Hours in Run | % Solids | pH | Eh | 4 M Base (gal) |
|---|---|---|---|---|---|---|
| F1 | 4:10 | 0:00:00 | | 3.032 | 691.2 | 35.5 |
| F2 | 5:10 | 1:00:00 | | 3.064 | 699.3 | 35 |
| F3 | 5:40 | 1:30:00 | | 3.044 | 706.2 | 34.75 |
| F4 | 6:10 | 2:00:00 | | 3.053 | 707.7 | 34.5 |
| | | | | | | |
| R1 | 4:10 | 0:00:00 | 0.3146 | 7.419 | 160.6 | |
| R2 | 5:10 | 1:00:00 | 0.2285 | 7.843 | 160.6 | |
| R3 | 5:40 | 1:30:00 | 0.2681 | 7.194 | 214.1 | |
| R4 | 6:10 | 2:00:00 | 0.2701 | 7.287 | 214.4 | |

Average pH and Eh values for the filtrate were 7.44 and 187, respectively. For the feed, the pH and Eh were 3.05 and 701.1, respectively. These numbers are noteworthy, in that the pH and Eh values for the feed were lower and higher, respectively, than seen when the solution was analyzed as the filtrate coming from PT0. These processing stages were performed a week apart, and the decrease in pH and increase in Eh indicate that the introduction of sodium hydroxide to the mixture increases the rate of oxidation. Thus, as more ferrous iron is converted to ferric, more hydroxide anions are consumed, the pH is lowered and residual unfiltered iron from PT0 is entering back into solution. For the filtrate pH, it should be noted that, while great care was taken to ensure the pH setpoint was kept as steady as possible during processing, drifts in the pH probe measurements led to a much higher average pH being attained. The results of this occurrence are shown below in Figure 3.7.34. While 97% of TREE content was recovered in the PT1-A cake, nearly 99.9% of the aluminum, 100% of the remaining iron content, 12% of the calcium content, and 33.3% of the magnesium content was also recovered. The recovery of rare earths is optimal, but the costs associated with the increased amount of contaminants due to the high pH of the precipitation stage and the lack of an aluminum/residual iron precipitation stage are troublesome.

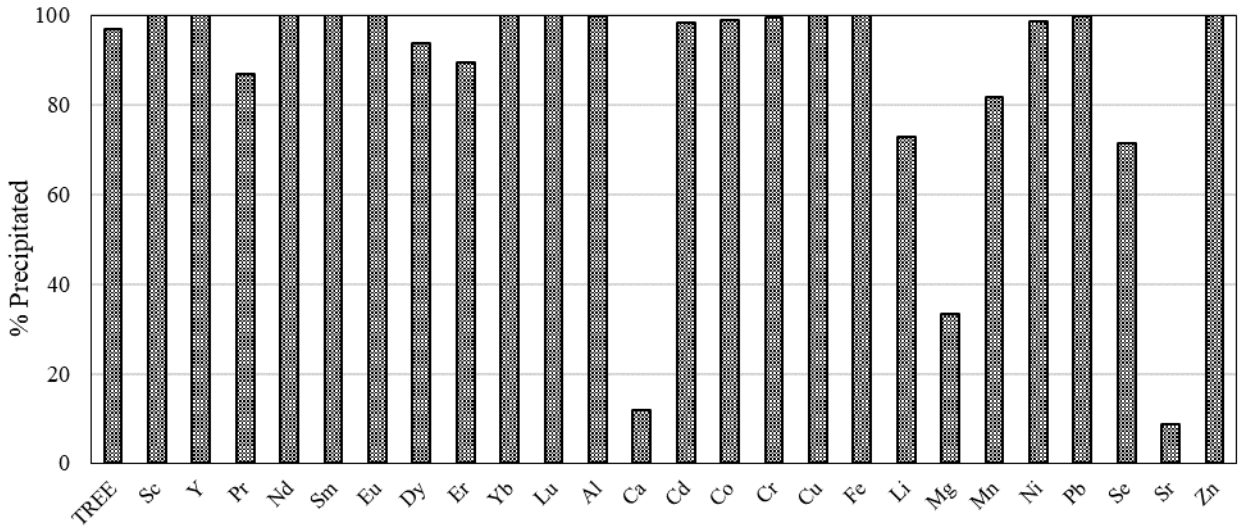



Figure 3.7.34. Elemental recovery in the filter cake during processing of the rare earth precipitation stage (PT1-A).

The cake retrieved from PT1-A was then used in initial laboratory experiments to determine the optimal solution molarity of sodium hydroxide to ensure near complete redissolution of aluminum. To determine these conditions, seven tests were run on 50 mL solutions consisting of PT1-A cake, deionized water, and 50 wt% NaOH, with amounts of water and NaOH added to so as to create samples of 0, 1, 2, 4, 6, 8, and 10 M concentrations. Two sets of these seven were created, with one set having enough cake to mimic a 15-gal dilution, and the other set a 25 gal dilution, so as to account for any effects solids content would have on the process. The results are shown below in Figure 3.7.35.

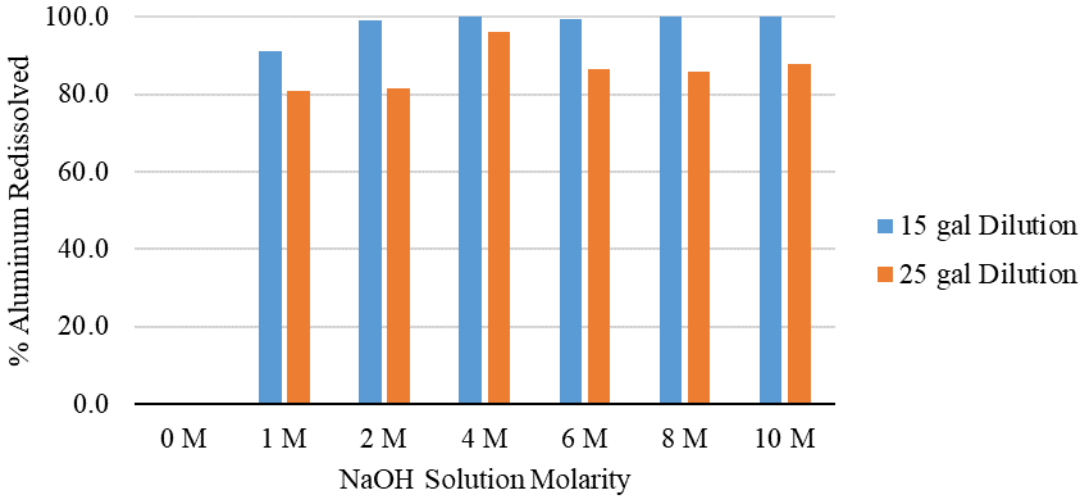



Figure 3.7.35. Aluminum redissolution testing on rare earth cake obtained from PT1-A. Samples were created to mimic 15 gal and 25 gal dilutions.

From the data it is clearly seen that little/to no improvement is seen in the percent of aluminum redissolved when increasing the molarity of the solution past 2M in either the 15-gal or 25-gal

dilution samples. However, aluminum redissolution seems to be improved when using less water, suggesting a 15-gal dilution would be optimal, even though the amount of base present in solution is greater in the 25-gal dilution per amount of cake. Variation in the data for the 25-gal dilution is also more pronounced than for that of the 15-gal dilution. Owing to the fact that a much smaller cake sampling was utilized in each of the seven samples for the 25-gal dilutions over the 15-gal dilutions, it is unsurprising to see this variation. Too, it is plausible, then, that the sample digested in acid to assess total aluminum content in the ICP-OES may not have been a representative sample, potentially skewing the aluminum recovery lower. Regardless, it was decided to use a 25-gal dilution of the cake in a 2M sodium hydroxide solution for aluminum redissolution. 25 gal was chosen over 15 gal primarily because the filter press used during processing leaves a considerable amount of filtrate trapped in the cake that is difficult to remove prior to retrieving the cake. At 25 gal, the amount of filtrate held in the filter would be 40% less concentrated in aluminum content than the filtrate from a 15-gal dilution, leaving less to contaminate the filter cake.

To test the 25-gal dilution, 2M NaOH concentration conditions on the cake, similar to lab testing procedures, the entirety of the filter cake obtained from PT1-A was dumped into a 50-gal mixing tank, followed by 10 L of 50 wt% NaOH (equivalent to 2M in 25 gal), and then the resulting slurry was brought to a total volume of 25-gal with deionized water. Mixing/reaction time was kept limited to 45 minutes to resemble more closely the preceding precipitation stages. A feed sample was taken for digestion, and the entire solution was then filtered through a small filter press. Samples of the filtrate were collected and analyzed using ICP-OES during the process. Cake samples were sent offsite for digestion, so a mass balance was tabulated around the redissolution unit, providing elemental recoveries in the cake, as well as cake composition, based on the analysis of the feed and filtrate samples. Figure 3.7.36 shows the results of this mass balance for the first redissolution stage.

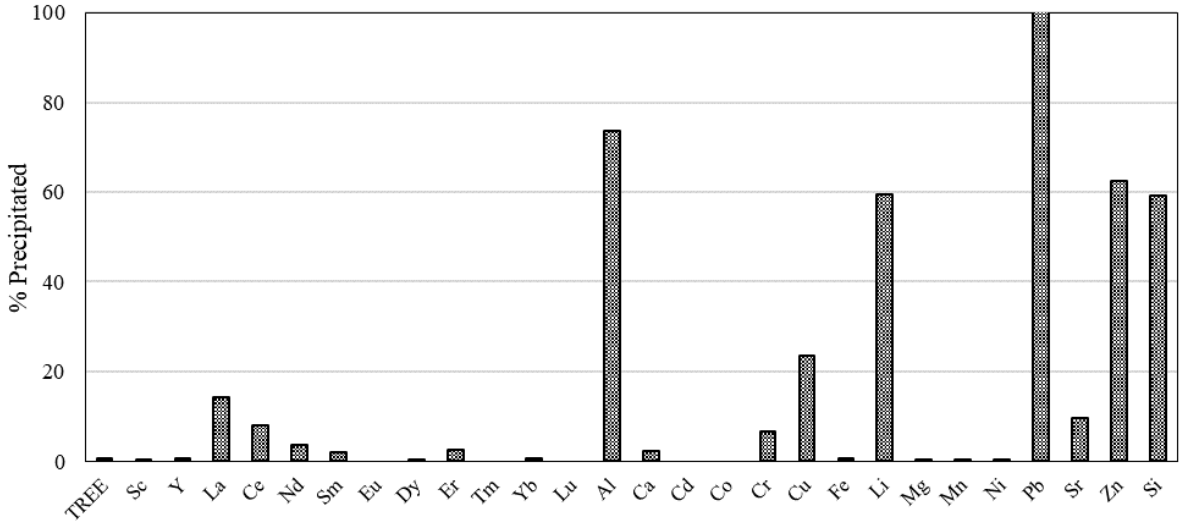



Figure 3.7.36. Elemental recovery in the filtrate during processing of the initial redissolution stage (RED-1).

Not as optimal as the initial laboratory testing, the aluminum redissolution stage using a 2M alkaline wash was still successful in removing nearly 74% of the total aluminum through the

filtrate, while still retaining over 99% of the total rare earth content in the filter cake. However, cake composition analysis for the first redissolution stage revealed that, while a substantial majority of aluminum was removed, the cake still had a 30:1 Al:TREE ratio, roughly 0.34% rare earths content and 11% aluminum content, seen below in Figure 3.7.37.

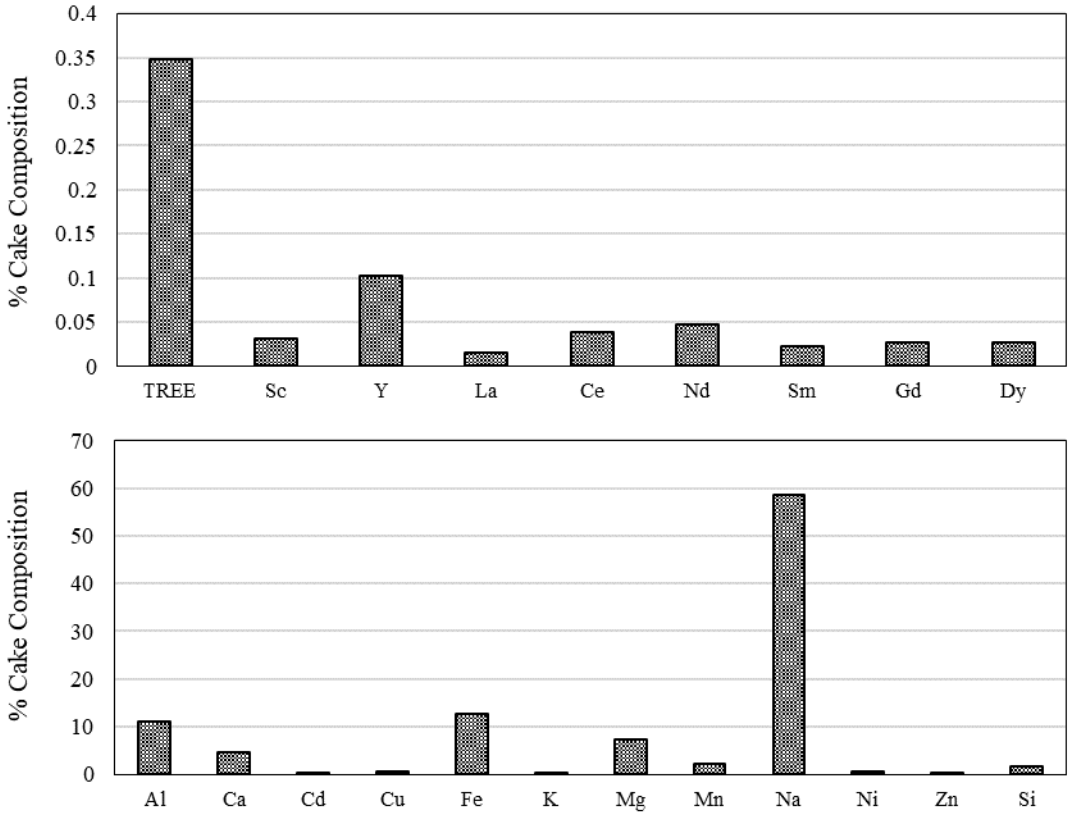



Figure 3.7.37. Percent cake composition from RED-1 aluminum redissolution stage on an elemental basis. Rare earth element content (top) and contaminant element content (bottom) are shown.

To improve the overall cake composition, a second round of aluminum redissolution using an alkaline wash stage was performed on the cake obtained from RED-1. The same procedure as described previously for RED-1 was performed, using fresh 50 wt% NaOH and deionized water so as to prevent saturation of the alkaline wash with aluminum and achieve maximum redissolution. It was found that another 58.6% of aluminum was removed through the filtrate, during this step, based on the mass balance around the feed and filtrate samples from this stage. Too, 100% of the TREE content stayed in the filter cake, as seen in Figure 3.7.38 below.

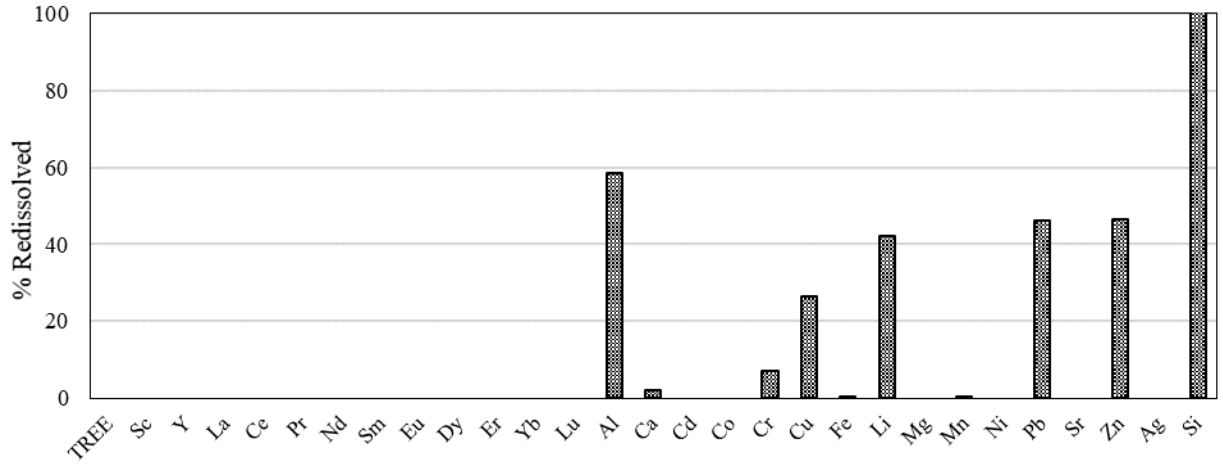



Figure 3.7.38. Elemental recovery in the filtrate during processing of the second redissolution stage (RED-2).

Before removing the cake from the filter, and based on the cake composition analysis from RED-1, it was desirable to do a "wash" step (BW-1) to help remove excess sodium, and aluminum, that may have been left behind in the cake dissolved in the residual filtrate. The reaction tank used for slurrying the mixture prior to filtering during the two redissolution steps was then filled with 25 gal of fresh deionized water, and it was passed through the lines and filter to rinse the cake and into a holding receptacle. ICP-OES sample analysis was performed on the filtrate, and a mass balance performed around the BW-1 stage itself, as well as the RED-2 and BW-1 stages as a single unit.

Using the cake compositions obtained directly from the BW-1 filter cake digestion and the filtrate sample analysis, elemental recoveries from the feed into the filtrate were calculated, as seen below in Figure 3.7.39. Roughly 82% of the sodium content appeared to be removed, which entailed roughly 300 g from a cake that weighed only 2.46 kg dry. In other words, a 7% reduction in contaminants as a result of the wash step. The final cake compositions are found below in Figure 3.7.39, as well. From the additional redissolution step, and the water wash step performed on the cake, the final TREE content in the product was increased from 0.35% to 0.91%, while aluminum content remained static at 12%. This narrowed the ratio of Al:TREE from 30:1 coming out of RED-1, to roughly 12:1. From the initial feed stock of PT0, where Al:TREE content was 265:1, this is a substantial improvement, as 96% of aluminum was successfully removed through the process. However, it is very concerning that iron still accounted for 35% of the final cake, as this is a primary contaminant and negatively impacts cost processing the rare earths further through oxalic precipitation stages.

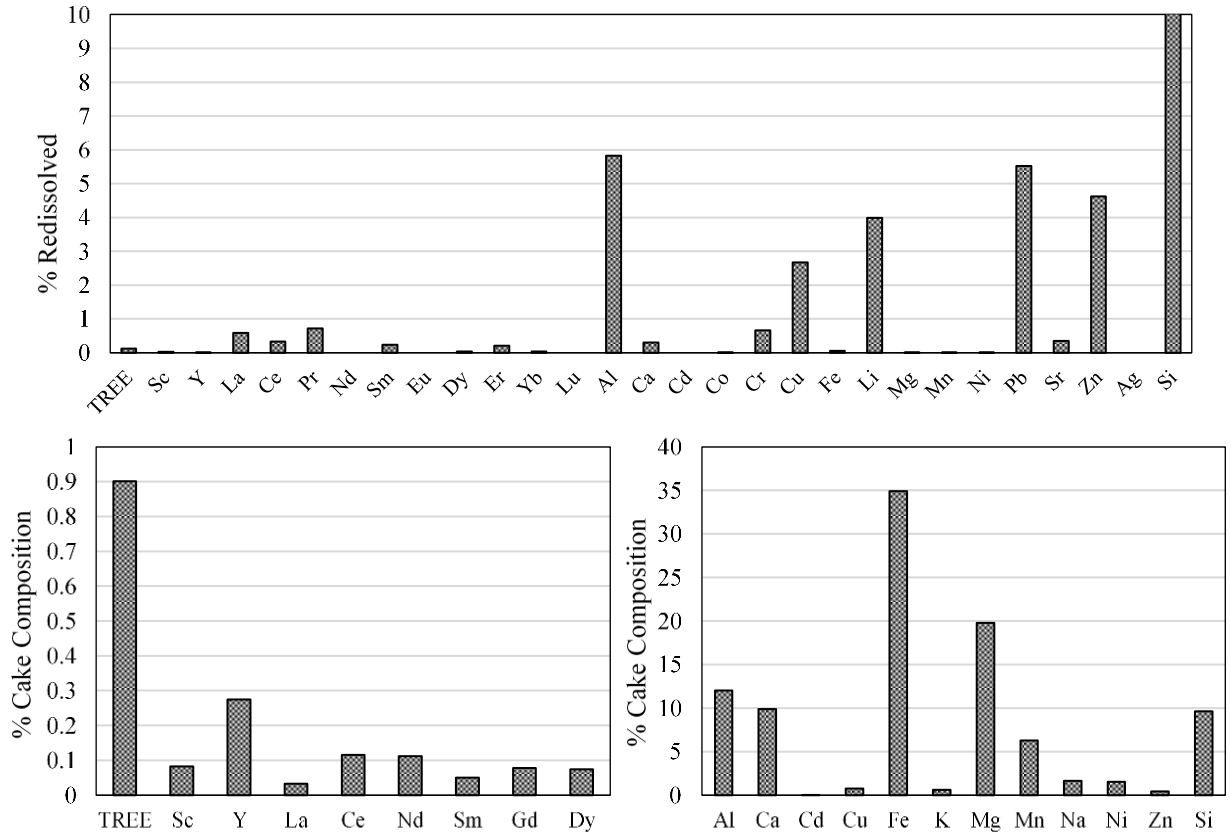



Figure 3.7.39. Percent cake composition from BW-1 aluminum redissolution stage on an elemental basis. Rare earth element content (top) and contaminant element content (bottom) are shown.

**Scenario 2 Processing:** An alternate scenario for implementing an alkaline wash to the circuitry for the removal of aluminum involved adding an intermediate aluminum precipitation step between the iron (PT0) and rare earth (PT1) precipitation stages. It was hypothesized that lowering the initial aluminum content going into the alkaline wash may prevent oversaturation of the solution, leading to a more complete removal. This precipitation stage has typically been performed at a pH of 4.4-4.5, as roughly 90% of aluminum precipitates at 4.5, as is shown in the initial precipitation testing discussed previously. However, scandium also precipitates in this range, and it is common to lose scandium in a 1:1 ratio along with aluminum. In addition, some rare earth elements, like neodymium, begin to precipitate at the high end of this interval. Therefore, it becomes necessary to retain tight control over the pH setpoint of the solution during processing for this stage, and it becomes desirable to rid the solution of most aluminum, but not all, so as to ensure no rare earth content is being lost.

To start, the second 300 gallons of DTK-SMI AMD saved for scenario two was sent through the proposed two-stage precipitation process at 2 gal/min, leading to a 45-minute reaction time, in-line with previous processing steps. Samples were taken in a similar fashion as to what has been described previously and were analyzed accordingly. The pH setpoint for PT1/2, or the aluminum precipitation stage, was selected to be 4.45, and the pH setpoint for PT1-B was kept at 6.8 like in PT1-A during Scenario 1. The pH, Eh, solids percentage of the filter feed after precipitation, and

base consumption for the two stages are listed below in Table 3.7.12. The average pH and Eh of the feed and filtrate during PT1/2 were 2.97 and 679, respectively, for the feed, and 4.40 and 388 for the filtrate, respectively. The attained pH of the filtrate was acceptable, as overshoot is more costly during the PT1/2 stage than undershoot, in terms of rare earths lost. For the feed and filtrate of the PT1-B stage, the pH and Eh of the feed averaged to be 4.38 and 283, respectively, and for the filtrate 6.87 and 17, respectively. For this run of the PT1 stage, much better pH control was maintained, and overshoot was mitigated, leading to a further reduction in contamination of the rare earth cake with calcium and magnesium, among others.

Table 3.7.12. Feed (F) and filtrate (R) samples taken during aluminum (PT1/2) and rare earth precipitation (PT1-B) processing.

| # | Time | Hours in Run | % Solids | pH | Eh | 4 M Base (gal) |
|---|---|---|---|---|---|---|
| F1 | 9:00 | 0:00:00 | | 2.973 | 671.5 | 34 |
| F2 | 9:30 | 0:30:00 | | 2.976 | 714.8 | 33.5 |
| F3 | 10:00 | 1:00:00 | | 2.99 | 640.9 | 33 |
| F4 | 10:30 | 1:30:00 | | 2.926 | 689.2 | 32.75 |
| R1 | 9:00 | 0:00:00 | 0.2541 | 4.463 | 420.3 | |
| R2 | 9:30 | 0:30:00 | 0.1862 | 4.408 | 580 | |
| R3 | 10:00 | 1:00:00 | 0.1493 | 4.371 | 226.3 | |
| R4 | 10:30 | 1:30:00 | 0.1837 | 4.361 | 324.8 | |
| # | Time | Hours in Run | % Solids | pH | Eh | 4 M Base (gal) |
| F1 | 4:10 | 0:00:00 | | 4.364 | 266.4 | 35.5 |
| F2 | 5:10 | 1:00:00 | | 4.38 | 263.7 | 35 |
| F3 | 5:40 | 1:30:00 | | 4.391 | 308.9 | 34.75 |
| F4 | 6:10 | 2:00:00 | | 4.386 | 292.6 | 34.5 |
| R1 | 4:10 | 0:00:00 | 0.0785 | 6.661 | 109.4 | |
| R2 | 5:10 | 1:00:00 | 0.0160 | 6.445 | 160.8 | |
| R3 | 5:40 | 1:30:00 | 0.0225 | 7.041 | 238.2 | |
| R4 | 6:10 | 2:00:00 | 0.0189 | 7.337 | 207.2 | |

Elemental recoveries in the cake are presented below in Figure 3.7.40 for both stages. It is seen in PT1/2, that nearly 67% of the total aluminum content was removed from the AMD solution into the filter cake. However, 56% of scandium content was lost. An upside to this stage is that 97% of the residual iron content, left over from PT0, was successfully removed, and only 8% of initial TREE content was lost as a result. In PT1-B, 90% of TREE content was removed in the cake from the AMD solution, while precipitating negligible amounts of calcium and magnesium. To add to this, 53% of cobalt and 17.5% of manganese were also precipitated, which could be collected in further downstream processing.

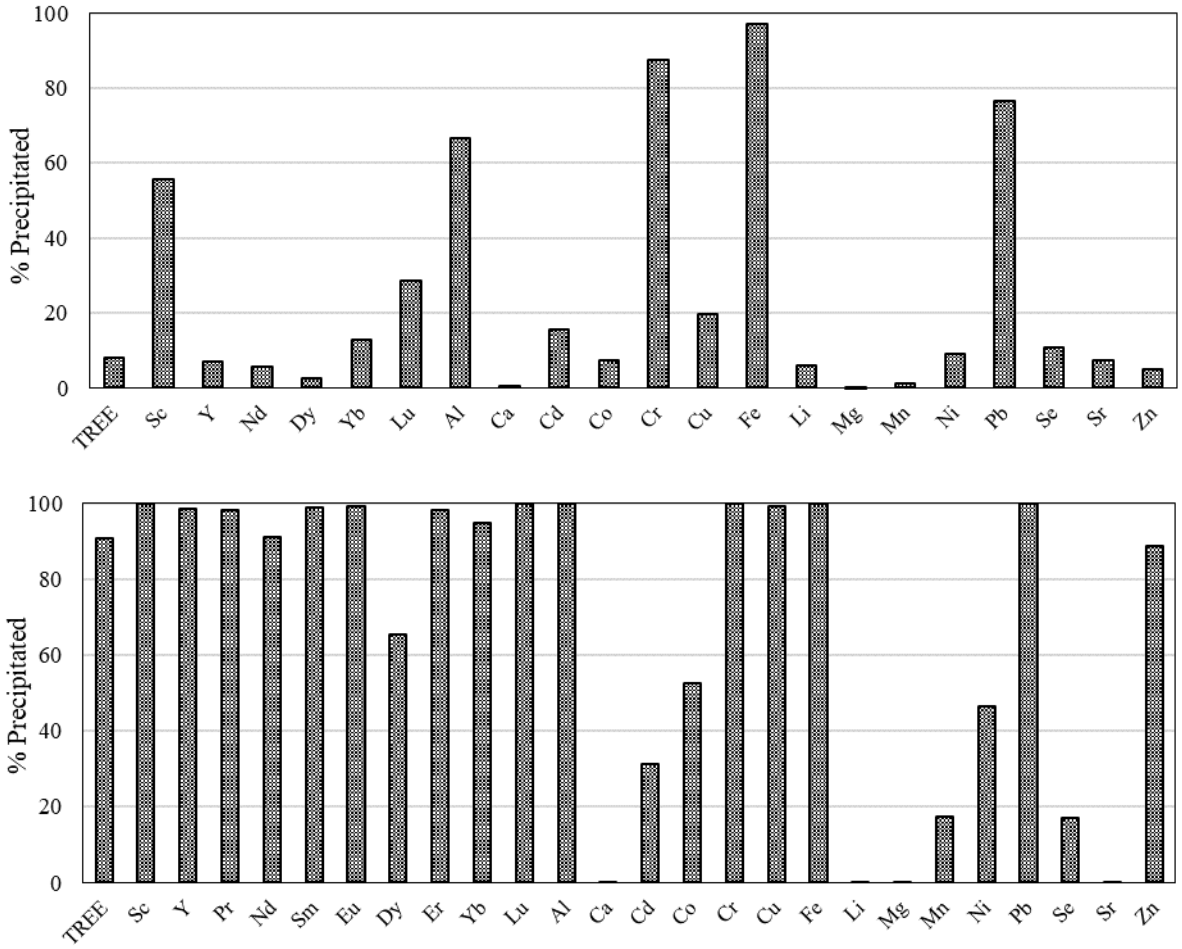



Figure 3.7.40. Elemental recoveries in the filter cake as a percentage of what was found in the feed. PT1/2 (top) was able to remove 67% of aluminum content and 97% of iron content, while only losing 8% of TREE content. PT1-B was able to remove nearly 91% of TREE content from solution. Magnesium and calcium contamination remained low, as well.

The rare earth cake, obtained from the filter after processing PT1-B, was sampled, and laboratory testing was again conducted to see if conditions of the alkaline wash needed to be adjusted due to the lower aluminum content. A slurry was formed to mimic a 25 gal dilution once again, but this time, a 400 mL sample was taken, and aliquots of 50 wt% NaOH were added to the slurry to achieve target pH points from 11 up to 13. From there, the appropriate amount of NaOH was added so as to bring the solution up to an initial molarity of 1M NaOH, and again to 2M NaOH. The results are shown below in Figure 3.7.41.

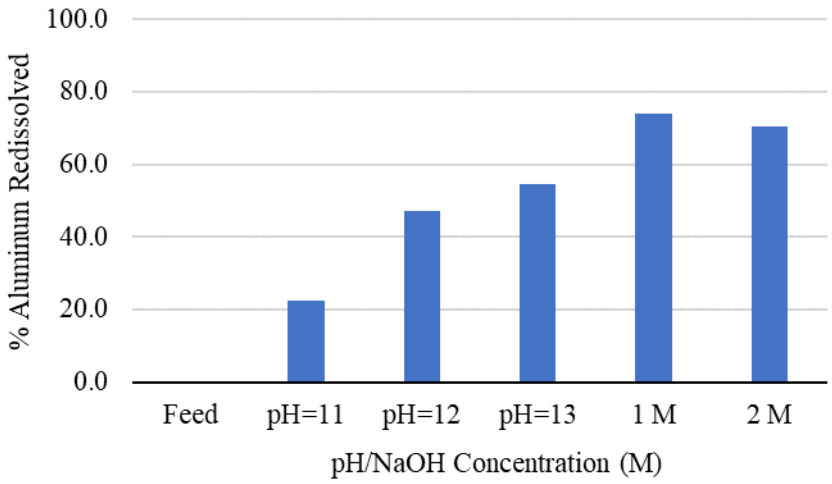



Figure 3.7.41. Aluminum redissolution testing on rare earth cake obtained from PT1-B.

Unsurprisingly, the lower aluminum content in the initial feedstock lowered the overall amount of base needed to achieve optimal aluminum redissolution. A 1 M alkaline wash solution redissolved 74% of the remaining aluminum in the rare earth cake, in-line with results obtained from RED-1 processing discussed earlier. A plateau was reached above 1 M, so in order to reduce costs associated with base, a 1 M, 25-gal wash was selected for the PT1-B cake redissolution step.

Similar procedures were employed in the redissolution stage (RED-3) as were used in the RED-1 and RED-2 stages. The cake was added to a mixing tank, along with 5 L of 50 wt% NaOH, and the remainder of the 25-gal volume was substituted with deionized water. The reaction was allowed to proceed for 45 minutes before filtering, and the filtrate was continuously recycled until the solution cleared up to the naked eye. This was to ensure maximum contact of the cake with the alkaline wash, as well as to ensure all undissolved cake was retained in the filter press. The filtrate was then passed through once more before being placed in a clean holding receptacle, and an additional 5 gal of water was passed through behind the last of the filtrate to clean out any slurry left in the feed and outlet lines. Feed and filtrate samples were taken and analyzed using ICP-OES. From the mass balance around the feed and filtrate samples, percent recoveries in the filtrate were calculated for the elements of interest and are seen below in Figure 3.7.42. In this stage of redissolution, 80% of the remaining aluminum content was successfully removed via the filtrate from the cake, all while losing negligible amounts of rare earth elements. Overall, this led to a 93% removal of aluminum from the initial AMD feedstock sent through PT0.

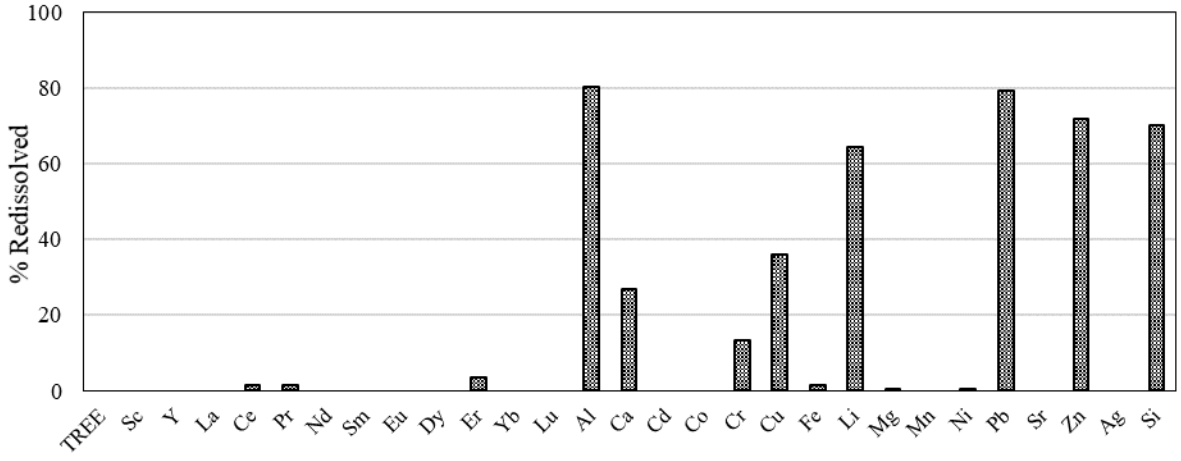



Figure 3.7.42. Elemental recovery in the filtrate during processing of the redissolution stage for Scenario 2 (RED-3).

Based on the success of the backwash stage in Scenario 1, a backwash step was added to Scenario 2, as well. However, the step was performed differently in that the cake was first removed from the filter, redispersed in 5 gal of deionized water, agitated for 15-20 min to ensure maximum contact of the water with the particulates in the cake, and then passed through the filter, undergoing recycling, before the final filtrate and wet cake were collected. A digestion was then performed on a sample of the cake, and the cake compositions were determined, as seen below in Figure 3.7.43.

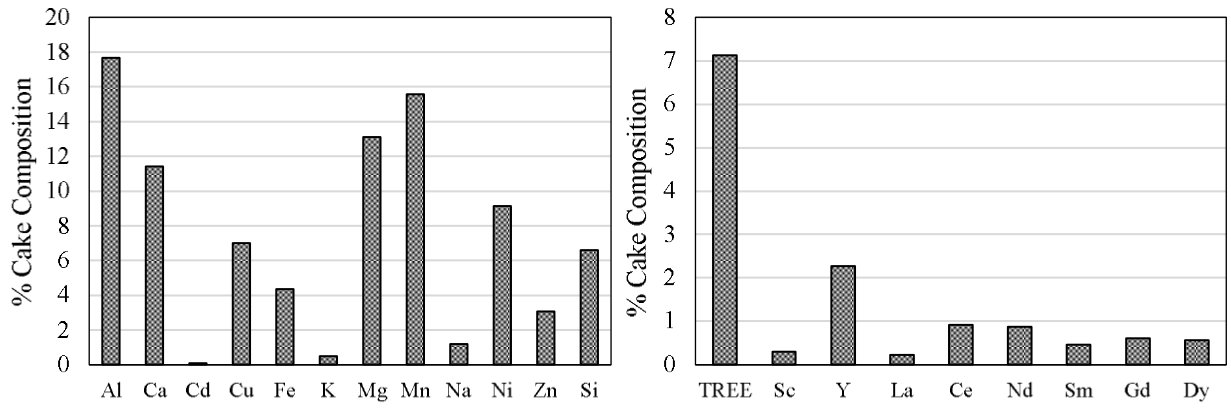



Figure 3.7.43. Cake compositions of the final rare earth product obtained from Scenario 2. Contaminant elements (top) and rare earths (bottom) are shown.

Overall, the final cake composition appeared to be 7% rare earths, with regard to the elements analyzed by the ICP-OES. To add to this, only 17.5% of the cake content was aluminum, and only 4.5% iron. Compared to the final cake produced in Scenario 1, the Al:TREE ratio was only a little over 2:1, versus 12:1. Iron content was also cut, in regards to cake composition, by almost 9 times. When assessing the efficacy and efficiency of aluminum removal between Scenario 1 and Scenario 2, three key items should be considered: overall rare earth recovery, overall aluminum removal, and overall iron removal. In Scenario 1, while 100% of the rare earth content was preserved throughout the process, including scandium, too much residual iron, that continued to oxidize and

precipitate, made its way into the final rare earth cake after PT0 processing. Two rounds of alkaline wash steps were also needed to achieve a +90% aluminum removal, with the alkaline wash consuming 25 gal of 2M NaOH both times. In Scenario 2, while a little less than 10% of the total rare earth content was lost, mainly in the form of scandium through the added aluminum precipitation step, less than 5% of the final cake content was iron, being less than the total mass of rare earths, and the final aluminum content was comparable to that of Scenario 1, even though only a quarter of the amount of base was needed. To add to this, scandium can easily be separated through solvent extraction techniques, which has been demonstrated in the previous studies. Further experimentation is necessitated, however, to determine the optimal cake dilution for slurrying, as well as a model of base strength needed to remove low versus high concentrations of aluminum when analyzing feedstocks.

### *1.1.7 Improvement on Fe precipitation through oxidation*

Bench scale tests were performed on a heap leach pregnant leaching solution to initiate the test plan on secondary feedstock in the pilot plant continuous circuit. A preliminary precipitation test was performed on the PLS to determine new setpoints for the precipitation circuit conditioning tanks in the pilot plant. Various concentration of NaOH solution was prepared to dope the 500 mL PLS sample that was collected from the heap. The sample was stirred constantly and pH and Eh were monitored through the whole test. Doses of 100 microliters were added to the AMD sample over time. After each dose was added, pH was allowed to stabilize before the next dose was added. 3 mL samples were collected at pH intervals of 0.5 and centrifuged. A 1 mL sample was collected from the supernatant of each precipitate sample and added to a 10 mL dilution for ICP analysis. Samples were submitted to UK main campus for more accurate testing with ICP-MS unit at CAER.

From the baseline test, it was determined that the PLS contained a higher level of ferrous iron in comparison to a typical AMD feed solutions that were mostly ferric iron. As can be seen in Figure 3.7.44, iron begins to precipitate at pH 2 where ferric iron would be expected to precipitate, then at pH 2.5 the precipitation stagnates and plateaus until 5.5 where the precipitation continues until 100% iron recovery is reached at pH 7. This behavior along with the initial Eh of 627 RmV indicate the presence of ferrous iron as ferrous iron typically precipitates around 6.0 or 6.5 pH. To reduce iron contamination downstream from the iron precipitation stage, oxidizing the PLS feed stream was proposed to convert ferrous iron to ferric and testing was performed.

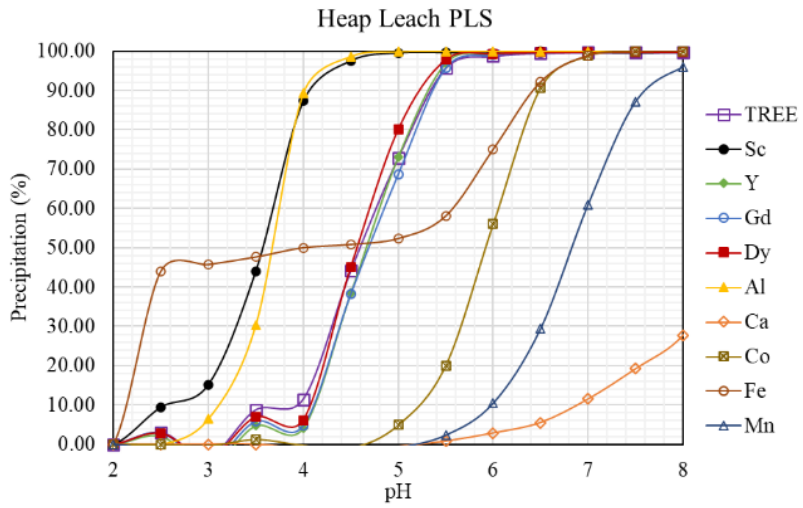



Figure 3.7.44. Recovery of heap leach pregnant leach solution after precipitation test performed in the lab.

To reduce the amount of iron reporting to the solvent extraction unit after iron precipitation and to reduce the amount of aluminum and iron contamination in the rare earth precipitation, oxidation of the iron was tested. Ferrous iron precipitates at pH 6 and based off the preliminary precipitation as seen in the figure above would contaminate the rare earth and critical materials cakes. And ferrous concentration in the solvent extraction will limit the loading efficiency of Sc as observed in the lab test. Oxidizing the ferrous iron to ferric iron would encourage Fe precipitation at lower pH and solve these problems.

Two different methods of dosing the hydrogen peroxide were performed. 30% hydrogen peroxide from Fischer-Scientific was used in the tests as the oxidizer. The first hydrogen peroxide test involved simply adding hydrogen peroxide to a 500 mL sample in 100 microliter doses and stirred vigorously with a magnetic stir bar at 700 rpm. As peroxide was added to the sample pH and Eh were monitored. pH rose slightly but never enough to precipitate any iron or other contaminants. Eh, however, rose significantly with every addition, indicating conversion of iron. Once Eh was stabilized another dose would be added until the Eh no longer changed with the addition of peroxide. At this point 12 M base was dosed into the PLS and the precipitation proceeded as normal. The next test utilized data gleaned from the initial precipitation test as well as the first hydrogen peroxide test. 12 M sodium hydroxide and hydrogen peroxide were added together incrementally in a ratio determined by the amount of base added during the baseline PLS precipitation test to reach pH 7 and the amount of hydrogen peroxide that was added during the first oxidation test. In this case, 46.5 mL of 12 M base were used in the baseline test and 1.6 mL of hydrogen peroxide were used. Thus, base and hydrogen peroxide were added to the new 500 mL PLS solution in a 46.5:1.6 ratio until pH 3.5 was reached. After 3.5, base was added as normal to pH 9.

When looking at the results between the two tests, precipitation behavior was very similar. In both cases, iron was seemingly totally converted with 100 % recovery of iron being achieved before pH 4. Additionally, all other elements fell within their normal precipitation ranges. The main difference between the two tests was the amount of reagent required for each test. In the oxidation

test 1, 1.6 mL of hydrogen peroxide was used with 55.8 mL of sodium hydroxide. In oxidation test 2, 0.88 mL of hydrogen peroxide were used with 73.16 mL of base. This change in material usage comes from the $Fe(OH)_2$ to $Fe(OH)_3$ conversion that occurred when sodium hydroxide was added with peroxide proceeded faster than the $Fe^{2+}$ to $Fe^{3+}$ conversion that occurred when only peroxide was added.

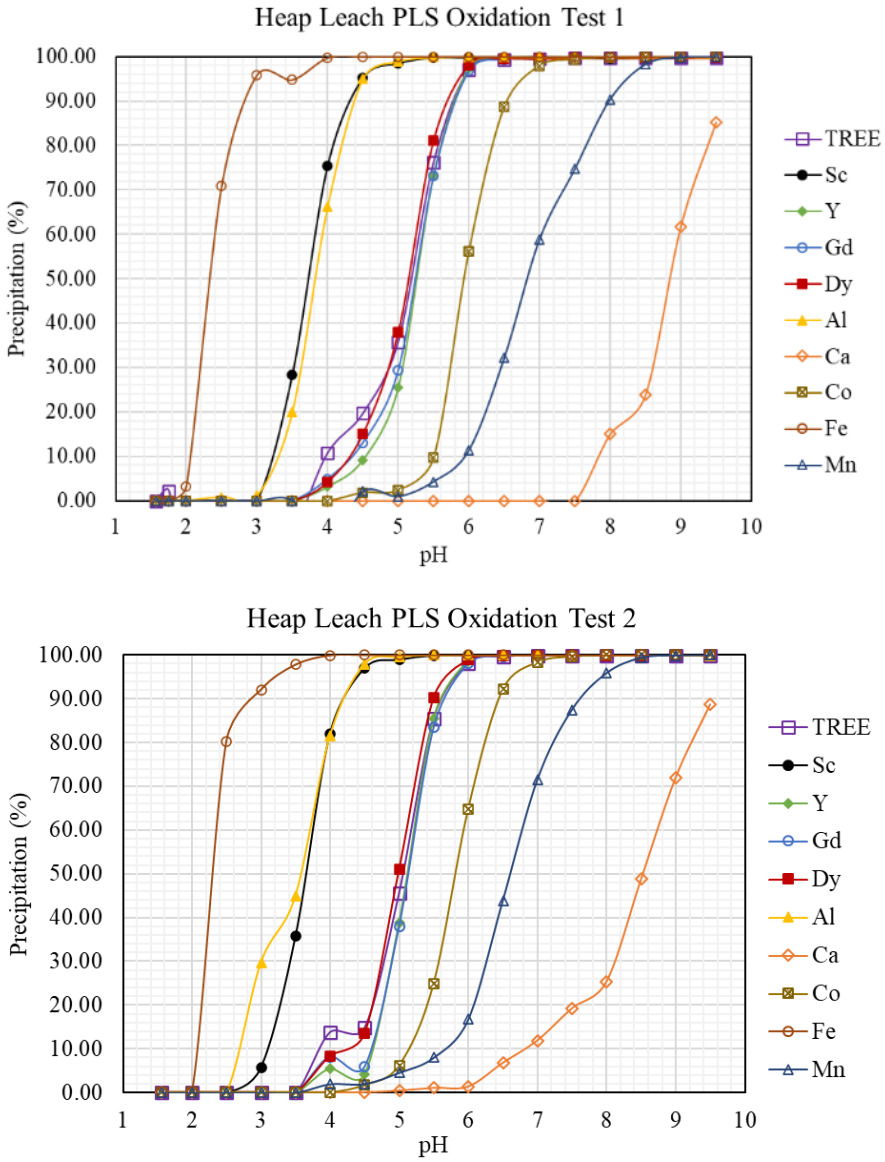
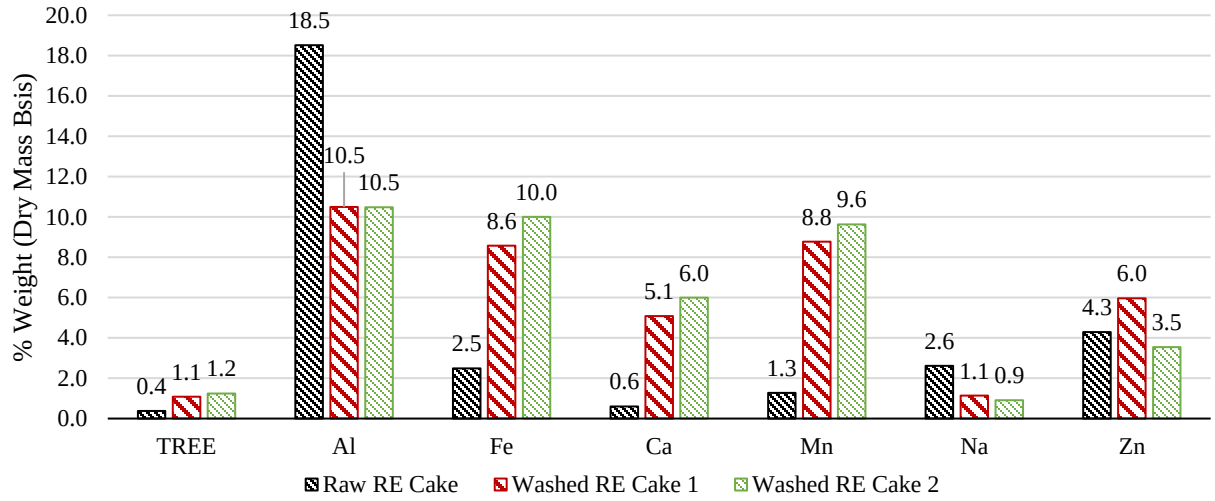



Figure 3.7.45. Percent recovery data from heap leach PLS tests using hydrogen peroxide. During Test 1, all hydrogen peroxide was added to the PLS before any base was added and precipitation occurred. In Test 2, base and hydrogen peroxide were added together incrementally in a 46.5:1.6 ratio.

### *1.1.8 Aluminum removal from REE precipitation cake*

**Alkaline Wash – RE Cake:** Rare earth precipitation cake was collected during the operation of the AMD pilot plant precipitation test, and it was determined that there were high levels of contaminant aluminum and sodium. To reduce the concentrations of these contaminants, an alkaline wash of the cake followed by a DI water wash was performed. Aluminum is redissolved into aqueous form when pH raises above 12-13 after initially precipitating at pH 4; thus, by mixing the rare earth cake in a concentrated base solution, aluminum will be removed from the rare earth cake. A 4 M sodium hydroxide solution was created for the alkaline bath and two ratios of cake to base were tested to compare efficacy. 20 g of RE cake were used in both cases and one test used 100 mL of base solution while the other used 200 mL. The cake and base solutions were mixed for 2 hours each using a magnetic stir bar at 700 rpm and were filtered afterward using a dead-end filtration cell to recover the cake. The waste base filtrate was collected for use in future tests. The washed cakes were then both washed in 500 mL DI water overnight to remove contaminant sodium from the cake. The solutions were filtered again to recover the final cakes which were dried overnight and digested for ICP analysis.




**Base Solution Reuse:** The base solution discharged from the alkaline wash was projected to be reused in the Fe and Al precipitation circuit. A lab test was conducted to use the base solution collected from the RE cake wash to precipitate Fe and Al from bioleach acid solution to determine the feasibility of recycling the waste base solution from a plant scale alkaline wash to be used in the precipitation stages of the plant. The use of this base would only be considered for the iron and aluminum precipitation stages as it would be undesirable to further contaminate any rare earth stream with possible aluminum. The base solution that was recovered from alkaline wash testing had a pH of 13 and was generated from the same original 4 M sodium hydroxide stock solution. Therefore, a precipitation test with fresh 4 M sodium hydroxide solution was performed as a control. The pH of the bioleach acid (BA) solution was elevated from the original pH of 1.1 to 4.5 to completely precipitate all possible iron and aluminum.

Table 3.7.13. Amount of base required to raise the Bio Acid to a measured pH benchmarks as well as the measured Eh at those benchmarks.

| BA pH | New Base (mL) | Used Base (mL) | New Base Eh (RmV) | Waste Base Eh (RmV) |
|---|---|---|---|---|
| 1.1 | 0 | 0 | 900.9 | 895.7 |
| 2.0 | 24.5 | 19.9 | 863.4 | 865.3 |
| 2.5 | 22.0 | 19.0 | - | 827.6 |
| 3.0 | 19.0 | 24.5 | 751.2 | 759.0 |
| 3.5 | 1.9 | 2.2 | 689.5 | 695.8 |
| 4.0 | 1.1 | 1.9 | 612.7 | 611.2 |
| 4.5 | 0.9 | 1.4 | 558.3 | 579.7 |

As in all precipitation tests performed, samples of the BA solution were collected at 0.5 increments for ICP analysis which show the efficiency of precipitation when using the two different bases. When comparing the amount of base required to complete the test, the used base test required less reagent, however with a different of only 0.5 mL of base being needed between the two tests one solution was not more effective at raising the pH than the other. As shown in Table 3.7.13, when using the fresh base, 65.5 mL reagent was needed to raise the pH from pH 1.0 to 3 when the most significant iron precipitation was occurring, whereas the used base required 63.4 mL. The opposite relationship can be seen from pH 3 to pH 4.5 where it took 5.5 mL of used base to raise the pH and only 3.9 mL of fresh base to complete the test and aluminum precipitation. This is likely due to the complexity of the Al precipitation behavior under high ionic concentration.

The recovery percentages of iron and aluminum of both tests are as shown in Figure 3.7.46 to compare the behavior. In the graph on the left, iron percent recovery is the same between the tests with both bases producing 100% recovery of iron at pH 3. As seen in the right graph, unlike iron, aluminum precipitation is significantly different between the two tests. Precipitation begins much sooner at pH 2.5 when using new base and aluminum is 100% recovered by pH 4.5 while the waste base does not begin recovering any aluminum until pH 3.5. In fact, before pH 3.5, aluminum concentration in the BA increases as the aluminum loaded waste base is added incrementally for the test. At pH 3.5 all the aluminum begins precipitating and is nearly 90% recovered at pH 4.5 where the test ends. The waste base is as effective at precipitating iron as fresh base however it seems to push the pH at which aluminum will begin and finish precipitating higher.

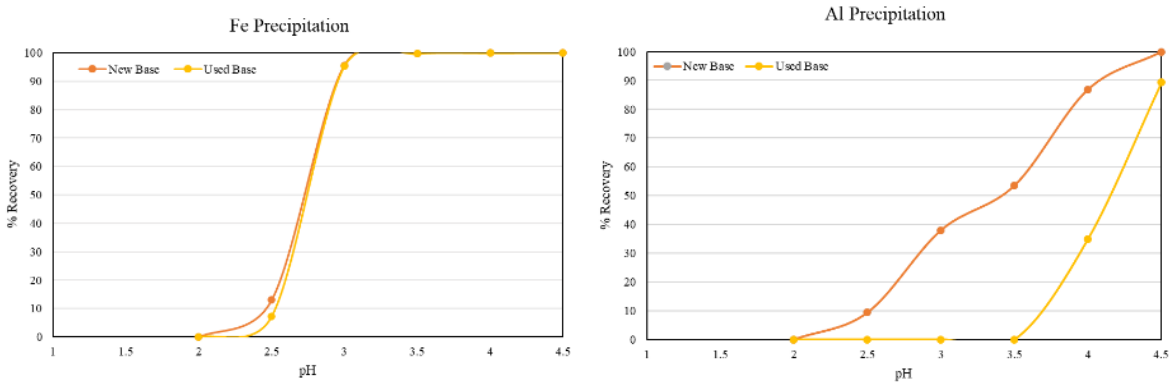



Figure 3.7.46. The percentage recovery of important contaminants aluminum and iron are compared between the new base precipitation test and used base test.

### *1.1.9 Oxalic Precipitation*

Oxalic acid precipitation was employed to selectively precipitate REEs from concentrated rare earth pregnant leachate solution (PLS) with presence of other contamination ions (i.e. Fe, Ca, and Al, etc.). The precipitation process is also called crystallization process which consists of two major steps, nucleation and crystal growth. In this study, a parametric investigation was performed to optimize the RE-oxalate precipitation efficiency with considering the impact of oxalic acid dosage, precipitation pH, temperature, and the ferric ion concentration on. The tests were conducted over a wide temperature range, and the solution pH was adjusted and maintained at different levels by the addition of pH modifiers. This study will assist in the identification of optimal conditions required to precipitate rare earth oxalate from a pregnant leach solution obtained from the leaching of coal.

**Materials:** The RE feedstock solution used in this study was generated from West Kentucky #13 coal seam coarse refuse material discarded from a coal preparation plant. The coarse refuse material was processed through physical beneficiation and hydrometallurgical circuit in a pilot scale plant. The material was air dried and processed through a X-Ray sorter where the RE content in solid was upgraded to the denser fraction. Then the upgraded solid material was crushed and ground through a jaw crusher and hammer mill, respectively, to acquire a top size of 1 mm particle. The hammer mill product was roasted in a continuous operated tube furnace with an inner tube temperature of 600 °C and a residence time of 20 minutes. The roasted material was then leached using 0.05 M sulfuric acid at 75 °C with solid concentration of 100 g/L. The pH value of the pregnant leachate solution (PLS) was 3.19±0.11 and contains about 9.5 ppm of total REEs (TREEs), 93 ppm of Al, and 800 ppm of Fe. The PLS was further upgraded using multiple stages of precipitation and redissolution. 45% of aluminum was removed by elevating the pH to 4.3 using 2M NaOH. Then the REEs were precipitated at pH 7.0 using 2M NaOH. The precipitated RE sludge which contains Al and Fe was then re-leached using HCl at pH of 2.5 to selectively dissolve REs in a concentrated solution. The re-leached PLS contains 34.6± 1.52 ppm of TREE, 60.8± 2.35 ppm of Al, 129.1± 10.54 ppm of Ca, and 149.3± 7.19 ppm of Fe, which is the feedstock solution used in this study. The distribution of the individual RE element in the feedstock solution is as shown in Figure 3.7.47.

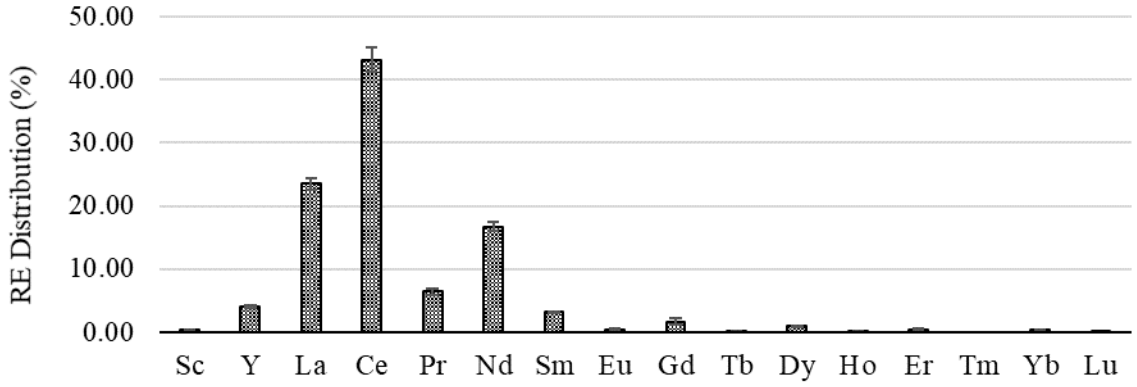



Figure 3.7.47. Rare earth element distribution in the feedstock solution.

**Methods and Methodology:** Each test started with the addition of 1% $Fe^{+3}$ solution into a 250 ml round-bottom flask from Pyrex. In the next step, a 100 ml pregnant leach solution was added and pre-heated to the required temperature. The oxalic acid was added to the solution after it reached the desired temperature. The pH of the solution was modified as per the test requirement using 4M NaOH alongside trace metal grade HCl and monitored with Thermo scientific Orion-Versastar pro meter after each adjustment. The collected samples were filtered promptly using $0.45\mu m$ pore size 33mm syringe filter by Fisherbrand. A 1 ml sample was collected from the filtered solution and diluted in 9 ml of 5% $HNO_3$. The glass-wares were cleaned and rinsed properly with DI water after each test, and the same procedure was repeated. Main factors investigated in this study are:

a. Oxalic Acid Dosage: The oxalic acid solutions for the tests were prepared at different concentrations of 40, 80, 120, and 160 g/L with DI water and dissolved completely with an ultra-sonic bath. Each test used a fixed 4ml concentration of the oxalic acid solution throughout the experimental plan.

b. pH: The acidity of the solution ranged from 0.5-2.5, as per the test constraint.

c. Temperature: The minimum temperature used in this study was 12.5 ºC, whereas the highest temperature was 62.5 ºC.

d. Iron contamination: The 1%Fe solution was prepared for the test plan and then added in the range of 0-4 ml, according to each test requirement.

e. Time: Every test was run for an hour, and continuous 2 ml samples were collected at 5,10,20,30, and 60 minutes throughout the test plan.

**Parametric Study:** A central composite design was established taking into account the oxalic acid dosage, pH, temperature, and iron content of the solution. The test parameters with their variable level used for the experiment design in Design Expert are shown in Table 3.7.14. The +1 sign indicates the upper-level value for the parameter, whereas the -1 value indicates the lower value used for the parameter.

Table 3.7.14 Operational variables and their levels for Central composite design of oxalic acid precipitation

| Factors | name | unit | Coded variable level | | | | |
|---|---|---|---|---|---|---|---|
| | | | lower | low | center | high | higher |
| A | Oxalic Acid Solution Concentration | g/L | 0 | 40 | 80 | 120 | 160 |
| B | Temperature | °C | 12.5 | 25 | 37.5 | 50 | 62.5 |
| C | Fe Stock Solution Addition | ml | 0 | 1 | 2 | 3 | 4 |
| D | pH | | 0.5 | 1 | 1.5 | 2 | 2.5 |

**Analysis (ICP):** The samples were analyzed using ICP-OES, and the REE precipitation efficiency values were calculated using;

$$Precipitation\ efficiency\ (\%) = \frac{C_f * V_f - C_l * V_l}{C_f * V_f} * 100$$

where $C_f$ and $C_t$ are the concentration of elements present in the feed, and the liquid sample, and $V_f$ and $V_t$ are the volume of feed and liquid sample, respectively.

**Results and Discussion:**

a. Effect of Oxalic Acid Concentration

The oxalic acid dosage has a direct effect on the RE precipitation efficiency. This association can be attributed to the fact that the higher oxalic acid concentration in the solution will introduce higher oxalate anion content that reacts with rare earth elements to form rare earth precipitates. According to (Han 2020), the rare earth elements in a sulfate media react with oxalate anions following the reaction mechanism shown in (2) at pH <2 and the potential atomic structure of resulting rare earth oxalate precipitate is shown in Figure 3.7.48.

$$2\mathrm{RE}(SO_4)_2^- + 3(HC_2O_4^-) + H^+ \rightarrow RE_2(C_2O_4)_{3(s)} + 4(HSO_4)^- \quad (2)$$

In this study, the RE precipitation efficiency was estimated at the oxalic acid concentrations of 0, 80, and 160 g/L, and the results are shown in Figure 3.7.48. It was observed that there is a significant variation in the recovery of rare earth elements when the reagent dosage is increased from 0-80 g/L. The RE precipitate recovery in the absence of the reagent was approximately 4.27%, rising to 95% recovery was witnessed at 80 g/L oxalic acid dosage. Moreover, there was no improvement in the RE precipitation efficiency with increased oxalic acid concentration from 80 g/L to 160 g/L. This observation is in agreement with the various other researchers (Güneş, Obuz, and Alkan 2019), who reported the ineffectiveness of increased oxalic acid dosage after a certain concentration.

Conversely, the addition of surplus oxalic acid, i.e., 160 g/L, increased the Fe-contamination of rare earth precipitates by as much as 5% as compared to lower dosages of the reagent discussed above. A possible reason behind this might be the reduction of Fe (III) to Fe (II). It has been established in previous studies that the iron precipitates as ferrous oxalates ((Panias et al. 1996; Sun and Qiu 2012; Zeng, Li, and Shen 2015)). As the oxalic acid is a mild reducing agent, the

surplus quantity of oxalic acid can reduce iron from ferric to ferrous and precipitate it as ferrous oxalates by undergoing reactions (3)-(5) (Bard, Parsons, and Jordan 1985). Further details on the precipitation of iron will be discussed later in the paper.

$$C_2O_4^{2-} \leftrightarrow 2CO_2 + 2e^- \quad (3)$$

$$2Fe^{+3} + 2e^- \rightarrow 2Fe^{+2} \quad (4)$$

$$Fe^{+2} + 3C_2O_4{}^{2-} \leftrightarrow Fe\ C_2O_4.2H_2O(s) + 2C_2O_4{}^{2-} \quad (5)$$

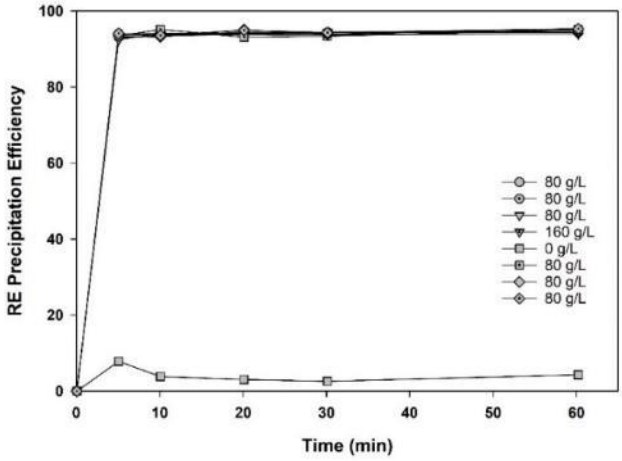



Figure 3.7.48 Comparison of RE precipitation efficiency at various concentrations of oxalic acid and fixed pH=1.5, temp=37.5 ℃, and 20,000 ppm Fe contamination

b. Effect of pH

The acidity of the solution has a significant impact on the recovery and purity of rare earth precipitates. According to (Bureau 2017), this influence can be explained by the impact of pH on the activity of oxalate anions, the primary precipitants for the rare earth precipitation. This investigation revealed a positive correlation of the solution's pH with the REE precipitation efficiency. High acidic conditions, i.e., 0.5 pH, did not produce any rare earth precipitates. Furthermore, the REE precipitation efficiency increased from 94.8% at pH 1.5 to 98.91% at pH 2.5.

The maximum recovery of 98.34% was attained at pH 2.5, and the lowest recovery was estimated to be 7.85% at pH 0.5, as shown in Figure 3.7.49. According to ((Chi and Xu 1999),(Bureau 2017),(Han 2020)), this significant change in the recovery can be explained by the significantly lower concentration of $C_2O_4^{2-}$ at pH < 1.5 and decreased in the affinity of rare earths for oxalate complexes as a function of reducing pH. However, the difference in the recovery of rare earth elements on pH 1.5 and 2.5 is only 3.6%. It can be claimed that most of the rare earth elements precipitate at pH 1.5. It is also worth mentioning that higher pH levels reduce the oxalic acid dosage required for REE-precipitation due to the increased oxalate anions concentration present in the less acidic solution.

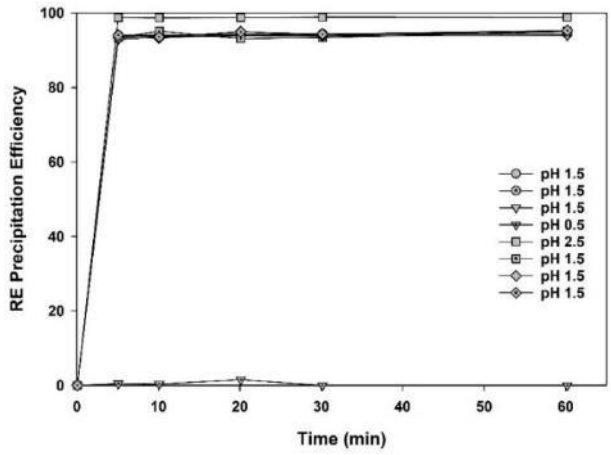



Figure 3.7.49 Effect of pH on the RE precipitation efficiency at 80g/L oxalic acid concentration, 20,000 ppm iron contamination, and 37.5ºC temperature

c. Effect of iron contamination:

Several researchers have reported the Fe reduction properties of oxalate anions ((Zeng, Li, and Shen 2015; Sun and Qiu 2012; Panias et al. 1996)). The oxalate anions have two oxygen atoms with unshared pairs of electrons, making the ion a strong complexing agent. In the presence of ferric and aluminum ions, the oxalate ions tend to form chelate complexes with these impurities, resulting in a soluble complex ((Sarı et al. 2020),(Christodoulou, Panias, and Paspaliaris 2001)). Ferric oxalates [$Fe_2(C_2O_4)_3$] have high stability in oxalic acid solution. However, ferrous oxalates ($FeC_2O_4.6H_2O$) are highly insoluble ((Önal et al. 2015; Yang et al. 2015)). Therefore, it can be concluded that Fe-contamination of RE-oxalates in an oxalate system is possible by the reduction of Fe(III) to Fe(II). The reduction of oxalate and, consequently, Fe is possible through reactions (3)-(5).

In this investigation, the Fe(III) contamination level was changed from 0 to 40,000 pm (0-4 ml) in the stock solution to examine the influence of Fe on the RE precipitation efficiency. It was determined from Figure 3.7.50 that the addition of iron up to 20,000 ppm does not significantly impact the RE precipitation efficiency. Further increase in the Fe-contamination to 40,000 ppm (4 ml) significantly drops the RE precipitation efficiency from 94.81% to 79%. As per Christodoulou et al., the formation of ferric oxalates as chelate complexes consumes oxalate anions (Christodoulou, Panias, and Paspaliaris 2001), decreasing the oxalate anion content available for RE-oxalate formation and thereby reducing the RE precipitation efficiency. According to Venkatesan et al., this problem can be addressed by increasing the oxalic acid dosage in the system ((Liu et al., n.d.),(Venkatesan et al., n.d.)).

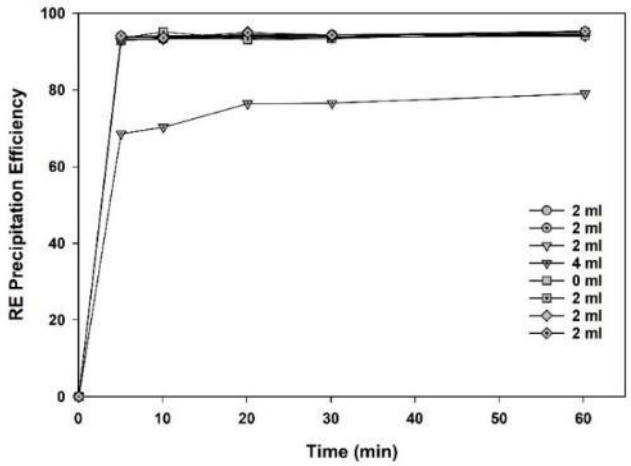



Figure 3.7.50 Impact of different Fe(III) contamination levels on the RE precipitation efficiency at 80g/L oxalic acid concentration, pH 1.5 and 37.5ºC

d. Effect of temperature:

The dependence of RE precipitation efficiency on temperature was ascertained by performing experiments at 12.5, 37.5, and 62.5 °C. It was determined that there is an inverse association between the temperature and precipitation of rare earth elements, as shown in Figure 3.7.51, indicating an exothermic reaction. The RE precipitation efficiency increases with a decrease in temperature with a maximum recovery of 99% obtained at 12.5 °C. The solubility of rare earth elements and oxalic acid has been reported to increase with an increase in temperature, owing to the dissociation of oxalic acid ((Apelblat and Manzurola 1987),(Hussain and Khan 2012)). Furthermore, according to Le Chlatelier's principle, increasing the reaction temperature shifts the reaction to the reverse direction. Consequently, the RE precipitation efficiency decreases from 99% to 95% at 37.5 °C, and ultimately, to its lowest point of 80% at 62.5 °C.

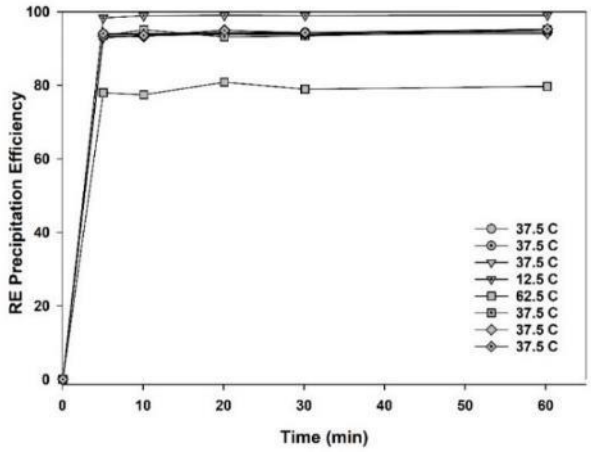



Figure 3.7.51 Effect of temperature on the RE precipitation efficiency at 80g/L oxalic acid concentration, pH 1.5, and 346 ppm of Fe.

**Model and 3-D Response Surface:** The results of the central composite factorial experimental design performed for the estimation of rare earth precipitation efficiency are given in table below.

. The quadratic regression model expression for the response was obtained from Design-Expert software, and the semi-empirical model after the elimination of insignificant coefficients can be expressed as follows:

$$\text{REE Precipitation efficiency} = 1.25 + 0.373 * \text{A} - 0.107 * \text{B} - 0.1461 * \text{C} + 0.222 * \text{D} + 0.161 * \text{AC} - 0.153 * \text{A}^2 - 0.1627 * \text{D}^2$$

where A is the oxalic acid dosage in ml, B temperature, C iron contamination in ml, and D pH. The significance of the model and individual parameters, along with their interactions, were tested against the null hypothesis that the coefficient for the variable is null. The parameters with a p-value < 0.05 were deemed significant. The forward elimination method was used to remove all insignificant parameters, and the model efficacy was determined through a higher adjusted $R^2$ value of the best fit model. Furthermore, the investigation of residual errors for the selected model revealed an independent distribution of errors with constant variance and zero mean.

The acquired quadratic regression model and its significance are shown in Table 3.7.16. The p-value <0.05 indicates that the overall model is highly significant for predicting REE precipitation efficiency. The $R^2$ and adjusted $R^2$ value of 85 and 80%, respectively, also suggests a good fit for the model. Additionally, as demonstrated in

Table 3.7.17, all the parameters used in this investigation were found to substantially impact the REE recovery. Figure 3.7.52 shows the impact of individual parameters on RE precipitation efficiency as predicted by the model. The oxalic acid dosage and pH showed a strong positive correlation, whereas the temperature and Fe content of the solution demonstrated a negative impact on the RE precipitation efficiency. The investigation also revealed an interaction between the oxalic acid dosage and the iron content of the solution. This interaction effect is shown in Figure 3.7.53 as 3-D response surface graph. It can be concluded that the higher Fe concentration present in the solution requires higher subsequent oxalic acid dosage to achieve high RE precipitation efficiency. Contrarily, lower oxalic acid dosage is required at smaller Fe concentration in solution.

Table 3.7.15. Central composite design experiments used for the oxalic acid test plan.

| Run | Oxalic Acid Concentration (g/L) | Temperature °C | Fe Addition (ml) | pH | TREE Recovery (%) |
|---|---|---|---|---|---|
| **1** | 40 | 50 | 1 | 1 | 0.83 |
| **2** | 80 | 12.5 | 2 | 1.5 | 99.06 |
| **3** | 40 | 50 | 1 | 2 | 68.98 |
| **4** | 80 | 37.5 | 2 | 0.5 | 0.00 |
| **5** | 40 | 25 | 1 | 1 | 49.96 |
| **6** | 80 | 37.5 | 2 | 1.5 | 94.47 |
| **7** | 80 | 37.5 | 2 | 1.5 | 94.55 |
| **8** | 40 | 50 | 3 | 2 | 0.00 |
| **9** | 0 | 37.5 | 2 | 1.5 | 4.28 |
| **10** | 120 | 25 | 3 | 1 | 91.56 |
| **11** | 80 | 37.5 | 2 | 1.5 | 94.02 |
| **12** | 120 | 25 | 1 | 1 | 94.97 |
| **13** | 40 | 25 | 3 | 2 | 0.62 |
| **14** | 80 | 37.5 | 0 | 1.5 | 94.26 |
| **15** | 120 | 25 | 1 | 2 | 99.88 |
| **16** | 40 | 50 | 3 | 1 | 5.45 |
| **17** | 120 | 50 | 1 | 1 | 68.84 |
| **18** | 80 | 37.5 | 4 | 1.5 | 79.05 |
| **19** | 120 | 50 | 3 | 2 | 96.56 |
| **20** | 40 | 25 | 3 | 1 | 0.82 |
| **21** | 120 | 50 | 3 | 1 | 68.22 |
| **22** | 120 | 25 | 3 | 2 | 97.34 |
| **23** | 80 | 62.5 | 2 | 1.5 | 79.71 |
| **24** | 120 | 50 | 1 | 2 | 97.51 |
| **25** | 80 | 37.5 | 2 | 1.5 | 95.37 |
| **26** | 80 | 37.5 | 2 | 1.5 | 95.34 |
| **27** | 160 | 37.5 | 2 | 1.5 | 94.61 |
| **28** | 40 | 25 | 1 | 2 | 95.51 |
| **29** | 80 | 37.5 | 2 | 1.5 | 95.12 |
| **30** | 80 | 37.5 | 2 | 2.5 | 98.91 |

Table 3.7.16 The analysis of variance (ANOVA) table for the suggested model

| Source | Sum of Squares | df | Mean | F-value | Significance |
|---|---|---|---|---|---|
| **Model** | 6.99 | 7.00 | 1.00 | 17.65 | 0.0001 |
| **Residual** | 1.24 | 22.00 | 0.06 | | |
| **Total** | 8.23 | 29.00 | | | |

Table 3.7.17 The ANOVA for the individual parameters used in the study.

| Source | Sum of Squares | df | Mean Square | F-value | p-value |
|---|---|---|---|---|---|
| A-Oxalic Dosage | 3.34 | 1 | 3.34 | 58.96 | < 0.0001 |
| B-Temperature | 0.2735 | 1 | 0.2735 | 4.83 | 0.0387 |
| C-Fe Addition | 0.5126 | 1 | 0.5126 | 9.06 | 0.0064 |
| D-pH | 1.18 | 1 | 1.18 | 20.80 | 0.0002 |
| AC | 0.4147 | 1 | 0.4147 | 7.33 | 0.0129 |

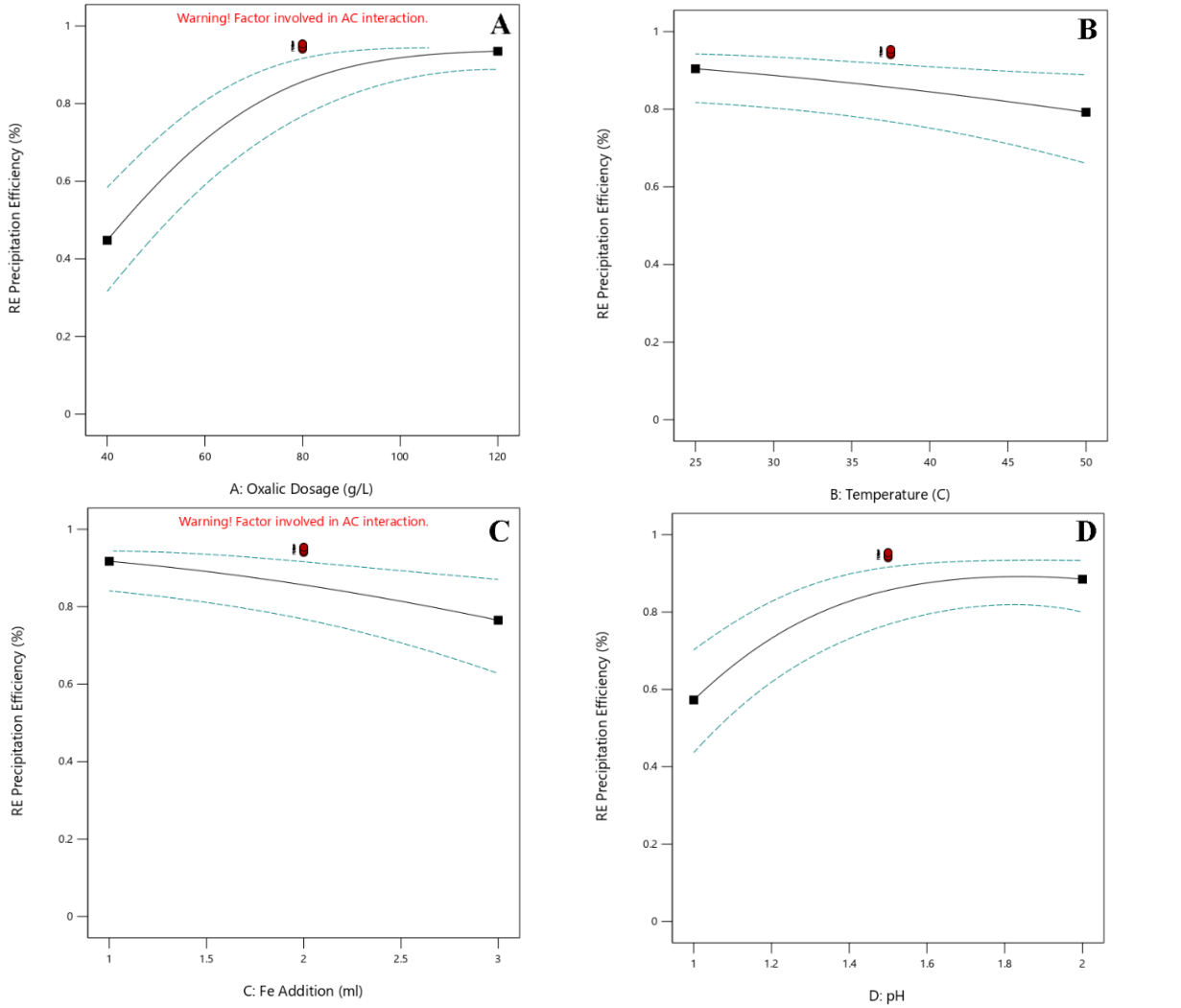



Figure 3.7.52 Impact of A: Oxalic acid dosage, B: Temperature, C: Fe Addition, and D:pH on RE precipitation efficiency.

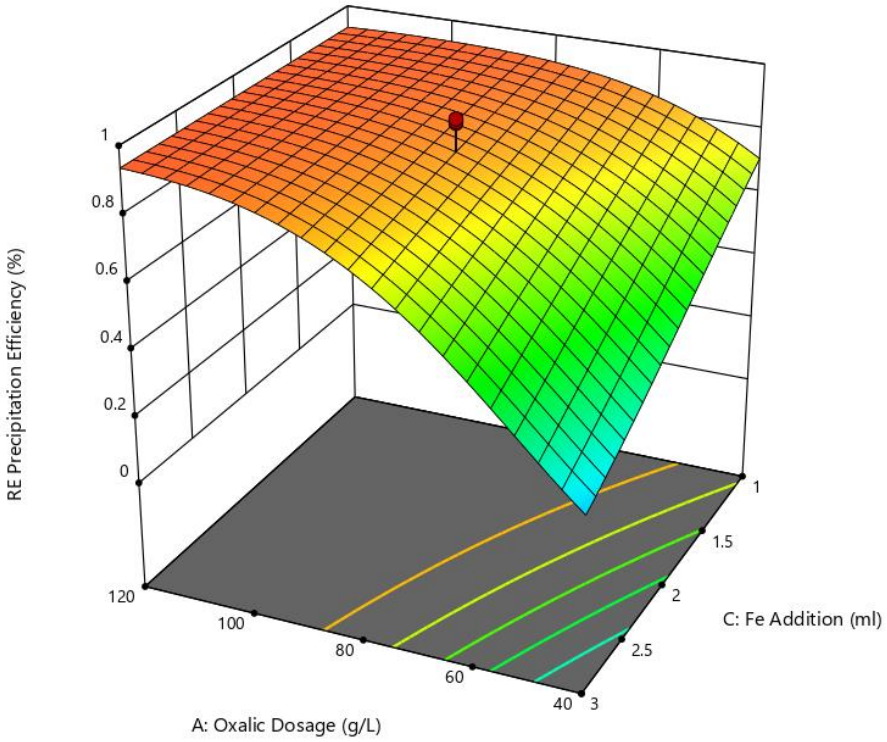



Figure 3.7.53 Response surface for optimization of rare earth precipitation efficiency as a function of oxalic acid dosage and Fe addition.

**Model Validation:** The semi-empirical quadratic regression model was validated under the conditions which minimized the required oxalic acid dosage and fixed the temperature at 25 ºC with a pH in 1-2 range at various Fe contamination levels. The validation test results, along with their conditions, are shown in Table 3.7.18. The predicted value for REE precipitation efficiency at Fe concentration of 1 and 2 ml were 83.94 and 76.37%, showing a good agreement with the actual test data, i.e., 72.48 and 79.84%, respectively. However, there was a significant disparity in the predicted and actual data when there was no Fe addition. This discrepancy implies that the model can accurately predict the REE precipitation efficiency only when the Fe-contamination is in the model input range, i.e., 1-3 ml.

Table 3.7.18. Tests performed for the model validation.

| Number | Oxalic Dosage | Temperature | Fe Addition | pH | REE Precipitation Efficiency (%) | |
|---|---|---|---|---|---|---|
| | | | | | Predicted | Actual |
| **1** | 35.71 | 25.00 | 1.00 | 1.84 | 83.94 | 82.42 |
| **2** | 49.64 | 25.00 | 2.00 | 1.84 | 76.37 | 79.84 |

**Production of mixed RE Oxides:** The information obtained from the oxalate precipitation study was implemented into production of a RE mixed oxide product. The composition of the RE oxide achieved is shown in Figure 3.7.54.

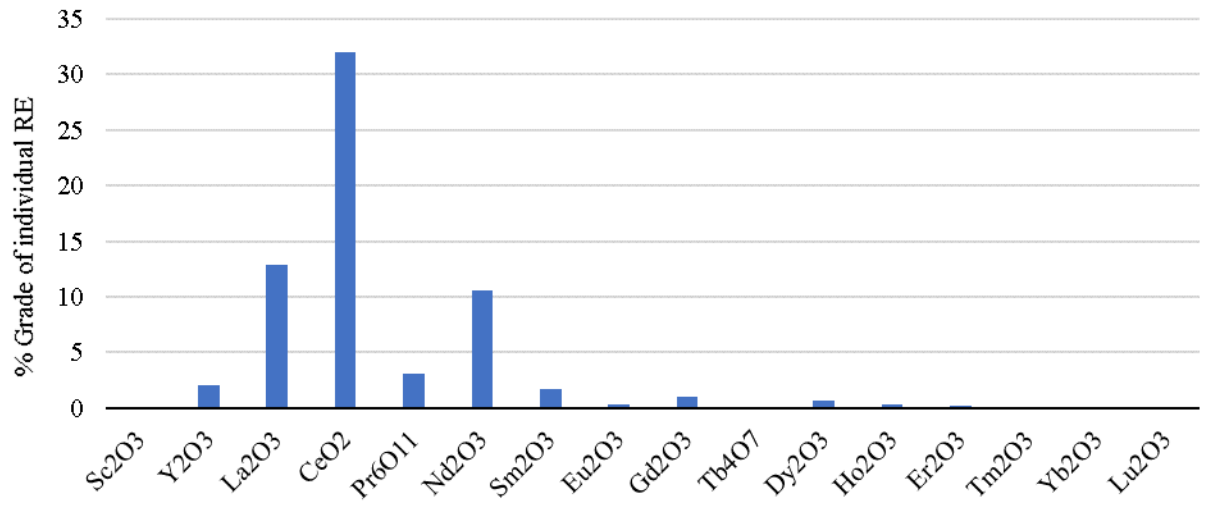



Figure 3.7.54. The grade of individual RE oxide in a mixed RE product generated from a sulfuric acid leach scenario. (DTK-C14-REO).

### *1.1.10 Optimization of RE purification circuit*

The collected wet RE cake from PLS 3 test was further processed through the RE purification circuit to generate high grade RE oxalate product. The cake was first processed through an aluminum redissolution stage using 4M NaOH solution to separate Al from REEs, then rinsed with water to remove sodium. Then REEs were re-leached using hydrochloric acid at pH 2.5 to redissolve REEs in solution. The redissolved and concentrated REE bearing solution was processed through oxalate precipitation to generate high grade REEs. The process was shown in Figure 3.7.55.

Total 324 lbs. of wet RE cake was collected from the precipitation circuit. In the Al redissolution process, the wet cake was added into 4M NaOH solution at a ratio of 2 lbs. wet cake per 1 gallon solution. The cake was mixed until the pH was stable then filtered using a press filter. The RE cake contains 11.6% of Al by weight, after Al dissolution, 5182 ppm of Al was leached into the alkaline solution, which is 19.6% of the Al in the cake. Due to the high viscosity of the base solution, water was added to dilute the system. After filtration, the cake was blended with water again to remove the excessive Na entrained in the cake, then filtered again using a press filter. The wet cake collected after rinsing is approximately 300 lbs. The cake was then blended with 150 gallon water. The pH was adjusted to be 2.5 using hydrochloric acid to redissolve the rare earth elements. The solution collected from RE redissolution contains 135 ppm of TREEs with 1970 ppm of Al, 340 ppm of Ca, 39 ppm of Fe, and 56 ppm of Co. The distribution of REEs in the re-leached REE PLS is shown in Figure 3.7.56.

The solution was then fed into the REE oxalate precipitation circuit. A parametric study was carried out in the lab to determine the optimized oxalic acid dosage and precipitation pH. The condition was determined to be 16g of oxalic acid per liter of PLS at pH of 1.2 adjusted using 1M NaOH solution. The oxalic acid was prepared in a concentration of 80g/L and pumped into the oxalate circuit using a peristaltic pump with controlled flowrate. To optimize the oxalate precipitation efficiency, seeding effect was examined in the pilot plant continuous operation. The oxalate circuit

utilizes three 15-gallon cone bottom mixing tank in sequence. The feed solution, oxalic acid solution, and base solution were added into the first reaction tank at a flowrate of 1.3 gpm, as shown in Figure 3.7.57. The first half of feed solution was processed without recirculation. Subsequently, the second half of the feed solution was processed while recirculating the underflow of reaction tank No. 3 back to the first tank to assist with crystallization of RE oxalate. Samples were collected from both tests and the recovery of REEs and major elements of interest were compared as shown in Figure 3.7.58.

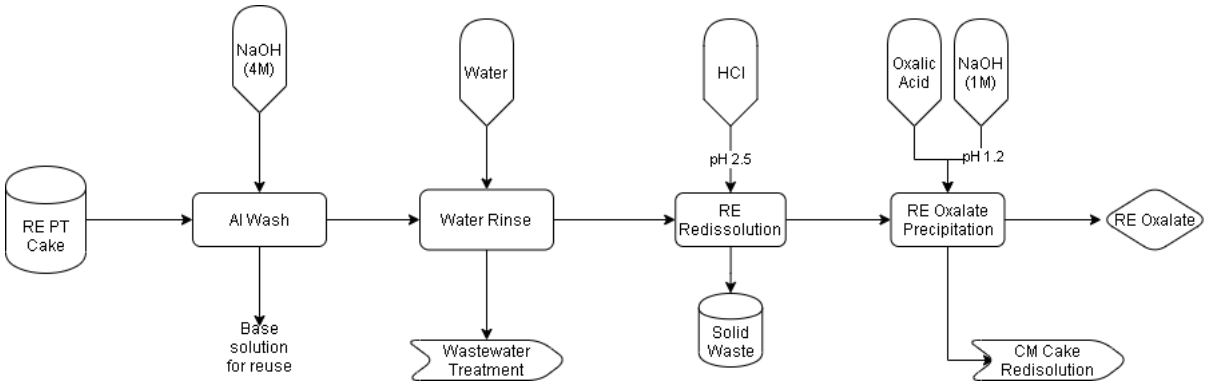



Figure 3.7.55. PLS Run #2 REE purification circuit.

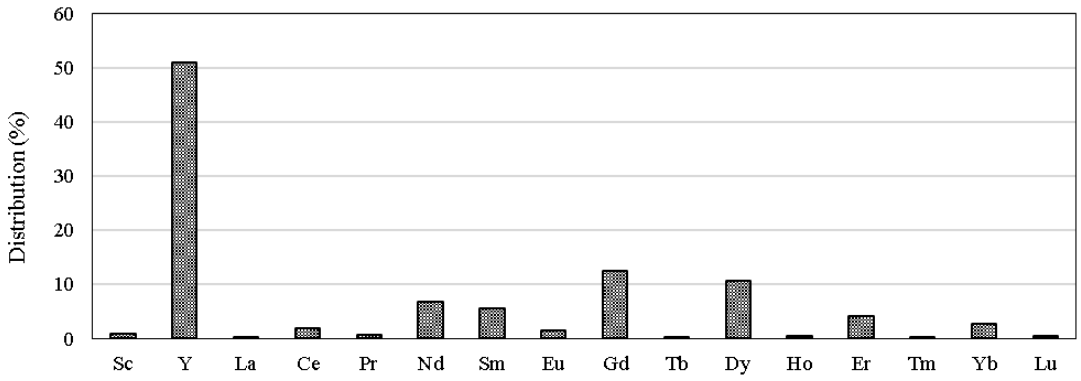



Figure 3.7.56. REE distribution in oxalate precipitation circuit feed.

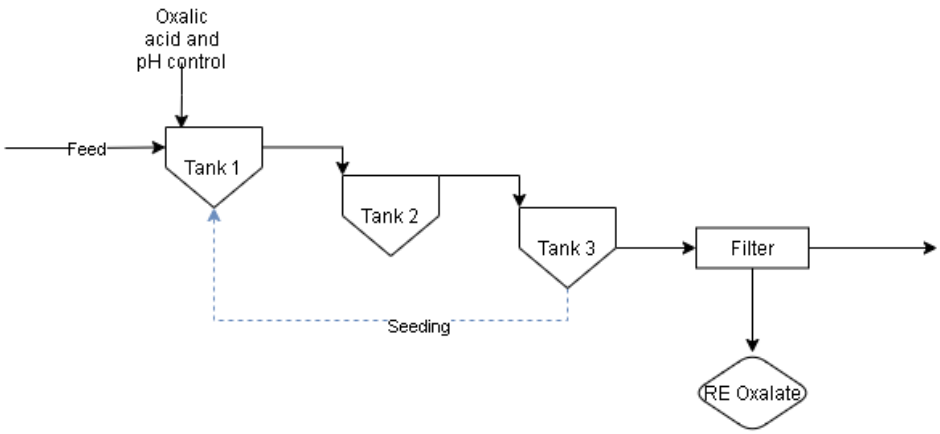



Figure 3.7.57. Schematic of rare earth oxalate precipitation circuit arrangement.

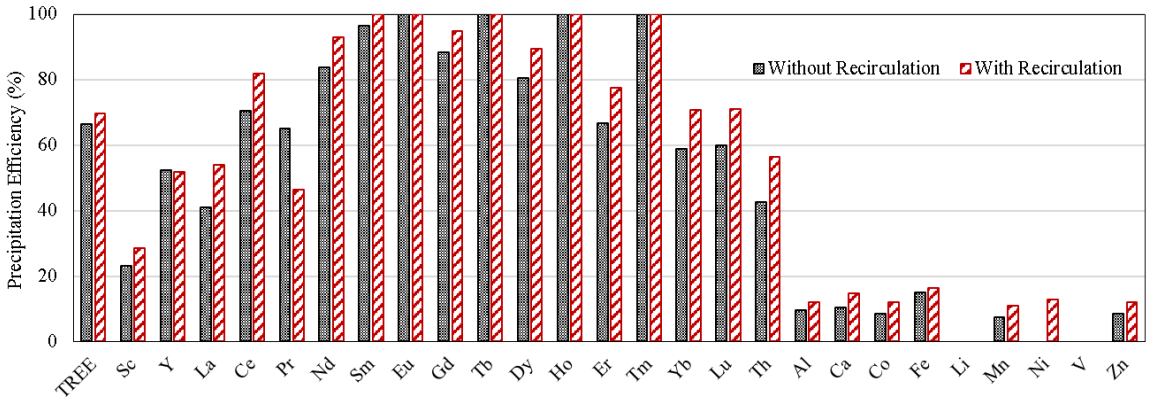



Figure 3.7.58. Comparison of rare earth oxalate precipitation efficiency with and without seeding.

### *1.1.11 Settling and filtration circuit improvement*

**<u>Improvement of settling rate with aid of flocculant</u>:**

Due to the high concentration of Fe and Al in the heap leach PLS solution, the amount of solids generated in the Fe and Al precipitation process required a significantly long settling time in the thickener. It was determined that a flocculant was needed for the thickeners in the precipitation stages to aid the solid-liquid separation and operate effectively. Based on the screening tests conducted by SGS Lakefield, BASF Magnafloc 333 (M333) was selected to be used for aluminum and iron precipitation and BASF Magnaflocc 338 (M338) was chosen for the Rare Earths precipitates. Settling and filtration tests were performed to determine the optimum dosages of each flocculant for each stage.

Solutions of M333 and M338 with concentration of 0.1% by weight were created for the tests as recommended by the manufacturer and SGS. Settling tests performed on 250- or 200-ml samples to accommodate for the limitations of the equipment at the pilot plant. Samples of precipitate were procured by performing 500-ml precipitation tests of the heap leachate that was to be used in the

plant. Testing started with using the recommended dosages of flocculant from SGS and were increased as necessary.

The recommended dose from SGS for iron precipitate was 0.5 mL of floc solution per 1 L of slurry which is equivalent to 0.125 ml floc per 250- ml slurry. The flocculation tests performed at SGS were conducted at 1% solids while the samples generated in the lab generally were 1.4% solids. Significantly more flocculant was required when hydrogen peroxide was used to oxidize iron although the solid percentage was not considerably higher after use. A dosage of 6 ml of floc per 250 ml of slurry was sufficient to effectively separate the solids from the liquid; however, iron precipitate with hydrogen peroxide required at least 45 ml floc per 250 ml slurry to produce a similar effect. This could be due to the bubbles generated from peroxide oxidation “lifting” the precipitate and slowing the ability of flocs to settle quickly. Overall, a greater amount of flocculant was required for iron than was recommended by SGS. The relationship between solid percentage and the amount of floc needed does not seem linear or predictable. Any slight difference in feed conditions changed requirements drastically across all flocculation tests.

Aluminum tests followed the same procedure as iron tests with the added step of filtering out the iron precipitate to obtain a pure aluminum sample after further precipitation. The leach solution was brought to a pH of 4.5 from 3.5 to ensure all aluminum was precipitated. All tests were done using precipitate obtained from a precipitation test that utilized hydrogen peroxide to remove all possible iron from the leach solution. This resulted in 1.9% solid in most tests in comparison to the testing at SGS conducted at 0.5% solids. As a result, more floc than suggested was needed once again. Aluminum precipitate required more flocculant than iron precipitate with no peroxide addition which could be due to residual effects from hydrogen peroxide usage or aluminum having a higher % solid overall. 7 ml of floc per 200 ml of slurry was determined to be the optimum dosage after 10 tests.

The M338 flocculant that was used for the rare earths is an anionic polymer whereas the polymer used for aluminum and iron was an anionic. Only a few rare earth tests were completed but the optimum dosage was close to the recommendations provided by SGS. Due to availability of rare earth samples, a 3% solid solution was used for testing at the pilot plant in comparison to the 0.1% solid tests at SGS. Interestingly, with a solution of 3% solids, only 15 ml of flocculant were required per 200 ml of slurry versus the 7.2 that was recommended for the 0.1% solution. This testing resulted in results that were much more in line with what was expected based on the testing performed by SGS. This polymer created very large flocs of precipitate that did not settle likely due to air pockets entrapped within the flocs. In a larger sample, with more liquid, settling was very fast and thus was a better representation of what would be seen in a thickener. However, once mixed or poured into another container, these large flocs break up very easily into much smaller flocs or simply back to precipitate so very low agitation of this solution is important.

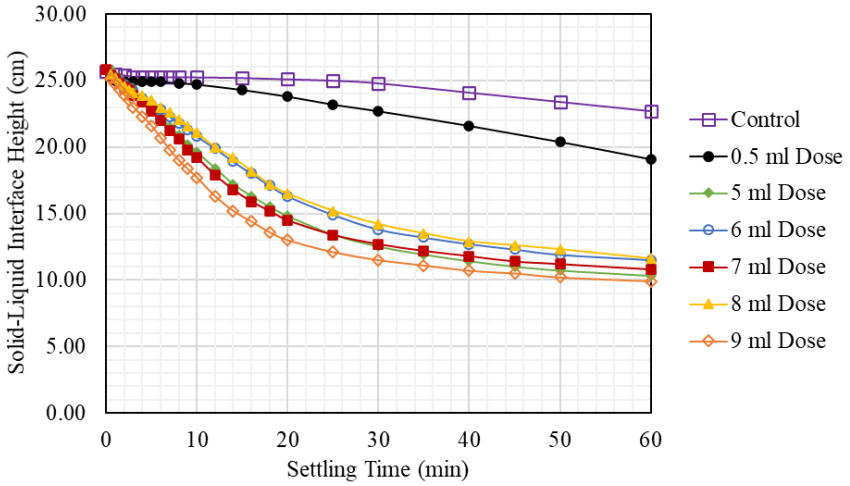



Figure 3.7.59. Fe precipitate settling velocity under various dosage of the 0.1% flocculant solution.

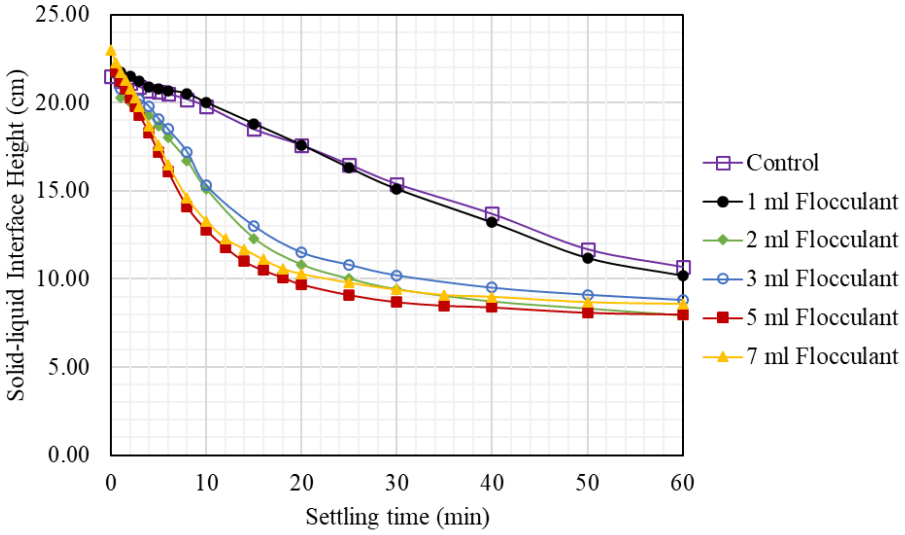



Figure 3.7.60. Al precipitate settling velocity under various dosage of the 0.1% flocculant solution.

**Filtration rate improvement testing:** Filtration tests were performed with aluminum and iron samples to determine the impact that flocculation would have on the filtration rates for both stages. Because flocculant increases the particle size of the material being filtered, filtration rate should increase, and the quality of filtration should improve to allow less breakthrough of solids at the beginning of filtration. All filtration tests were performed using a laboratory pressure filter at 30 psi and used a 10-micron filter paper. Precipitate samples for these tests were artificially generated in the lab. A 2-liter sample of heap leach solution was treated using hydrogen peroxide and sodium hydroxide to oxidize followed by iron precipitation. Generally, a pH of 3.5 was sufficient to recover 100% of the iron. To test only iron filtration, at this point, the optimal amount of flocculant for 2 liters was added with gentle mixing to the sample. After allowing the precipitate to settle for an hour, the beaker was decanted such that the bed of precipitate remained undisturbed and only around 200 ml of leach solution remained in addition to the precipitate bed. This solution was then

carefully stirred up to not break up any of the flocs and 250 ml of the slurry was used for filtration. To create an aluminum sample, the same method was employed, however, after iron precipitation was completed, all the iron was filtered from the leach solution using a vacuum filter. The filtrate was then collected and brought up to pH 4.5 to precipitate aluminum.

As seen in previous iron filtration without flocculant, the filtration rate of the precipitated material was extremely slow. The iron precipitating from this heap leach solution is small in particle size and quickly blinds a filter so that cakes with very high resistance are formed. This causes problems with filter presses filling up very quickly and taking a long time to air out. With polymer at the correct dosage and gentle mixing, filtration rate of the iron stage was improved. The flocs formed with effective flocculation of the iron stage are large and reduce the blinding effect of un-flocked solution. The filtration rate more than doubled from 0.002-gpm to 0.005-gpm and the quality of filtration was better as well.

Addition of the polymer to the aluminum stage seems to have the opposite effect on the aluminum stage. Filtration went very well with a sample of just precipitate: the filtration rate was fast and minimal particles broke through the filter before an adequate bed was built up. Once polymer was added, the flocs looked very healthy; however, they packed down very tightly or were very dense because the filtration rate was much slower. Aluminum filtration began at 0.017-gpm but reduced to below 0.002-gpm with the addition of flocculant. The quality of the filtration did improve and the filtrate of the test with polymer was completely clean of any breakthrough particles throughout the entire filtration. This suggests that filtering aluminum particles from the PLS is very easy but resistance from the cake will cause filters to appear to fill up very quickly as pumps will not be able to provide enough pressure to push through filtrate at the setpoint flowrate. Additionally, the cakes will be difficult to air out again because of the high cake resistance.

## 3.8 OPTIMIZATION OF CRITICAL MATERIAL RECOVERY (TASK 10)

Process circuitry was developed and optimized to co-extract critical materials from the coal refuse materials in a manner that is synergistic with the existing rare earth recovery circuitry. The process evaluated the recovery of Sc by solvent extraction. Tests were performed to identify the optimum extractant and circuit arrangement needed to concentrate Sc from the pregnant leach solution. In addition, processes and procedures were tested to develop a means to recover and concentrate Co, Ni and Mn. A process flowsheet was developed that is the subject of a patent application.

### *3.8.1 Process Development for the Recovery of Critical Materials*

Co/Mn recovery efforts primarily focused on acid mine leachates collected from a coal preparation plant processing Western Kentucky No. 13 coal. The acid mine leachates were generated due to pyrite oxidation under the function of oxygen and bacteria in the natural environment. Experimental laboratory tests were performed on the leachates to optimize the removal of major contaminants, as well as the concentration and purification of Co and Mn. Staged precipitation, selective re-dissolution, and selective precipitation methods were employed as the primary means to achieve the objectives. Co and Mn concentrates of greater than 2% purity were ultimately obtained from the leachates.

Moreover, since leach solutions generated from coal refuse through acid leaching have a similar elemental matrix as the acid mine leachates, it is expected that the flowsheets developed in this period for Co/Mn recovery and purification are also applicable to the leach solutions. This assumption will be tested in the next budgeting period. At the beginning, Co/Mn recovery and purification tests were performed on an acid mine leachate, which only had a total volume of around 10 gallons and was insufficient for completing a comprehensive experimental program. Therefore, another acid mine leachate of the same source as the first leachate was collected. The second leachate had a total volume of around 100 gallons, providing enough feed that finally led to the successful production of high-grade Co and Mn products. Detailed experimental procedures, results, and findings are presented below. To be more orderly, the presentation is divided into two subsections: Acid Mine Leachate I and Acid Mine Leachate II. The first section focuses on the tests performed on the first acid mine leachate, while the second section focuses on those performed on the second leachate.

**Acid Mine Leachate I:** An acid mine leachate of around 10 gallons was collected from a waste pond of Dotiki mine located in western Kentucky. Full elemental analysis was performed on the leachate, showing the presence of several critical metals, in addition to REEs, such as Co, Mn, and Li. As shown in the proposed process flowsheet (see Figure 3.4.1), REEs and other critical metals will be concentrated first using the staged precipitation method. Therefore, systematic precipitation tests were performed on the leachate to evaluate the viability of this method and update the process flowsheet per laboratory experimental findings if necessary.

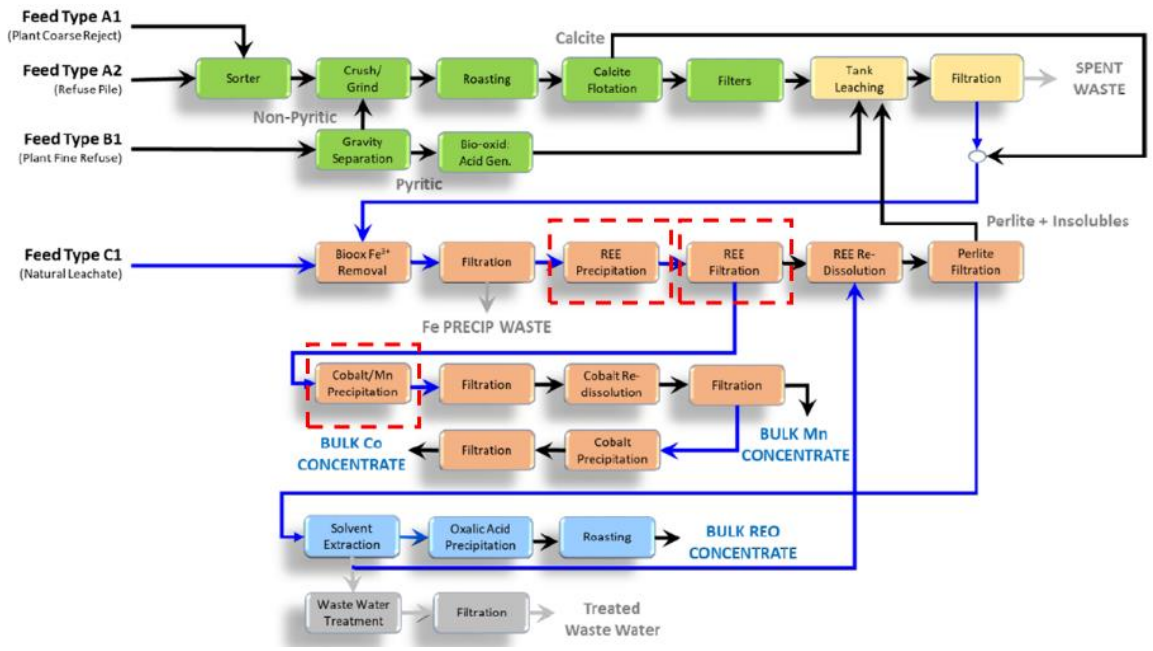



Figure 3.8.1.Integrated process flowsheet that will be tested in this project.

Immediately upon being collected, the leachate was stored in two 5-gallon buckets, which were tightly sealed and sent to the Mineral Processing Lab at Virginia Tech. The leachate was filtered using 2.5 µm pore size filter paper (GE Healthcare Whatman Grade 5) to remove any particulate matter. The resultant liquid was split and stored in buckets of a smaller volume, which were tightly sealed to reduce oxidation. The appearance of the samples is shown in Figure 3.8.2. Based on the color of the leachate, it can be told that the majority of iron contained in the material had been oxidized to ferric ions ($Fe^{3+}$).

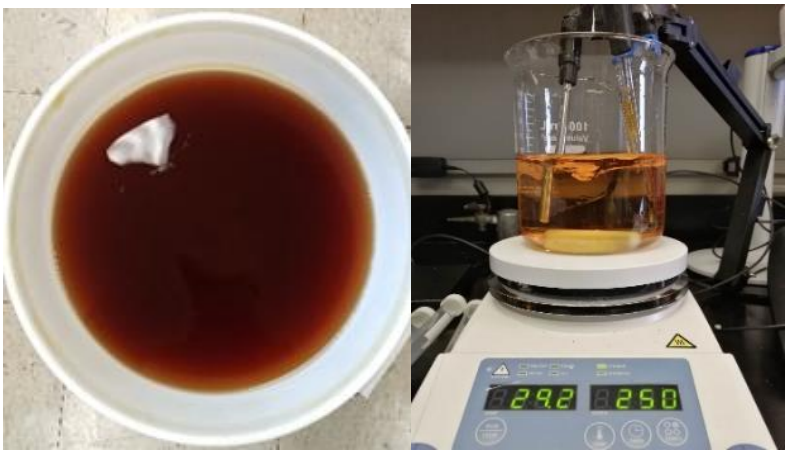


Figure 3.8.2. Acid mine leachate collected from western Kentucky: (left) prior to filtration; (right) after filtration.

Sequential precipitation tests were performed on the leachate to investigate the precipitation behavior of REEs and other valuable elements (Cu, Zn, Ni, and Co) as well as contaminant metal ions (Al, Fe, etc.). The tests were conducted by gradually adding alkaline solution into a 0.5 L

sample of the leachate. Three alkalis, including sodium hydroxide, sodium carbonate, and calcium oxide, were tested for comparison purposes. Small incremental pH adjustments were achieved by adding 0.25 mL NaOH solution, 0.25 mL $Na_2CO_3$ solution, or 0.05 g CaO followed by a certain period of magnetic stirring at 250 rpm. The purpose of the mixing was to eliminate non-selective localized precipitation caused by localized concentrations of the basic solution. Longer mixing was used for CaO relative to the other alkalis (5 min versus 1 min) since it takes a longer time for the pH to reach equilibrium when using CaO. Solution pH after each addition and mixing was recorded using a pH meter manufactured by Thermo Fisher Scientific (Orion™ Versa Star Pro™). Representative samples were collected during the test, followed by centrifuging at 4,000 rpm. The resultant supernatants were acidified and sent to the pilot plant for elemental analysis using ICP-OES. Precipitation recovery ($R\%$) of element $i$ is calculated based on the following equation:

$$R_i\% = \frac{C_{s,i} \times DF}{C_{f,i}} \times 100$$

where $C_{s,i}\ (ppm)$ and $C_{f,i}\ (ppm)$ represent elemental concentration in the supernatant and feed leachate, respectively; and $DF$ means dilution factor caused by adding the alkaline solutions. The equipment used for the staged precipitation tests is shown in Figure 3.8.3.



Figure 3.8.3. Experimental apparatus used for the sequential precipitation tests.

The leachate had a pH of 2.19, and its elemental concentration is shown in Error! Reference source n ot found.. The dominant metal ions in the solution were Fe (1962 ppm), Al (639 ppm), Ca (566 ppm), and Mg (535 ppm). Moreover, the leachate contained 5.9 ppm of total REEs (TREEs) with 3.32 ppm of heavy REEs (HREEs) and 2.58 ppm of light REEs (LREEs). The ratio of heavy to light REE concentration reached as high as 1.28, indicating a better resource of REEs compared with selected commercial rare earth ores (e.g., monazite). In addition, the leachate also contained 1.79 ppm of Co, 1.95 ppm of Li, and 85.15 ppm of Mn, all of which have been listed as critical mineral commodities in Executive Order 13817. Therefore, this sample can be used as an ideal feed for the study of critical metal recovery and purification. Few other elements, including Cu, Zn, and Ni, were also found in the solution, which will increase the economic viability of the overall process if they are recovered as byproducts.

Table 3.8.1.Elemental concentration (ppm) of the leachate analyzed using ICP-OES.

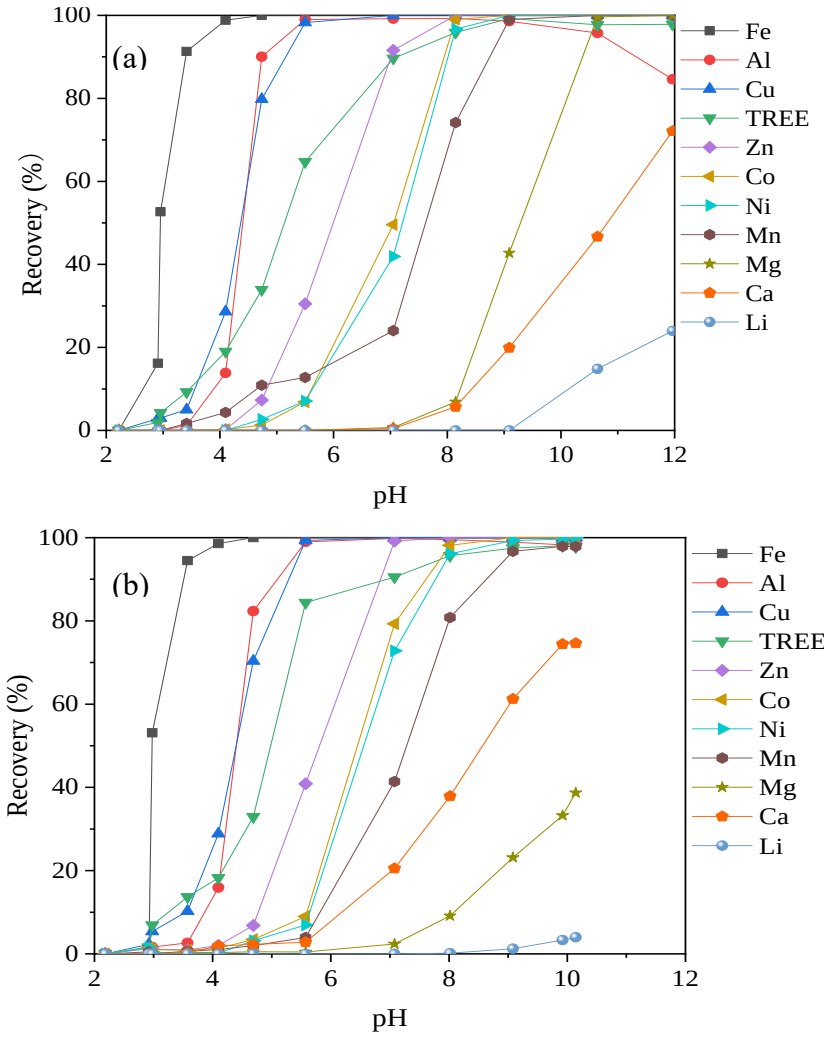
| Elements | TREEs | LREEs | HREEs | Sc | Y | La | Ce | Pr | Nd |
|---|---|---|---|---|---|---|---|---|---|
| Concentration | 5.90 | 2.58 | 3.32 | 0.12 | 1.35 | 0.14 | 0.95 | 0.37 | 0.60 |
| Elements | Sm | Eu | Gd | Tb | Dy | Ho | Er | Tm | Yb |
| Concentration | 0.41 | 0.098 | 0.44 | 0.067 | 0.93 | 0.12 | 0.14 | 0.033 | 0.12 |
| Elements | Lu | Co | Cu | Ni | Zn | Li | Mn | Fe | Al |
| Concentration | 0.029 | 1.79 | 1.91 | 5.45 | 11.84 | 1.95 | 85.15 | 1962 | 639 |
| Elements | Ca | Mg | | | | | | | |
| Concentration | 566 | 535 | | | | | | | |

Precipitation recovery of Fe, Al, Cu, TREEs, Zn, Co, Ni, Mn, Mg, Ca, and Li using 5 M NaOH, 3 M $Na_2CO_3$, and CaO as pH regulators is shown in Figure 3.8.4. As shown in Figure 3.8.4 (a), Fe was first precipitated with nearly 90% recovery at approximately pH 3.5. Thereby the majority of Fe can be removed from the leachate without losing a considerable portion of the REEs and other critical elements. Al and Cu started to precipitate after the majority of iron were removed, with 100% precipitation recovery occurred at pH 5.5. TREE and other critical elements (Co and Mn) as well as few valuable elements (Zn and Ni) were precipitated in the pH range of 4.0-9.0. However, in this pH range, certain amounts of Al, Mg, and Ca were also precipitated, which were considered as contaminant metal ions, and need to be removed to obtain high-grade final concentrates of Co and Mn.

Comparisons between Figure 3.8.4 (a) and (b) show that when $Na_2CO_3$ was used, similar precipitation patterns were observed as NaOH. However, Ca ions were more easily precipitated, with approximately 60% being recovered at pH 9.0, much higher than NaOH precipitation (20%). This can be explained by the formation of calcite due to the presence of carbonate anions in solution dissociated from sodium carbonate. Similar to NaOH and $Na_2CO_3$, the majority of iron was first precipitated from the leachate when CaO was used (Figure 3.8.4 (c)), which suggests the viability of using CaO as a cheap precipitant to remove iron from the solution. However, the precipitation curve of the other element shifted to the left, indicating that they were more likely precipitated. For example, 90% of Mn was precipitate at around pH 7, whereas less than 50% were recovered under the same pH when using NaOH and $Na_2CO_3$.

In order to compare the three precipitants more directly, the precipitation curves of Al and REEs were selected and put in the same figure. As shown in Figure 3.8.5, 100% recovery of Al occurred at a slightly lower pH when using CaO compared with the other two precipitants (pH 4.6 versus pH 5.3). However, nearly 70% of TREEs were co-precipitated at pH 4.6 with Al when using CaO. Thereby CaO is not a satisfactory precipitant for REE recovery. In addition, undissolved CaO particles were observed in the solution, which will report into REE-enriched precipitates collected in specific pH ranges, such as 4.5-8.0, causing problems for downstream purification. Titration curves using the three precipitants are shown in Figure 3.8.6. Two plateaus can be observed for all of the three curves, and the plateaus occurred at nearly the sample pH values, namely pH 3.0 and pH 4.3, which corresponds to the precipitation of Fe and Al, respectively (see Figure 3.8.4). Based on the curve of 3 M NaOH, approximately 0.04 mole NaOH is required to precipitate Fe, and another 0.04 mole is required to elevate the solution pH to 9.0, where all of the REEs and critical metals can be recovered. Therefore, at least half of the consumed NaOH can be replaced with CaO, which is cheaper and renders the process more economical viability.

As Figure 3.8.4 shows, it is difficult to separate REEs with other critical metals (Co and Mn) through staged precipitation. Therefore, in the next reporting period, a pre-concentrate containing REEs, Co, and Mn was prepared by staged precipitation. Then oxidants such as hydrogen peroxide were added to convert Co and Mn to higher valences ($Co^{3+}$ and $Mn^{4+}$), the oxides/hydroxides of which are relatively insoluble, thereby REEs can be separated from Co and Mn using selective re-dissolution. Lithium remaining in the residual solution of the sequential precipitation step was recovered using an ion-sieve synthesized in the laboratory. Co and Mn in the undissolved solid were separated using pH/Eh-controlled dissolution. A description of the process is shown in Figure 3.8.7.



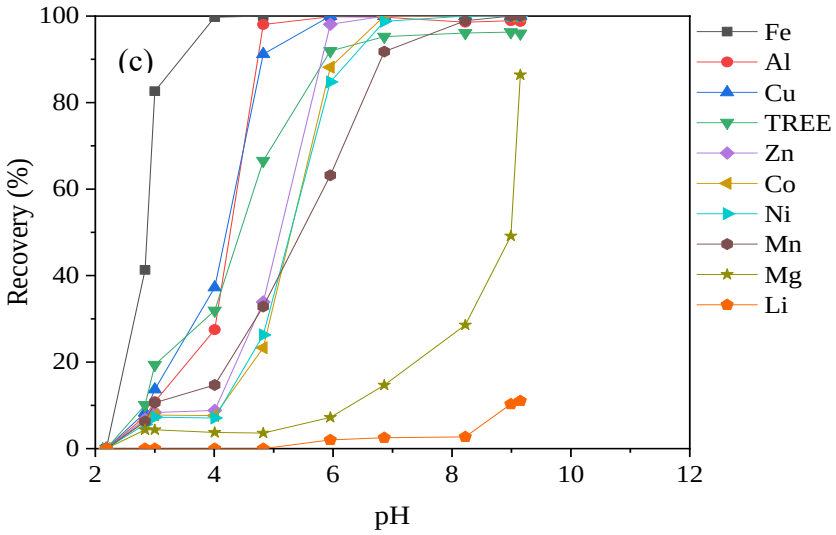



Figure 3.8.4. Precipitation recovery of rare earth, other critical, and major elements from the leachate using different precipitants: (a) 5 M NaOH, (b) 3 M $Na_2CO_3$, and (c) CaO powder

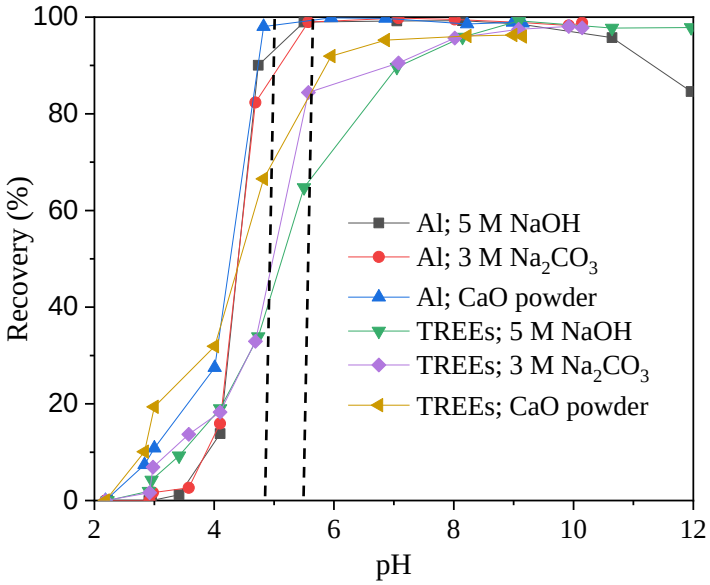



Figure 3.8.5. Precipitation recovery of Al and TREEs using the three different precipitants (dotted lines indicate where 100% recovery of Al occurs).

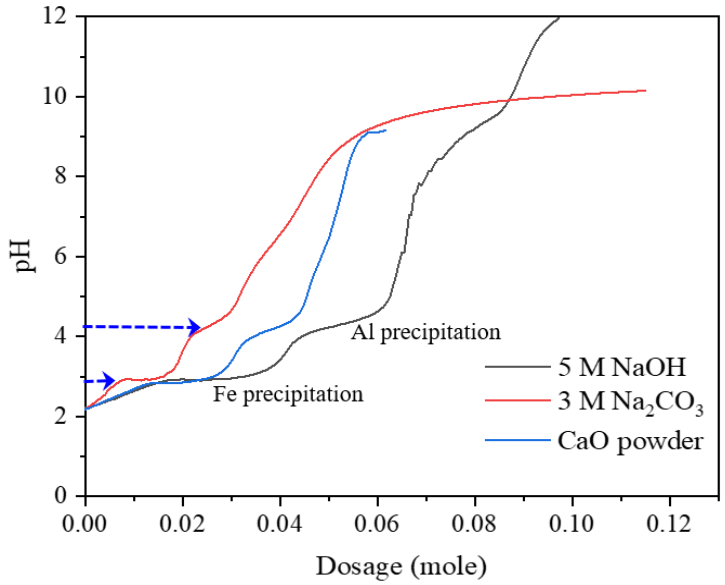



Figure 3.8.6. Titration curves using the three precipitants.

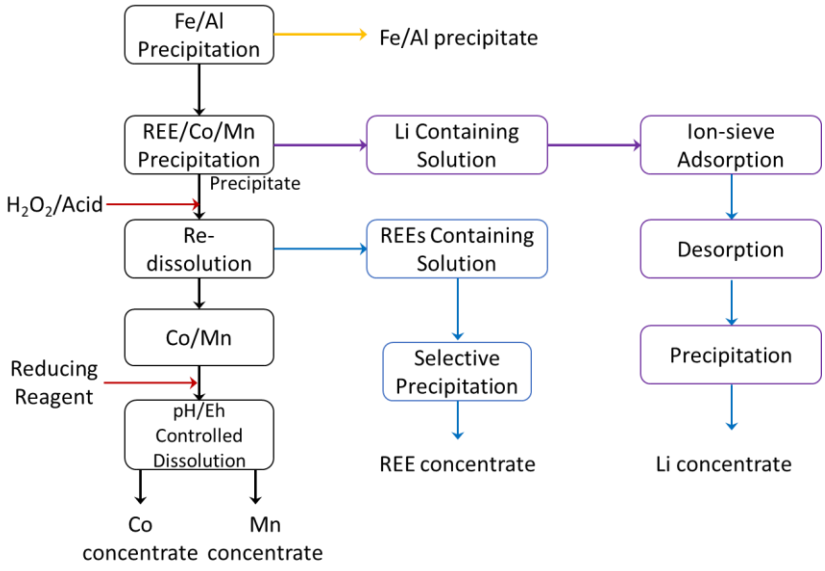



Figure 3.8.7. An updated flowsheet for REEs, Co, Mn, and Li recovery.

**Acid Mine Leachate II:** A separate acid mine leachate sample was collected from a source that originated from the oxidation of West Kentucky No. 13 coarse refuse. The total volume of the sample delivered to the mineral processing laboratory of VT was around 100 gallons. Given the sufficient volume, we used this sample as feedstock for systematic REEs, Li, Co, and Mn recovery studies. In the process flowsheet that was tested in the pilot plant, contaminant metal ions, such as Fe and Al, are first removed from the solution through staged precipitation, resulting in a pre-concentrate of REEs and other critical metals. Therefore, we repeated staged precipitation tests on this sample using different precipitants.

The concentration of the major and trace elements in the sample was measured using ICP-MS at Virginia Tech. As shown in Table 3.8.2, the leachate contained 6.6 ppm of TREEs, of which critical REEs, including Nd, Eu, Tb, Dy, and Y, account for 16.6%, 1.0%, 0.7%, 4.2%, and 20.5%, respectively. Therefore, the leachate can be used as a good resource of REEs, especially critical REEs. In addition, the leachate also contained 1.65 ppm Co, 1.39 ppm Li, and 58 ppm Mn, which have been coded as critical materials by the Department of Interior. Major contaminant metal ions occurring in the sample were Fe (884 ppm), Al (350 ppm), Ca (417 ppm), and Mg (302 ppm).

Staged precipitation tests were performed to remove the contaminant metal ions and preconcentrate the REEs and other critical metals. Sodium hydroxide (NaOH) and sodium carbonate ($Na_2CO_3$) were used as precipitants for comparison purposes. Furthermore, hydrogen peroxide ($H_2O_2$) was also added into the leachate at the beginning of selected tests, which served as an oxidizing agent to ensure all the Fe occurred as ferric ions. The staged precipitation test procedures have been presented in Acid Mine Leachate I; thus, they are not repeated in this section.

Table 3.8.2. Elemental concentration (ppm) of the acid mine leachate sample.

| Element | Sc | Y | La | Ce | Pr | Nd | Sm | Eu | Gd | Tb | Dy | Ho |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Concentration | 0.13 | 1.35 | 0.79 | 1.67 | 0.25 | 1.09 | 0.29 | 0.07 | 0.36 | 0.05 | 0.27 | 0.05 |
| % of TREE | 2.0 | 20.5 | 12.0 | 25.4 | 3.70 | 16.6 | 4.5 | 1.0 | 5.4 | 0.7 | 4.2 | 0.7 |
| Element | Er | Tm | Yb | Lu | LREE | | HREE | | TREE | | HREE/LREE | |
| Concentration | 0.12 | 0.01 | 0.08 | 0.01 | 4.23 | | 2.36 | | 6.60 | | 0.56 | |
| % of TREE | 1.8 | 0.2 | 1.2 | 0.2 | 64.2 | | 35.8 | | 100 | | – | |
| Element | Co | Cu | Ni | Zn | Li | Mn | Fe | Al | Ca | Mg | | |
| Concentration | 1.65 | 2.42 | 4.73 | 7.63 | 1.39 | 58 | 884 | 350 | 417 | 302 | | |

The precipitation recovery of TREEs and other elements was first analyzed, which will inform the viability of using staged precipitation to preconcentrate REEs and other critical metals. As Figure 3.8.8(a) shows, when 5 M NaOH was used as the precipitant without adding $H_2O_2$, TREEs started to precipitate at pH around 4.0, and complete precipitation was achieved when the pH was elevated to around 7.0. Furthermore, 100% of Al was precipitated at pH 5.1, under which condition, only 25% of TREEs was precipitated. Therefore, Al can be effectively separated from TREEs by staged precipitation without a considerable loss of the TREEs. The precipitation behavior of Fe is different from Al and REEs. As shown, when pH was increased to 4.0, nearly 23% of Fe was precipitated, whereas, as pH increased from 4.0 to 6.0, increases in the recovery were slower than pH < 4.0. Moreover, when pH further increased to above 6.0, the Fe was rapidly precipitated. Therefore, it can be concluded that ferrous and ferric ions simultaneously occurred in the leachate. Al lower pH, ferric ions were preferentially precipitated, while ferrous ions tended to precipitate at higher pH. Given the fact that Fe is the most abundant metal ion in the leachate, staged precipitation with 5 M NaOH alone cannot provide satisfactory removal. For comparison purposes, 3 M $Na_2CO_3$ was also used as a precipitant. As shown in Figure 3.8.8 (b), the precipitation characteristics of the elements, except Ca and Mg, are similar to those observed with 5 M NaOH. When using $Na_2CO_3$, nearly 100% of Ca was precipitated at pH 10, which is attributable to the formation of calcite. In order to convert the Fe from ferrous to ferric forms, $H_2O_2$ was added at the beginning of the staged precipitation tests. As shown in Figure 3.8.8 (c) and (d), nearly 100% of the Fe was precipitated at pH 4.0, regardless of whether $H_2O_2$ dosage was 0.5 vol.% or 1 vol.%. Therefore, in order to achieve

a higher-grade preconcentrate of REEs, the ferrous ions need to be oxidized to ferric ions prior to staged precipitation.

The recovery values of the individual elements obtained using the different reagent schemes were plotted in the same figure to provide more intuitive comparisons. As shown in Figure 3.8.9, the recovery curve of NaOH plus 0.5 vol.% $H_2O_2$ overlaps the curve of NaOH plus 1 vol.% $H_2O_2$, indicating that $H_2O_2$ dosage of 0.5 vol.% is sufficient to oxidize the ferrous ions in the acid mine leachate completely. Replacing NaOH with $Na_2CO_3$ did not cause considerable differences in the recovery of Al and Fe. However, at pH greater than 8.0, calcium recovery using $Na_2CO_3$ was much higher than NaOH, which introduced more calcium into the preconcentrate of TREEs, Co, and Mn obtained in the pH range of 5.0 to 9.0. Based on the titration curves shown in Figure 3.8.10, the molar consumption of $Na_2CO_3$ is less than NaOH, which can be explained by the two-step hydrolysis of $CO_3^{2-}$ of the $Na_2CO_3$. After reaching pH 9.0, adding $Na_2CO_3$ did not noticeably change the pH, primarily due to the formation of calcite, which consumed the $CO_3^{2-}$.

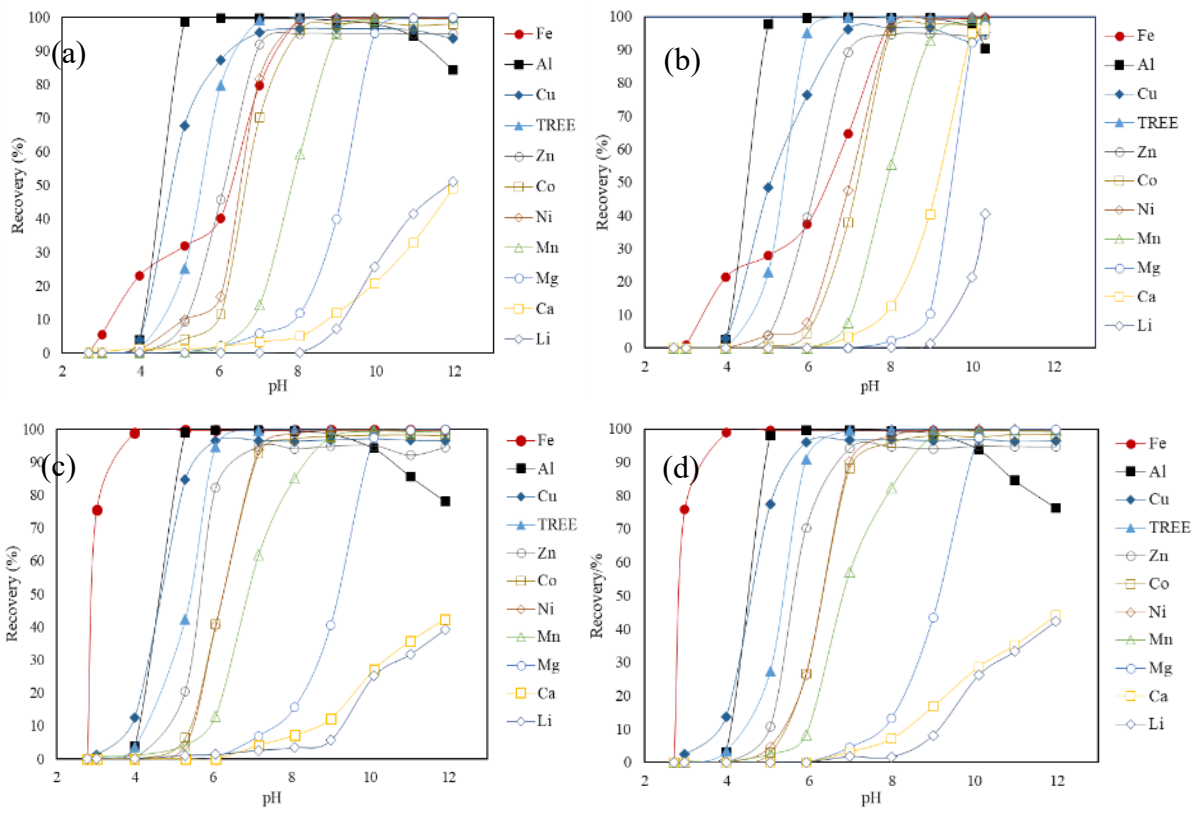



Figure 3.8.8. Staged precipitation recovery of TREEs and other elements using different reagents: (a) 5 M NaOH; (b) 3 M $Na_2CO_3$; (c) 5 M NaOH plus 0.5 vol.% $H_2O_2$; and (d) 3 M NaOH plus 1 vol.% $H_2O_2$.

Based on the titration curve of NaOH, it can be observed that pH gradually elevated with the increases in the amount of precipitant added into the solution (Figure 3.8.10). Slower pH increases occurred in four pH ranges: 3.1-3.3, 4.0-4.5, 6.0-7.8, and 8.0-9.4, which correspond to the precipitation of ferric, aluminum, ferrous, and magnesium ions, respectively, per the results shown in Figure 3.8.8(a). When hydrogen peroxide was added into the leachate, the third range

disappeared. This finding corroborates the conclusion that the ferrous ions present in the leachate were completely transformed into ferric ions by adding 0.5 vol.% hydrogen peroxide.

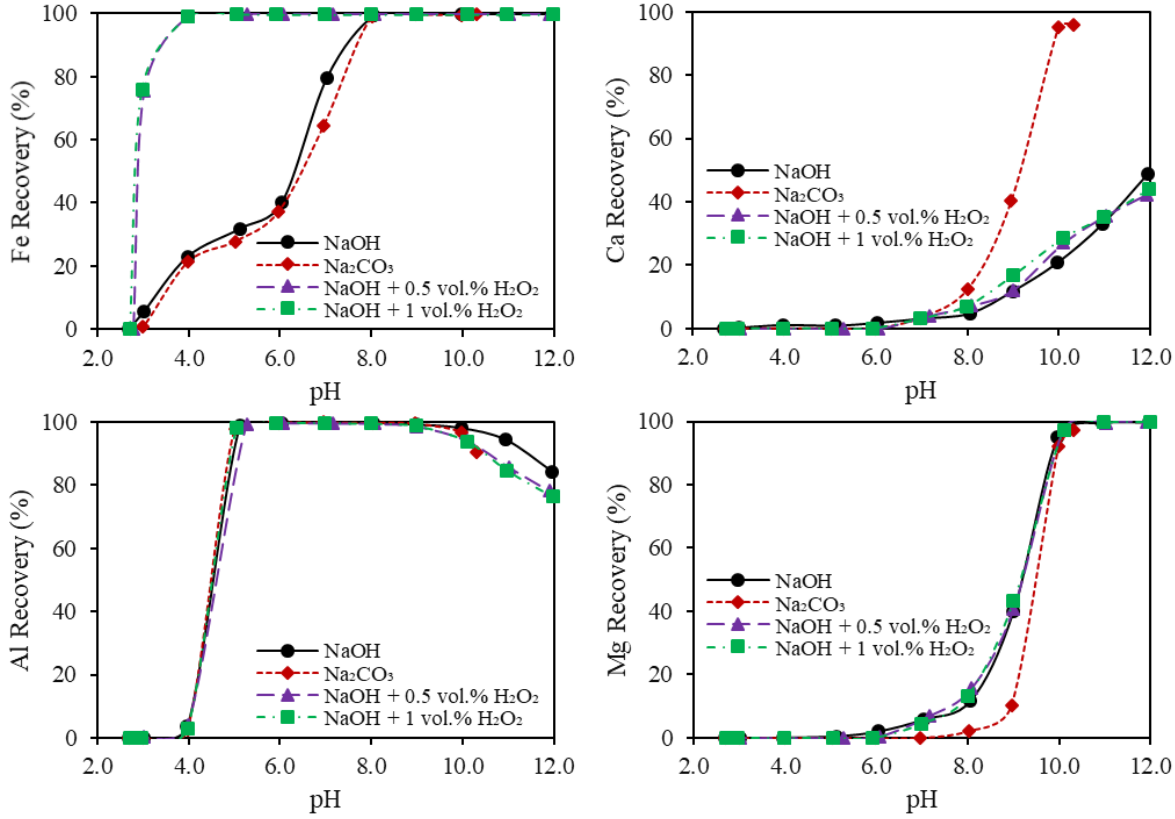



Figure 3.8.9. Staged precipitation recovery of Fe, Al, Ca, and Mg.

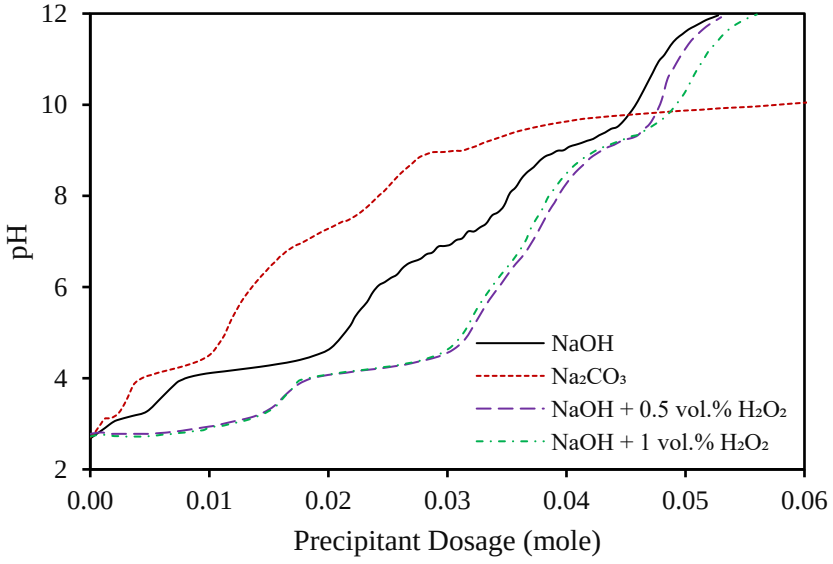


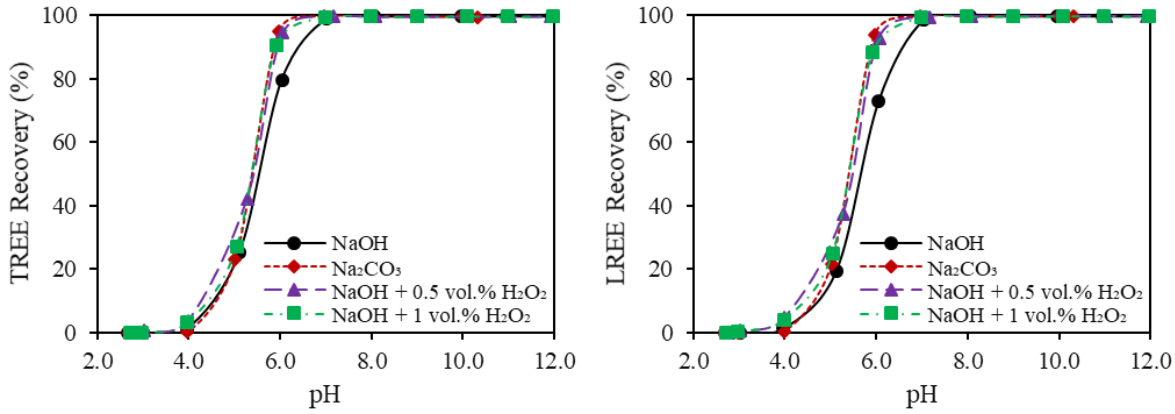

Figure 3.8.10. Titration curves of the acid mine leachate using different reagents.

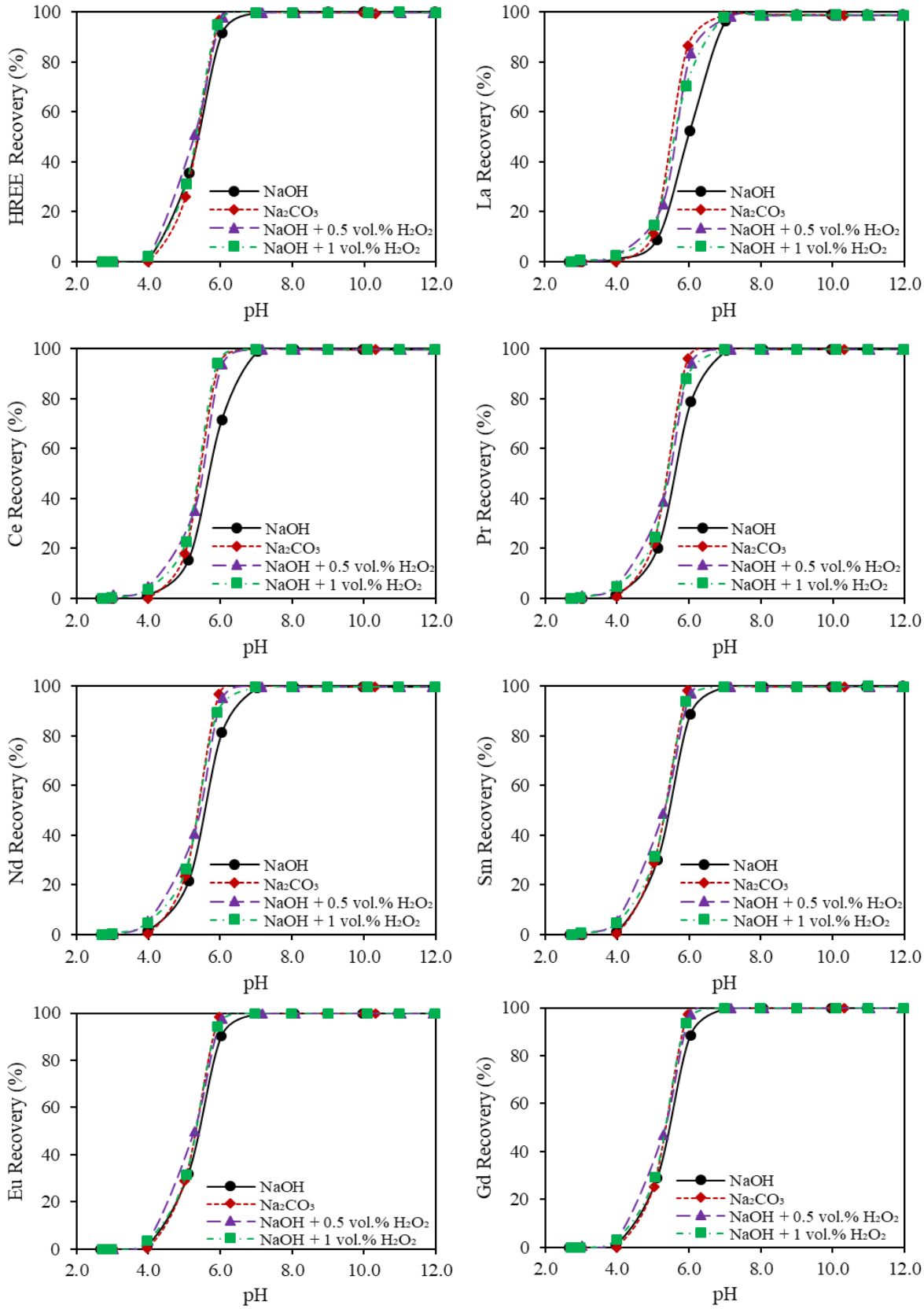
The impact of the different reagent schemes on the precipitation recovery of REEs is shown in Figure 3.8.11. TREE recoveries obtained using $Na_2CO_3$ and NaOH plus $H_2O_2$ are higher than using NaOH. As shown, the former two reagent schemes provided nearly 95% recovery of the TREEs at pH 6.0; whereas, only 80% of the TREEs were precipitated at the same pH when using NaOH. Comparisons between the precipitation behavior of LREEs and HREEs show that the same phenomenon occurred for the LREEs. However, no noticeable differences in recovery values of the HREEs were observed. Therefore, lower recoveries of the TREEs are due to the LREEs.

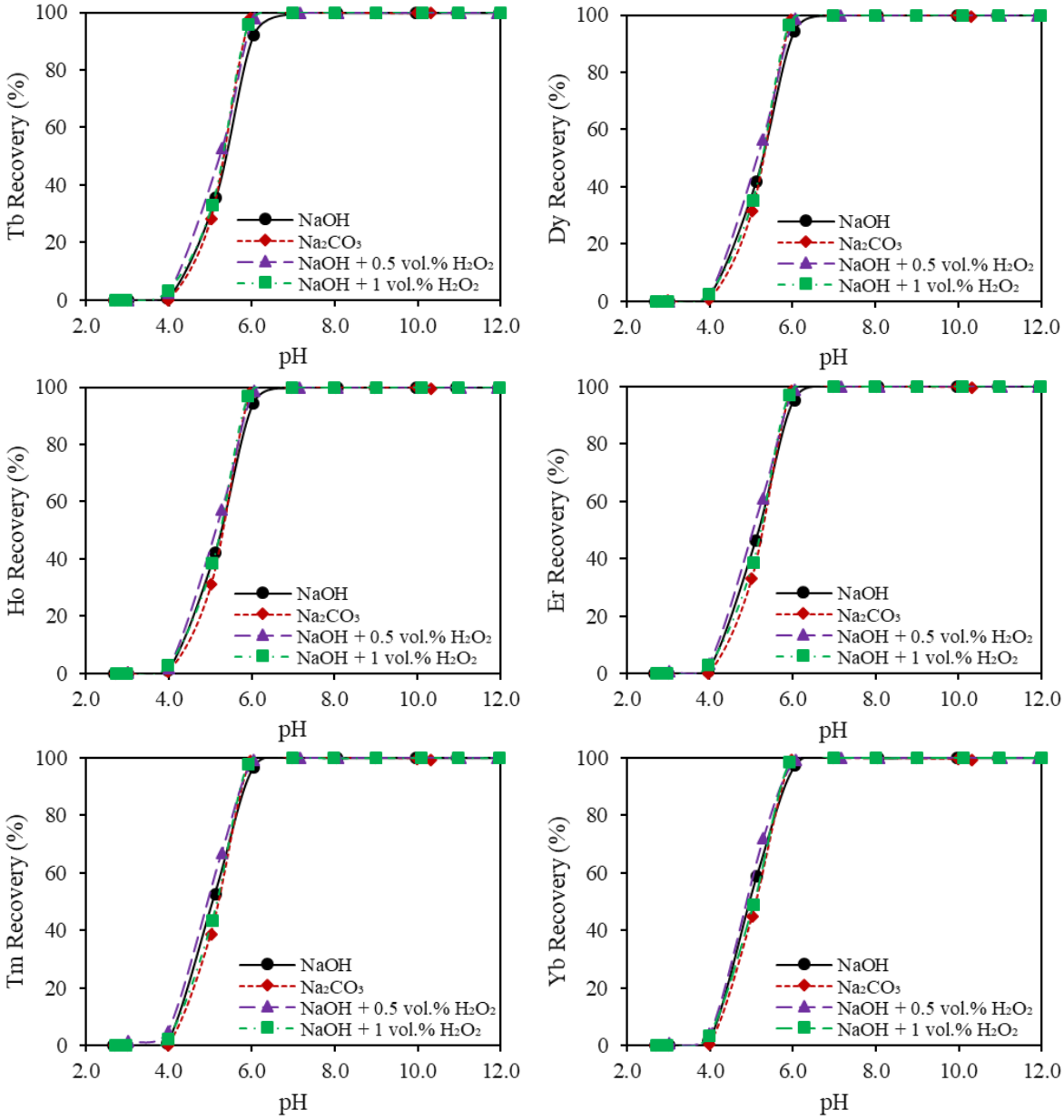
Two mechanisms can potentially explain the lower recoveries of the LREEs: (1) the most abundant LREE, Ce, was oxidized from 3+ to 4+ valence by hydrogen peroxide, which is insoluble even under acidic conditions; (2) more ferric precipitate was formed at pH < 4.0 in advance of the REE precipitation due to oxidation, thereby the LREE recovery was increased due to adsorption. However, the recovery of the individual LREEs showed that not only Ce but also other LREEs had higher recoveries when hydrogen peroxide was employed. Therefore, rather than the oxidation of Ce, the higher recoveries are more likely due to the adsorption of the LREEs on the surface of the ferric precipitate.

The results shown in Figure 3.8.9 indicated that the precipitation behavior of Fe was not changed when using $Na_2CO_3$ as the precipitant relative to NaOH. However, using $Na_2CO_3$ increased the recovery of LREEs (see Figure 3.8.11). Therefore, the increases in recovery were not due to adsorption on ferric precipitate. Furthermore, the hypothesis of Ce oxidation is also untenable since $Na_2CO_3$ does not have the oxidizing capability. Given these facts, the higher recovery of LREEs is likely due to the precipitation of LREEs in the form of rare earth carbonates.

The recovery of individual REEs using 5 M NaOH plus 0.5 vol.% $H_2O_2$ is shown in Figure 3.8.12. At the same pH, recovery of LREEs was lower than HREEs. Despite being included as LREEs, Sc was the most preferentially precipitated REEs. As shown, more than 95% of Sc was precipitated at around pH 5, however, only 23% of La was precipitated at this pH.




HREE Recovery (%)
La Recovery (%)
Ce Recovery (%)
Pr Recovery (%)
Nd Recovery (%)
Sm Recovery (%)
Eu Recovery (%)
Gd Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2


Tb Recovery (%)
Dy Recovery (%)
Ho Recovery (%)
Er Recovery (%)
Tm Recovery (%)
Yb Recovery (%)
pH
NaOH
Na2CO3
NaOH + 0.5 vol.% H2O2
NaOH + 1 vol.% H2O2

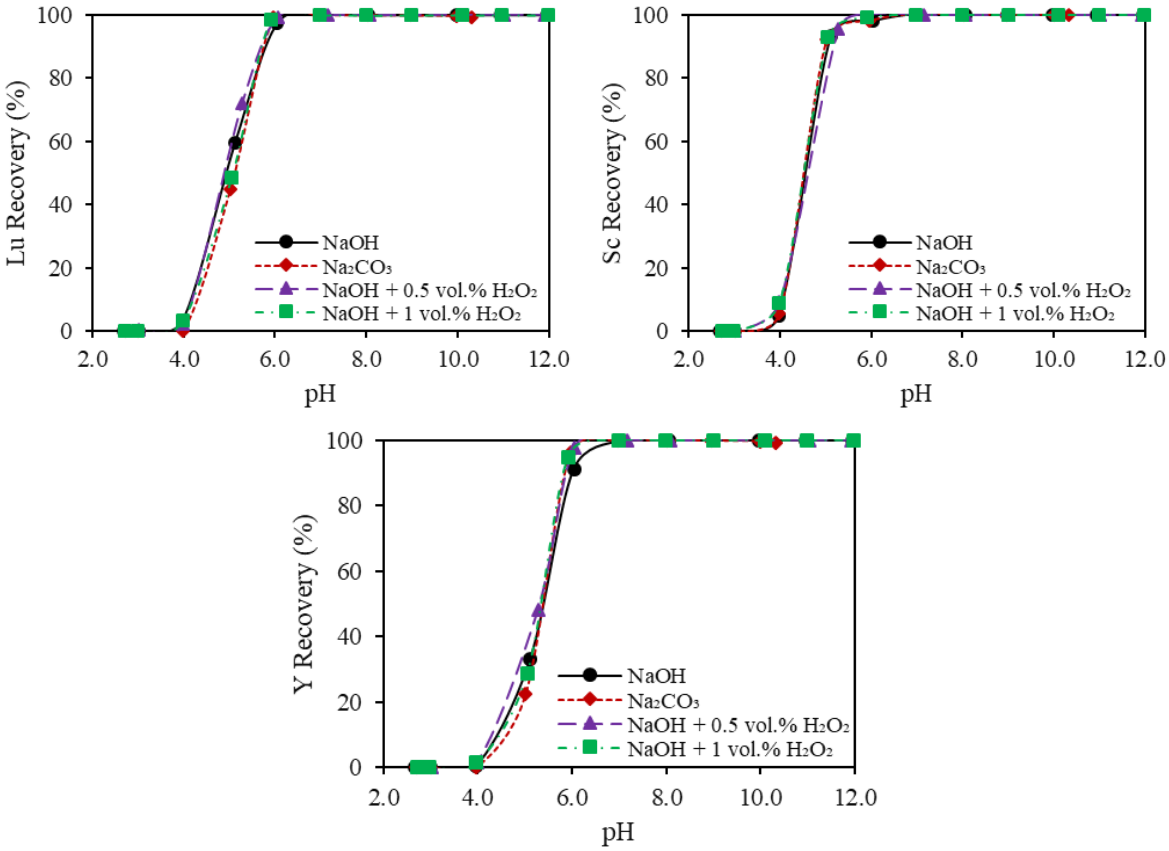



Figure 3.8.11. REE recovery as a function of pH using different reagents.

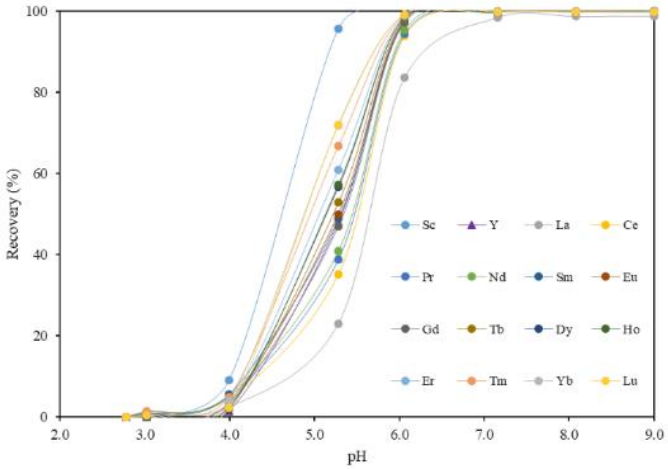



Figure 3.8.12. Recovery of individual REEs as a function of pH using 5 M NaOH plus 0.5 vol.% $H_2O_2$.

Co and Mn can be recovered together with REEs from acid mine leachate. As the staged precipitation results show, nearly all Fe and a portion of Al can be eliminated from acid mine leachate by elevating pH to around 4.0. Further increases in the pH to 9.0 precipitate nearly all REEs, Co, and Mn resulting in a preconcentrate, which needs to be processed to separate and

purify the three components. A selective dissolution approach was used in this reporting period to achieve the purpose. As shown in Figure 3.8.13, both Mn and Co occur as high valence states in a wide pH range under oxidizing environments. Hydroxides and oxides of high valence Mn and Co are difficult-to-dissolve in water. Therefore, in the presence of oxidizing agents, Mn and Co stay as precipitates, while REEs are dissolved into solution under acidic conditions. In this case, Mn and Co can be separated from REEs. Selective dissolution tests were performed on a preconcentrate prepared from an acid mine leachate (see Figure 3.8.14).

For each test, 50 mL of the preconcentrate occurring as the slurry form was used. Re-dissolution performance was evaluated using the amount (in mg) of metals that were dissolved into the solution. As shown in Figure 3.8.15, without adding oxidizing agents, the amount of Mn was gradually dissolved with the decrease in solution pH. The dissolution was increased when 1 vol.% $H_2O_2$ was added. The effect can be more intuitively observed in Figure 3.8.16. Therefore, rather than the oxidizing agent, hydrogen peroxide in the slurry served as a reducing agent, which transformed the high-valence Mn originally occurred in the slurry into low-valence species. Unlike hydrogen peroxide, $Na_2S_2O_8$ decreased the amount of Mn dissolved into the solution. The decrease indicates that the transformation of Mn from low-valence to high-valence occurred when using $Na_2S_2O_8$. However, the amount of dissolved Mn was only reduced from around 9.5 mg to 7.5 mg; thus, the oxidization is incomplete. As shown in Figure 3.8.17, adding oxidizing agents did not considerably reduce the dissolution of Co. Therefore, rather than separating Mn and Co from REEs, it is more viable to separate Mn from Co and REEs through selective re-dissolution. After being dissolved into solution, REEs can be separated from Co through oxalic acid precipitation and/or solvent extraction.

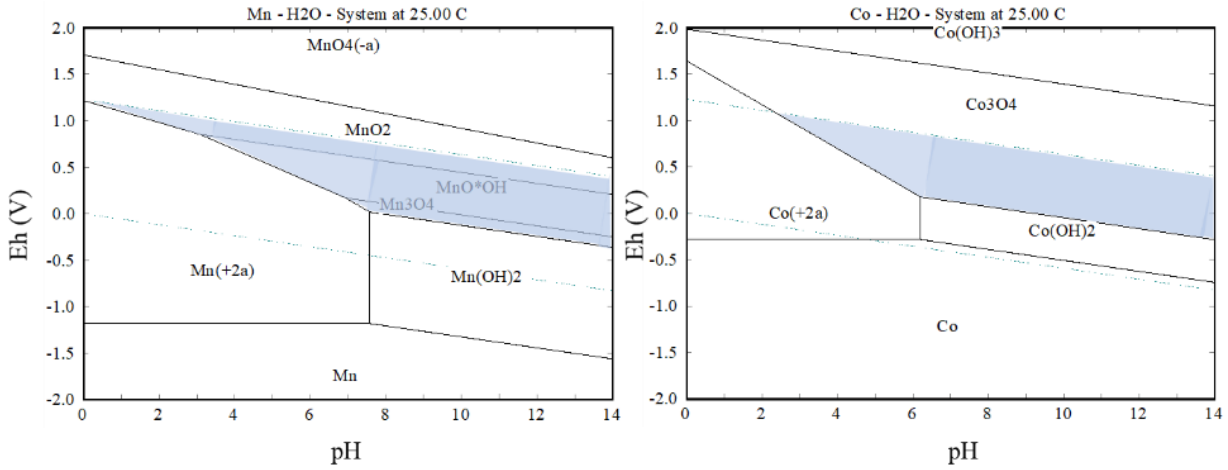



Figure 3.8.13. Eh-pH diagrams of 1 ppm Mn (left) and 1 ppm Co (right) in aqueous system at 25℃.

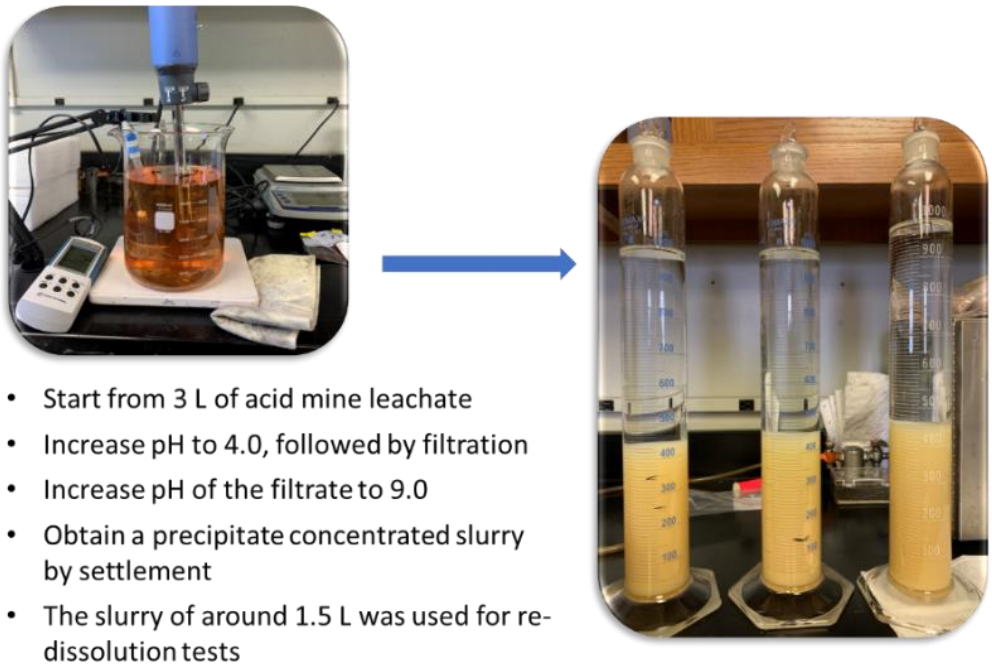



Figure 3.8.14. Preconcentrate preparation from an acid mine leachate.

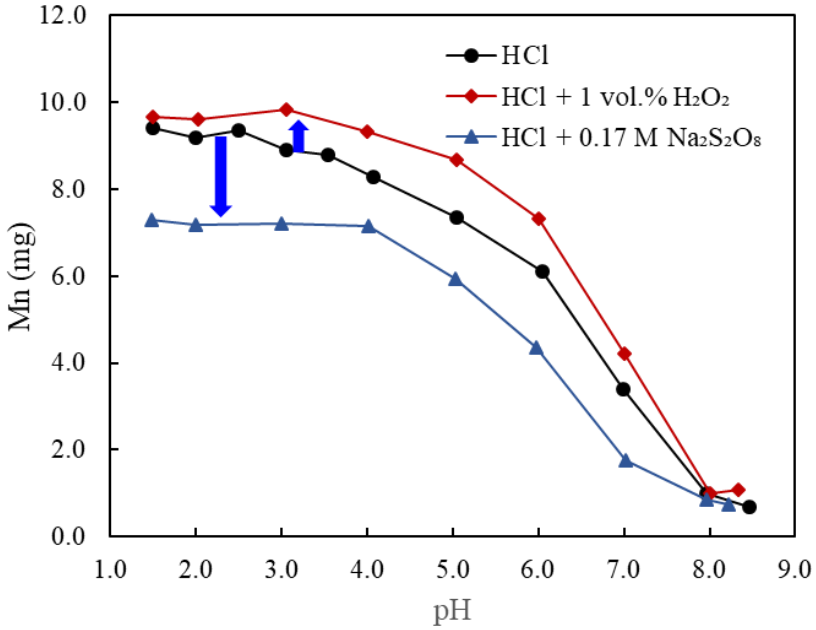



Figure 3.8.15. Re-dissolution of Mn as a function of solution pH.

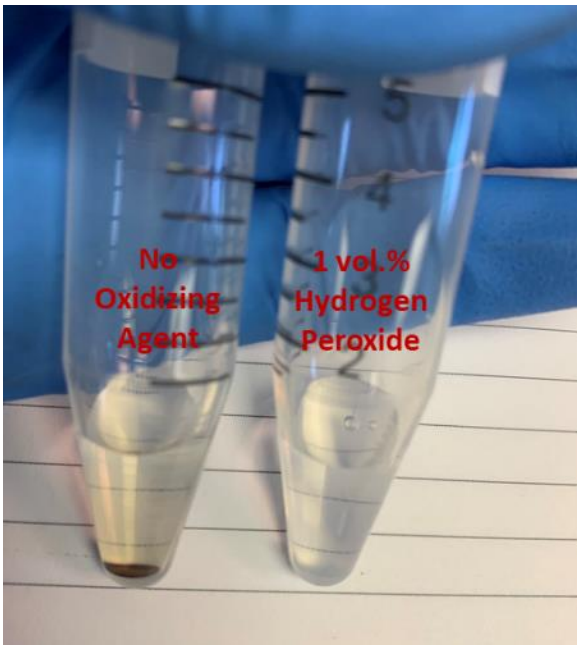



Figure 3.8.16. Appearance of the slurry at pH 1.5 without and with the addition of hydrogen peroxide.

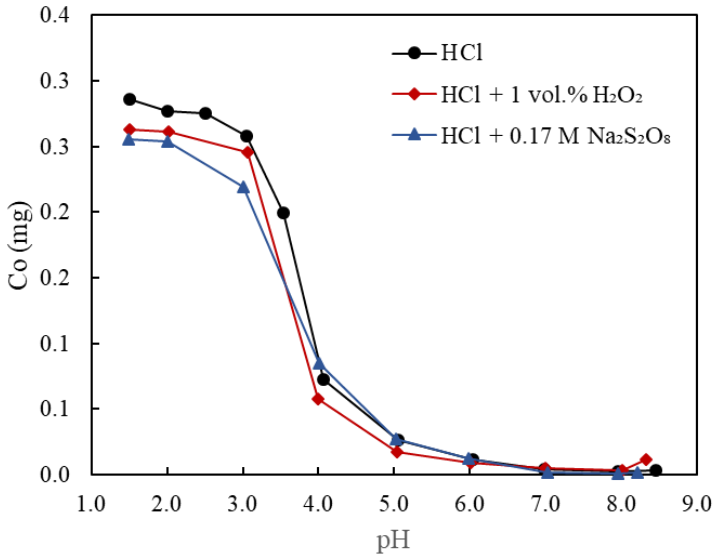



Figure 3.8.17. Re-dissolution of Co as a function of solution pH.

A preconcentrated slurry of 1.5 L was obtained from 3 L of acid mine leachate by collecting precipitates formed in the pH range of 4.0 – 9.0. Re-dissolution tests were performed on the preconcentrated slurry using $H_2O_2$ and $Na_2S_2O_8$ as oxidizing agents. The re-dissolution recovery of Mn and Co as a function of pH was shown in Figure 3.8.15 and Figure 3.8.17, respectively. Elemental analysis of the undissolved precipitates obtained from the re-dissolution tests was conducted. The elemental compositions of the undissolved solids are shown in Figure 3.8.18. Regardless of which oxidizing agent was used, the precipitates contained around 10% of Mn and

0.1% of Co. The Mn content has exceeded the targeted grade of 2 %, while Co needs to be further upgraded to reach the target. The precipitates also contained considerable amounts of contaminants, primarily Al and Fe. The presence of contaminants is due to the fact that during the initial staged precipitation process, the preconcentrated slurry was collected in the pH range of 4.0 – 9.0, under which condition Fe and Al cannot be completely precipitated. Therefore, optimum staged precipitation pH ranges were determined in order to maximize the recovery of Mn and Co while minimizing the recovery of contaminant metal ions.

To achieve a maximum recovery of Mn and Co as well as a minimum recovery of contaminants, optimization of the staged precipitation process was performed on the acid mine leachate. Figure 3.8.19 shows the experimental procedures used for the optimization. Specifically, for each test, the pH of the acid mine leachate was incrementally elevated by adding 5 M NaOH. After reaching an appropriate pH whereby Fe was precipitated, the slurry was filtered using 2.5 μm pore size filter paper. The filtrate was subjected to further pH increases until Al was precipitated, and similarly, the slurry was filtered. After the first two steps of precipitation-filtration, the majority of Fe and Al were expected to be removed from the solution. The pH of the filtrate resulting from the second step was further increased to an appropriate value to precipitate Mn and Co. Similarly, the resultant slurry was filtered, and the filter cake was dried and digested for elemental analysis. Three tests referred to as Test 1, Test 2, and Test 3 were performed using different pH cuts: Test (pH 4.0, 6.0, and 9.0), Test 2 (pH 4.0, 6.5, and 9.5), and Test 3 (pH 4.0, 7.0, and 10.0).

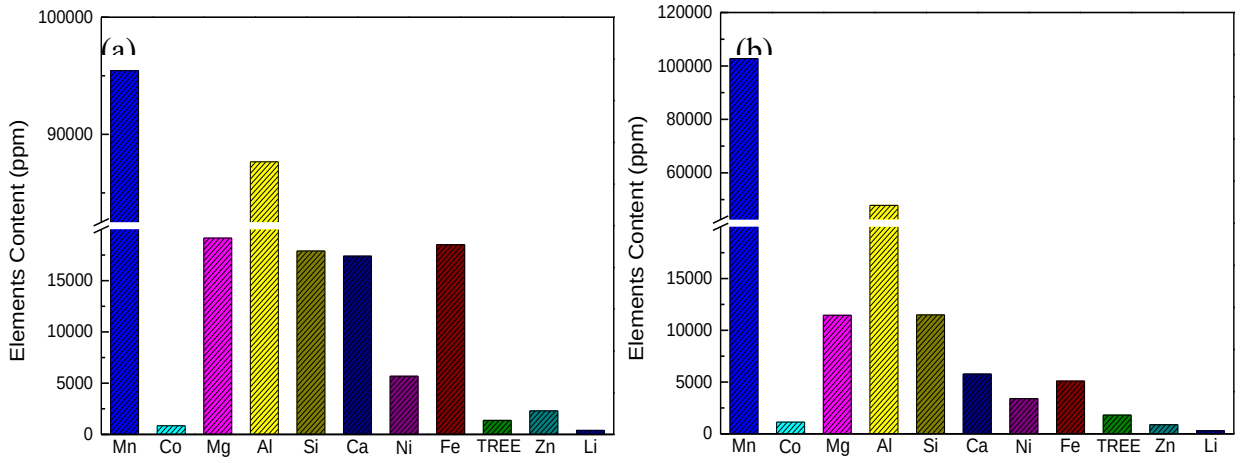



Figure 3.8.18. Elemental compositions of the undissolved solids obtained using different oxidizing agents: (a) 1 vol.% $H_2O_2$, and (2) 1 M $Na_2S_2O_8$.

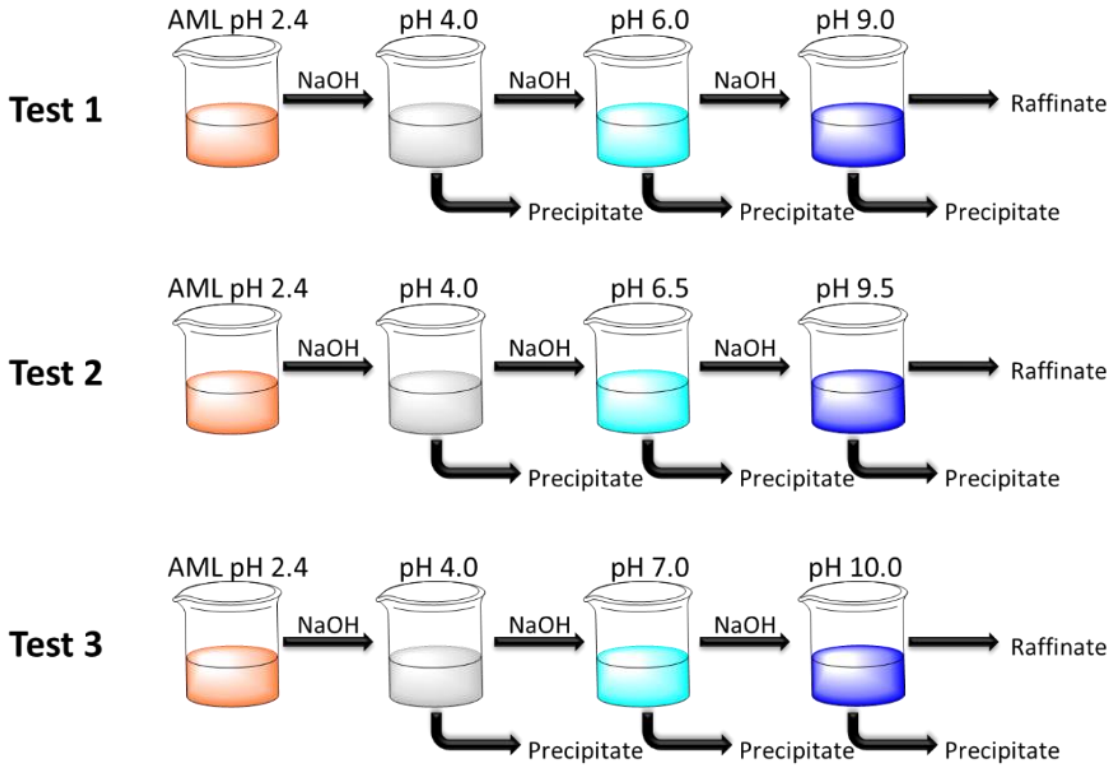


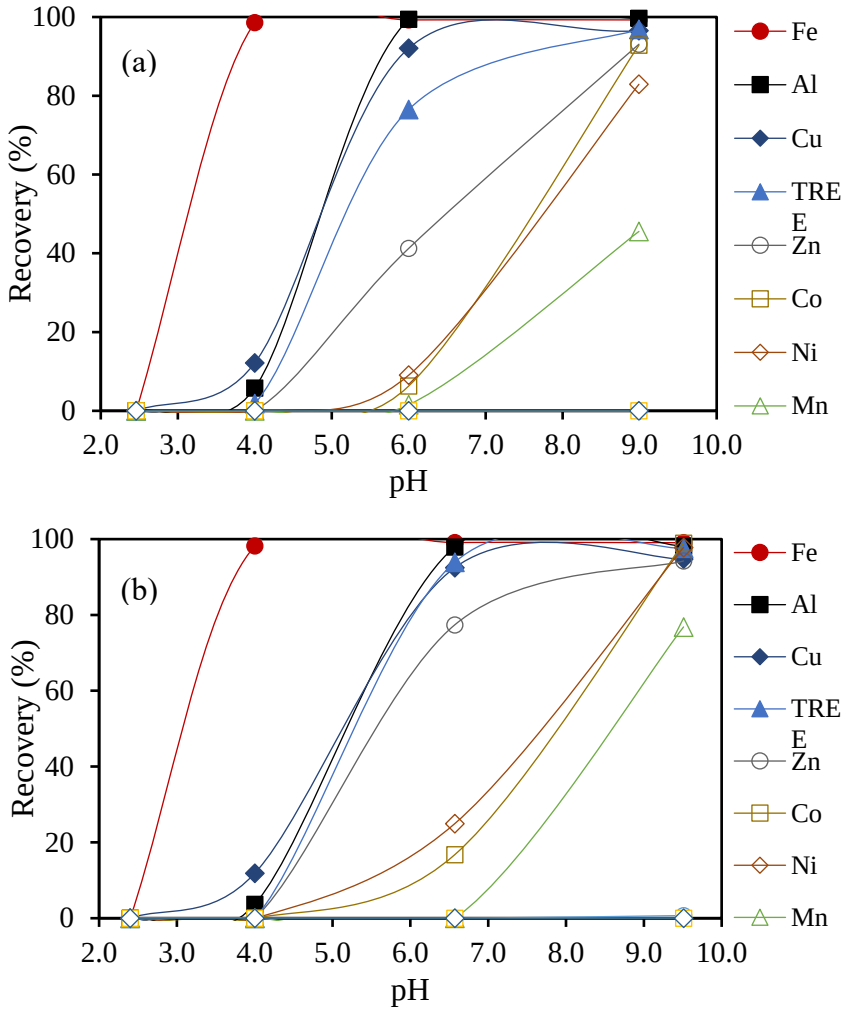

Figure 3.8.19. Schematic showing the experimental scenario used to optimize the staged precipitation tests.

Elemental recoveries of Mn, Co, REEs, Al, Fe, and other metals as a function of pH obtained from the three tests were shown in Figure 3.8.20. For all the three tests, nearly 100% of Fe was precipitated at pH 4.0 with negligible losses of critical materials, such as REEs, Co, and Mn. Elevating the pH to 6.0 led to the precipitation of nearly 100% of Al and 80% of TREEs. As shown in Figure 3.8.20(b), when elevating the pH to 6.5, nearly a complete precipitation of the TREEs was achieved. At pH 6.5, the majority of Mn still stayed in the solution, while around 20% of Co was simultaneously precipitated together with REEEs. Using a pH cut of 7.0, the separation of REEs from Co and Mn was deteriorated since around 25% of Mn and 80% of Co were precipitated with REEs. Therefore, the optimum pH range for precipitating Al and REEs was selected as 6.5. Comparisons among the recoveries at pH 9.0, 9.5, and 10.0 suggested that a pH of 10.0 was required to achieve 100% recovery of Mn. Overall, based on the recoveries obtained from the three tests, it can be concluded that pH 6.5-10.0 is the optimum range where the recovery of Mn and Co was maximized, while the recovery of Fe, Al, and TREEs was minimized. It is worth noting that a portion of Co was introduced into the Al/TREE precipitate; thus, Co purification needs to be considered for two streams in subsequent upgrading processes: REEs concentrated stream and Mn concentrated stream.

Elemental compositions of the precipitates obtained in the pH range of 6.0-9.0 from Test 1, 6.5-9.5 from Test 2, and 7.0-10.0 from Test 3 are presented in Figure 3.8.21. Mn contents of as high as 22.1% and 22.5% were obtained from Test 2 and Test 3, respectively. Test 2 provided a higher Co content of 0.58% compared with Test 3 (0.16%), which is likely due to the increase in the

precipitation recovery of Mg. Compared with Test 2 and Test 3, the precipitate generated from Test 3 contained the least amount of Mn, primarily due to the lowest recovery of Mn at pH 9.0 (around 50%, see Figure 3.8.20(a)).

A Co/Mn preconcentrated slurry was produced from the acid mine leachate using the optimum pH range of 6.5-10.0. A schematic presenting the procedures for preparing the slurry is shown in Figure 3.8.22. The slurry was redissolved into solution by decreasing pH to 1.50 in the absence of an oxidizing agent. The resultant suspension was filtered using a 2.5 µm pore size filter paper. The filtrate was collected and used as a feedstock for Co and Mn separation tests. The filtrate contained around 700 ppm of Mn and 20 ppm of Co. The primary contaminants in the solution included around 1,500 ppm of Mg and 110 ppm of Si.



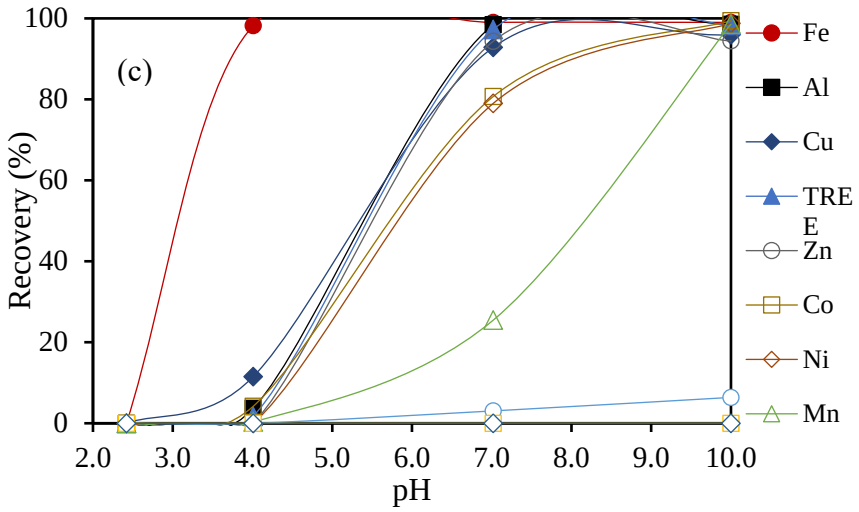
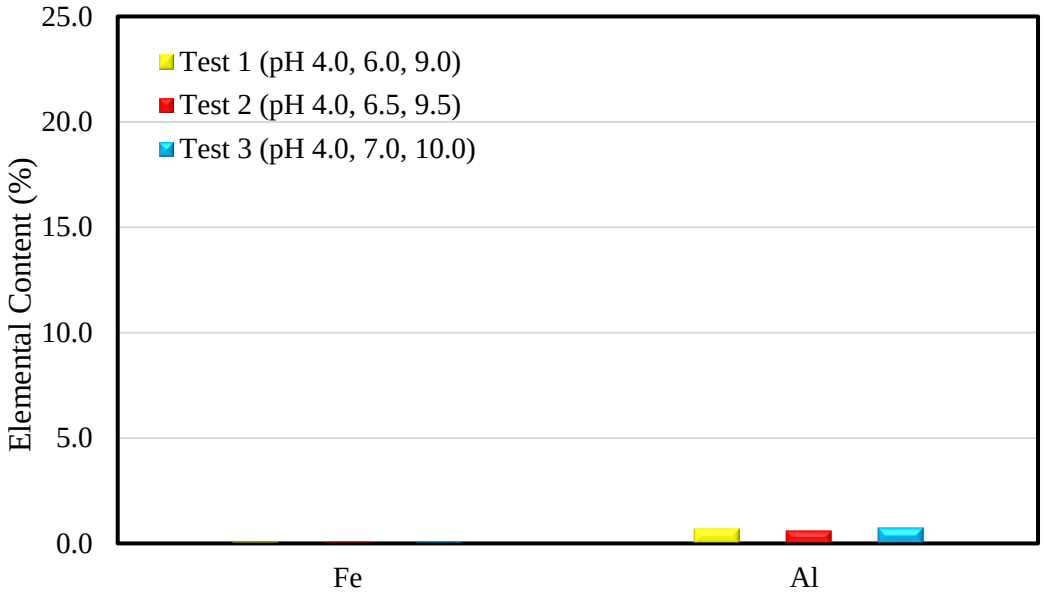



Figure 3.8.20. Elemental recoveries as a function of pH obtained from (a) Test 1, (b) Test 2, and (c) Test 3.



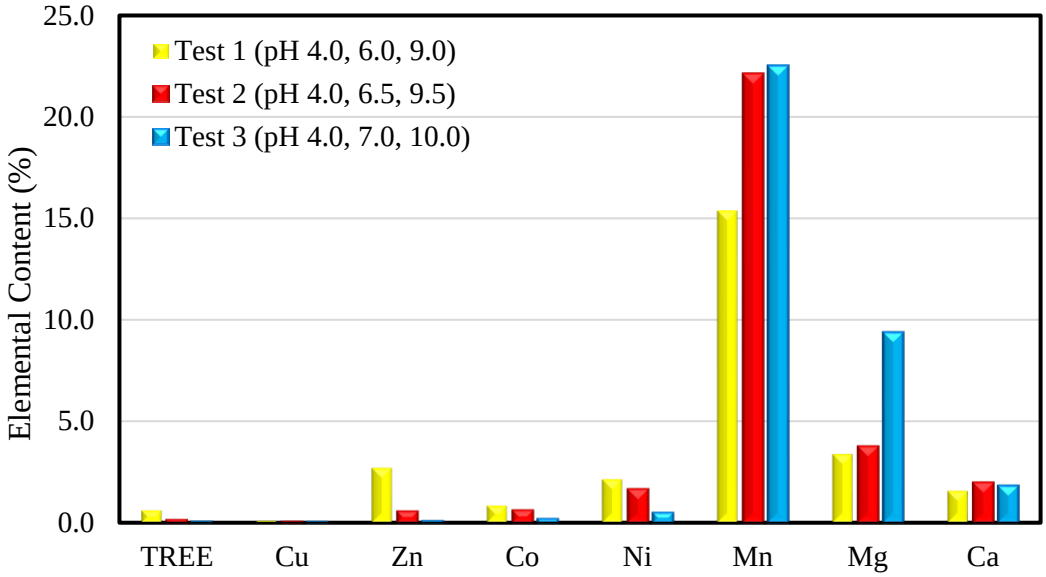



Figure 3.8.21. Elemental compositions of the precipitates obtained in the pH range of 6.0-9.0 from Test 1, 6.5-9.5 from Test 2, and 7.0-10.0 from Test 3.

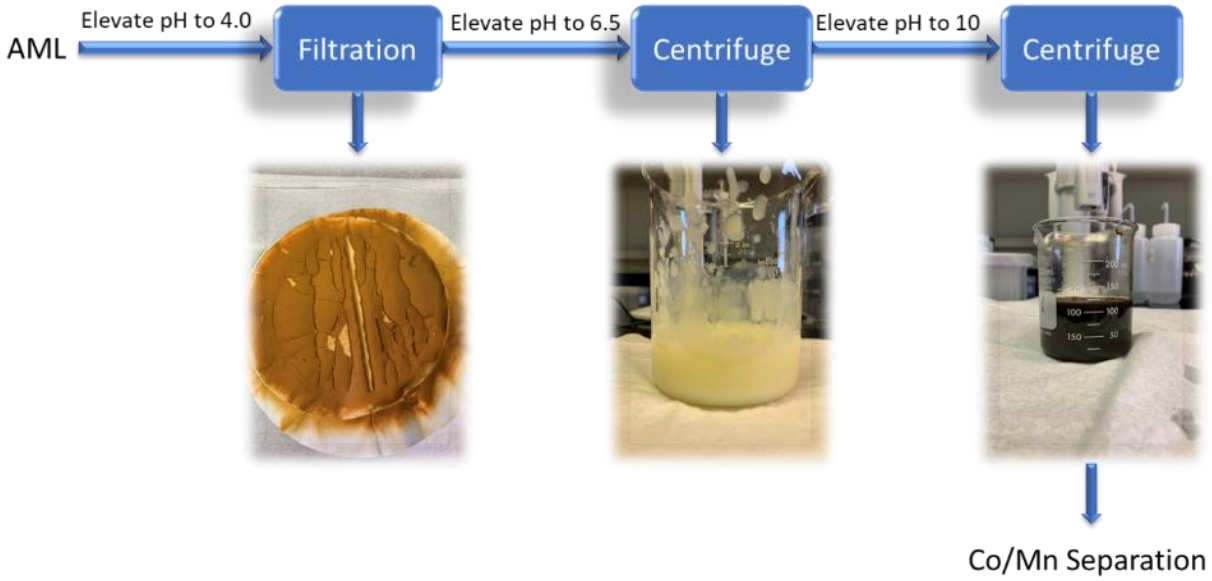



Figure 3.8.22. Schematic showing the procedures of preparing the Co/Mn preconcentrated slurry.

Oxalic acid precipitation tests were performed in a beaker of 15 mL maximum volumetric capacity. For each test, 7.5 mL of the solution was added into the beaker together with a predetermined volume of oxalic acid solution. The pH was then adjusted by adding sodium hydroxide and/or hydrochloric acid. After pH adjustment, mixing of the suspension was lasted for 15 min to provide sufficient time for the precipitate to crystallize and grow. The suspension was then centrifuged with the supernatant being collected for elemental recovery calculation. As shown in Figure 3.8.23, as the pH and oxalic acid dosage increased, the suspensions gradually showed a pink color, which

suggested the precipitation of cobalt. However, Figure 3.8.24 shows that Co was barely recovered under the investigated oxalic dosages and pH values. This contrast may be due to the formation of Co-oxalate chelates rather than precipitates during the reaction. Moreover, as shown, REEs were first precipitated from the solution followed by Ca, while Mn and Co were the least prone to precipitation. Therefore, it can be concluded that oxalic acid precipitation cannot afford satisfactory performance in the separation and upgrading of Co and Mn.

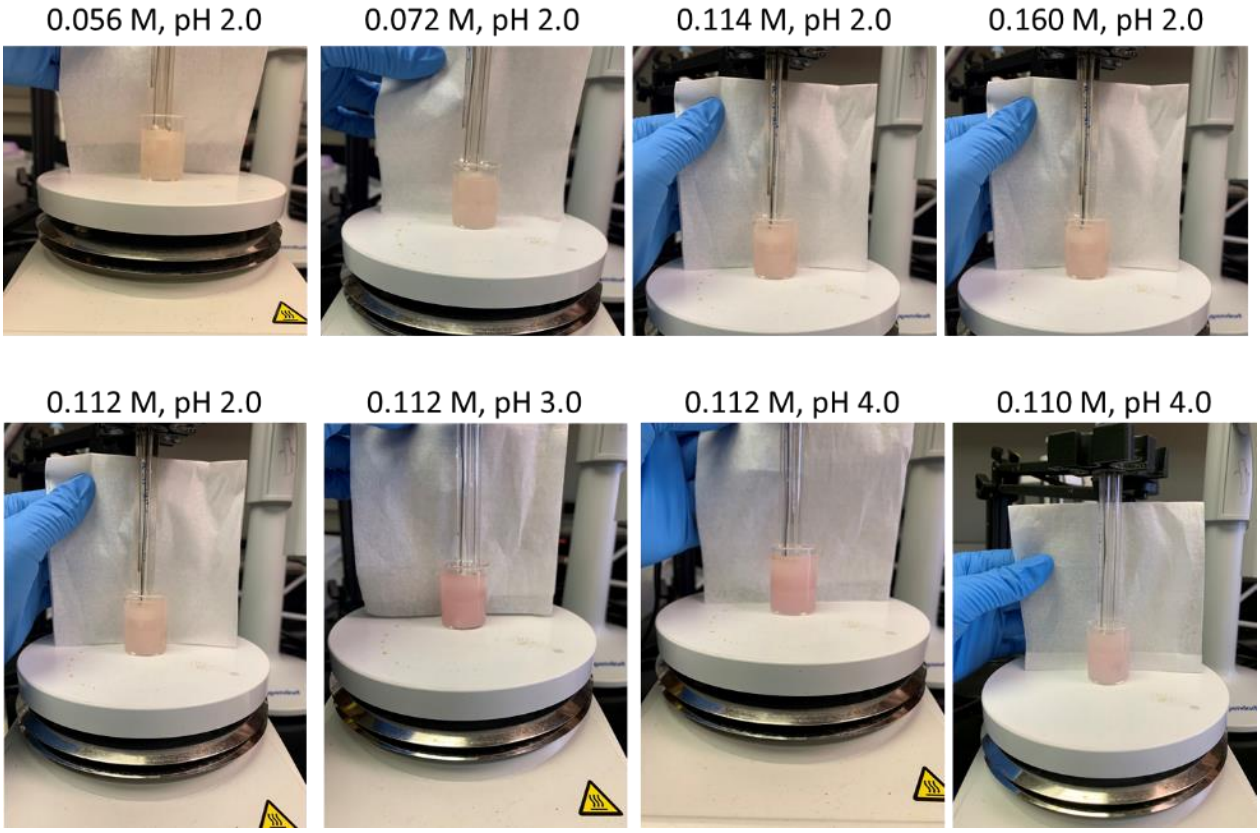



Figure 3.8.23. Appearance of the suspensions obtained from oxalic acid precipitation tests using different dosages of oxalic acid at different pH values.

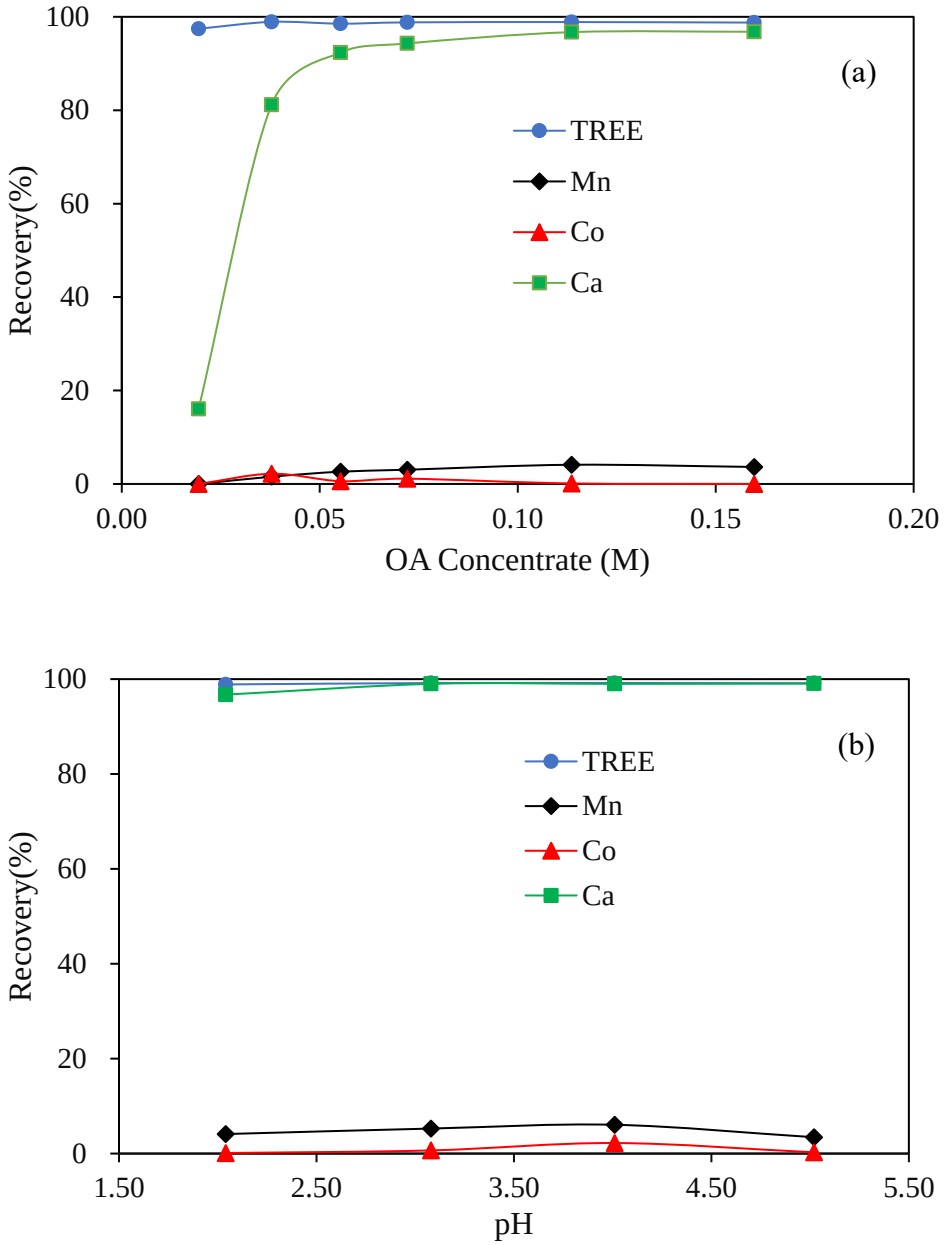



Figure 3.8.24. Oxalic acid precipitation recovery of TREE, Mn, Co, and Ca as a function of (a) oxalic acid dosage and (b) pH.

Since oxalic acid precipitation did not effectively separate and upgrade Mn and Co, more exploratory tests were performed using other methods. Selective dissolution and second stage precipitation were employed to achieve the objective. A new preconcentrated slurry was generated by using a larger volume reduction, leading to higher Mn and Co contents in the slurry (2,700 ppm Mn and 66 ppm Co). The slurry was used as a feedstock for the re-dissolution tests. Basically, during the re-dissolution process, the majority of Mn and Co were dissolved into solution, while a portion of the metal ions remained in the undissolved precipitates (Figure 3.8.25). The incomplete dissolution of Mn and Co is mainly due to the fact that a portion of the metal ions occurred as

higher valences, the oxides of which were difficult-to-dissolve. Based on this concept, it may be viable to convert all the Mn and Co into high-valence species; thereby, all the contaminants will be dissolved into solution, while the Mn and Co will remain undissolved. In this case, a concentrate of Mn and Co can be obtained with negligible contents of contaminant metal ions.

As shown in Figure 3.8.26, after adding 1 M $Na_2S_2O_8$ into the slurry at the beginning of the re-dissolution test, the re-dissolution recovery of Mn and Co was largely reduced to around 40% and 35%, respectively. Using higher oxidizing agent dosages, it is expected that nearly all the Mn and Co will end up as an undissolved material after the re-dissolution test. As shown, the contaminant metal ions, such as Al, Mg, and Ca, were completely dissolved into solution when pH was reduced to around 3.0. Therefore, the content of Mn and Co in the undissolved precipitate is relatively higher. Moreover, when using 1 M $Na_2S_2O_8$, a portion of the Mn and Co were dissolved into solution. Thereby the redissolved solution was obtained through filtration, which was used as a feedstock for the second stage precipitation tests.

The precipitation recovery of Mn, Co, and other metal ions from the redissolved solution is presented in Figure 3.8.27. As pH gradually increased, REEs were first precipitated from the solution, followed by Co and Mn. Nearly complete precipitation of Mn and Co was achieved when the pH was elevated to 9.0. At this pH, only small fractions of Mg and Ca were simultaneously precipitated. Therefore, the second stage precipitation of the redissolved solution effectively separated Mn and Co from contaminant metal ions. Moreover, as shown in the figure, nearly 80% of Co was precipitated at pH 8.0, under which condition less than 30% of Mn was precipitated. Therefore, the second stage precipitation contributed to the separation of Co from Mn at a certain level.

As aforementioned, re-dissolution tests were performed on a pre-concentrated slurry in the absence and presence of oxidizing agent (see Figure 3.8.25 and Figure 3.8.26). The undissolved materials obtained from the tests were collected, dried, and digested for elemental composition analysis (Figure 3.8.28). Mn and Co contents of as high as 39% and 0.78% as well as 35% and 0.89% were achieved from the tests in the absence and presence of oxidizing agent, respectively. The Mn contents have far exceeded the target grade of 2%. However, Co needs to be further upgraded in order to reach the target. Based on the recovery data shown in Figure 3.8.25 and Figure 3.8.26, it can be told Mg in the pre-concentrated slurry was completely dissolved into solution; whereas, around 2% of Mg still existed in the undissolved materials. Centrifuging was used to separate the undissolved materials, thereby the re-dissolved solution was not completely separated from the solids. This fact likely explained the appearance of Mg in the materials.

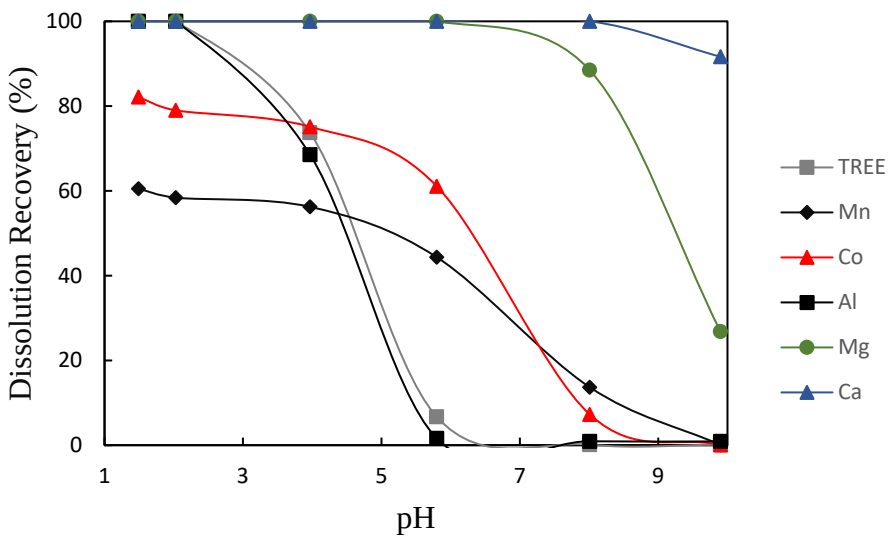



Figure 3.8.25. Re-dissolution test results of the preconcentrated slurry without adding any oxidizing agents.

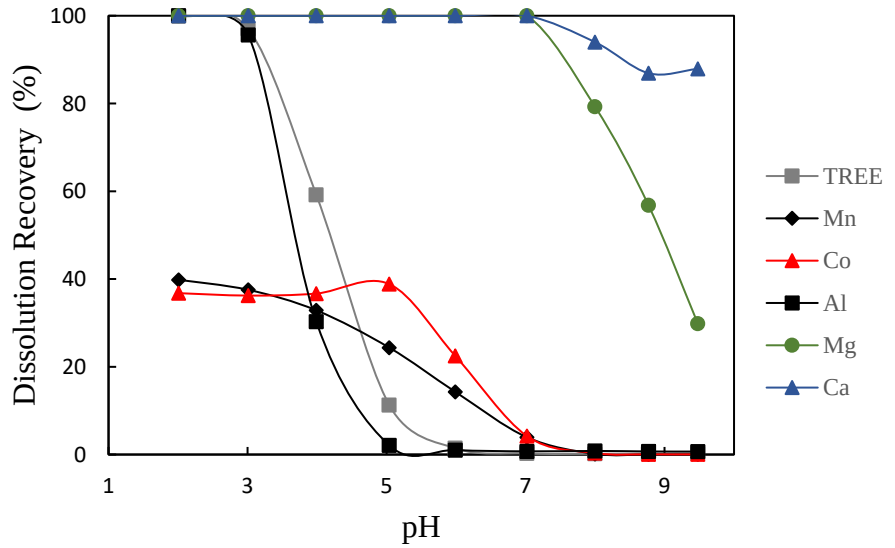



Figure 3.8.26. Re-dissolution test results of the preconcentrated slurry with 1 M $Na_2S_2O_8$ being added at the beginning of the test.

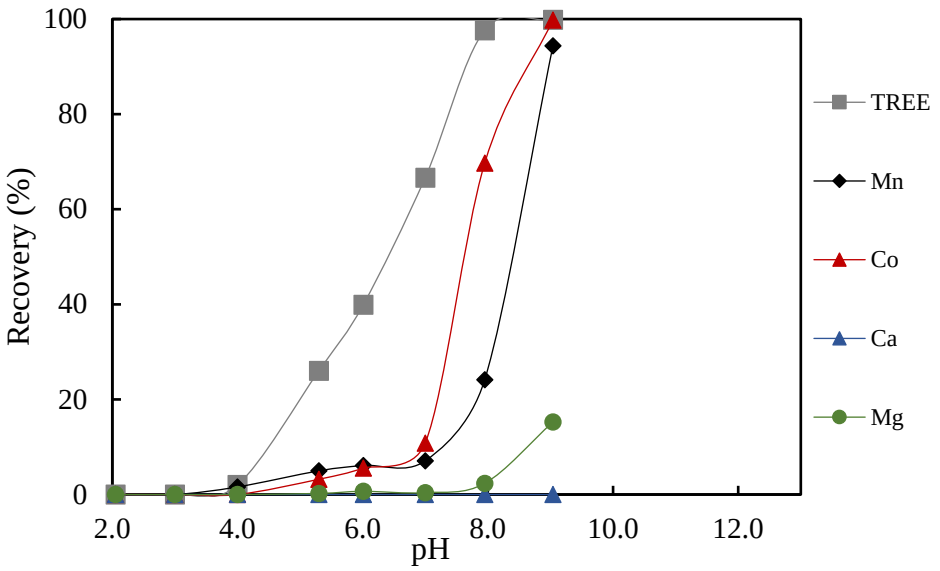



Figure 3.8.27. Precipitation recovery of Mn, Co, and other metal ions from the redissolved solution as a function of pH.

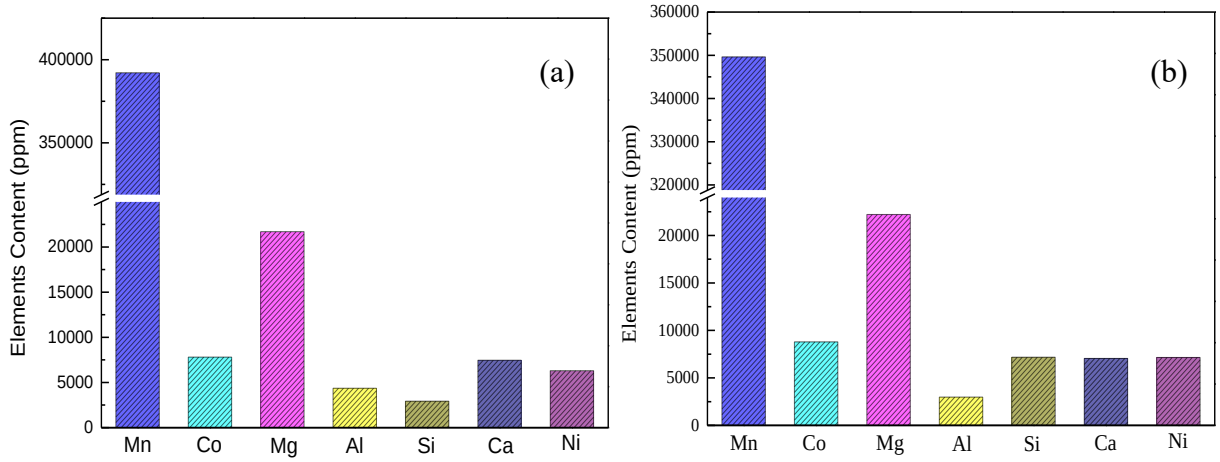



Figure 3.8.28. Elemental compositions of undissolved materials obtained from re-dissolution tests: (a) In the absence of oxidizing agent; and (b) In the presence of oxidizing agent.

Given the fact that the solid/liquid separation through centrifuging failed to completely separate the undissolved material from the redissolved solution. A washing step was incorporated into the re-dissolution process to thoroughly remove the contaminants, primarily Mg. Moreover, as shown in Figure 3.8.25, in the presence of 1 M $Na_2S_2O_8$, when pH decreased to 7.0, nearly 100% of Mg and Ca were redissolved into solution, while Mn and Co stayed with the precipitate. Therefore, reducing the slurry pH to 7.0, followed by solid/liquid separation, was performed to remove the contaminants. A schematic explaining the modified re-dissolution procedures is present in Figure 3.8.29. As shown, the preconcentrated slurry has a pH of around 9.0. After adding $Na_2S_2O_8$, the slurry pH was incrementally reduced to 7.0 by adding hydrochloric acid. The resultant suspension was subjected to centrifuging, and the supernatant was decanted, while the concentrated slurry was washed with deionized water three times. After washing, the slurry pH was further decreased to

5.0, followed by centrifuging. As shown in Figure 3.8.25, in the pH range of 5.0-7.0, Mn and Co were partially dissolved, while the dissolution of contaminants was negligible. Therefore, the dissolved solution in this pH range was expected to contain more Mn and Co relative to the contaminants. After the solid/liquid separation at pH 5.0, the slurry was subjected to further pH decreases until 1.5. Ultimately, an undissolved material and a redissolved solution were obtained. The undissolved material was dried and digested for elemental composition analysis using ICP-MS. The redissolved solutions obtained at pH 5.0 and 1.5 were subjected to a second stage precipitation tests.

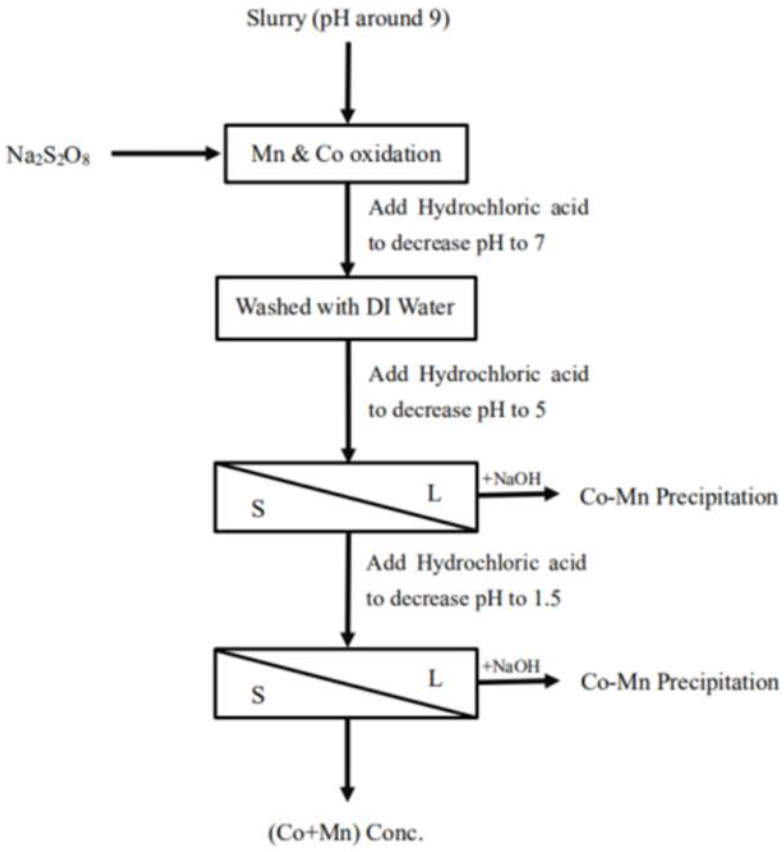



Figure 3.8.29. The modified re-dissolution experimental procedures.

As shown in Figure 3.8.30, using the modified re-dissolution procedures, the undissolved material contained up to 40.4% Mn and 0.97% Co. The contents of contaminants, especially Mg, were largely reduced. Second stage precipitation tests were performed on the redissolved solution obtained at pH 5.0 and 1.5, respectively. The pH of the solutions was incrementally elevated to 10.0. The precipitates formed in the pH range of 5.0-10.0 and 1.5-10.0 were collected, dried, and digested for elemental composition analysis. The analytical results are shown in Figure 3.8.31. The precipitate obtained in the pH range of 5.0-10.0 contained 31.4% of Mn and 1.04% of Co, while the precipitate obtained in the pH range of 1.5-10.0 contained 19.1% of Mn and 0.51% of Co. Overall, all the Mn contents have exceeded the target grade, however, all the Co contents have not met the target relying upon the revised re-dissolution process.

Based on the precipitate compositions, it can be told that a Co content of greater than 2% cannot be achieved even all the contaminants are removed from the precipitates. In order to obtain a qualified Co concentrate, certain degrees of Mn and Co separation need to be achieved. For example, in the second stage precipitation process, two precipitates can be collected from the redissolved solution in the pH range of 1.5-8.0 and 8.0-9.0, respectively (see Figure 3.8.27). In this case, the first and second precipitates are expected to contain a relative higher content of Co and Mn, respectively. Therefore, a Co precipitate of greater than 2% grade will likely be generated.

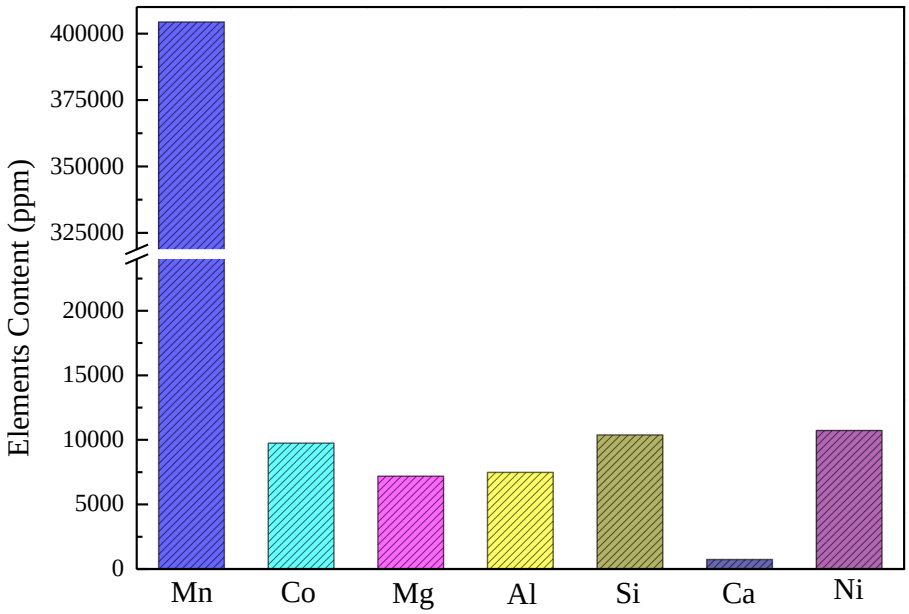



Figure 3.8.30. Elemental compositions of the undissolved materials obtained from the re-dissolution process.

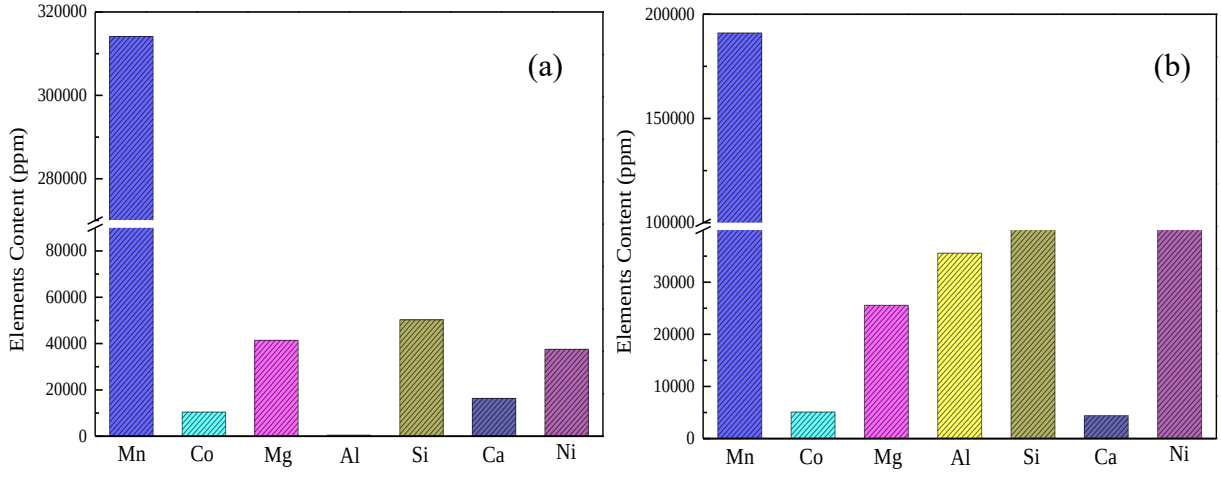



Figure 3.8.31. Elemental compositions of the precipitates obtained by the second stage precipitation of the redissolved solutions obtained at (a) pH 5.0 and (b) pH 1.5.

Based on the testing results of the process flowsheet shown in Figure 3.8.29, three updated process flowsheets were formulated to improve the Co/Mn recovery and purification, which are presented

in Figure 3.8.32. In Process 1, pH of the preconcentrated slurry was directly reduced to 1.5 in the presence of an oxidizing agent, $Na_2S_2O_8$. In this case, a suspension containing undissolved precipitates was obtained. A reducing agent, $NH_2OH{\cdot}HCl$, was added into the suspension, which led to a clear solution with all the precipitates being dissolved. The clear solution was subjected to pH increases to 9 and then to 10. The resultant precipitates in the pH ranges of 1.5-9.0 and 9.0-10.0 were collected and analyzed in terms of elemental content. In Process 2, rather directly to 1.5, pH of the suspension was first reduced to 7.0, which results in the selective dissolution of Ca and Mg. With the pH fixing at 7.0, the suspension was washed once by mixing with deionized water. After washing, pH of the suspension was brought to 1.5, leading to a suspension containing undissolved precipitates. The suspension was subjected to centrifuging, which will lead to a raffinate depleted in Co/Mn and a solid enriched in Co/Mn. The solid was redissolved into solution by adding $NH_2OH{\cdot}HCl$ at pH 1.5. The resultant redissolved solution was further subjected to pH increases to 9.0 and then to 10.0, which leads to two precipitates. The precipitate was dried and analyzed in terms of elemental composition. As shown, the first two steps of re-dissolution in Process 3 are similar to those of Process 2, however after the pH being reduced to 1.5, the resultant suspension with undissolved precipitates was completely dissolved by adding $NH_2OH{\cdot}HCl$. The resultant solution was subjected to the precipitation test, and two preciapites were sequentially obtained in the pH ranges of 1.5-9.0 and 9.0-10.0, respectively. The precipitates were dried and analyzed in terms of elemental composition.

Elemental compositions of the precipitates obtained using Process 1, 2, and 3 are presented in Figure 3.8.33. As shown, Mg contents in the precipitate resulting from Process 2 and 3 were lower than those from Process 1. Therefore, the washing step at pH 7.0 during the re-dissolution process largely reduced the alkaline earth contaminant metal ions, especially Mg. The precipitate collected at pH 9.0 during the staged precipitation process contained 1.3% of Co, which is lower than the target grade of 2%. However, the corresponding precipitates generated from Process 2 and 3 contained 5.2% and 2.4%, respectively. The content levels have met the expected target of 2%. It has been mentioned that due to the low concentration of Mn and Co in the preconcentrate slurry, only a small amount of precipitates were generated and digested for elemental analysis. Therefore, the contents of greater than 2% may partially result from experimental errors. However, this concern can be dispelled by the fact that a higher grade of around 12% was obtained using an updated flowsheet formulated based on Process 1, 2, and 3.

Collecting precipitates at pH 9.0 during the staged precipitation step of Process 2 and 3 led to Co products that meet the target set in the proposal of this project. However, more efforts were conducted to get Co products of higher purity. As shown in Figure 3.8.33, Mn was the primary constituent of the Co products, and other elements only accounted for a small fraction in mass. Therefore, in order to improve Co content, Mn needs to be separated from Co. Moreover, as suggested from the staged precipitation curves, it can be told that Co cannot be efficiently separated from Mn by controlling precipitation pH. In this case, other methods need to be tested to achieve the objective. After comprehensive literature research, the sulfide precipitation method was selected. The reasonability of this selection can be justified because cobalt sulfide has a much lower solubility product relative to manganese sulfide (see Table 3.8.3).

Table 3.8.3. Log Ksp of selected sulfide precipitates.

| CoS | NiS | ZnS | MnS | FeS | CuS |
|---|---|---|---|---|---|
| -21.64 | -19.55 | -23.63 | -9.6 | -16.88 | -34.62 |

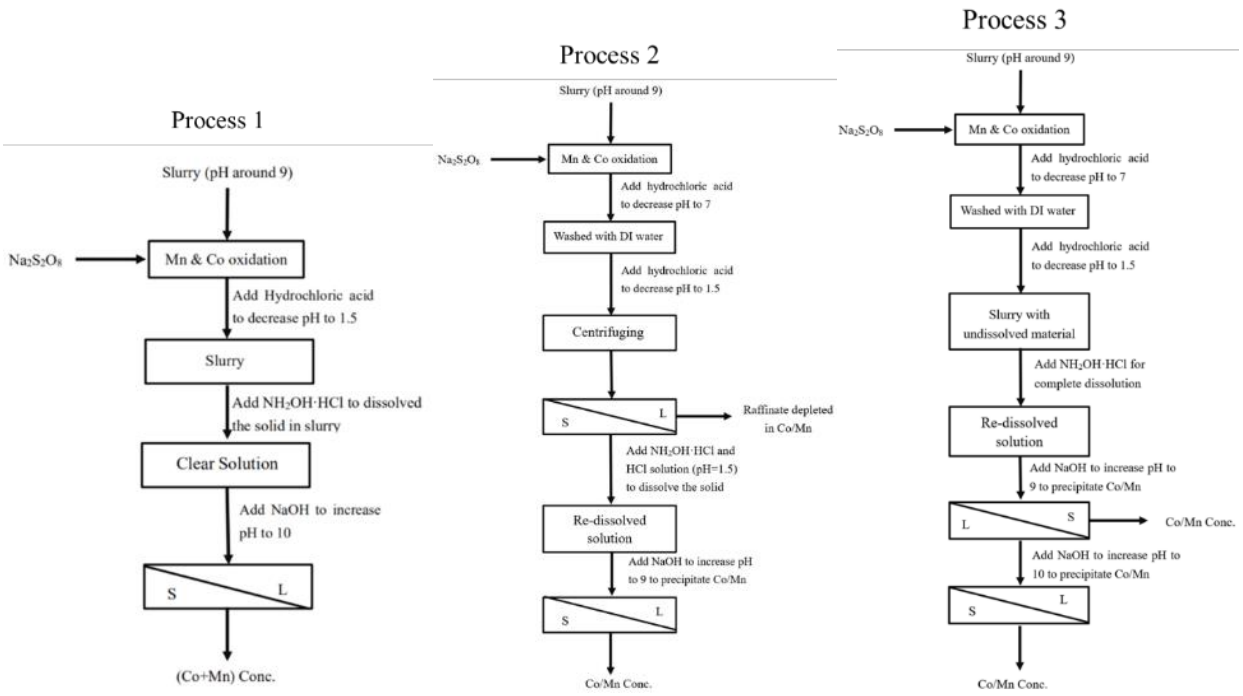
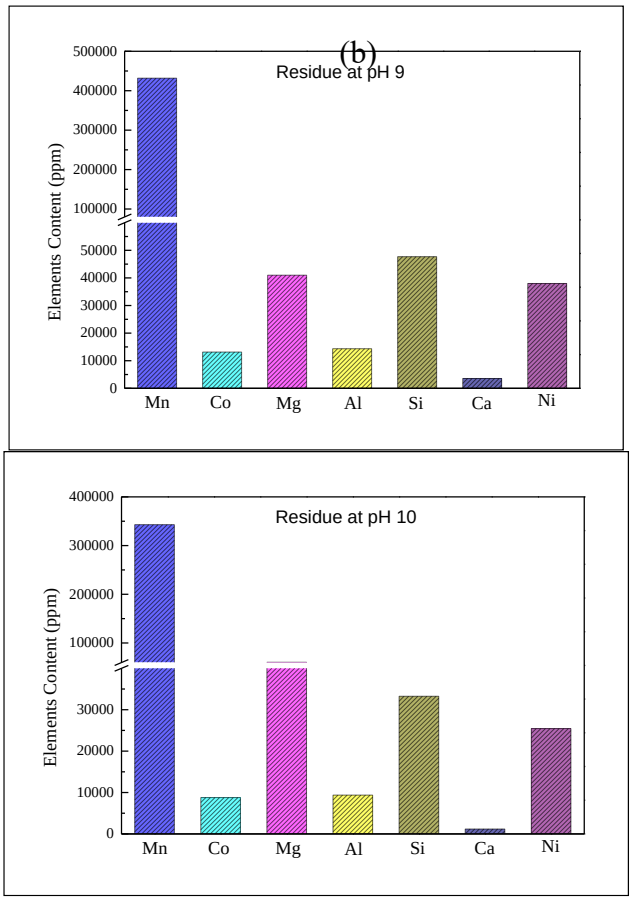
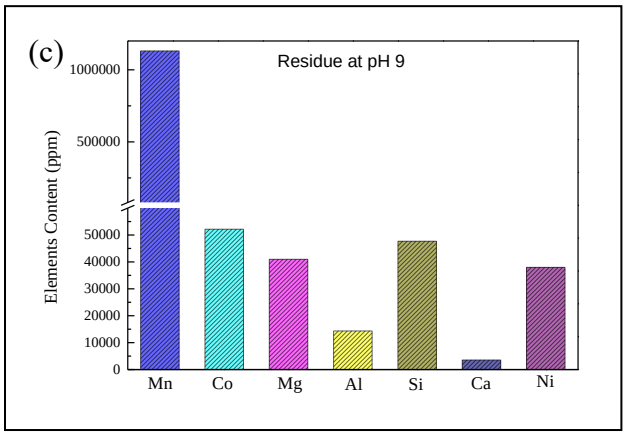
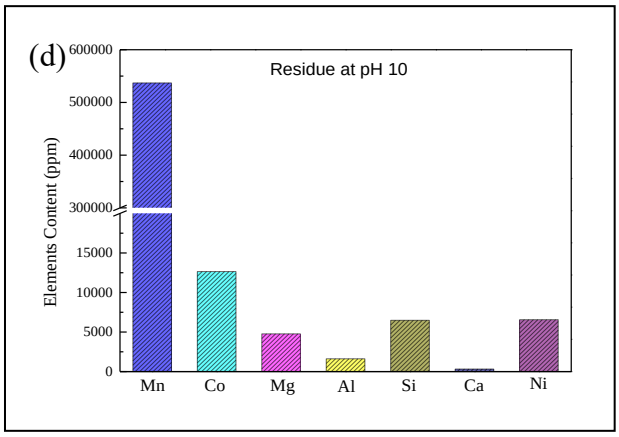



Figure 3.8.32. Three process flowsheets formulated for Co and Mn recovery.

(a)


(b)
Residue at pH 9
Elements Content (ppm)
500000
400000
300000
200000
100000
50000
40000
30000
20000
10000
0
Mn
Co
Mg
Al
Si
Ca
Ni
Residue at pH 10
Elements Content (ppm)
400000
300000
200000
100000
30000
20000
10000
0
Mn
Co
Mg
Al
Si
Ca
Ni


(c)
Residue at pH 9
Elements Content (ppm)
1000000
500000
50000
40000
30000
20000
10000
0
Mn
Co
Mg
Al
Si
Ca
Ni


(d)
Residue at pH 10
Elements Content (ppm)
600000
500000
400000
300000
15000
10000
5000
0
Mn
Co
Mg
Al
Si
Ca
Ni

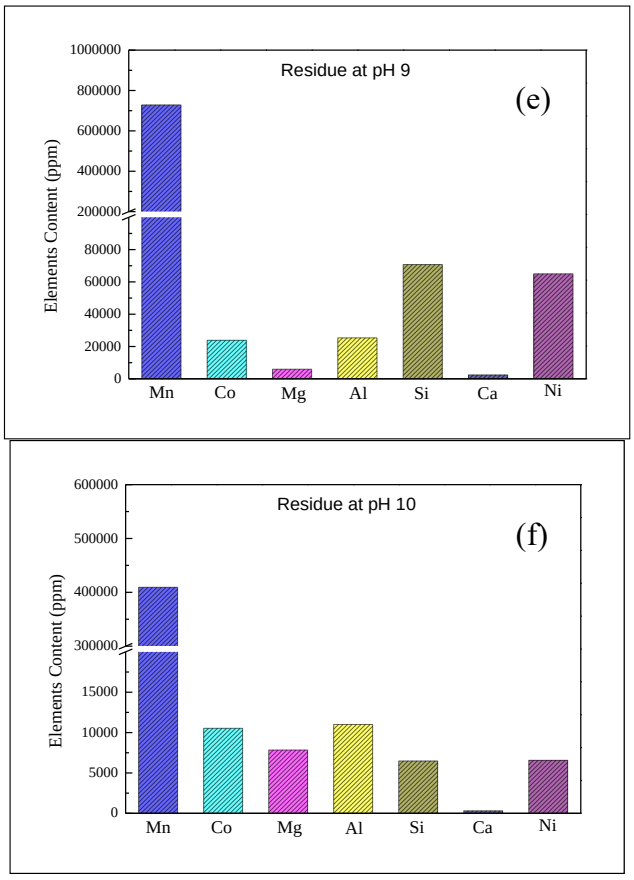



Figure 3.8.33. Elemental compositions of the precipitates formed form Process 1 (a and b), Process 2 (c and d), and Process 3 (e and f).

The way that how sulfide precipitation was incorporated into the overall Co/Mn recovery and separation process is presented in Figure 3.8.34. A solution enriched in Mn and Co was first generated from the preconcentrated slurry through pH reduction and re-dissolution. As shown in Table 3.8.4, the solution contained around 4453 ppm Mn and 125 ppm Co. In addition, with a concentration of around 406 ppm and 111 ppm, respectively, Zn and Ni are also promising valuable elements that can be recovered from the solution. Four tests using different sets of the experimental condition were formulated to optimize the sulfide precipitation performance (Test 1-4, see Table 3.8.5).

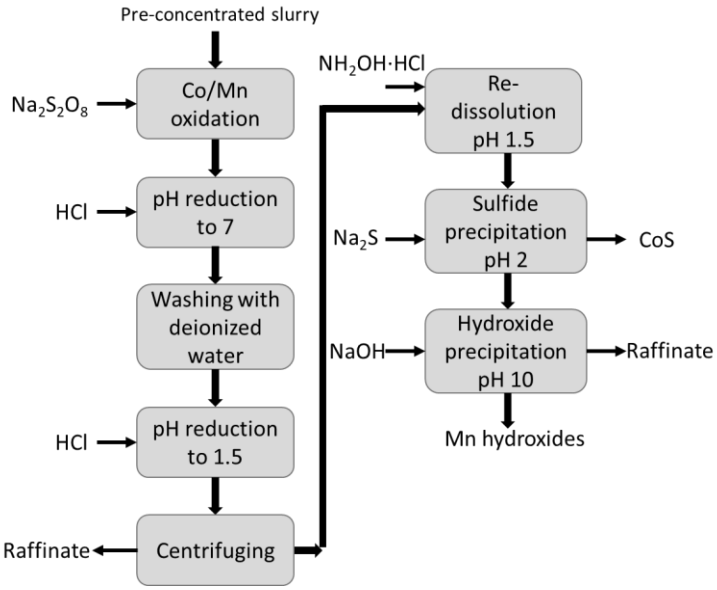



Figure 3.8.34. Process flowsheet incorporating sodium sulfide precipitation.

Table 3.8.4. Elemental composition of the solution used for the sulfide precipitation tests.

| TREE | Mn | Co | Ni | Cu | Zn | Mg | Al | Si |
|---|---|---|---|---|---|---|---|---|
| 17.7 | 4453.2 | 125.3 | 111.1 | 1.65 | 405.75 | 97.5 | 37.8 | 177.5 |

Table 3.8.5. Experimental conditions used for the four different tests.

| Test Code | Initial pH | Final pH | Temp. (°C) | Total precipitation (min) |
|---|---|---|---|---|
| 1 | 2 | 2 | 50 | 60 |
| 2 | 3 | 3 | 50 | 60 |
| 3 | 2 | 2 | 25 | 60 |
| 4 | 3 | 3 | 25 | 60 |

The sulfide precipitation tests were performed in a beaker of 50 mL maximum volumetric capacity. After precipitation, the suspension was subjected to centrifuging. Both the supernatant and precipitate were collected and analyzed to calculate the recovery of Mn, Co, and the other metal ions and determine the resultant precipitates' elemental composition. As shown in Figure 3.8.35(a), Mn recovery is negligible for all the four tests, while more than 50% of Co was precipitated. This phenomenon suggested that Co can be selectively recovered from the solution through sulfide precipitation. In addition to Co, other trace valuable metals, such as Ni, Cu, and Zn, were also selectively recovered. The high selectivity of sulfide precipitation resulted in products containing as high as 12% of Co, 10% of Ni, and 32% of Zn (see Figure 3.8.35 (b)). The appearance of certain amounts of Mn in the products was observed, which is likely due to the exceedingly higher concentration of Mn relative to the other metal ions in the solution. In this case, even a small recovery of Mn can lead to a high content in the product. Moreover, a small amount of suspension remained in the centrifuge tube after centrifuging. The Mn dissolved in the suspension will be

reported to the precipitate after drying in the tube placed in a drying oven. The appearance of cobalt sulfide precipitate is presented in Figure 3.8.36.

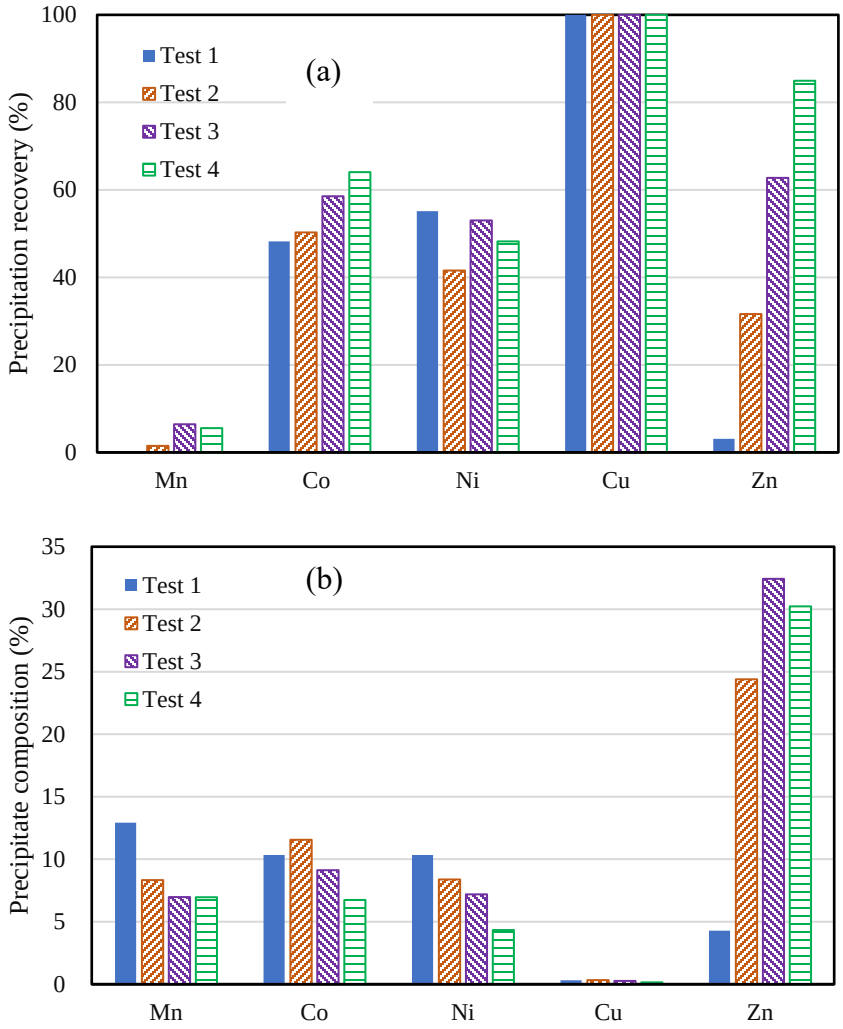



Figure 3.8.35. Precipitation recovery (a) and precipitate composition (b) obtained using sulfide precipitation.

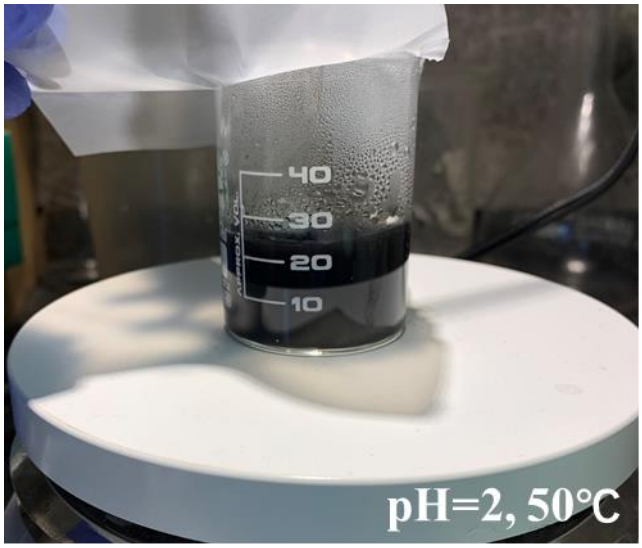



Figure 3.8.36. The appearance of the cobalt sulfide precipitate obtained at pH 2 and 50°C (Test 2).

The sulfide precipitation results suggested that this method is very selective for the separation of Co from Mn, and alkaline earth metals, such as Mg and Co. Therefore, the flowsheet shown in Figure 3.8.34 can be simplified into the flowsheet shown below (Figure 3.8.37). In the simplified flowsheet, all the metal ions contained in the preconcentrated slurry, such as Mn, Co, Al, Ca, and Mg, were dissolved into a solution, which was then reported to the sulfide precipitation step whereby Co was recovered. After that, the resultant supernatant was subjected to pH increases to 10.0, leading to a Mn product of high purity (greater than 40% per results from prior tests).

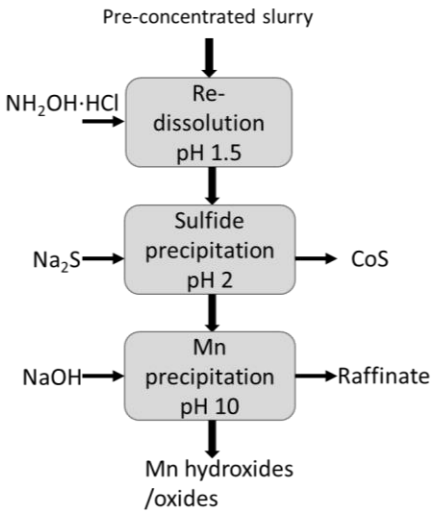



Figure 3.8.37. The simplified Co/Mn recovery and purification process.

### *3.8.2 High-Purity Critical Metal Compound Production*

Two Co/Mn recovery flowsheets were proposed based on bench-scale experimental results: a complex flowsheet and a simplified flowsheet (Figure 3.8.38). The complex flowsheet was proved to be able to generate Co and Mn products of greater than 2 wt% purity. However, experimental data of the tests relevant to the simplified flowsheet was not obtained, therefore, the effectiveness of the second flowsheet was not assessed. The data was obtained and used to assess the effectiveness. As Figure 3.8.39 shows, the Mn hydroxide precipitate and the Co sulfide precipitate that were produced using the simplified flowsheet contained around 27 wt% of Mn and 6 wt% of Co, respectively, which are greater than 2 wt%. Therefore, the target purity level proposed in the proposal can be achieved using the simplified flowsheet. In this flowsheet, the pre-concentrated slurry obtained through the staged precipitation of leachates containing REEs is re-dissolved by decreasing pH to 1.5. A reducing agent, hydroxylamine hydrochloride ($NH_2OH{\cdot}HCl$), is added in the re-dissolution process to achieve a complete dissolution of the precipitates in the slurry. After re-dissolution, a Co concentrate is generated through sulfide precipitation by using sodium sulfide ($Na_2S$) as the precipitant. The cobalt sulfide precipitate is then collected through solid-liquid separation. The resultant solution is then subjected to pH increases, and a Mn concentrate is generated from the solution at pH 10. Overall, the simplified flowsheet consists of three segments: re-dissolution, sulfide precipitation, and hydroxide precipitation. The purity level of the products obtained using the simplified flowsheet is greater than the target level (2 wt%), however, it is challenging to obtain products of higher purity (e.g., 90 wt%) using the precipitation method alone. Therefore, we investigated the separation of Co from the major contaminant metal ions, including Ni, Cu, and Zn, through solvent extraction.

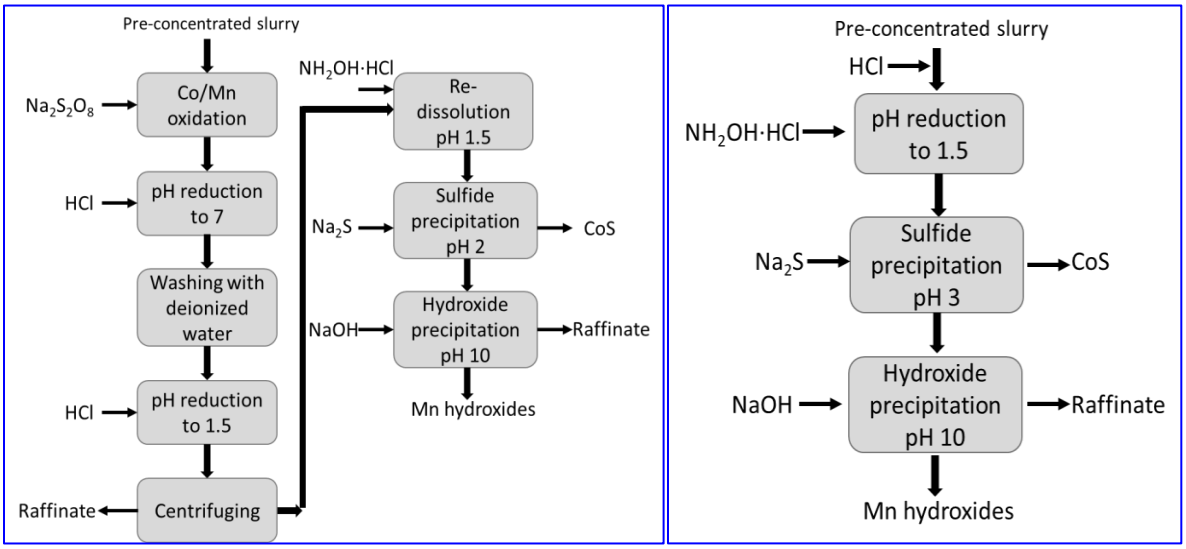



Figure 3.8.38: The complex (left) and simplified (right) flowsheets for Co/Mn recovery.

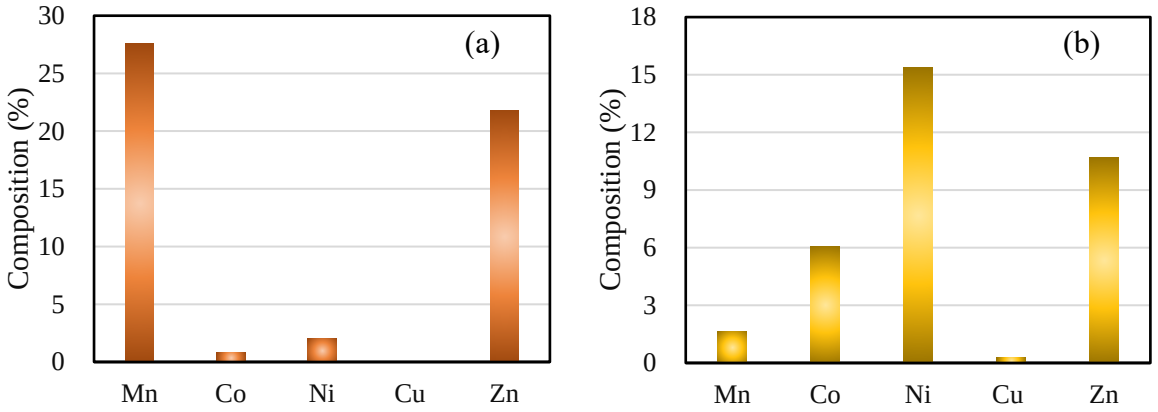



Figure 3.8.39. The elemental composition of (a) the Mn precipitate and (b) the Co precipitate obtained using the simplified flowsheet.

**Solvent Extraction Experiments:** Due to the low Co content in the initial feedstock (acid mine leachate) that was used in this study, it is difficult to obtain a CoS precipitate of large amount for solvent extraction tests. Therefore, a stock solution, containing 1 g/L Co, 1 g/L Ni, 1 g/L Cu, and 1 g/L Zn, was prepared and used as a feedstock for the solvent extraction tests in this reporting period. In the extraction tests, five different types of extractants were employed, including DEHPA, CYANEX 272, Versatic Acid 10, IL1, and IL2, which were dissolved in the same organic phase (kerosene). The first three extractants are commonly used in metal extraction industry and have been frequently reported in the literature, while the last two extractants are ionic liquids and have rarely been studied. All the extractants were obtained from regent supply companies. The aqueous solution was mixed with the organic phase in a beaker, and the beaker was placed on a magnetic stirrer. The system was continuously mixed for a period of 20 min, which was then transferred into a separating funnel and stood for 5 min (Figure 3.8.40). The aqueous phase was collected from the bottom of the funnel, and subjected to elemental analysis using ICP-MS. Recovery of the different metal ions was calculated based on the elemental concentration of the feed and extracted aqueous solutions.

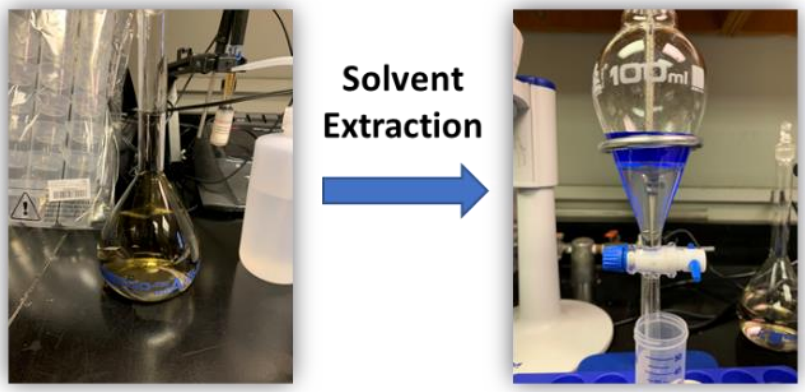
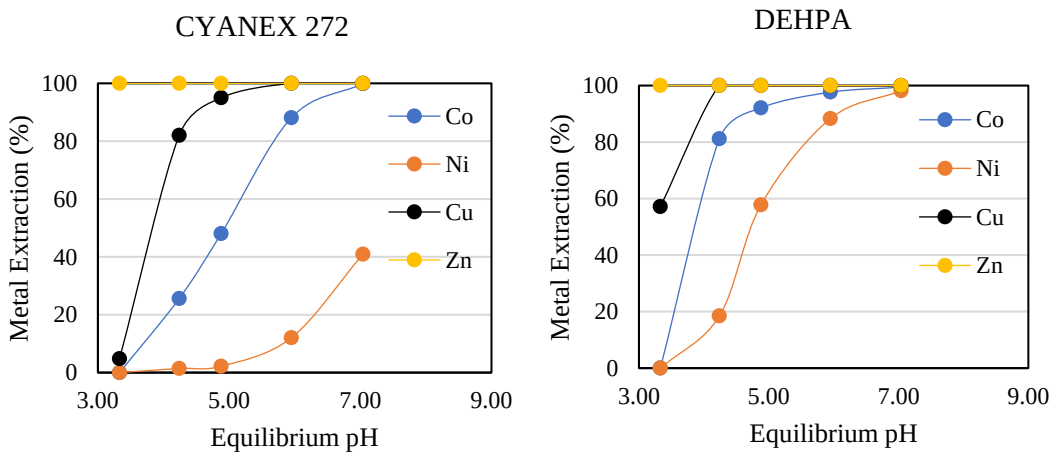



Figure 3.8.40. Procedures of the solvent extraction tests.

Extraction recoveries of Co, Ni, Cu, and Zn from the synthetic solution using the different extractants are presented in Figure 3.8.41. The experiments were conducted at room temperature

with an aqueous to organic volumetric ratio (A/O) of 1. As shown, CYANEX 272, DEHPA, and IL 1 provided satisfactory separation of Co from the other metal ions. For example, at an equilibrium pH of around 3.0, nearly 100% of Zn and 60% of Cu were recovered using DEHPA, while the recovery of Co and Ni is negligible. However, using a higher equilibrium pH of around 4.5 with the same extractant, around 80% of Co and only 20% of Cu were extracted. The performance of IL 1 is also encouraging. As the figure shows, at an equilibrium pH of 6.0, Zn and Cu were completely extracted from the solution, while the extraction of Ni and Co is close to zero percent. Based on the performance of the three extractants, a strategy for the separation of Co from Ni, Cu, and Zn can be formulated: Complete removal of Zn and Cu first using IL 1 followed by Co and Ni separation using either DEHPA or CYANEX 272. Compared with these three extractants, the performance of the other two extractants, Versatic Acid 10 and IL 2, are less satisfactory. For example, the extraction recovery curve of Co nearly overlaps with that of Ni using Versatic Acid 10, while IL 2 had strong extractability towards to Zn, Co, and Cu, and as a result, it is very different to achieve separation among the three metal ions using this extractant.

In order to evaluate whether the metal ions loaded in the organic phase can be removed back into an aqueous solution. Stripping tests were performed for the metal ions that are extracted by DEHPA. As Figure 3.8.42 shows, using a sulfuric acid concentration of as low as 0.1875 mol/L, all the metal ions loaded in the organic phase can be recovered. This acid concentration level corresponds to a solution pH of 1, thereby, using DEHPA as the extractant, the metal ions can be efficiently stripped. Moreover, acid concentrations greater than 0.5 mol/L causes negative impacts on the stripping. The same phenomenon has been reported in the literature.



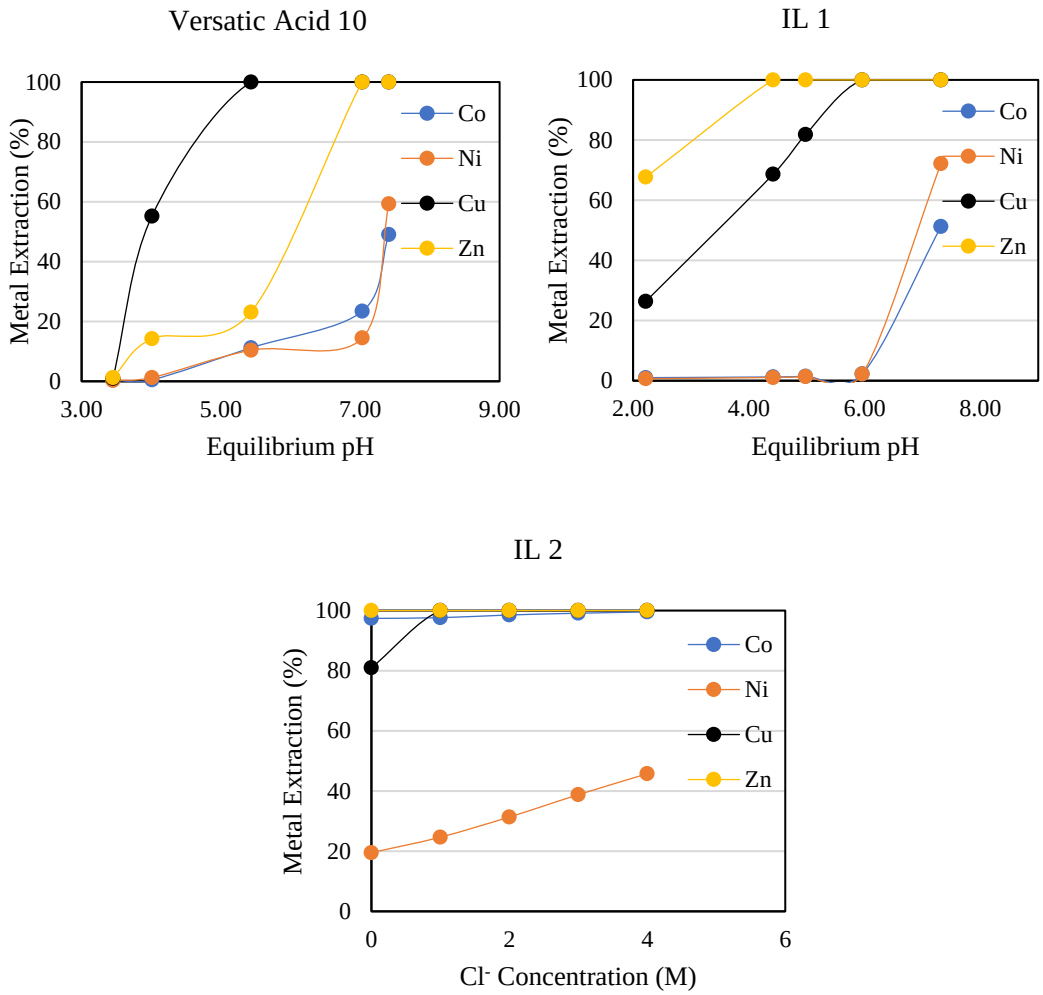



Figure 3.8.41. Extraction recoveries of Co, Ni, Cu, and Zn using different extractants (room temperature, A/O = 1).

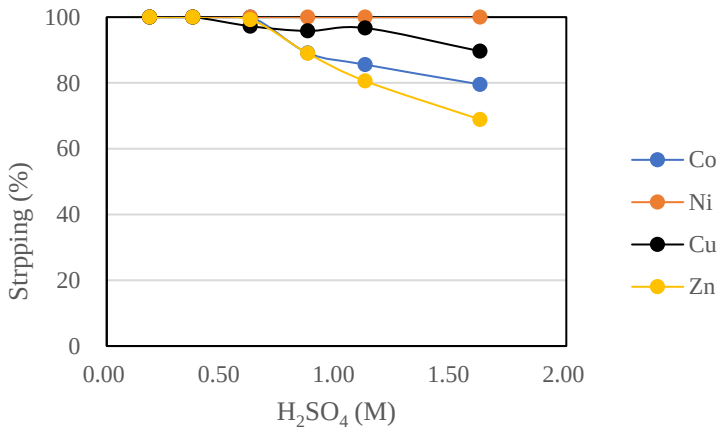



Figure 3.8.42. Stripping recovery of the metal ions loaded in the organic phase by DEHPA.

**Concentrated Slurry Preparation:** Based on the solvent extraction test results, it can be inferred that high-purity Co products can be generated from leachates obtained from coal-related materials, since the efficient separation of Co from Ni, Cu, and Zn can be achieved using proper extractants. However, to confirm this inference, extraction tests need to be performed on Co solutions generated from the leachate obtained from coal-related materials. Therefore, in this reporting period, efforts were made to generate a sufficient amount of real solutions containing Co. A certain amount of precipitates containing Co and Mn were produced in the pilot plant and shipped to the laboratory at Virginia Tech. Re-dissolution tests were performed on the precipitates to generate a solution of large volume and containing Co. As Figure 3.8.43 shows, two re-dissolution tests were performed under different conditions. In the first re-dissolution test, a solids concentration of 1% (w/v) was used, and $NH_2OH{\cdot}HCl$ was added to serve as a reducing agent, around 18% of the precipitates were re-dissolved, which is lower as compared with the dissolution observed before (nearly 100%). A much lower dissolution of 4.5% was obtained in the second test, which employed a higher solids concentration of 20% (w/v) without the addition of the reducing agent. Overall, the re-dissolution of the precipitates was ineffective. This phenomenon is likely due to that the precipitates had been dried in an oven prior to being shipped to Virginia Tech. Drying at high temperature caused oxidation of certain metal ions in the precipitate, and based on the phenomenon observed in the re-dissolution tests, once the metal ions are oxidized, they can hardly be reduced again.

Re-dissolution of the precipitates failed to provide a sufficient volume of solutions containing Co. Therefore, efforts were made in this reporting period to prepare a sufficient volume of pre-concentrated slurry from an acid mine leachate. The preparation followed the same procedure as the staged precipitation test. Around 150 L acid mine leachate was processed in this reporting period, which led to around 600 ml pre-concentrated slurry (see Figure 3.8.86). In the next reporting period, attempts will be made to produce high-purity Co products from the pre-concentrated slurry through re-dissolution, sulfide precipitate, and solvent extraction methods.

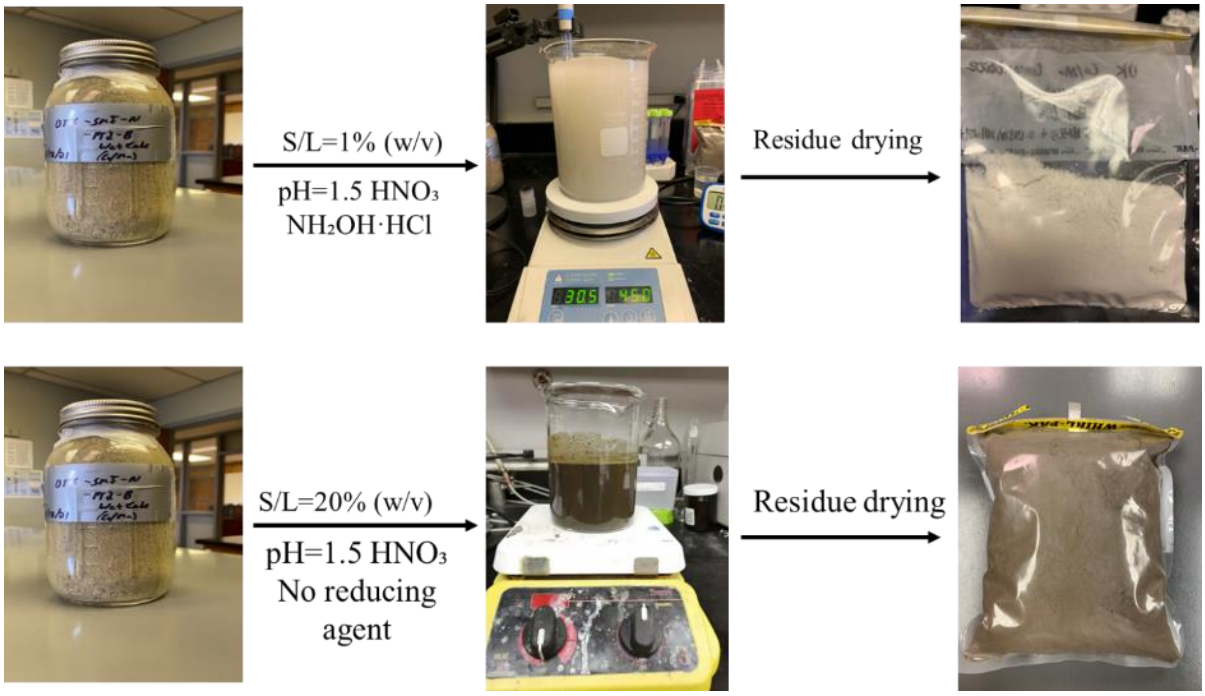



Figure 3.8.43. Re-dissolution tests performed on the precipitates generated in the pilot plant.



Figure 3.8.44. Pre-concentrated slurry prepared from an acid mine leachate.

**CoS Re-dissolution:** The re-dissolution of CoS obtained through sulfide precipitation is necessary, since the re-dissolved solution needs to be subjected to solvent extraction to further remove the contaminant metal ions. In this reporting period, re-dissolution tests were performed on a CoS precipitate prepared from a synthetic solution. In addition to Co, the solution also contained certain concentrations of the contaminant metal ions. As Figure 3.8.45 shows, precipitates were immediately formed after mixing the solution containing Co and other metal ions with a sodium sulfide solution. The precipitates were collected through filtration and completely dried in an oven.

Re-dissolution tests were performed on the dry sample at room temperature. Moreover, the effect of roasting on the re-dissolution was also evaluated. In this case, the sample was first roasted at different temperatures for 2 hours prior to re-dissolution. As shown in Figure 3.8.46, around 87% of the raw precipitates were dissolved by reacting with 1.2 M HCl at room temperature. Roasting of the precipitates in the absence of any additives contributed to the re-dissolution. For example, after roasting at 400ºC for 2 hours, the precipitates were almost completely dissolved. The positive impacts of roasting were likely due to the conversion of metal sulfides into amorphous metal oxides. The amorphous species are expected to be more leachable as compared with the sulfides. Overall, the experimental results suggested that cobalt sulfide precipitates can be effectively dissolved.

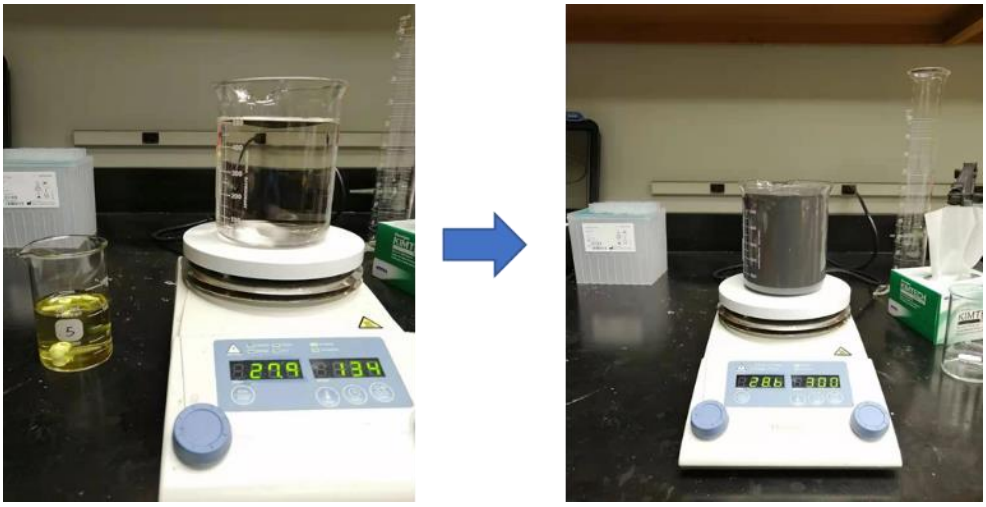


Figure 3.8.45. CoS precipitate preparation.

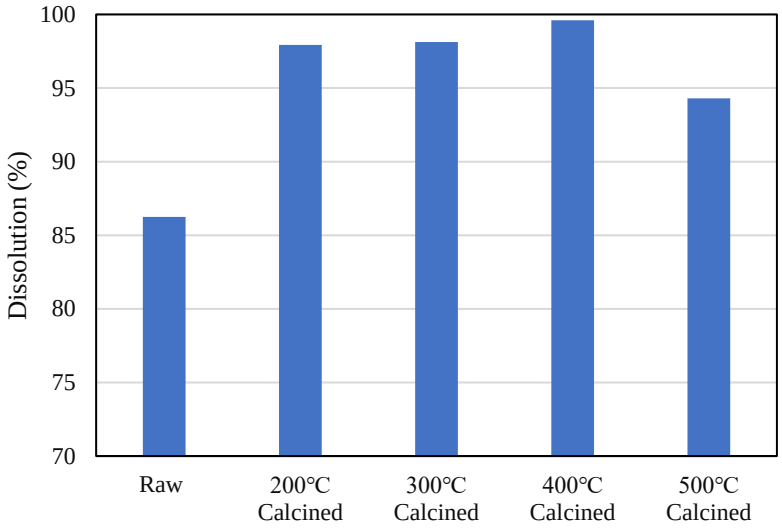



Figure 3.8.46. Re-dissolution of the raw and calcined precipitates with 1.2 M HCl at room temperature.

**Optimization of Sulfide Precipitation:** A certain amount of precipitate containing Co and Mn was produced in the pilot plant and shipped to the Mineral Processing Laboratory at Virginia Tech. Re-dissolution tests were performed on the precipitate to generate a large volume of solution containing Co. The solution was used as a feed for the sulfide precipitation tests. A solid concentration of 7 % (w/v) was used, and the pH of the slurry was decreased to 1.5 by adding $HNO_3$. $NH_2OH{\cdot}HCl$ was added as a reducing agent to reduce Mn and Co and make them easier to dissolve. Around 17% of the precipitate was re-dissolved, and the elemental concentration of the re-dissolved solution is shown in Figure 3.8.47. It can be seen that the solution contained around 2500 ppm Mn and 51 ppm Co. In addition, the concentration of Zn and Ni in the solution was around 162 ppm and 58 ppm, respectively. Relative high concentrations of Al and Fe were present in the solution due to incomplete removal in the staged precipitation process.

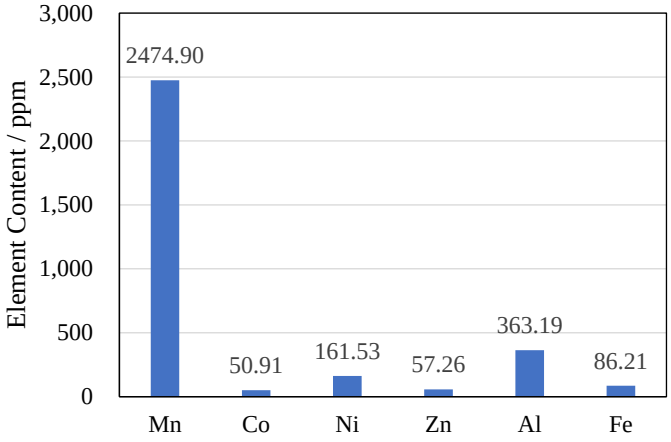



Figure 3.8.47. Elemental concentration of the re-dissolved solution.

The optimal pH and concentration of sodium sulfide solution for the sulfide precipitation process were investigated to separate Mn and Co effectively. Sulfide precipitates were immediately formed after sodium sulfite solution was added into the re-dissolved solution. The pH of the solution increased significantly since sodium sulfide solution is alkaline; Therefore, hydrochloric acid was needed to adjust the pH to a target value. The slurry was stirring at the target pH value for 15 minutes, and the suspension was then subjected to centrifuging for solid/liquid separation. The supernatant was diluted with 5% $HNO_3$, and the elemental concentration of the supernatant was measured using ICP-MS to calculate the recovery of Mn, Co, and the other metal ions.

The sulfide precipitation tests were performed in a beaker of 50 mL maximum volumetric capacity. As stated in the previous reports, the high selectivity of sulfide precipitation could still be observed that the recovery of Mn was significantly lower than other valuable elements, such as Co, Ni, and Zn, and most of the Mn remained in the solution, which can be recovered in the form of hydroxide precipitation. As shown in Figure 3.8.48, the recovery of elements was significantly enhanced when raising the pH from 2 to 3, except Mn. When the concentration of sodium sulfide solution was 0.04 mol/L (see Figure 3.8.48(a)), the recovery of Mn was below 6% for all the three different pH values. The highest recovery of Co and Ni was obtained at pH 3, which was 71% and 63%, respectively. In addition to Co and Ni, Zn was also selectively recovered. The recovery of Zn was 88% at pH 2, and it was improved to 98% when the pH raised to 3. As shown in Figure 3.8.48 (b),

when using 0.06 mol/L sodium sulfide solution for precipitation, the recovery of Co, Ni, and Zn was lower than 90% at pH 2, and it was improved to greater than 95% at pH 3. Similar results were obtained when increasing the concentration of sodium sulfide solution to 0.08 mol/L. Therefore, the optimal pH value for sulfide precipitation is 3.

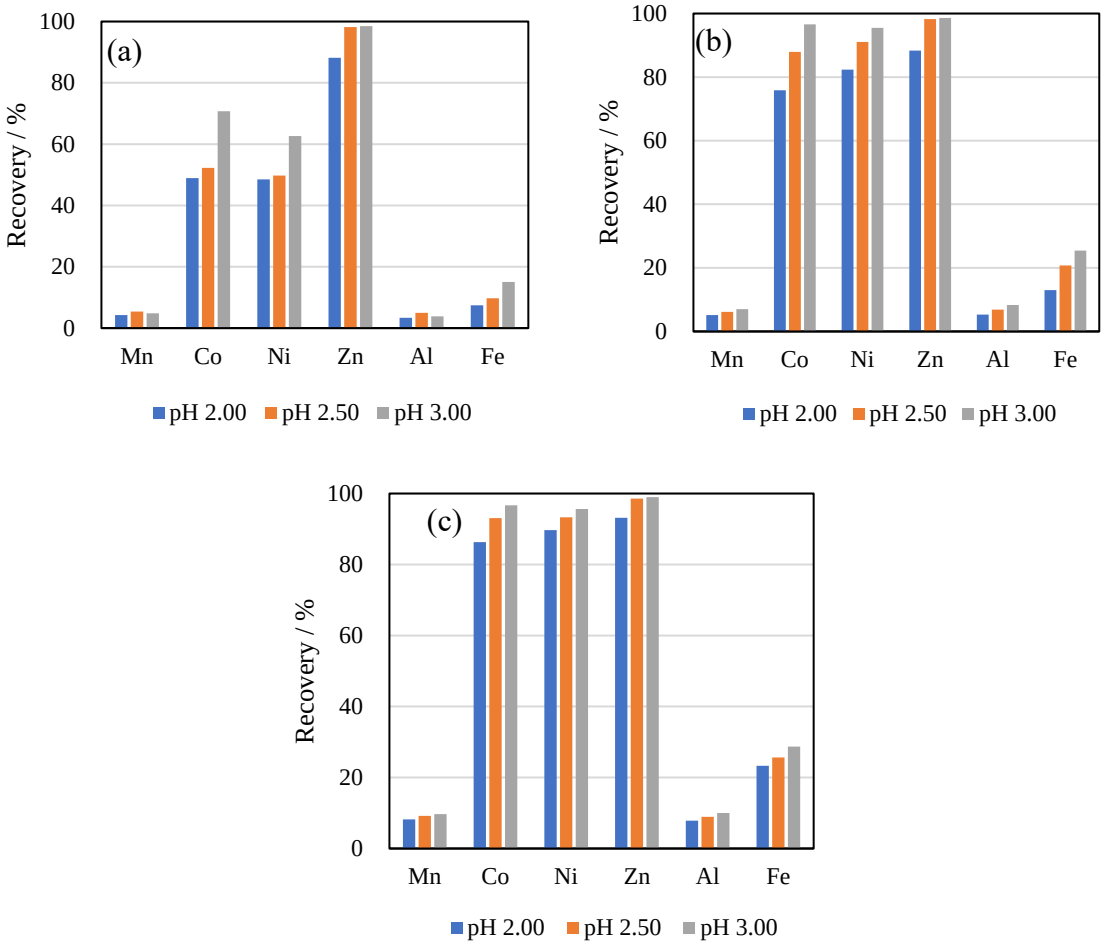



Figure 3.8.48. Effect of pH on sulfide precipitation recovery of metal ions (a) 0.04M $Na_2S \cdot H_2O$ (b) 0.06M $Na_2S \cdot H_2O$ (c) 0.08M $Na_2S \cdot H_2O$

The effect of sodium sulfide solution concentration on the recovery of metal ions at different pH values is presented in Figure 3.8.49. The recovery of Co and Ni was below 50% at pH 2.0 and 2.5 (see Figure 3.8.49(a) and (b)) when the concentration of sodium sulfide solution was 0.04 mol/L. Even when the pH was raised to 3.0, the recovery of Co and Ni was elevated to only around 70% and 60%, respectively (Figure 3.8.49(c)). When the concentration of sodium sulfide solution was increased to 0.06 mol/L, the recovery of Co and Ni increased significantly, with more than 95% of the Co, Ni, and Zn being recovered from the re-dissolved solution at pH 3. However, a further increase in the concentration of sodium sulfide solution to 0.08 mol/L did not noticeably improve the recovery of Co, Ni, and Zn. Based on the experimental results, it can be concluded that the optimal sodium sulfide concentration for sulfide precipitation is 0.06 mol/L.

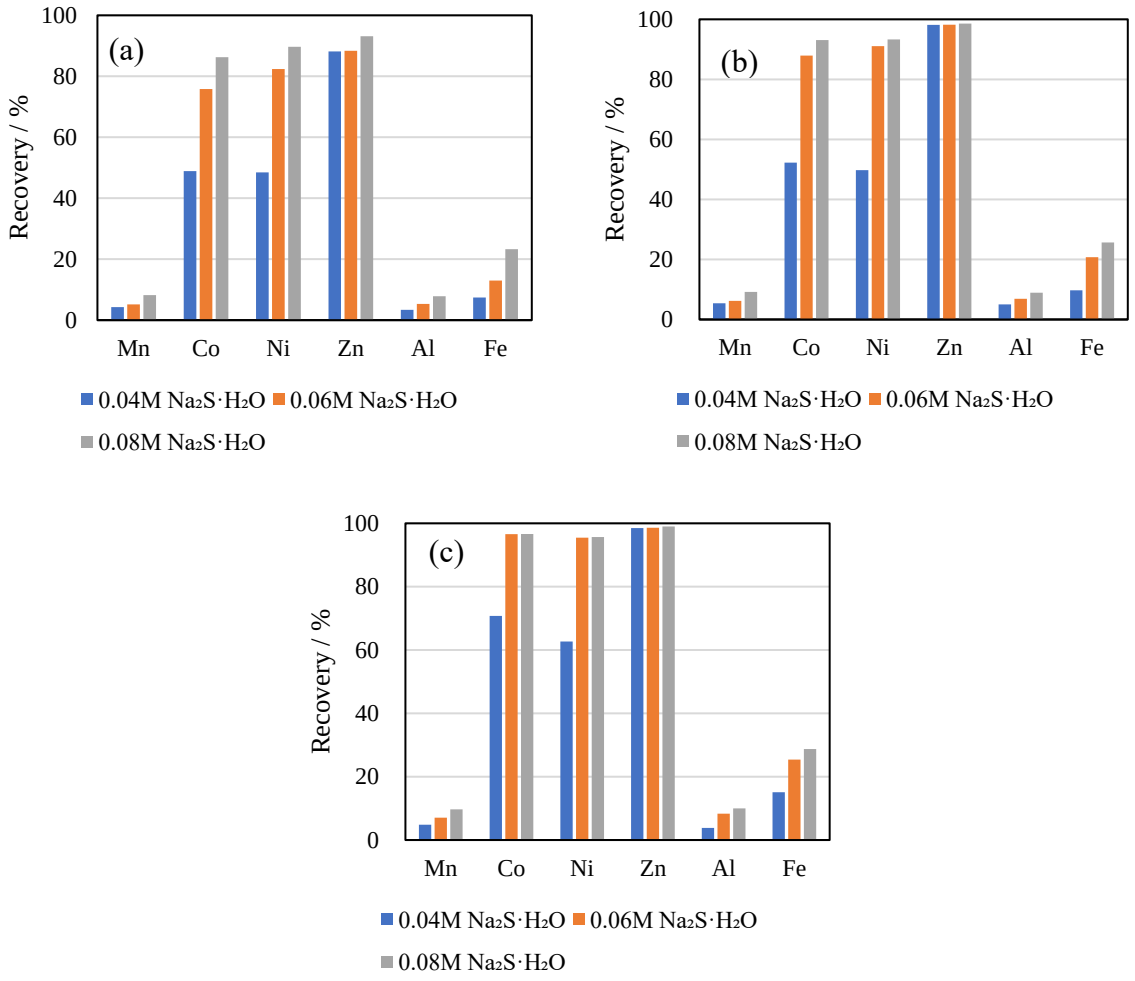



Figure 3.8.49. Effect of sodium sulfide solution concentration on sulfide precipitation recovery of metal ions (a) pH=2 (b) pH=2.5 (c) pH=3.

**Solvent Extraction of the Re-dissolved Solution of the Sulfide Precipitate:** Dissolution of the sulfide precipitate obtained through sulfide precipitation is required since the dissolved solution needs to be subjected to solvent extraction to get a high-grade cobalt product. The sulfide precipitation process was conducted under the optimal conditions as stated in the prior section. The sulfide precipitate was collected through centrifuging and completely dried at 60 °C in an oven. In the last reporting period, the effect of roasting on the dissolution was evaluated. Based on findings from the last reporting period, the sample was first roasted at 200 °C for 2 hours in the present study. Around 97% of the roasted precipitate was dissolved by reacting with 1.2 M HCl at room temperature. The pH of the dissolved solution was then increased to 6.0 by adding a sodium hydroxide solution to precipitate a small amount of Al and Fe existing in the solution.

After raising the solution pH to 6.0, a clear solution was obtained by centrifuging, which served as the feed of the solvent extraction tests. The aqueous solution was mixed with the organic phase in a beaker, which was placed on a magnetic stirrer. The system was continuously mixed for a period of 20 min, which was then transferred into a separating funnel and stood for 5 min. The

aqueous phase was collected from the bottom of the funnel and subjected to elemental analysis using ICP-MS. Recovery of the different metal ions was calculated based on the elemental concentration of the feed and extracted aqueous solutions.

Procedures of two-stage solvent extraction tests using the DEHPA as the extractant are shown in

Figure 3.8.50. Extraction recoveries of Mn, Co, Ni, and Zn from the dissolved solution are presented in Figure 3.8.51(a) and (b). The experiments were conducted at room temperature with an aqueous-organic volumetric ratio (A/O) of 1. As shown in Figure 3.8.51(a), in the first stage of solvent extraction, nearly 100% of Zn and 51% of Mn were recovered using DEHPA at an equilibrium pH of around 3.0, while the recovery of Co and Ni was negligible. At the second stage of solvent extraction with the same extractant, the equilibrium pH was raised to around 4.0, and around 100% of Mn, 63% of Co, and only 27% of Ni were extracted from the aqueous phase obtained from first stage solvent extraction. Co and Ni can be separated to some extent; however, more solvent extraction stages are needed to remove Ni and obtain a solution only containing Co.

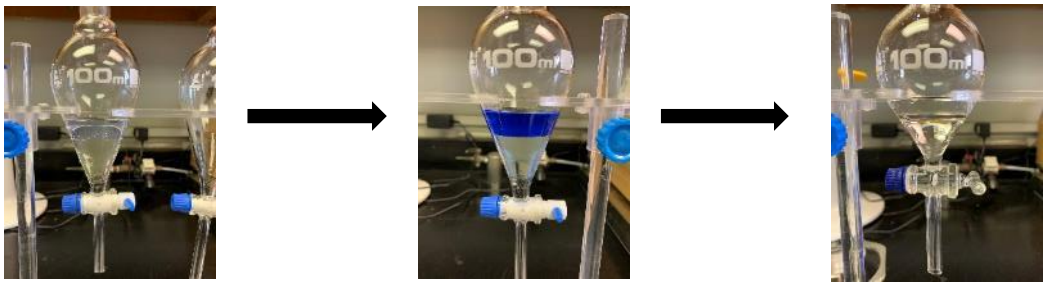



Figure 3.8.50. Procedures of the solvent extraction tests (2 stages).

The stripping of elements from the loaded organic solution was achieved using 0.3 M $H_2SO_4$, and the elemental content in stripped solution is shown in Figure 3.8.51 (c). After the solvent extraction process, a considerable proportion of Mn and nearly all Zn were removed, and a purified solution rich in Co and Ni were obtained. The concentration of Co and Ni in the solution was increased since the impurities were removed. Therefore, with a further sulfide precipitation test conducted on the stripped solution, a precipitate containing $MnO_2$ 22%, CoO 30%, and NiO 40% was obtained (see Figure 3.8.52Figure 3.8.42).

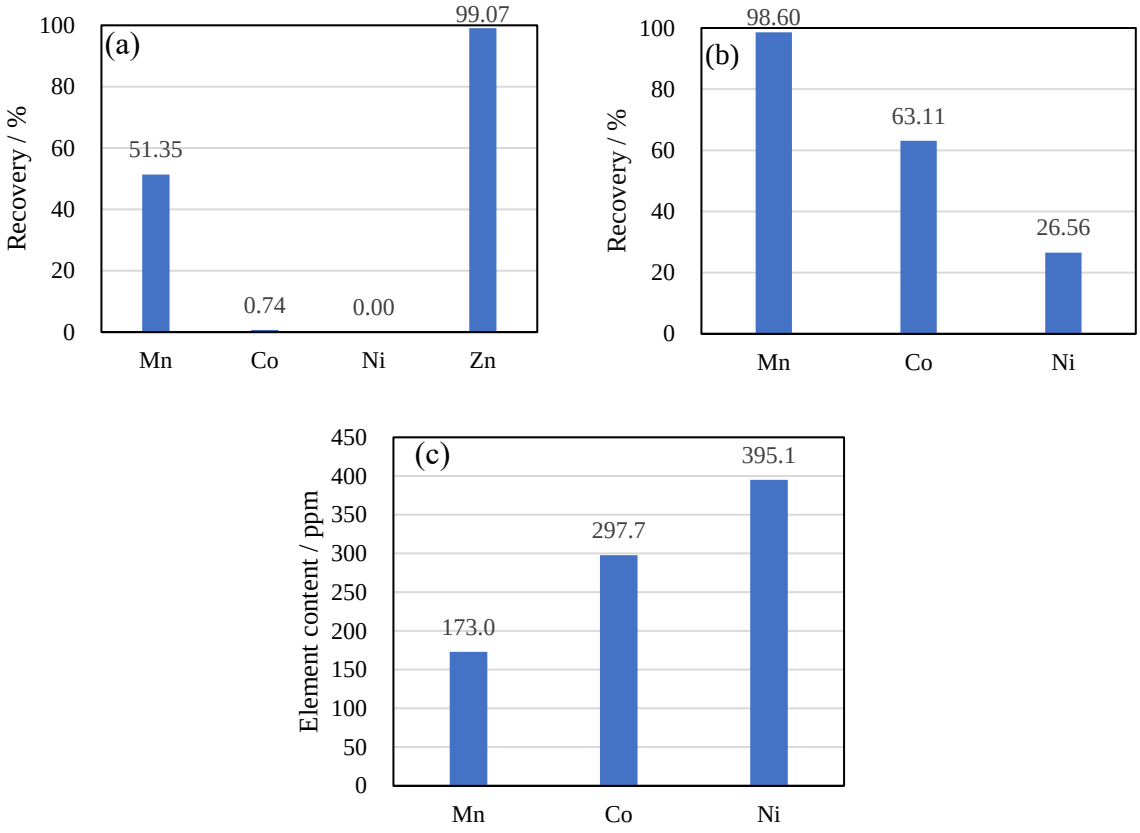



Figure 3.8.51. Recoveries and element content of Mn, Co, Ni, and Zn in solvent extraction and stripping solution (a) Recovery of 1st stage solvent extraction, Eq.pH=2.976; (b) Recovery of 1st stage solvent extraction, Eq.pH=3.033; (c)Elemental content in tripping solution, 0.3M $H_2SO_4$.

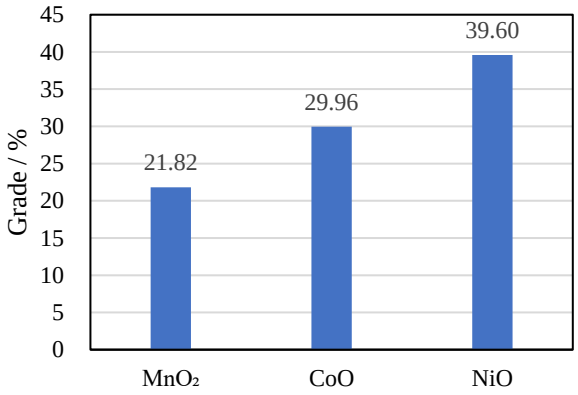



Figure 3.8.52. Elemental content of the alkaline precipitate obtained after solvent extraction.

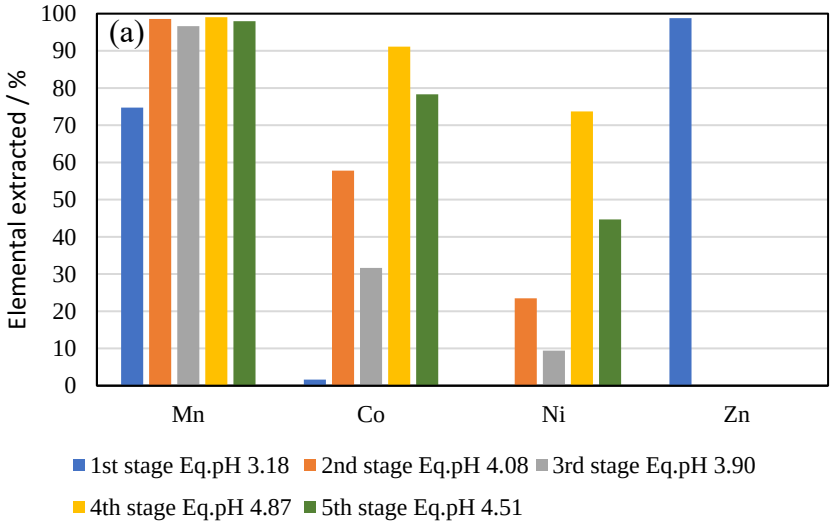
Based on the results obtained from previous solvent extraction tests, more stages for solvent extraction are needed to get a cobalt product with high purity. Therefore, a 5-stages solvent extraction test was conducted on the solution prepared by redissolution of a sulfide precipitate followed by Fe and Al removal through pH increases to 6.0. The 5-stages solvent extraction test procedures are shown in figure below..



| 1st stage | 2ed stage | 3rd stage | 4th stage | 5th stage |
|---|---|---|---|---|
| 0.135M DEHPA | 0.135M DEHPA | 0.135M DEHPA | 0.135M DEHPA | 0.135M DEHPA |
| Eq. pH 3.18 | Eq. pH 4.08 | Eq. pH 3.90 | Eq. pH 4.50 | Eq. pH 4.51 |

Figure 3.8.53. Procedures of the solvent extraction tests (5 stages).

Mn, Co, Ni, and Zn were the dominant elements in the solution, and their extraction recoveries are shown in Figure 3.8.54(a). As shown in Figure 3.8.54 (a), in the first stage of solvent extraction, nearly 100% of Zn and 75% of Mn were recovered using DEHPA at an equilibrium pH of around 3.2, while the recovery of Co and Ni was negligible. In the following four stages of solvent extraction, the recovery of Co and Ni was varied due to the different equilibrium pH values. The recovery of Co was only 32% at an equilibrium pH of 3.90. When raising the equilibrium pH value to 4.08, the recovery of Co reached 58%, which further increased to 92% when the equilibrium pH was elevated to 4.87. The recovery of Ni was also increased when raising the equilibrium pH; however, it was always lower than that of Co in the same stage, indicating that Co and Ni can be effectively separated. Stripping was conducted on the loaded organic phase after the second stage and the subsequent (3rd, 4th, and 5th) stages of solvent extraction, and more than 80% of the Mn, Co, and Zn loaded in the organic phase were stripped using 0.3 M $H_2SO_4$ (see Figure 3.8.54(b)).



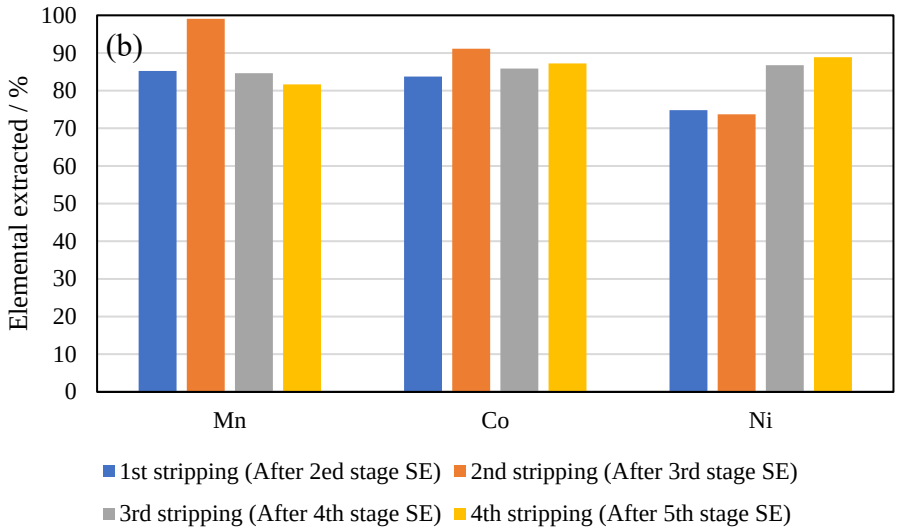



Figure 3.8.54. Recovery of the metal ions obtained during multi-stage solvent extraction and stripping process (a) Elemental recovery loaded in the organic phase; (b) Percentage of ions stripped by 0.3M $H_2SO_4$.

Sulfide precipitation was conducted on the stripped solution. Elemental concentrations of the solution before and after precipitation is shown in Figure 3.8.55. Due to the selectivity of sulfide precipitation, the concentration of Mn was barely unchanged, while most Co was precipitated. Based on concentration differences of the solution before and after precipitation, elemental contents of the sulfide precipitate generated from the solution were estimated. As shown in Figure 3.8.56, a high purity cobalt product with a purity of 92% was expected to obtain; however, the cobalt sulfide precipitate was insufficient to run digestion and elemental content analysis.

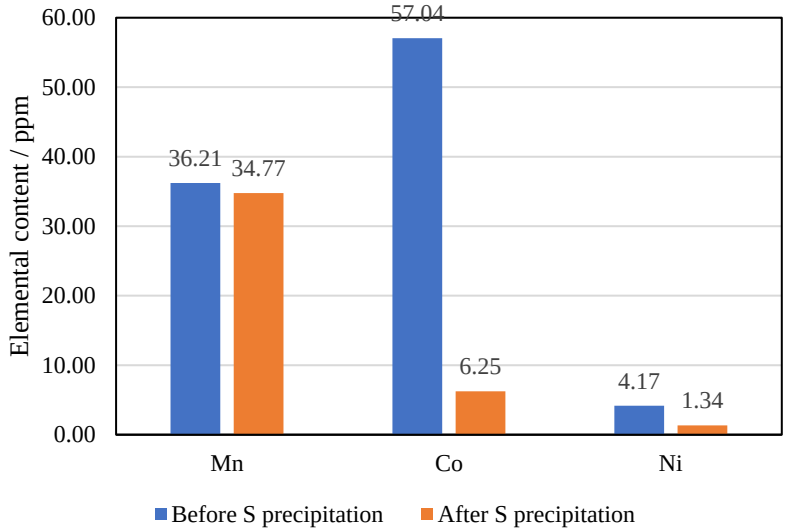



Figure 3.8.55. Elemental content of the solution before and after sulfide precipitation ("S" represents sulfide).

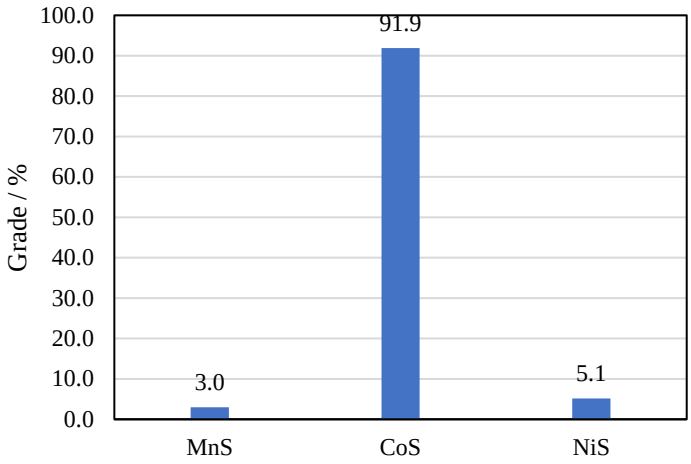



Figure 3.8.56. Elemental content of the sulfide precipitate conducted after solvent extraction.

A high-purity product containing CoS of 92% was obtained from the pre-concentrated precipitate produced in the pilot plant, suggesting that the efficient separation of Co from Mn, Ni, and Zn can be achieved by combining sulfide precipitation with multi-stage solvent extraction. To verify the results, similar purification tests were performed on a pre-concentrated slurry generated from an acid mine leachate through staged precipitation in the laboratory. The procedures used to prepare a sulfide-dissolved solution from pre-concentrated slurry are shown in below.

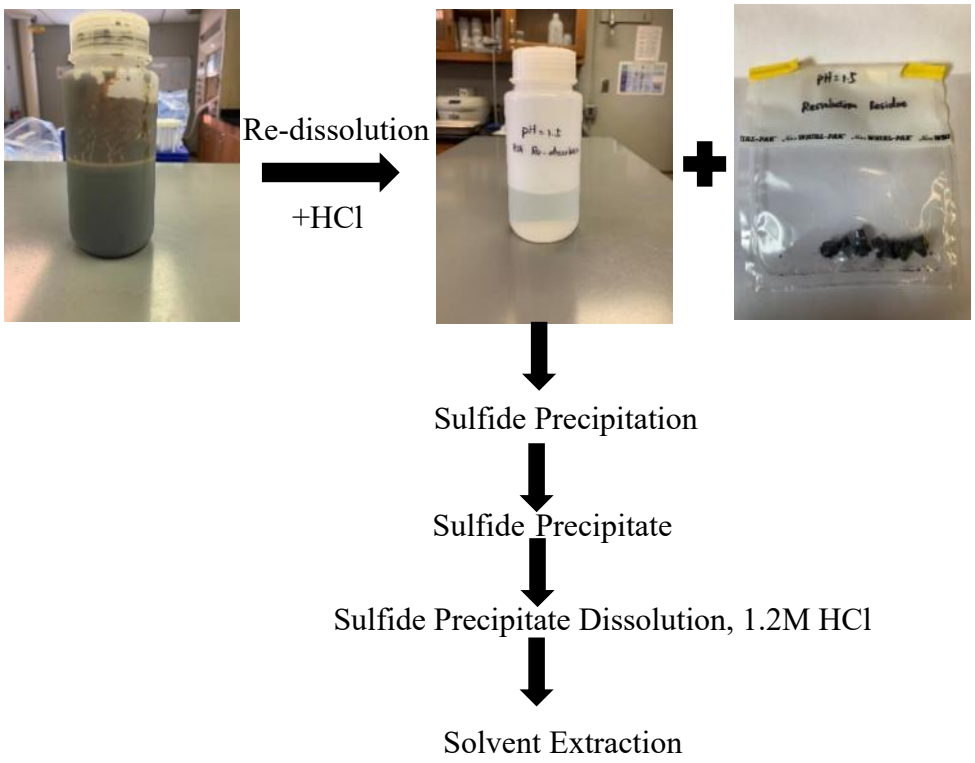
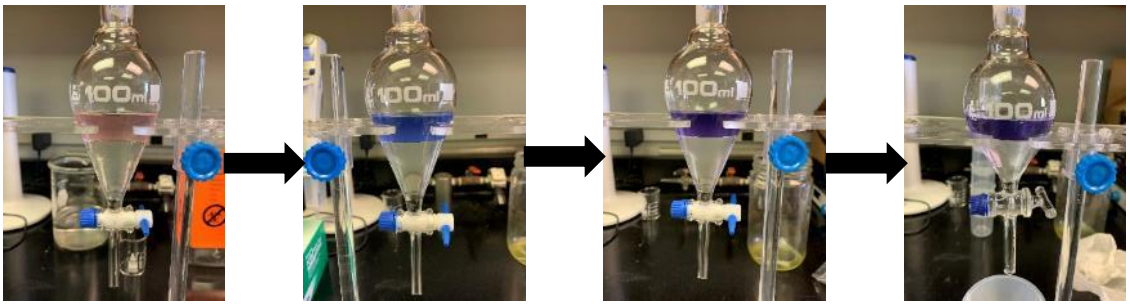


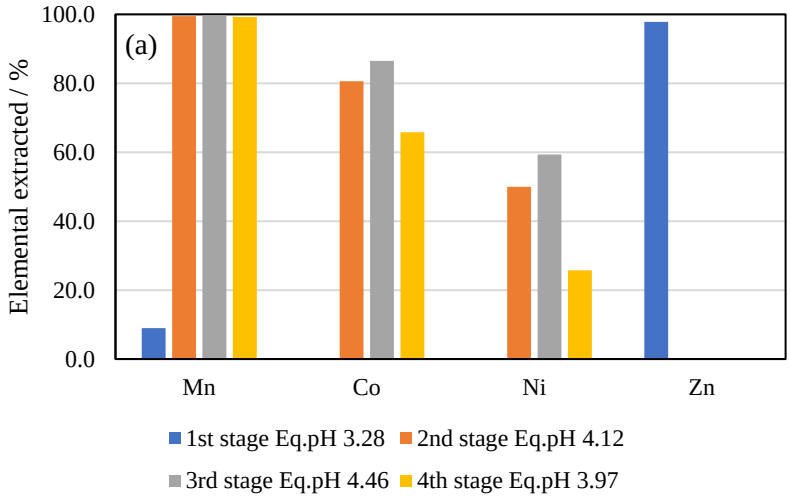

Figure 3.8.57. Procedures of preparing the sulfide-dissolved solution with pre-concentrated slurry obtained from an acid mine leachate.

A 4-stages solvent extraction test was conducted on the sulfide-dissolved solution. The procedures are shown in Figure 3.8.58. As shown in Figure 3.8.59, in the first stage of solvent extraction, nearly 100% of Zn was recovered using 0.135 M DEHPA at an equilibrium pH of 3.28, while the recovery of Mn was only 9%. The reason for this phenomenon could be that the concentration of Zn in the sulfide-dissolved solution was around 3000 ppm, and 0.135 M DEHPA was insufficient to remove both Zn and Mn. Therefore, the concentration of DEHPA needs to be improved, and the equilibrium pH also needs to be optimized to remove Zn and Mn as much as possible in the first stage of solvent extraction. In the subsequent three stages of solvent extraction, the recovery of Co and Ni was varied due to the different equilibrium pH values. The recovery of Co was only 65.8% at the equilibrium pH of 3.98. When raising the equilibrium pH to 4.12, the recovery of Co was 80.6%, and it was still lower than 90% when the equilibrium pH was 4.46. Stripping was conducted on the loaded organic phase after 2nd, 3rd, and 4th stage solvent extraction. More than 85% of the Mn, Co, and Zn loaded in the organic phase were extracted using 0.3 M $H_2SO_4$ (see Figure 3.8.59(b)).



| 1st stage | 2nd stage | 3rd stage | 4th stage |
|---|---|---|---|
| 0.135M DEHPA | 0.135M DEHPA | 0.135M DEHPA | 0.135M DEHPA |
| Eq. pH 3.28 | Eq. pH 4.12 | Eq. pH 4.46 | Eq. pH 3.97 |

Figure 3.8.58. Procedures of the solvent extraction tests (4 stages).



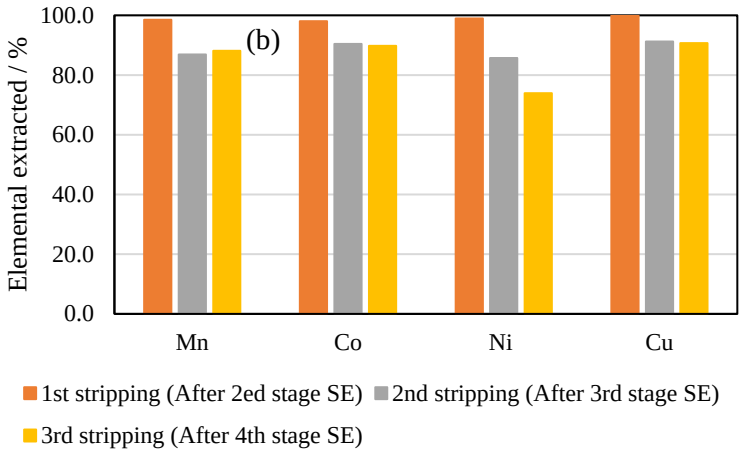



Figure 3.8.59. Recovery of the metal ions obtained during multi-stage solvent extraction and stripping process (a) Elemental recovery loaded in the organic phase; (b) Percentage of ions stripped by 0.3M $H_2SO_4$.

Sulfide precipitation was conducted on the stripped solution, and elemental contents of the solution before and after the sulfide precipitation test are shown in Figure 3.8.60. Because Mn was not completely removed in the first stage of solvent extraction, the concentration of Mn was significantly higher than other elements. A certain amount of Zn of 63.4ppm remained in the solution, which also significantly influenced the purity of the final product. The elemental contents shown in Figure 3.8.61 were calculated based on the difference in elemental concentration of the stripped solution before and after precipitation (see Figure 3.8.60). Cobalt content in the product was only 26%, which was below the value obtained from the pre-concentrated precipitate obtained from the pilot plant. Based on a careful analysis of the data, it was found that most Mn and Zn need to be removed in the first stage of solvent extraction at a proper equilibrium pH, and in order to recover most of the Co in the following stages of solvent extraction, the equilibrium pH also needs to be optimized.

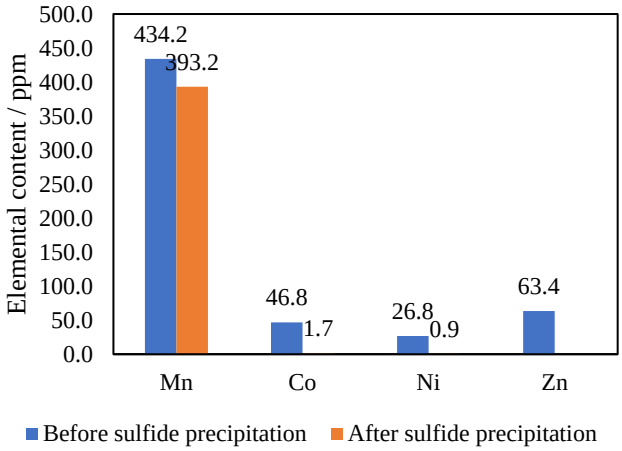



Figure 3.8.60. Elemental content of the solution before and after sulfide precipitation.

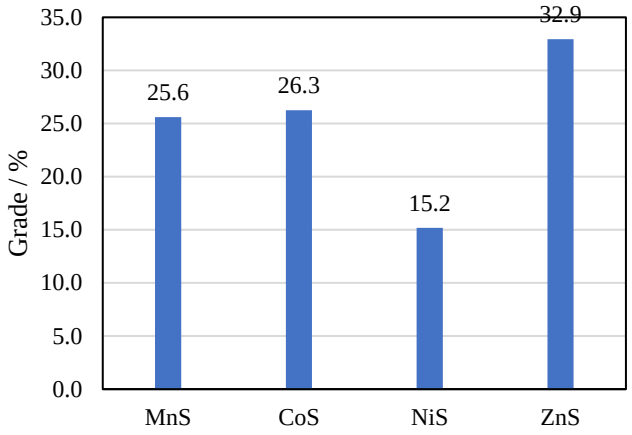
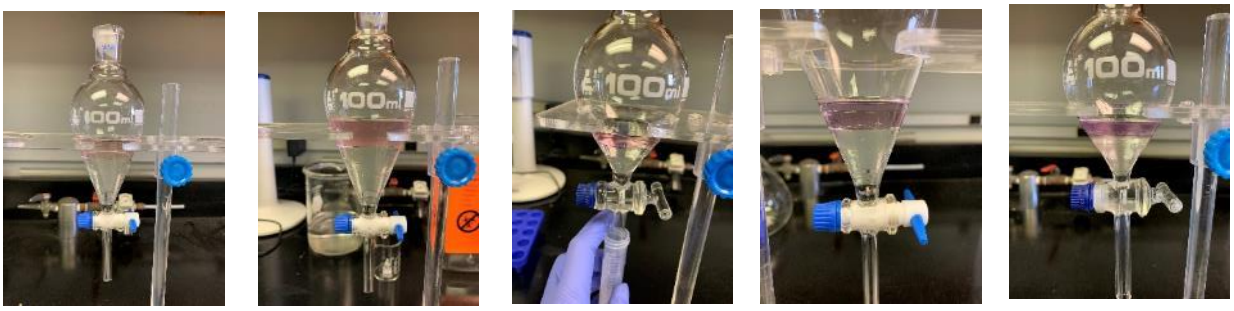



Figure 3.8.61. Elemental content of the sulfide precipitate conducted after solvent extraction.

Procedures of the solvent extraction pH optimization tests using DEHPA as the extractant are shown in

Figure 3.8.62, and the extraction recoveries of Mn, Co, Ni, and Zn from the solution are presented in Figure 3.8.63. As stated in the last section, equilibrium pH is crucial for achieving a high recovery of Mn and Zn in the first stage of solvent extraction, which partially determines the grade of Co in the final product. As shown in Figure 3.8.63, the recovery of Zn and Mn was increased when raising the equilibrium pH in the range of 3 to 4, while the recovery of Co and Ni is negligible. Thereby, the optimum equilibrium pH for the first stage solvent extraction is estimated to be close to 4 to get as much as Mn and Zn extracted.



0.135M DEHPA
Eq. pH 3.00

0.135M DEHPA
Eq. pH 3.28

0.135M DEHPA
Eq. pH 3.55

0.135M DEHPA
Eq. pH 3.65

0.135M DEHPA
Eq. pH 3.86

Figure 3.8.62: Procedures of the solvent extraction pH optimization tests.

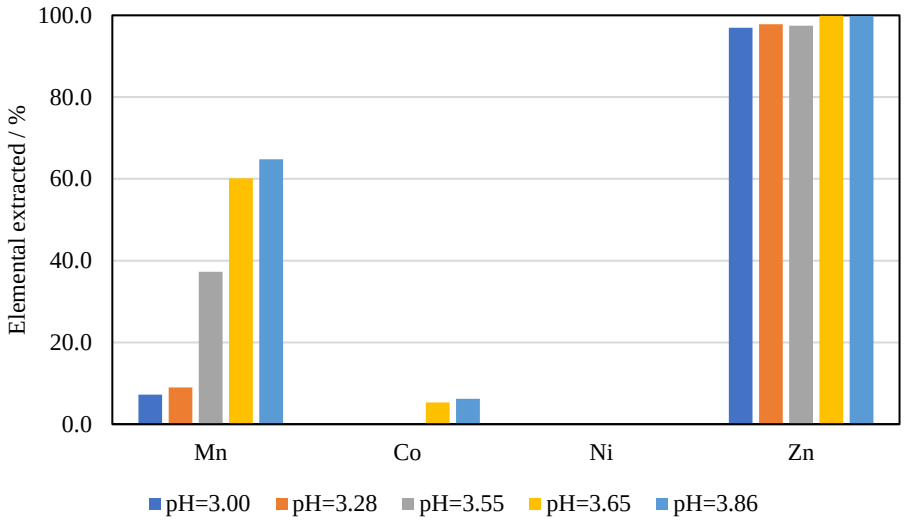
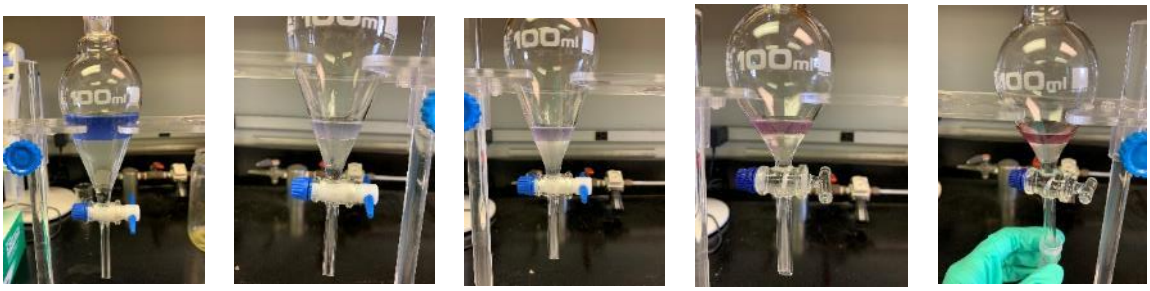



Figure 3.8.63: Recovery of the metal ions obtained during solvent extraction in the pH range of 3 to 4.

Before conducting the second stage solvent extraction, the first stage solvent extraction was carried out with the optimized equilibrium pH. In the second stage of solvent extraction with DEHPA, the equilibrium pH was raised in the range of 4.0 to 5.0 (see Figure 3.8.64). As shown in Figure 3.8.65, the separation of Co and Ni was not satisfactory when raising the equilibrium pH to close to 5. To separate Co and Ni more effectively, the equilibrium pH should be controlled at around 4.0. A few other types of extractants will be tested in the next reporting period to obtain a high-purity cobalt product.



| 0.135M DEHPA Eq. pH 4.12 | 0.135M DEHPA Eq. pH 4.44 | 0.135M DEHPA Eq. pH 4.53 | 0.135M DEHPA Eq. pH 4.67 | 0.135M DEHPA Eq. pH 4.78 |
|---|---|---|---|---|

Figure 3.8.64: Procedures of the solvent extraction pH optimization tests.

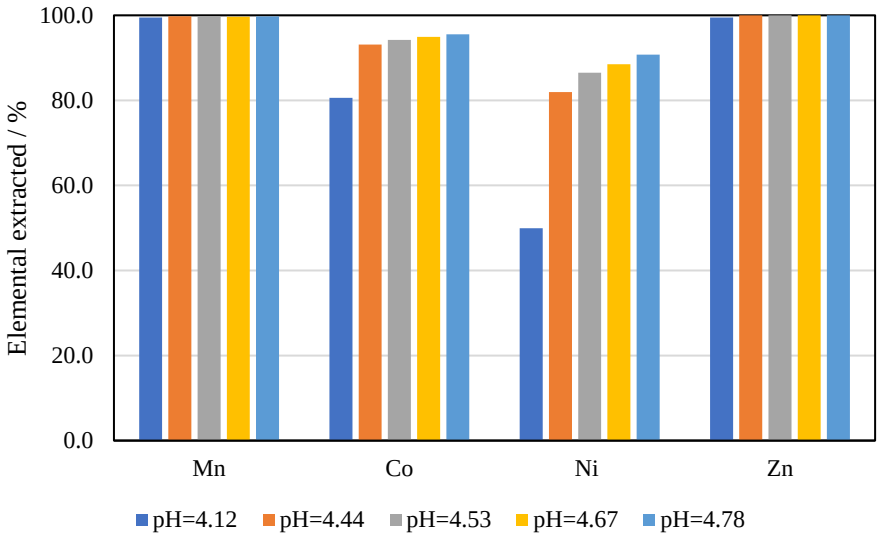



Figure 3.8.65: Recovery of the metal ions obtained during solvent extraction in the pH range of 4 to 5.

A synthetic solution with major constituents similar to the real materials was prepared as the feed of the solvent extraction tests, which contained 400 ppm Mn, 80 ppm Co, 400 ppm Ni, and 2100 ppm Zn. The aqueous solution was mixed with the organic phase in a beaker, which was placed on a magnetic stirrer. The experiments were conducted at room temperature with an aqueous to the organic volumetric ratio (A/O) of 1. The pH adjustments were performed by introducing NaOH dropwise using a pipette until the desired equilibrium pH was attained. A concentrated NaOH of 19 M was used to change the volume of the system minimally. After the equilibrium state was achieved, the solution was kept stirring for 20 minutes to stabilize the pH. As shown in Figure 3.8.66, the separation of the aqueous and organic phase was then carried out using a separating funnel based on the specific gravity difference between the organic and the aqueous solution. The aqueous phase was collected from the bottom of the funnel and subjected to elemental analysis using ICP-MS. The percent of the different metal ions extracted by the organic phase was calculated based on the elemental concentration of the feed and extracted aqueous solutions.

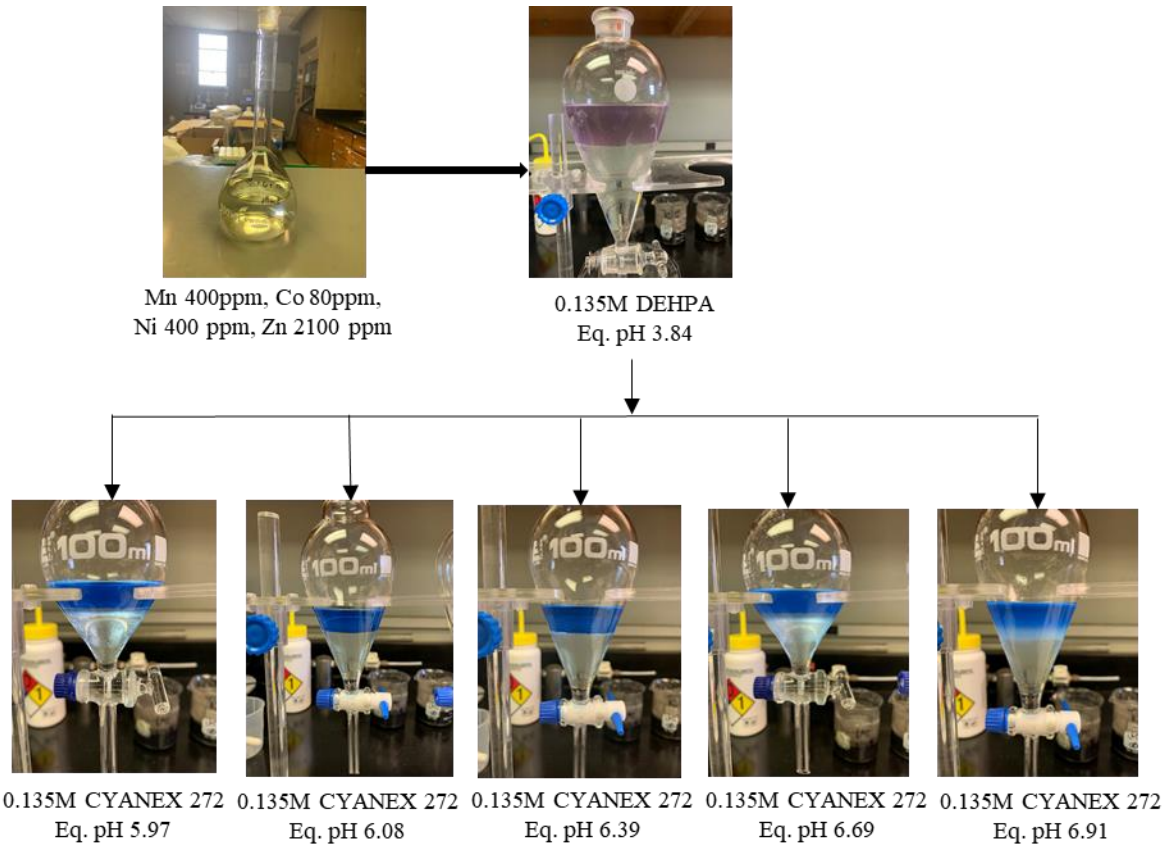



Figure 3.8.66. Procedures of the Cyanex 272 solvent extraction pH optimization tests.

The equilibrium pH for Mn and Zn extraction in the first stage solvent extraction using D2EHPA was optimized, which is in the range of 3 to 4. As shown in Figure 3.8.67, around 100% of Zn and 70% of Mn were extracted from the solution at an equilibrium pH of 3.84. D2EHPA was also applied in the second stage solvent extraction to separate Co and Ni in the last reporting period; however, it was found that the D2EHPA cannot effectively separate Co and Ni. Therefore, the effect of another extractant, Cyanex 272, was tested in the second stage solvent extraction to separate Co and Ni. Since Zn was completely extracted in the 1$^{st}$ stage of solvent extraction, its recovery was not considered in the second stage of solvent extraction. As presented in Figure 3.8.68, 100% of Mn and Co were extracted from the solution in the pH range of 6 to 7, and the equilibrium pH did not influence the recoveries of Mn and Co; however, the recovery of Ni was increased when raising the equilibrium pH. At an equilibrium pH of 5.97, around 40% of Ni was extracted, and when elevating the equilibrium pH value to 6.91, the recovery of Ni reached 98%.

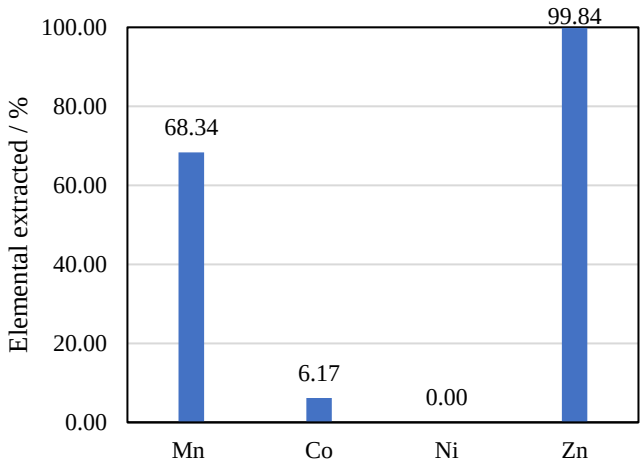



Figure 3.8.67: Extraction recoveries of the metal ions using 0.135M DEHPA during 1st stage solvent extraction (Equilibrium pH=3.84, A/O=1).

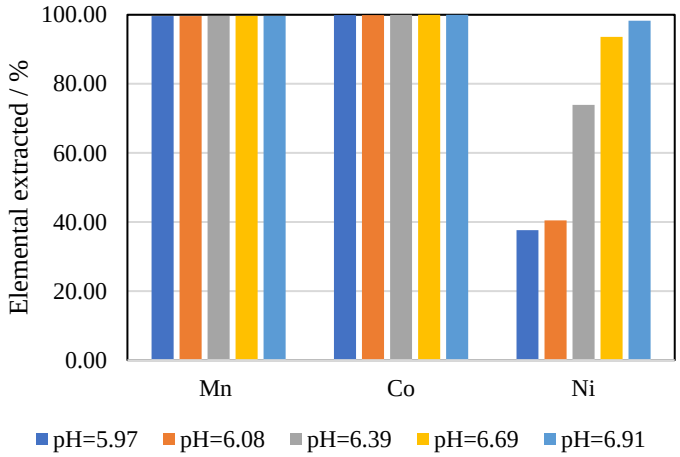



Figure 3.8.68: The effect of equilibrium pH on extracting metal ions using Cyanex 272 ($C_{Cyanex\ 272}$=0.135M, A/O=1).

To further optimize the separation of Co and Ni using Cyanex 272, the tests were conducted in the pH range of 5 to 6, and the results are shown in Figure 3.8.69. In the pH range of 5 to 6, Mn and Co were still effectively extracted by 0.135M Cyanex 272; however, comparing the results obtained in the pH range of 6 to 7 (Figure 3.8.68), and the separation efficiency of Co and Ni was improved. When the equilibrium pH was 5.14, the recovery of Co reached 100%, while the recovery of Ni was only around 7%. Increasing the equilibrium pH also increased the recovery of Ni, with around 40% of Ni being extracted at pH 5.94. Thereby, to effectively separate Co and Ni using Cyanex 272, the optimum pH should be in the range of 5 to 6.

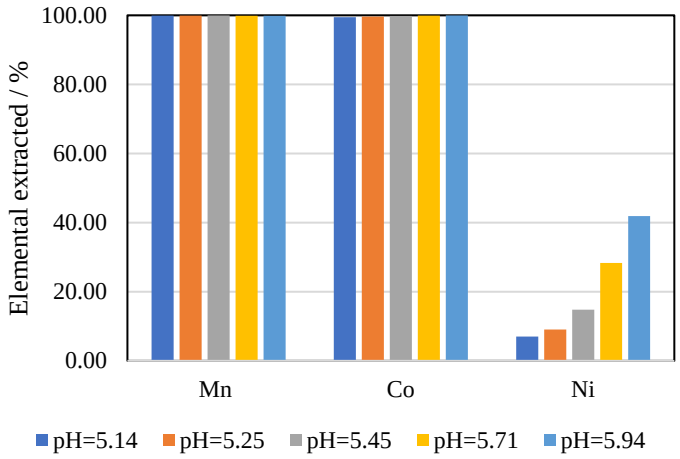



Figure 3.8.69: The effect of equilibrium pH on extracting metal ions using Cyanex 272 ($C_{Cyanex\ 272}$=0.135M, A/O=1).

The concentration of extractant is another factor that could significantly influence the recovery of metals ions. In previous tests, it was found that around 70% Mn and 100% Zn were extracted using 0.135 M DEHPA at pH 3.84. To get more Mn extracted, the concentration of D2EHPA was increased to 0.2 M, 0.5 M, 1 M, and 2 M, respectively, and the equilibrium pH was maintained at around 3.8. As presented in Figure 3.8.70, the recovery of Mn was improved with increasing the concentration of DEHPA; however, Co and Ni were also extracted with D2EHPA solutions of higher concentrations. Thereby, 0.135 M D2EHPA was the optimum condition for extracting Mn and Zn without losing Co and Ni.

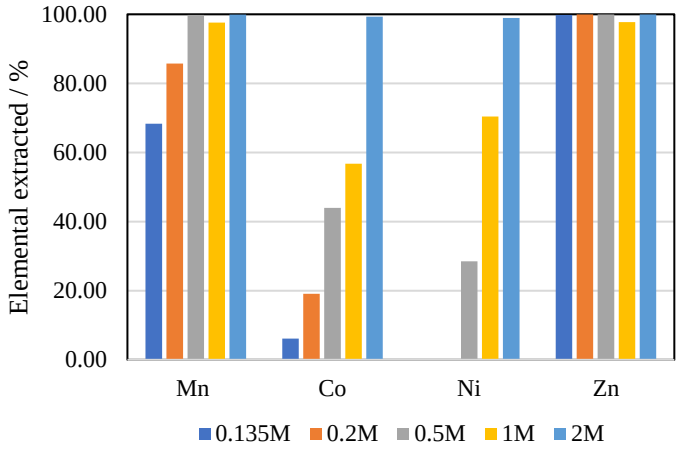



Figure 3.8.70: The effect of DEHPA concentration on extracting metal ions (Equilibrium pH around 3.8, A/O=1).

The effect of the aqueous to organic volumetric ratio (A/O) of using D2EHPA in the first stage solvent extraction was investigated. In previous tests, the aqueous to organic volumetric ratio was fixed at 1. Based on the results shown in Figure 3.8.71, it was found that increasing the A/O ratio above 1 caused less Mn extracted, and decreasing the A/O ratio to 0.5 got Co and Ni extracted as

well. Therefore, the optimum A/O ratio for using 0.135 M D2EHPA in the first stage solvent extraction was 1.

The effect of the aqueous to organic volumetric ratio (A/O) for using Cyanex 272 in the second stage solvent extraction was also investigated (see Figure 3.8.72). Mn and Co were completely extracted from the solution with the aqueous to organic volumetric ratios of 0.5, 1, and 1.5. The recovery of Ni decreased with increasing the A/O ratio from 0.5 to 1.5. Around 20% of Ni was extracted from the solution when the A/O ratio was 0.5, while the recovery of Ni decreased to 5.7% with an A/O ratio of 1.5. Further increasing the aqueous to organic volumetric ratio to 2 led to a decrases in Co recovery to 80%. It can be concluded that choosing the A/O ratio of 1.5 for using 0.135 M Cyanex 272 in the second stage solvent extraction could effectively separate Co and Ni.

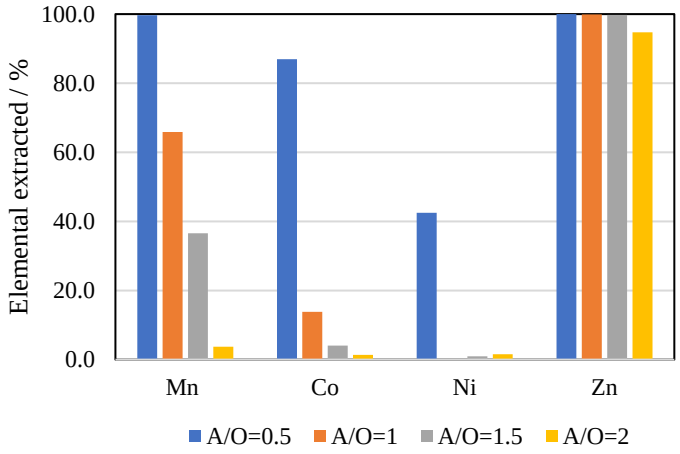



Figure 3.8.71: The effect of the aqueous to organic volumetric ratio (A/O) on extracting metal ions using DEHPA ($C_{D2EHPA}$=0.135 M, Equilibrium pH around 3.7).

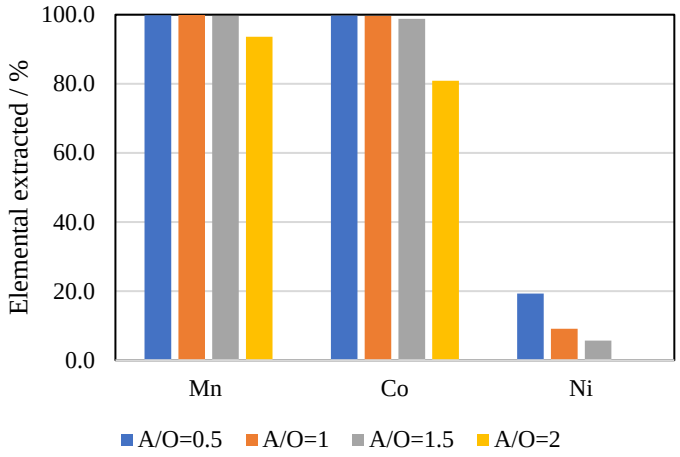



Figure 3.8.72: The effect of the aqueous to organic volumetric ratio (A/O) on extracting metal ions using Cyanex 272 ($C_{Cyanex}$=0.135 M, Equilibrium pH around 5.2).

**Production of high purity product:** Based on the results obtained from optimization tests, a two-stage solvent extraction process was conducted with the volumetric ratio of aqueous to organic phase (A/O) of 1 and the extractant dosage ($C_{Solvent}$) of 0.135 M. After extraction, Ni was in the aqueous phase. The loaded organic phase was stripped with a 0.3 M $H_2SO_4$ solution, leading to a Co-enriched solution. To generate high-purity Co and Ni products, sulfide and alkaline precipitation were performed on the solutions. The optimized sulfide precipitation conditions were determined in previous tests. Ni in the aqueous phase was precipitated by adding 5 M NaOH to increase the pH to 10.00. The elemental contents of Co and Ni in the corresponding precipitate were calculated based on the elemental composition of the feed solution and the raffinate after precipitation.

Based on the aforementioned results, a two-stage solvent extraction test was conducted on the solution obtained by dissolving the calcined sulfide precipitate. In the first stage, 100% of Zn and 70% of Mn were removed from the solution with 0.135 M D2EHPA at an equilibrium pH of 3.32. Co and Mn were extracted by 0.135 M Cyanex 272 at an equilibrium pH of 5.39 in the second stage while leaving Ni in the aqueous phase. As shown in Figure 3.8.73(a), Ni was the only element in the aqueous phase after the second-stage solvent extraction, and the concentration of Mn, Co, and Zn was less than 1 mg/L. Co and Mn were in the organic phase; thereby, the organic phase was stripped with 0.3 M $H_2SO_4$, and elemental concentrations in the stripping solution are shown in Figure 3.8.73(b). For producing a high-purity Co product, further treatment is needed to separate Mn and Co.

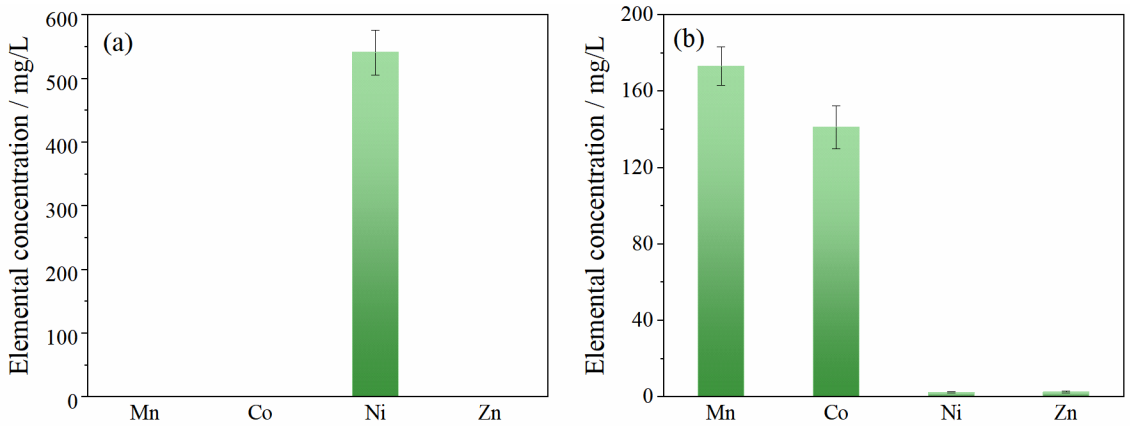



Figure 3.8.73. Elemental concentrations of (a) the aqueous phase after two-stage solvent extraction and (b) the stripping solution.

After the two-stage solvent extraction, Ni in the aqueous phase was precipitated by adding 5 M NaOH to increase the pH to 10.00. Based on elemental concentrations in the solution before and after precipitation (Table 3.8.6), it can be concluded that more than 99% of the Ni was precipitated, and the purity of the products was nearly 100% since Ni was the only valuable element detected in the solution. To produce a high-purity Co product, Mn and Co in the stripping solution were separated by sulfide precipitation, which was conducted at pH 4.00 with a sulfur to metal molar ratio of 1. Based on the elemental concentrations shown in Table 3.8.7, it could be calculated that more than 90% of Co was recovered from the solution; whereas, only 5% of Mn was precipitated. In addition, based on elemental concentrations in the solution before and after precipitation, the purity of the Co sulfide product was estimaited to be around 93.6%.

Table 3.8.6. Elemental concentrations of Ni in the aqueous phase before and after precipitation.

| Element | Ni |
|---|---|
| Concentration (before precipitation, mg/L) | 575.90 |
| Concentration (after precipitation, mg/L) | 2.25 |

Table 3.8.7. Elemental concentrations of Mn and Co in the solution before and after sulfide precipitation.

| Element | Mn | Co |
|---|---|---|
| Concentration (before precipitation, mg/L) | 173.18 | 141.20 |
| Concentration (after precipitation, mg/L) | 164.46 | 14.14 |

A process flowsheet was developed to generate high-purity products of certain critical elements (i.e., Mn, Co, and Ni) from AMD. As Figure 3.8.74 shows, the primary contaminants, including Fe and Al, are initially removed by staged precipitation at pH 4.00 and 6.50, respectively. By further increasing the solution pH, a pre-concentrate of Mn, Co, Ni, and Zn is obtained in the pH range of 6.50-10.00. A slurry is then prepared by mixing the pre-concentrate with a small volume of the raffinate obtained from the third-stage precipitation. Around 100% of Ni and Zn, 85% of Co, and 45% of Mn are dissolved into solution by adding hydrochloric acid to decrease the pH of the slurry to approximately 5.00. The undissolved residue obtained from the re-dissolution process is an Mn-enriched product with high purity (30 wt.% Mn content). The solution containing re-dissolved elements was routed to a sulfide precipitation process to further separate Mn from the other critical elements. The sulfide precipitate is calcined and dissolved in hydrochloric acid for further purification using solvent extraction. The solvent extraction is performed in two stages, and D2EHPA and Cyanex 272 are used in the first- and second-stage, respectively. In the first stage, around 70% of Mn and 100% of Zn in the feed solution are extracted, resulting in a solution comprised of Co, Ni, and a small amount of Mn. In this case, the second stage solvent extraction is applied to separate Co and Ni. Co and Mn in the organic phase are stripped with sulfuric acid and further separated with sulfide precipitation to obtain a cobalt product. Ni in the aqueous phase is recovered by raising the pH to 10.00. Ni and Co products with ash high as 100% and 94% purity can be obtained using this flowsheet.

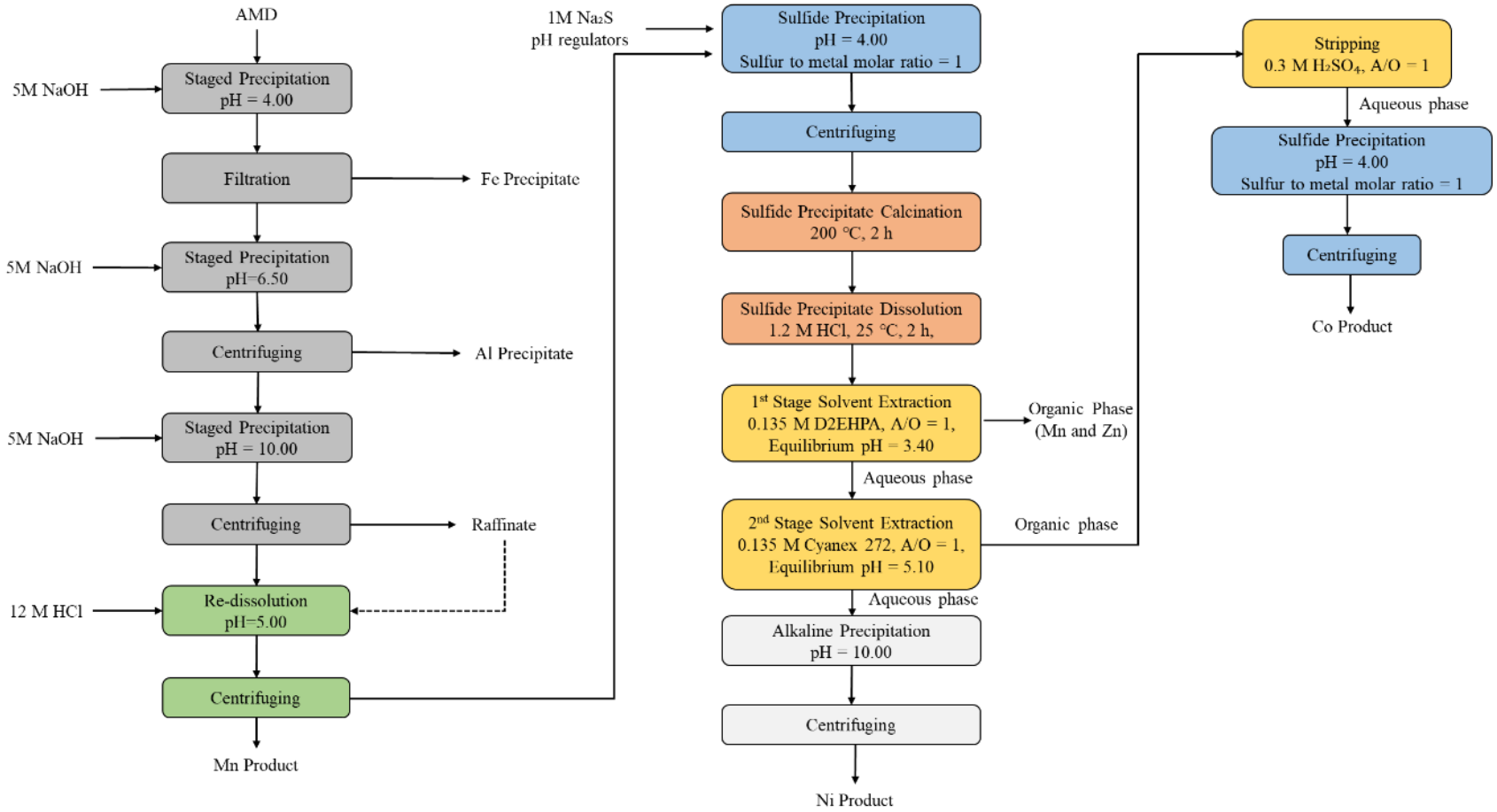



Figure 3.8.74. Flowsheet for generating high-purity products of certain critical elements (i.e., Mn, Co, and Ni) from AMD.

Around 100 lbs of Co/Mn sludge (see Figure 3.8.75) produced from coal-related materials was collected from the pilot plant, which served as the feed to produce the bulk of Mn, Co, and Ni high purity products. The liquid content in the sludge is about 90%. The elemental content of the sludge was analyzed, which is shown in Figure 3.8.76. Compared to the slurry prepared in the lab, the Mn/Co sludge produced in the pilot plant had a higher content of Fe and Al because it is difficult to accurately control pH in actual production.



Figure 3.8.75. Mn/Co sludge collected from pilot plant.

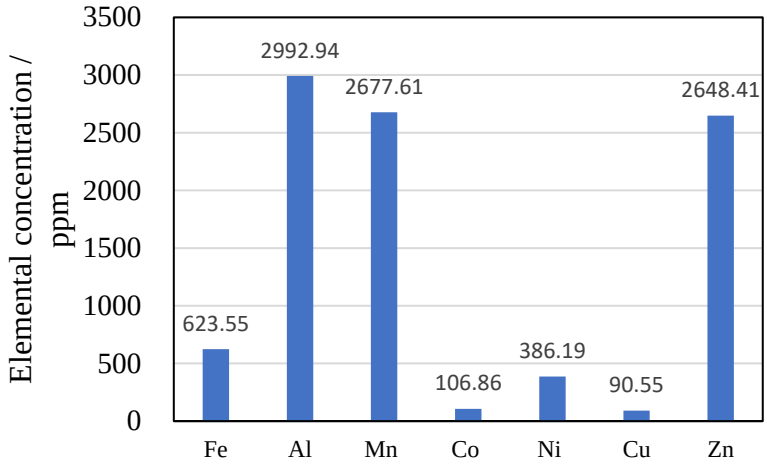



Figure 3.8.76. Elemental concentrations of the Mn/Co sludge.

Based on the developed process, a preliminary re-dissolution test was conducted with 400 g of the sludge. The sludge was re-dissolved by adding HCl. The Dissolubility of the elements is shown in Figure 3.8.77. The results of the test were consistent with the previous experimental results. The critical elements started to dissolve at pH 7.00, and a considerable dissolution occurred within the pH range of 7.00 to 3.00. However, to avoid Al and Fe from dissolving into the solution and affecting the purity of the product, pH=4.00 was chosen as the final pH for the re-dissolution process. With the results obtained from the preliminary test, 3000 g sludge was re-dissolved, and 2 L of the re-dissolved solution was obtained. The elemental content of the re-dissolved solution is shown in Figure 3.8.78. The solution was mainly enriched in Mn, Co, Ni, and Zn, and the content of Al and Fe was significantly decreased compared to the feed sludge.

After the re-dissolution, a sulfide precipitation test was conducted on the re-dissolved solution at pH 4.00 with sulfur to metal molar ratio of 1. Based on the elemental concentrations shown

in Figure 3.8.79, it could be calculated that almost 100% of Co, Ni, and Zn but only around 17% of Mn were precipitated. Co, Ni, and Zn were effectively separated from the Mn. The sulfide precipitate was dried and calcined at 200 ℃ for 2 h. The calcined precipitate was dissolved in 1.2 M HCl for further purification. Solvent extraction will be conducted on the dissolved solution in the following report period.

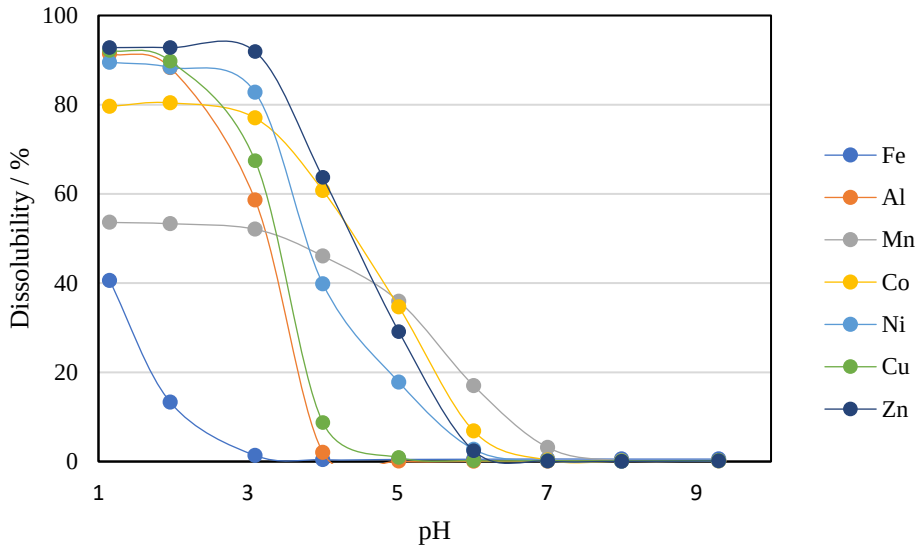



Figure 3.8.77. Dissolubility of the elements in the sludge.

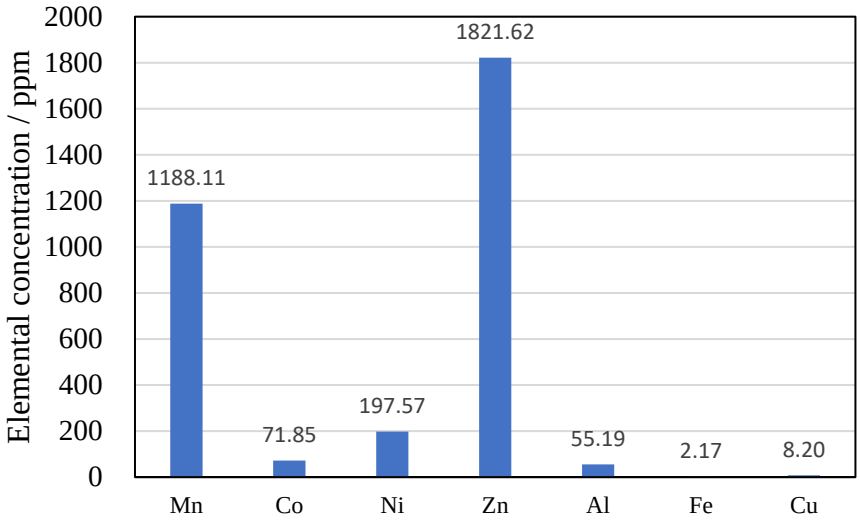



Figure 3.8.78. Elemental concentrations of the re-dissolved solution.

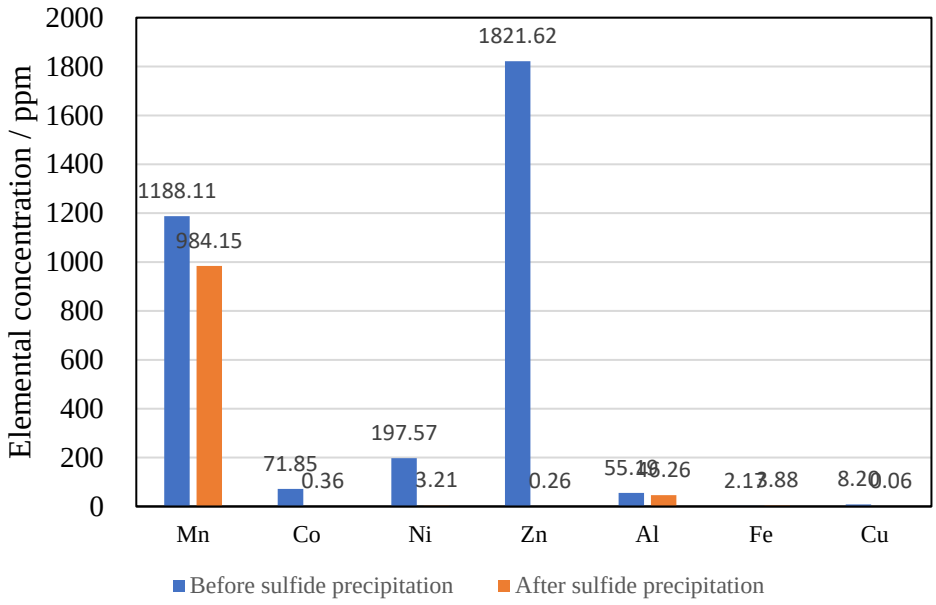



Figure 3.8.79. Elemental concentration before and after the sulfide precipitation.

### *3.8.3 Mineralogical Characterization*

**Materials***:* A representative sample of the feed to a coal preparation plant treating the Fire Clay coal was collected in eastern Kentucky, USA. The sample was air-dried and then dry sieved using a sieve with an aperture of 10 mm. The fraction coarser than 10 mm was subjected to float-sink tests using a dense medium bath of 1.4 specific gravity, which was prepared by mixing magnetite powder finer than 48 µm with tap water. After separation, the float material was rinsed with tap water to remove the entrained magnetite particles, then air-dried at room temperature, and finally pulverized to a top particle size of 0.18 mm using a jaw crusher and a hammer mill. Proximate analysis showed that the pulverized material had an ash content of 5.31%. The material was used as feeds for low temperature plasma (LTP) treatment process. Chemicals used for the tests include nitric acid (HNO3), hydrochloric acid (HCl), and hydrofluoric acid (HF). All the acids were of trace metal grade and purchased from Thermo Fisher Scientific, USA. Deionized water with a resistivity of 18.2 MΩ·cm at room temperature was used for all experiments.

**Methods:** *Low Temperature Plasma Treatment*: A benchtop low temperature plasma system (PE-100 RIE, Plasma Etch Inc, Carson City, Nevada, USA) was used to peel away the organic matter and liberate the REEs from the coal organic matrix. An aluminum vacuum chamber was incorporated with the instrument to resist the plasma etching. The chamber houses a reactive ion etching (RIE) electrode to create plasma. Uniform plasma is generated through a 0-600 W 13.56 MHz direct contact radio frequency (RF) power supply with a variable power setting. The vacuum controller controls the vacuum conditions inside the aluminum chamber by a 2-stage direct drive oil pump with an oil mist filter. The oxygen gas flow controller and water-cooling temperature controller regulate and control the gas flow rate and the base electrode plate temperature, respectively. The entire system is operated by Plasma Etch, Inc. software preloaded on a laptop for fully automatic system control, operation, process sequencing, and multiple recipe storage.

For the low temperature plasma treatment, the coal sample was uniformly spread on a watch glass of 15 cm diameter. Due to the limited exposure to the plasm, the top layer of the sample was more likely oxidized, while the bottom layer remained unchanged. Therefore, the sample

was manually mixed every two hours. The operation temperature for low temperature plasma treatment was 80 ℃. After being treated for around 40 hours, the ash content for the coal sample reduced to around 70%. pH-dependent leaching tests and SEM-EDS analysis were performed on the low temperature plasma ash to understand the mineralogy of REEs in the Fire Clay coal.

**pH-Dependent Leaching Test:** pH-dependent leaching tests demonstrate the effect of pH on the leaching recovery of investigated elements from solids. In the present study, pH-dependent leaching tests were performed on the low temperature plasma ash to evaluate the leachability of REEs, which informs the mineralogy of REEs in the raw coal. The tests were carried out in a three-neck round-bottom flask of 100 mL maximum volumetric capacity. A condenser was connected to the flask to reduce water loss due to evaporation, and the flask was immersed in a water bath to control the reaction temperature. For each test, 50 mL deionized water was placed into the flask first, and 0.5 g of solid was then added once the temperature of the water reached 75 °C. The slurry was continuously agitated using a magnetic stirrer to ensure sufficient mixing and dispersion. The pH of the slurry was continuously measured with a pH meter (AP110, Fisher Scientific, USA), and concentrated HCl was frequently added to adjust the pH. After each addition, 5 minutes of agitation was applied until the pH was stabilized, and then representative slurry samples were collected and immediately centrifuged at 5000 rpm for 10 min. The supernatants were diluted in 5% $HNO_3$ solution for elemental concentration measurement using an inductively coupled plasma mass spectrometry (ICP-MS). The solid residues were first dried at 60 °C for 12 h and then digested following a modified ASTM D6357-11 method. The digestion solutions were subjected to ICP-MS analysis, and the analysis results were used to back calculated elemental contents in the solids. The leaching recovery of REEs was calculated based on the content of REEs in the digestion solutions and solid residues.

**SEM-EDS Analysis:** SEM-EDS analysis was performed on the low temperature plasma ash to understand the mineralogy of REEs in the coal sample. A method reported in previous studies by Bin et al. (Ji et al., 2022a, 2022c) was adopted to find many REE-bearing particles and reach statistically reasonable conclusions about the REE mineralogy. The analysis was performed using a JSM-IT500HRE SEM (JEOL, MA, USA) equipped with an Ultim Max EDS detector (Oxford Instruments, Abingdon, UK). Two different scan modes, Map and Point & ID, were employed during the EDS analysis. Potential REE-bearing particles were first found from SEM specimens using Map mode at low magnification. Then, the existence of REEs in the potential REE-bearing particles was confirmed using Point & ID mode at high magnification. SEM images, EDS spectra, and EDS elemental contents were collected during the analysis and used to determine the REE mineralogy. A detailed description of the SEM-EDS analysis method is presented in the supplementary spreadsheet and can be referred to previous studies by Bin et al. (Ji et al., 2022a, 2022c) and Qi et al. (Li et al., 2022).

**Results and Discussion**

1. pH-Dependent Leaching Test

pH-dependent leaching tests were performed on the low temperature plasma ash (LTA) to evaluate the extractability of REEs, which also contributed to the understanding of REE mineralogy in the Fire Clay coal. The leaching recovery of total REEs (TREEs; Sc, Y, La, Ce, Pr, Nd, Sm, Eu, Gd, Tb, Dy, Ho, Er, Tm, Yb, Lu), (LREEs; Sc, La, Ce, Pr, Nd, and Sm), heavy REEs (HREEs; Y, Eu, Gd, Tb, Dy, Ho, Er, Tm, Yb, Lu), and individual REEs as a function of pH is shown in Figure 3.8.80. The initial pH of the slurry without adding any pH regulators was around 3.3, at which around 19% of TREEs, 10% of LREEs, and 40% HREEs were leached. This phenomenon suggested that a portion of the REEs, particularly HREEs, can be easily leached under weakly acidic conditions. Slight increases in the recovery occurred with

decreases in the pH, while after the pH reached around 1.2, the recovery of both LREEs and HREEs was dramatically increased with decreases in the pH.

The occurrence modes of REEs in coal can be divided into two general types: organic association and inorganic association. Organic association refers to REEs that are chemically bound by organic matter or atomically dispersed in the organic matrix of coal. In the present investigation, the majority of the organic matter was eliminated by low temperature plasma treatment; thus, the organically associated REEs in the coal were released and ended up in the ash product. Since a low temperature of 80 °C was employed during the plasma treatment, those REEs likely existed in amorphous states, such as rare earth oxides/hydroxides, rather than crystallizing into mineral species, and as such, those REEs had high leachability even under mild leaching conditions. As Figure 3.8.80(b) and 1(c) show, the recovery of individual REEs followed a trend that the REEs with larger atomic numbers (on the right side of the periodic table) had high recoveries than the REEs with smaller atomic numbers (on the left side of the periodic table). This trend is consistent with the fact that the binding constant of REEs with organic acids increases with the decrease in ionic radius and the increase in atomic number (Sonke and Salters, 2006; Stern et al., 2007). Therefore, it can be concluded that the recoveries obtained at pH 3.3 were predominantly contributed by the organically associated REEs.

Although the recovery of both LREEs and HREEs was noticeably improved when the pH was lower than 1.2, the improvement in LREE recovery was more significant than the improvement in HREE recovery (see Figure 3.8.80(a)). After reaching the final pH of -0.05, the recovery of LREEs exceeded that of HREEs. Therefore, the leaching behavior of REEs at pH < 1.2 cannot be explained by the organically associated REEs, while is more likely determined by the dissolution of REE-bearing minerals that are dispersed in the organic matrix of the coal sample. Scandium is included in the LREEs, however, as Figure 3.8.80 shows, the leaching behavior of Sc is considerably different from that of the other REEs. Around 45% of Sc was leached at pH 3.3, and as the pH reduced to only 1.8, the recovery was improved to 87%. This result indicated that the major of Sc in the low temperature ash originated from organic associations in the coal.

The pH-dependent leaching test results suggested that both organically and inorganically associated REEs in the Fire Clay coal contributed to the recovery of REEs from the LTA. SEM-EDS analysis was performed on the low temperature ash to better understand the REE mineralogy. It has been realized that SEM-EDS has several disadvantages in this type of application; for example, nanoscale REE-bearing particles below the detection limit of SEM have been frequently found in coal, and additionally, SEM-EDS cannot provide direct information about the crystal structure of minerals. However, compared with other analytical techniques, such as TEM-EDS, SEM-EDS is more cost effective and do not need to lift out REE-bearing particles and mill them into thin slices. In addition, meaningful conclusions about REE mineralogy can still be obtained using SEM-EDS by finding many REE-bearing particles from a coal sample and statistically analyzing the SEM and EDS results. After the low temperature plasma treatment, the organically associated REEs are predicted to remain atomically and/or nano dispersed in the ash product. This prediction can be supported by the result shown in the next section that for most REE-bearing particles found in the LTA using SEM-EDS, the REEs existed in rare earth phosphate minerals instead of oxides/hydroxides.

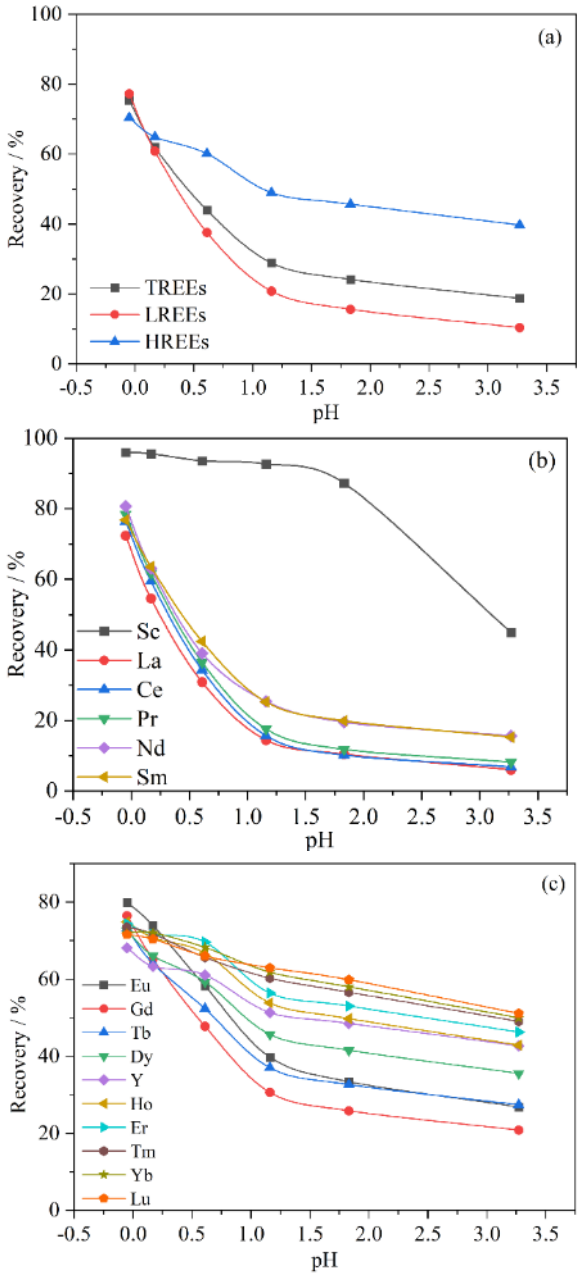



Figure 3.8.80. The leaching recovery of (a) TREEs, LREEs, and HREEs, (b) individual LREEs, and (c) individual HREEs as a function of pH.

2. SEM-EDS Analysis

Raw Low Temperature Plasma Ash:

SEM-EDS analysis was conducted to find REE-bearing particles from the LTA of the Fire Clay coal, and the REE mineralogy of these particles was evaluated based on EDS elemental composition data. A total of 94 REE-bearing particles referred to as FC1-1 to -94 were found. Elemental compositions, EDS spectra, and SEM images of the particles are presented in the supplementary spreadsheet. Based on elemental composition, the particles can be divided into two groups: group H (FC1-1 to -13) and group L (FC1-14 to -94). The REEs in the H group particles were predominantly HREEs, and the content of major LREEs (e.g., La and Ce) was negligible; whereas the REEs in the L group particles were predominantly LREEs, and the content of major HREEs (e.g., Y) was negligible.

Based on elemental composition, the H group particles can be further divided into three subgroups: subgroup H1 (FC1-1 to -5), subgroup H2 (FC1-6 to -11), and subgroup H3 (FC1-12 to -13). The particles of H1 subgroup were characterized by high Zr content and relatively low REE content. In addition, small amounts of P were detected in selected H1 subgroup particles. These phenomena suggested that the REEs in the H1 subgroup particles either existed as ionic substitutes in Zr minerals such as zircon, or in the form of rare earth phosphate minerals that are physically associated with Zr minerals. Hafnium with a content of 1.3% was detected in particle FC1-2, agreeing with the fact that Hf isotopes are ubiquitous in zircon and have been widely used to study planetary crust-mantle differentiation (Iizuka et al., 2017; Kinny and Maas, 2003). The H2 subgroup particles were characterized by high REE and P contents, and thus, the REEs primarily existed as rare earth phosphate minerals, such as xenotime. Unlike the particles of H1 and H2 subgroups, the H3 subgroup particles did not contain Zr and P. Particle FC1-13 of the H3 subgroup contained 12.9% Ta, 2.0% Nb, and 4.5% Ti, suggesting that the REEs existed in minerals rich in Ta, Nb, and Ti. Although belonging to the same subgroup, Ta and Nb were not detected in particle FC1-12, and the content of Ti was also negligible. Based on the elemental composition of FC1-12, the REEs likely existed in rare earth carbonate, silicate, oxide, or hydroxide minerals.

The TREE content and TREE to P content ratio (TREE/P) of L group particles were plotted against the P content. As Figure 3.8.81(a) shows, all L group particles contained P, and the TREE content gradually increased with the increase in the P content. Therefore, the REEs in the L group particles primarily existed in the form of rare earth phosphate minerals. In addition, the L group particles can be further divided into two subgroups: subgroup L1 and subgroup L2. The particles of the two different subgroups are distributed around the dashed and dotted lines in Figure 3.8.81(a), respectively. The slope of the dashed line is larger than that of the dotted line; therefore, with similar P content, the REE content of the L1 subgroup particles is higher than that of the L2 subgroup particles. Correspondingly, the TREE/P of L1 subgroup particles is larger than that of the L2 subgroup particles (see Figure 3.8.81(b)).

In previous studies by (Ji et al., 2022a, 2022b), particles containing LREEs and P were found in the coal refuse of the Fire Clay and Western Kentucky No. 13 seams, and it was concluded that the particles with TREE/P greater than 2 were monazite, while the particles with TREE/P less than 2 were crandallite group minerals. This judgment method was verified through TEM-EDS analysis, as well as SEM-EDS analysis of pure florencite, a typical crandallite group mineral (Ji et al., 2022a; Li et al., 2022). In the present investigation, a TREE/P of 2 was also employed to separate the L1 and L2 subgroup particles. As Figure 3.8.82(a) shows, the Al content of L1 subgroup particles decreased with the increase in the P content, while the Al content of L2 subgroup particles except FC1-89 remained at a high level. This phenomenon is consistent with findings from the REE mineralogical characterization of the Fire Clay and Western Kentucky No. 13 coal refuse (Ji et al., 2022a). In addition, as Figure 3.8.82(b) shows, several L2 subgroup particles contained certain amounts of Sr, and Sr also found in selected particles with TREE/P less than 2 in the Fire Clay and Western Kentucky No. 13 coal refuse. These consistencies indicated that it is reasonable to using a TREE/P of 2 to separate the two subgroups. As Figure 3.8.83 shows, particle FC1-89 was on top of a large particle, which is rich in Si but deficient in Al based on the elemental maps. This kind of association explained the much lower Al content but higher Si content of FC1-89 relative to the other particles in L2 subgroup.

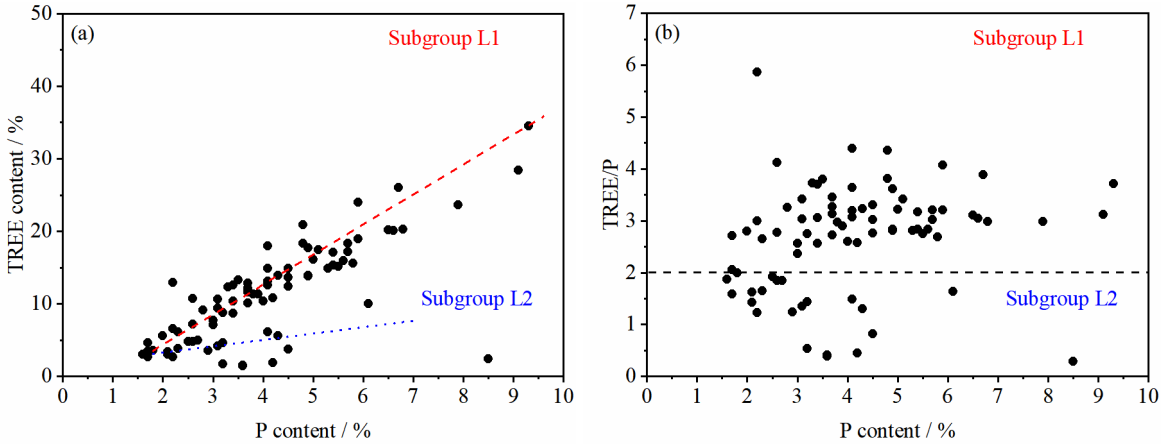



Figure 3.8.81. Changes in (a) the TREE content and (b) the TREE/P as a function of the P content for the L group particles found in the LTA of the Fire Clay coal.

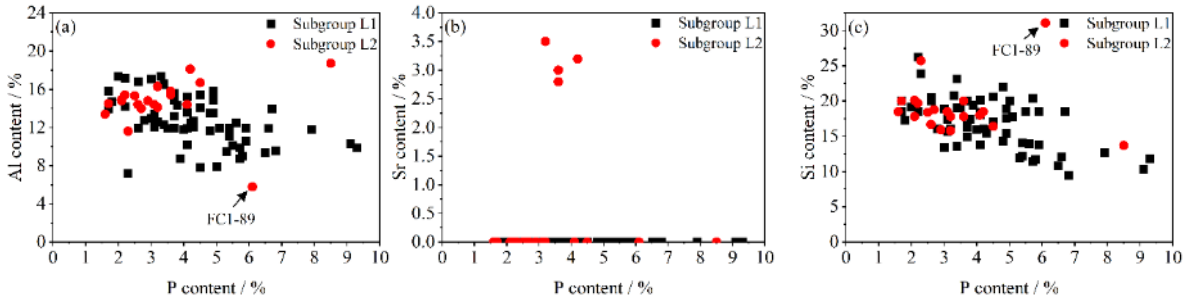



Figure 3.8.82. Changes in the content of (a) Al, (b) Sr, and (c) Si as a function of the P content for the L group particles found in the LTA of the Fire Clay coal.

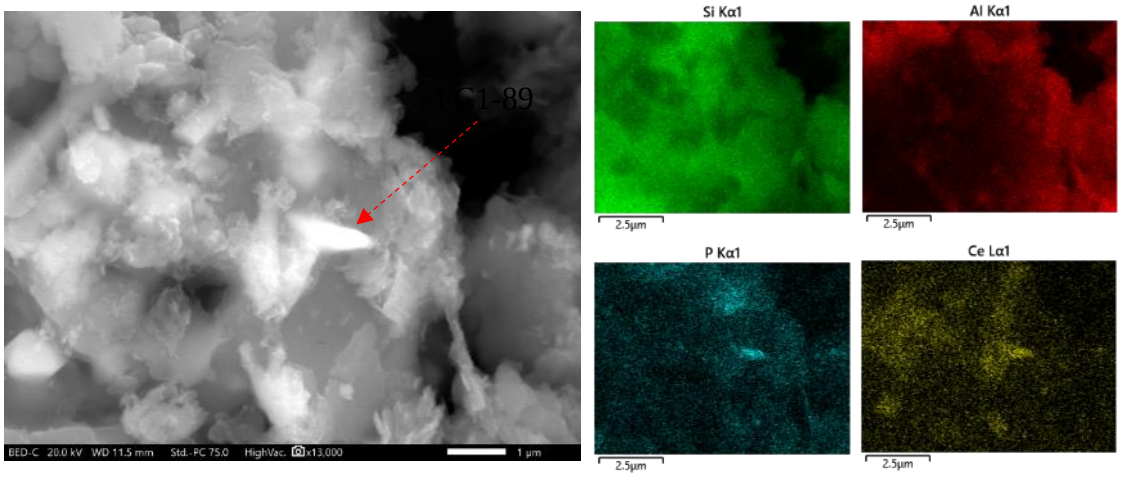



Figure 3.8.83. BSE (backscattered electron) image and EDS elemental maps of particle FC1-89.

Besides REEs, Al, Si, Sr, P, and O, other elements, such as Ti, Fe, S, Ca, Mg, Ba, Na, and K, were also frequently found in the L group particles. As shown in the supplementary spreadsheet, the content of Na and K in all the L group particles was low, less than 1% for Na and 2% of K; thus, Na and K were excluded from the EDS data analysis. The content of Ti, Fe, S, Ca, Mg, and Ba was plotted against the P content. As shown in Figure 3.8.84, there is no obvious relationship between the content of Ti, Fe, S, Ca, Mg, and Ba and the content of P, however, particles with relatively high content of Ti, Fe, S, Ca, Mg, and/or Ba existed. This phenomenon suggested that these particles were physically associated with Ti-, Fe-, S-, Ca-,

Mg-, and/or Ba-bearing minerals, or Ti, Fe, S, Ca, Mg, and/or Ba are constituent elements of the REE-bearing minerals. For example, as Figure 3.8.85 shows, particle FC1-76 was surrounded by several rutile particles, leading to the high Ti content in this particle analyzed by EDS. Particle FC1-79 contained 4.2% of Ba, which is much higher than the Ba content of other L group particles. In addition, it can be observed from the elemental maps shown in Figure 3.8.85 that similar to P and Ce, Ba was distributed throughout particle FC1-79, therefore, Ba was a constituent element of the REE-bearing mineral. The Al content of FC1-79 is much higher than Si (18.7% versus 13.7%), and the Al to Si content ratio is the highest among all the REE-bearing particles found in the raw LTA. These results indicated that the REE-bearing mineral is likely gorceixite ($BaAl_3(PO_4)_2(OH)_5 \cdot H_2O$), which is rich in Ba, Al, and P.

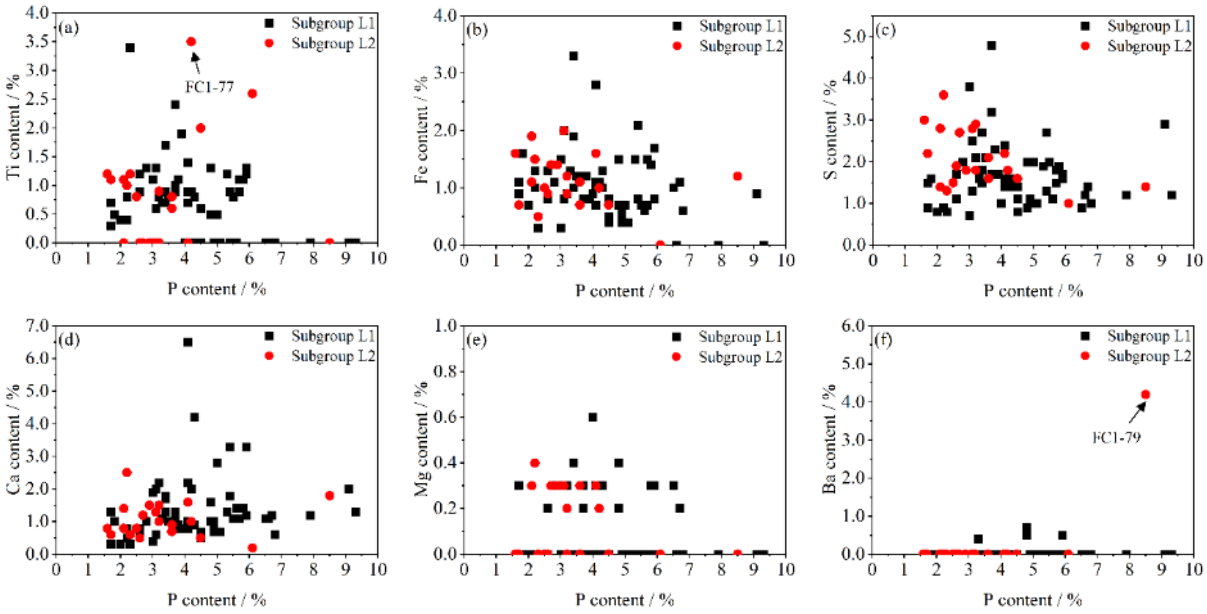



Figure 3.8.84. Changes in the content of (a) Ti, (b) Fe, (c) S, (d) Ca, (e) Mg, and (f) Ba as a function of the P content for the L group particles found in the LTA of the Fire Clay coal.

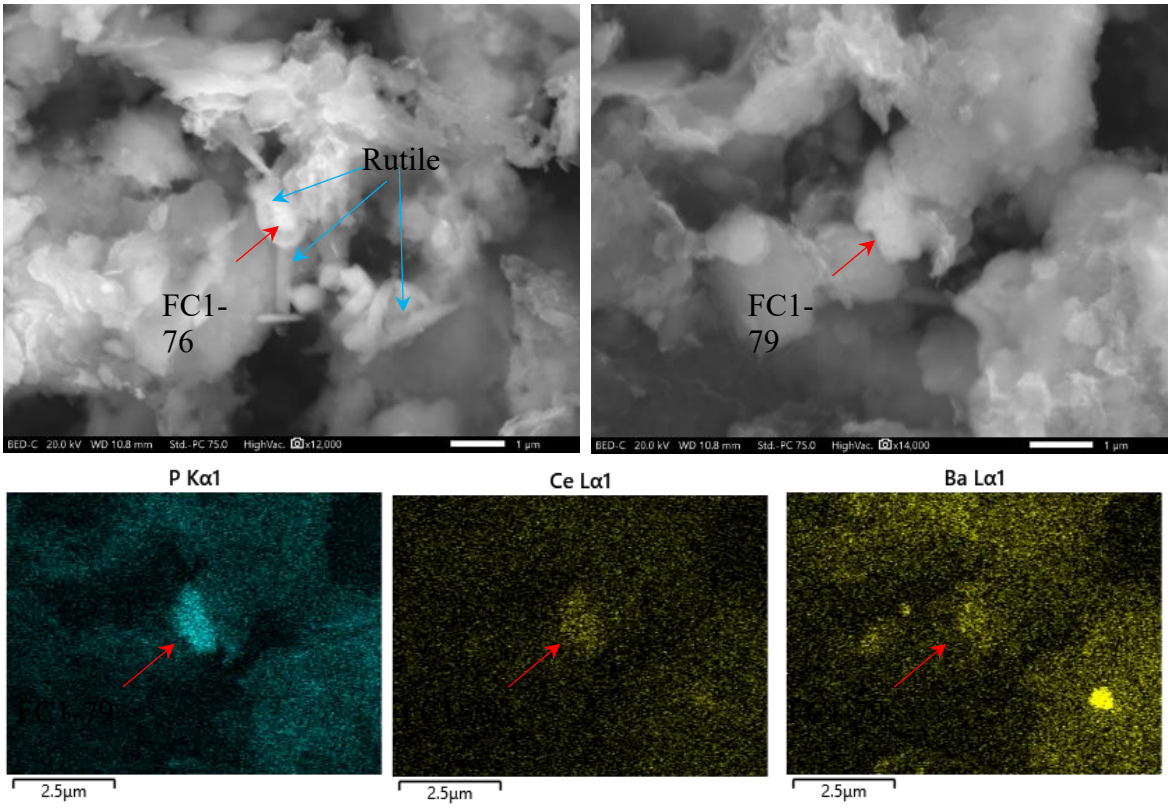



Figure 3.8.85. BED images of FC1-76 and FC1-79, and elemental maps of FC1-79.

Leaching Residue of the LTA:

SEM-EDS analysis was also performed on the final leaching residue of the Fire Clay low temperature ash obtained from the pH-dependent test (i.e., the solid residue obtained by gradually reducing slurry pH to -0.05). The same as the raw LTA, a total of 94 REE-bearing particles referred to as FC2-1 to -94 were found from the final leaching residue, and the particles were divided into two major groups (group H rich in HREEs and group L rich in LREEs) and five subgroups (subgroup H1 with high Zr content, subgroup H2 with high HREE and P contents, subgroup H3 without P, subgroup L1 with TREE/P greater than 2, and subgroup L2 with TREE/P less than 2). Elemental compositions, EDS spectra, and SEM images of the particles are presented in the supplementary spreadsheet.

The P content of H1 subgroup particles (FC2-1 to -11) varied from 0% to 4.9%, suggesting that the REEs either existed as ionic substitutes in Zr minerals such zircon, or in the form of rare earth phosphate minerals that are physically associated with Zr minerals. Hf was found in several H1 subgroup particles, such as FC2-1 (1.6% Hf) and FC2-2 (0.9% Hf). These phenomena agreed with the findings from the SEM-EDS analysis of the raw LTA. Particle FC2-9 contained 11.4% of Ti, which is close to the content of Zr (11.7%). As Figure 3.8.86 shows, a relatively large Ti-bearing particle presented around FC2-9, likely resulting in background noise and thereby the high Ti content in FC2-9 measured by EDS.

The H2 subgroup particles (FC2-12 to -32) were characterized by high HREE and P contents, therefore, the REEs primarily existed in rare earth phosphate minerals. This conclusion is supported by the gradual increase of the REE content with increases in the P content (see Figure 3.8.87(a)). As shown in Figure 3.8.87, compared with the other H2 subgroup particles with similar P contents, particles FC2-31, FC2-20 and -32, and FC2-30 had higher TREEs, Ti, and Fe contents, respectively. The SEM images and EDS analyses of the four particles are presented in Figure 3.8.88. Both P and F were detected in spectrum 8 of FC2-31, while only P was

detected in spectrum 407; therefore, the elemental composition of FC2-31 was not uniform, and rare earth phosphate likely intergrew with rare earth fluoride or fluoride carbonate (e.g., bastnaesite) in this particle. The whole FC2-20 particle was rich in Ti per the elemental map of Ti, while only a portion of the particle contained REEs and P; therefore, rare earth phosphate likely presented in a Ti-bearing mineral in this particle. The whole FC2-32 was also rich in Ti, but the mineralogy of this particle is more complicated than that of FC2-20. Besides a portion rich in REEs and P (spectrum 4), a portion rich in REE, Nb, and Ta but deplete in P was found in FC2-32. Thus, multiple REE-bearing minerals, such as rare earth phosphate, Ti-bearing, and Nb, Ta-bearing minerals, likely co-existed in this particle. Based on the elemental maps of FC2-30, it can be concluded that the REEs of this particle existed in a rare earth phosphate mineral; however, the upper left portion of the particle was also rich in Fe. Therefore, an Fe-bearing particle likely encapsulated in FC2-30. Overall, the complex REE mineralogy and complex physical associations of REE minerals with other minerals resulted in the high contents of certain elements in these four particles.

As shown in the supplementary spreadsheet, the H3 subgroup particles (FC2-33 to -38) did not contain P, and only one particle FC2-38 of this subgroup contained Zr, but the Zr content is much lower than that of the H1 subgroup particles. Thus, the REE mineralogy of this subgroup is different from that of H1 and H2 subgroups. This conclusion is also confirmed by the fact that all the H3 subgroup particles contained certain amounts of Nb and Ta, which were only found in a portion of particle FC2-32 in H2 subgroup.

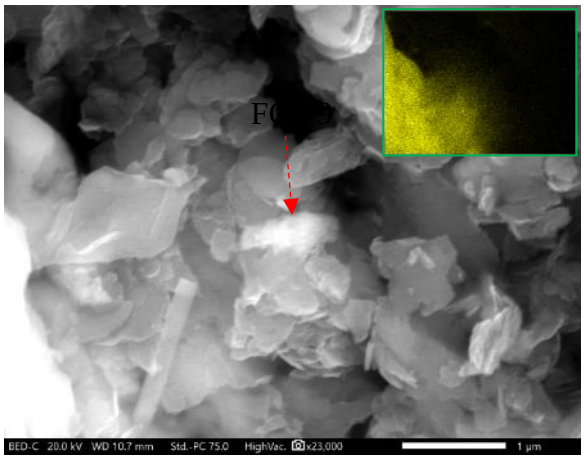


Figure 3.8.86. BSE image of particle FC2-9 and Ti map (up right corner) of the BSE image area.

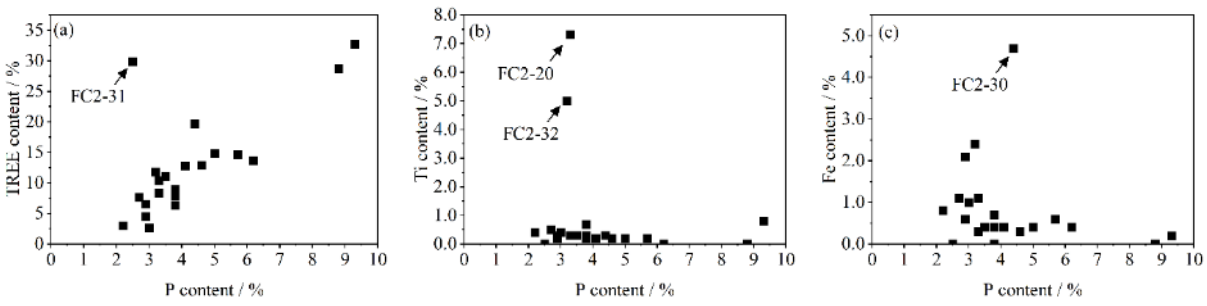



Figure 3.8.87. Changes in the content of (a) TREEs, (b) Ti, and (c) Fe as a function of the P content for the H2 group particles found in the leaching residue of the LTA of the Fire Clay coal.

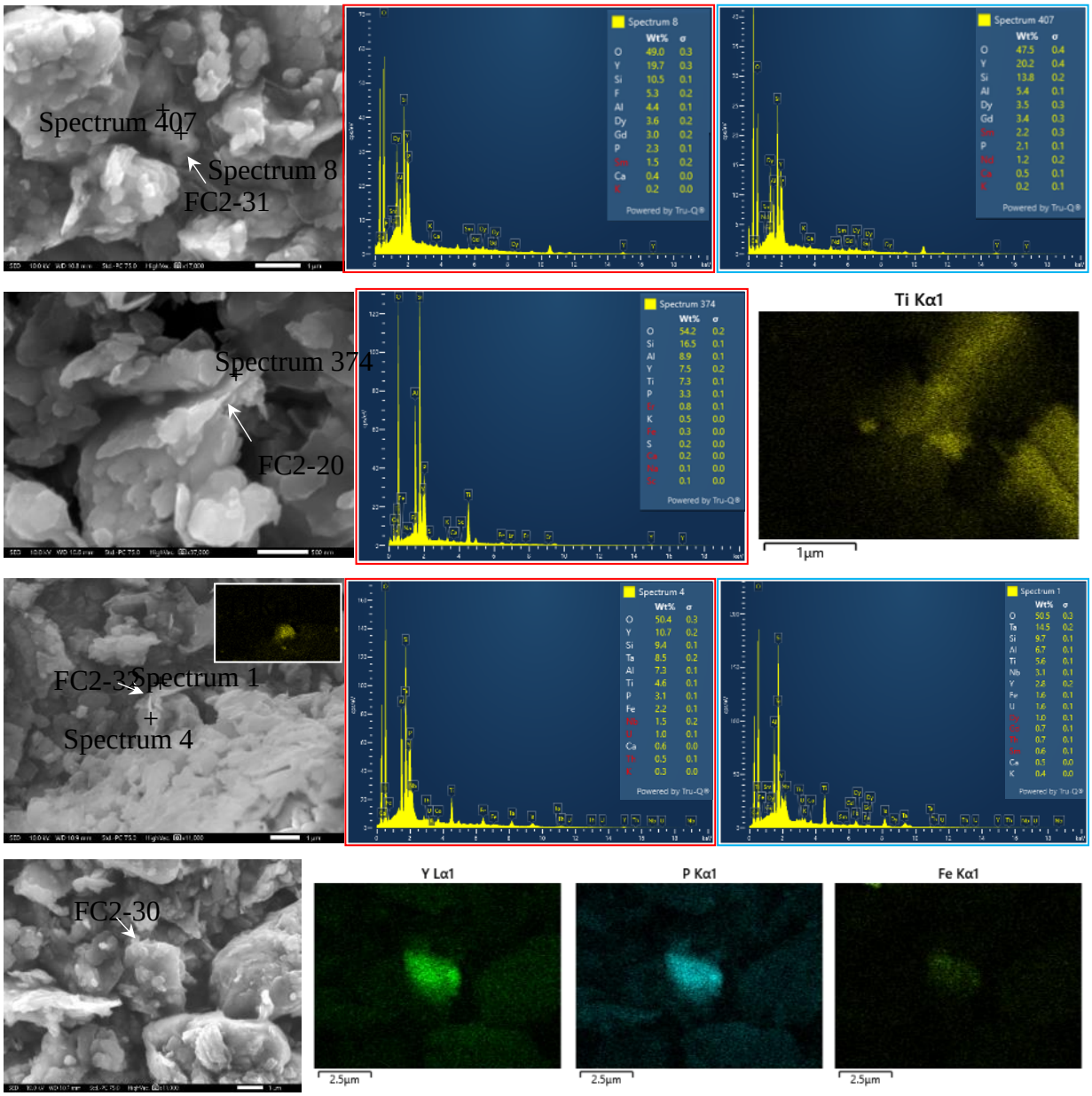



Figure 3.8.88. SEM image and EDS spectra of particle FC2-31, -20, -32, and -30.

The TREE content and TREE to P content ratio (TREE/P) of L group particles were plotted against the P content. As Figure 3.8.89(a) shows, the L1 and L2 subgroups particles are distributed around two lines of different slopes. This phenomenon is similar to that observed in the L group particles of the raw LTA (Figure 3.8.81(a)). However, comparisons between Figure 3.8.89(b) and Figure 3.8.81(b) showed that less particles are concentrated around the dashed line corresponding to TREE/P of 2 in Figure 3.8.89(b). Therefore, it is more effective to use TREE/P of 2 to separate the L group particles in the leaching residue than the raw low temperature ash. The linear relationships between the TREE content and the P content suggested that the REEs of L group particles predominantly existed in rare earth phosphate minerals. However, as Figure 3.8.89(a) shows, particle FC2-67 did not contain P but had an exceedingly high TREE content of 55.3%, suggesting that the REE mineralogy of FC2-67 is different from that of the other L group particles. No F, Zr, Ti, Nb, and Ta were found in this particle, and the Al to Si content ratio is in the range of other REE-bearing particles, therefore, the REEs likely existed in a rare earth oxide/hydroxide mineral (see Figure 3.8.90). As Figure 3.8.91 shows, the Al content of L1 subgroup particles gradually decreased with increases in the

P content, while the Al content of L2 subgroup particles remained high even at high P content, and additionally, many of the L2 subgroup particles contained varying amounts of Sr. These phenomena are consistent with those observed from the raw LTA and the coal waste of the Fire Clay seam, suggesting that classification into the L1 and L2 subgroups is viable.

Changes in the content of Ti, Fe, S, Ca, Mg, and Ba of the L group particles as a function of the P content are shown in Figure 3.8.92. The Ca and Mg contents were low for all the particles, while several particles had higher Ti, Fe, S, and/or Ba contents relative to the other L group particles with similar P contents. Based on the SEM images and elemental maps shown in Figure 3.8.93, it can be concluded that some of the high contents of Ti, Fe, and S were due to the appearance of particles rich in these elements around the REE-bearing particles (e.g., FC2-93, -82, -44, and -80). In addition, two REE-bearing particles themselves, FC2-74 and -70, were found to contain S and Ba, respectively, suggesting that some of the REE phosphate minerals of L2 subgroup particles contained S and Ba.

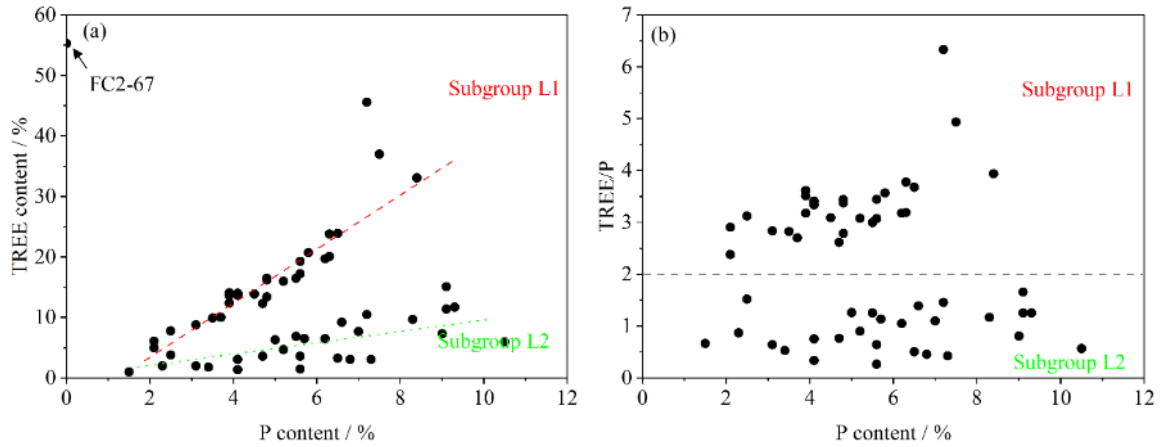



Figure 3.8.89. Changes in (a) the TREE content and (b) the TREE/P as a function of the P content for the L group particles found in the leaching residue of the LTA of the Fire Clay coal. (FC2-67 is excluded in Figure 3.8.89(b) since it does not contain P.)

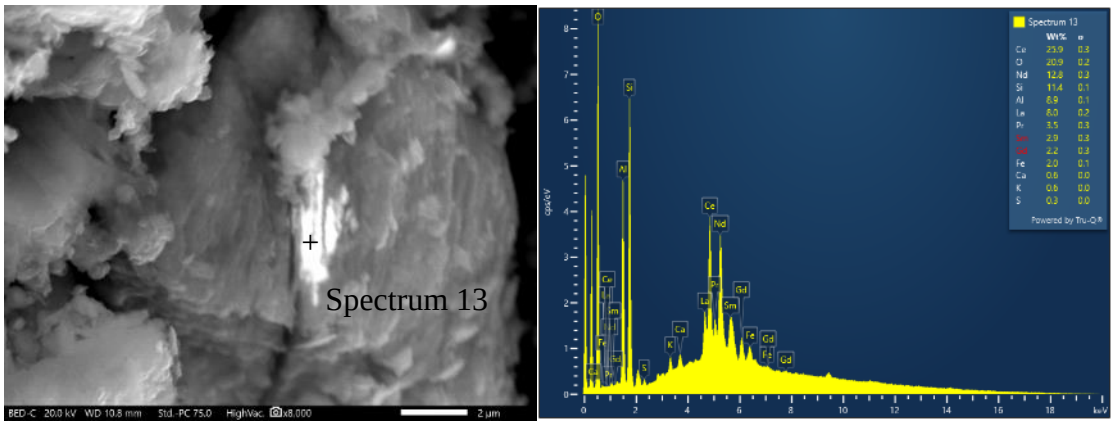



Figure 3.8.90. BSE image and EDS spectrum of particle FC2-67.

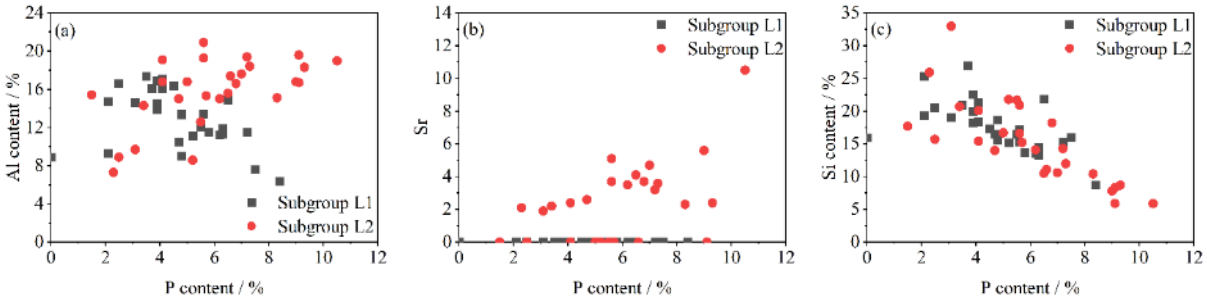



Figure 3.8.91. Changes in the content of (a) Al, (b) Sr, and (c) Si as a function of the P content for the L group particles found in the leaching residue of the LTA of the Fire Clay coal.

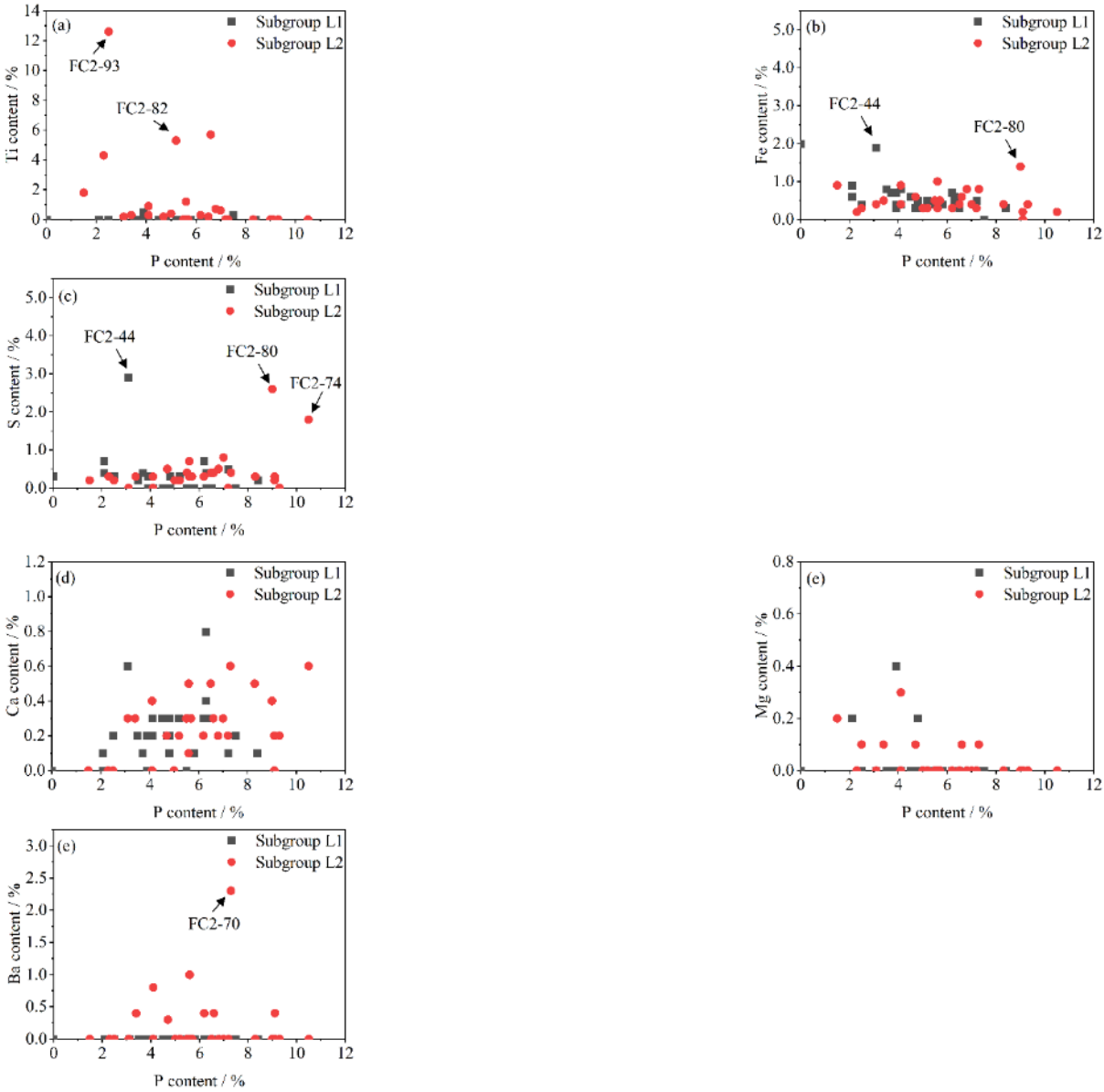



Figure 3.8.92. Changes in the content of (a) Ti, (b) Fe, (c) S, (d) Ca, (e) Mg, and (f) Ba as a function of the P content for the L group particles found in the leaching residue of the LTA of the Fire Clay coal.

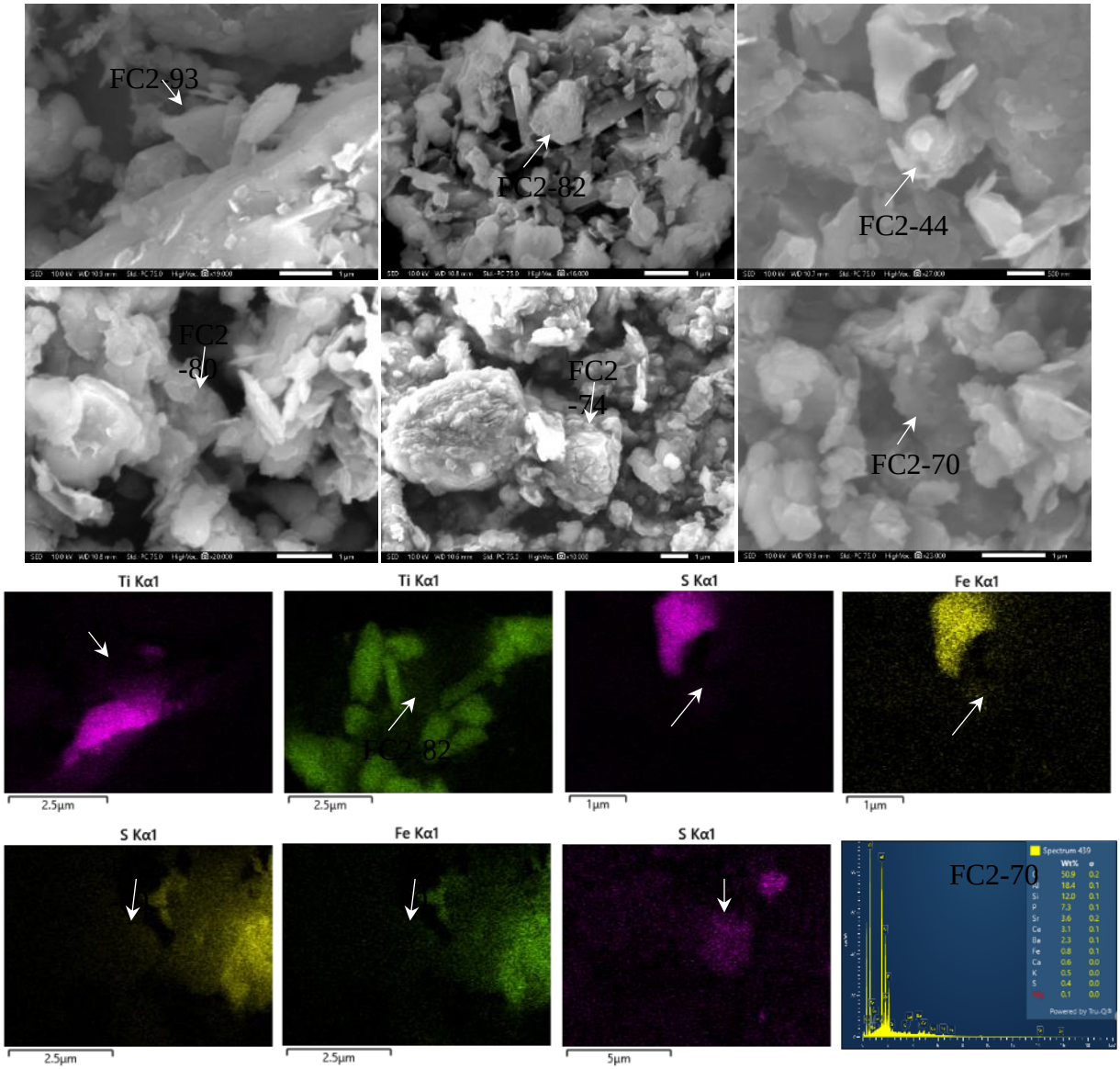



Figure 3.8.93. SEM images and EDS analyses of selected particles marked in Figure 3.8.92.

**Comparison between the Raw Material and the Leaching Residue:** SEM-EDS results of the raw and leached low temperature plasma ashes revealed complex REE mineralogy in the Fire Clay coal. The REE-bearing particles found from both the raw and leached materials were divided into two major groups, groups H and L, which were rich in HREEs and LREEs, respectively. Different occurrence modes were found for the HREEs, such as HREE-bearing Zr minerals (subgroup H1), HREE phosphate minerals (subgroup H2), and minerals rich in Ti, Nb, and/or Ta (subgroup H3). In addition, the REEs in selected H1 subgroup particles existed in rare earth phosphate minerals that are physically associated with Zr minerals. Fluoride and/or fluoride carbonate minerals, Ti minerals, and minerals rich in Nb/Ta also contributed to the REE content of a few subgroup H2 particles, and these minerals were physically associated with rare earth phosphate minerals in these particles. The L group particles were divided into L1 and L2 subgroups according to TREE/P: TREE/P > 2 for subgroup L1 while < 2 for subgroup L2. The different TREE/P values between these two subgroups indicated different REE mineralogy. Previous studies on a Fire Clay coal refuse concluded that LREE-bearing particles with TREE/P > 2 are monazite, while LREE-bearing particles with TREE/P < 2 are crandallite-group minerals. This conclusion partially applies to the L group particles found in the low temperature plasma. Many L2 group particles presented in Figure 3.8.81 and Figure

3.8.91 were found to contain certain amounts of Sr, and Ba was also detected in a few L2 group particles presented in Figure 3.8.84 and Figure 3.8.92. The appearance of Sr and Ba suggested the existence of goyazite ($SrAl_3(PO_4)_2(OH)_5 \cdot H_2O$) and gorceixite ($BaAl_3(PO_4)_2(OH)_5 \cdot H_2O$), respectively, which are typical endmembers of the crandallite-group minerals.

SEM images of all the REE-bearing particles found in the raw and leached materials are presented in the spreadsheet. It can be observed that the REE-bearing particles of both materials were very fine, and most of the particles had a particle size of around 1 μm. Therefore, acid leaching did not significantly alter the size of the REE-bearing particles. Based on the total number of REE-bearing particles and the number of REE-bearing particles of each subgroup, the proportion of each subgroup was calculated. Since no noticeable differences in particle size were observed among the different subgroups, the proportion values qualitatively described the distribution of REEs in the different types of mineralogy. As Figure 3.8.94 shows, the sum of the proportions of the H subgroups notably increased after leaching, accompanied by the decrease in the sum of the proportions of the L subgroups. This phenomenon suggested that the LREE-bearing particles were preferentially leached relative to the HREE-bearing particles. pH-dependent leaching tests showed that a higher recovery of LREEs was obtained at pH -0.05 than HREEs, and as the pH decreased from 3.3 to -0.5, the increase in the recovery of LREEs was much higher than the increase in the recovery of HREEs (around 67% versus 30%). Therefore, the leaching results are consistent with the proportion changes caused by leaching.

It was mentioned previously that organically-associated REEs likely transformed into amorphous rare earth oxides/hydroxides during the plasma treatment process at a relatively low temperature of 80 °C. However, rare earth oxides/hydroxides were barely found from the SEM-EDS analysis. This inconsistency suggested that the rare earth oxides/hydroxides were atomically and/or nano dispersed in the ash product, which were too small to be found using SEM-EDS. The recoveries obtained at high pH (e.g., 3.3) were predominantly attributed to the dissolution of amorphous rare earth oxides/hydroxides, while the considerable increases in the recovery obtained at low pH were due to the dissolution of REE-bearing minerals that were dispersed in the organic matrix of the coal prior to the low temperature plasma treatment. The dissolution of REE-bearing minerals is confirmed by the proportion changes of the different subgroups caused by leaching (see Figure 3.8.94). However, most of the REE minerals found in the LTA, such as monazite in the L1 subgroup and crandallite-group minerals in the L2 subgroup, were insoluble even under strongly acidic conditions. For example, monazite and crandallite-group minerals are major REE-bearing minerals in the coal refuse of the Western Kentucky No. 13 seam, and the leaching recovery of REEs was only around 20% at pH < 0 with 1.2 M HCl (Zhang and Honaker, 2020). This phenomenon is inconsistent with the pH-dependent leaching result: the recovery of TREEs notably increased from around 19% to 75% as the pH reduced from 3.3 to -0.05. It was found from the SEM-EDS analysis that the majority of the REE-bearing particles in the LTA had irregular shapes and were formed by the intergrowth of small spheres and/or flakes (see Figure 3.8.95). These characteristics suggested that the REE-bearing minerals are of authigenic origin and likely have lower crystallinity compared with accessory REE-bearing minerals in coal refuse, which explained the higher leaching recovery obtained at pH -0.05 and the inconsistency between the pH-dependent leaching test and SEM-EDS analysis results. A cerium phosphate precipitate was synthesized and calcined at different temperatures. As Figure 3.8.96 shows, the diffraction peaks of monazite appeared in the 600 ºC-calcined material, and the intensity of the peaks increased with elevations in the calcination temperature, suggesting increases in the crystallinity of monazite in the material. Due to the increases in crystallinity, the leaching recovery of cerium reduced with elevations in the calcination temperature. This finding confirmed that the leachability of rare earth minerals is negatively correlated to their crystallinity.

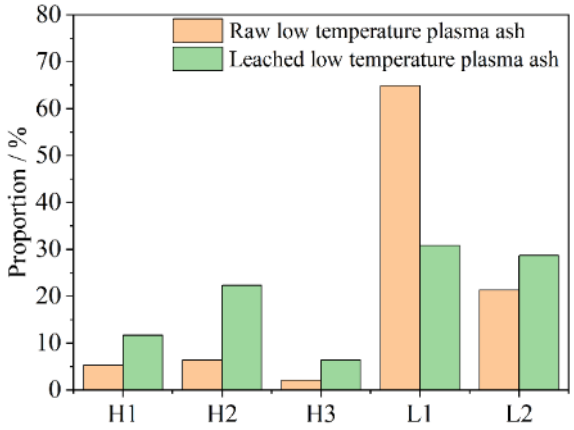



Figure 3.8.94. The proportions of each subgroup in the total REE-bearing particles.

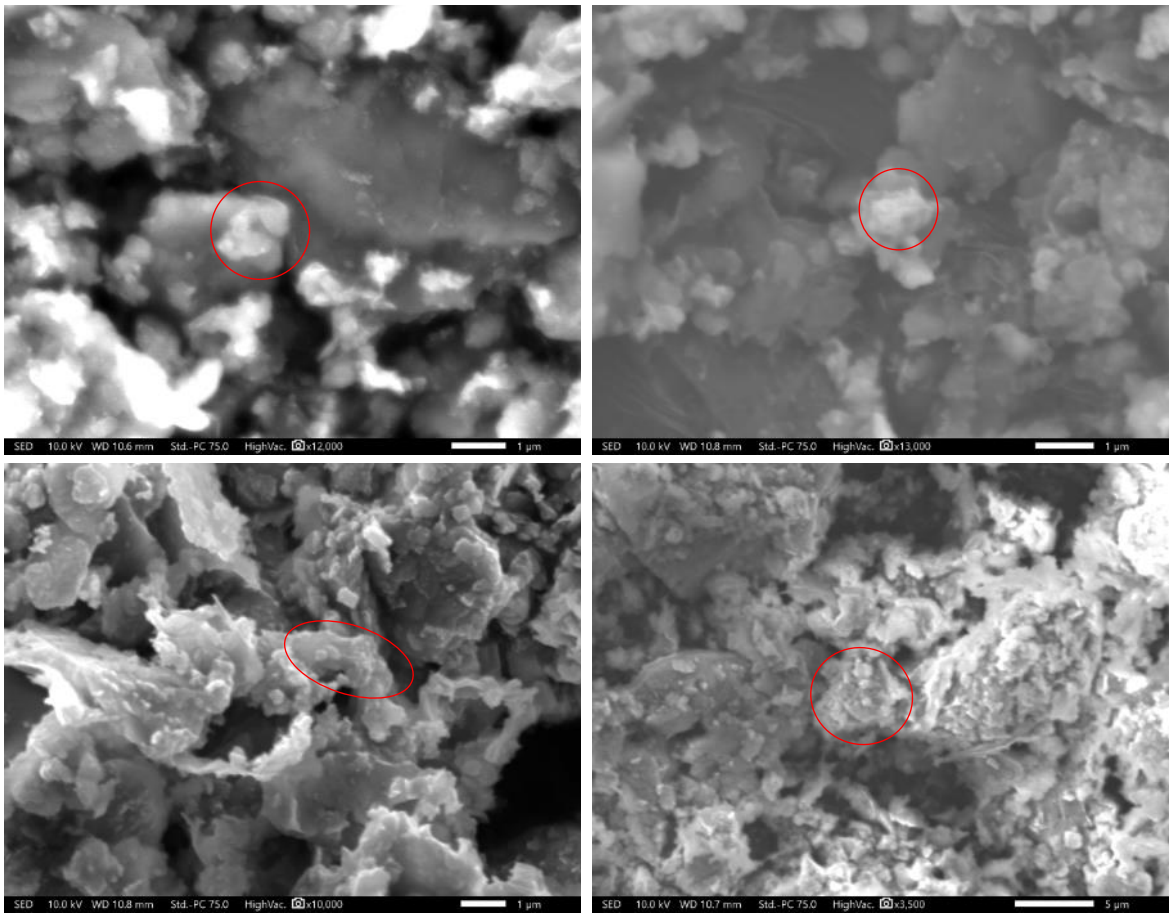


Figure 3.8.95. SEM images of REE-bearing particles found in the raw LTA.

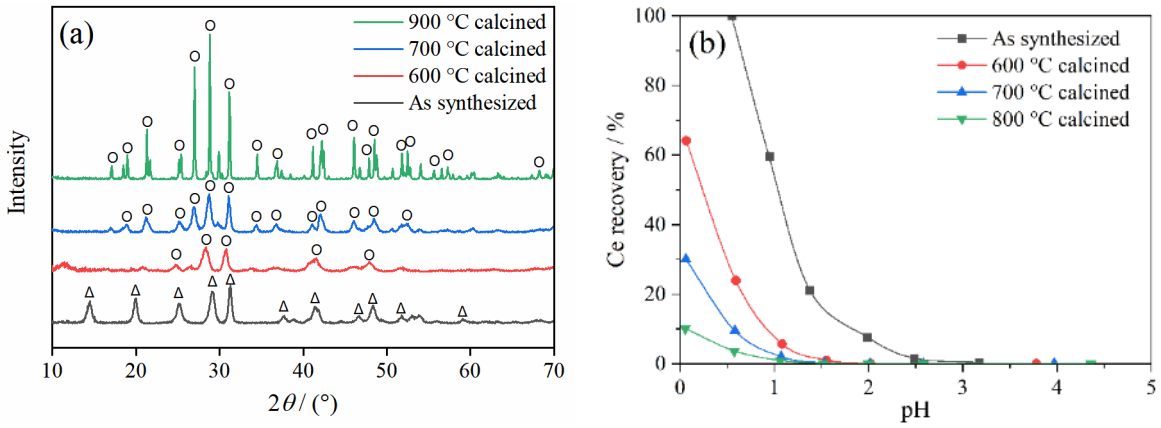



Figure 3.8.96. XRD patterns (a) and cerium recoveries (b) from a synthetic cerium phosphate precipitate and its calcination products (Triangles and circles represent the diffraction peaks of $Ce(PO_4)\cdot H_2O$ (PDF#35-0614) and monazite(Ce) (PDF#32-0199), respectively).

**Conclusions:** A coal sample of the Fire Clay seam was collected and subjected to low temperature plasma treatment. pH-dependent leaching tests were performed on the obtained ash material to investigate the leachability of REEs. Around 19% of TREEs, 10% of LREEs, and 40% of HREEs were leached from the material at a relatively high pH of 3.3, while as the pH decreased to -0.05, the recoveries increased to 75%, 77%, and 70%, respectively. The leaching recovery of individual REEs at high pH followed a similar order as the binding constant of REEs with organic acids; therefore, the recoveries at pH 3.3 were ascribed to the organically-associated REEs in the coal. Considerable increases in the leaching recovery of REEs, particularly LREEs, occurred in the pH range of 1.2 to -0.05, which were explained by the dissolution of REE-bearing minerals that were dispersed in the organic matrix of the coal sample prior to the low temperature plasma treatment. SEM-EDS analysis was performed on both the raw and leached LTA materials to understand the mineralogy of REEs in the Fire Clay coal. Many REE-bearing particles were found from the materials using an efficient SEM-EDS analysis method reported in previous studies. Since the coal was treated with oxygen plasma at a relatively low temperature of 80 ºC, the majority of the organically-associated REEs are expected to end up in the ash material as amorphous rare earth oxides/hydroxides. However, rare earth oxides/hydroxides were barely found from the SEM-EDS analysis, suggesting the REEs in the REE-bearing particles were inorganically associated in the coal sample.

Based on the EDS elemental composition data, the REE-bearing particles were classified into two major groups (H and L groups) and five subgroups (H1, H2, H3, L1, and L2 subgroups). The REE-bearing particles of the H and L groups were rich in HREEs and LREEs, respectively. Different REE mineralogy was observed in the H-group particles, including Zr minerals (subgroup H1), rare phosphate minerals (subgroup H2), and minerals rich in Ti, Nb, and/or Ta (subgroup H3). In addition, the REEs in selected H1 subgroup particles existed in rare earth phosphate minerals that are physically associated with Zr minerals. Fluoride and/or fluoride carbonate minerals, Ti minerals, and minerals rich in Nb/Ta also contributed to the REE content of a few H2 subgroup particles, and these minerals were physically associated with rare earth phosphate minerals in these particles. The L group particles were divided into L1 and L2 subgroups according to TREE/P: TREE/P > 2 for subgroup L1 while < 2 for subgroup L2. The different TREE/P values between these two subgroups indicated different REE mineralogy (e.g., monazite in the L1 subgroup and crandallite-group minerals in the L2 subgroup).

SME images showed that all the REE-bearing particles were very fine, and most of the particles had a particle size of around 1 µm. Comparisons between the SEM-EDS results of the raw and leached materials, it was found that the LREE-bearing particles were preferentially leached

compared with the HREE-bearing particles. This finding is consistent with the pH-dependent leaching test result that more considerable recovery increases were obtained for LREEs in the pH range of 1.2 to -0.05. The leaching recoveries at pH -0.05 were much higher than those obtained from coal refuse under similar conditions. This contrast was explained by the authigenic origin and low-crystallinity nature of most REE-bearing particles in the LTA.

### *3.8.4 Metal Sulfide Precipitation*

A flowsheet for the purification and separation of Ni, Co, Mn, and Zn from coal-based leach solutions was developed and optimized based on lab-scale tests (Figure 3.8.74). However, when applying the process in the production of the pilot plant, the consumption of sodium sulfide in the sulfide precipitation step was much higher than the theoretical value. It was found that in the production of the pilot plant, the contaminant elements (e.g., Fe and Al) were not completely depleted in the stage precipitation stage. Thereby, the influence of contaminant elements in the sulfide precipitation was investigated in this reporting period.

**Materials:** Manganese (II) nitrate tetrahydrate ($Mn(NO_3)_2{\cdot}4H_2O$, 98.0 wt.%), nickel nitrate hexahydrate ($Ni(NO_3)_2{\cdot}6H_2O$, 98.0 wt.%), zinc nitrate hexahydrate ($Zn(NO_3)_2{\cdot}6H_2O$, 99.0 wt.%), cobalt nitrate hexahydrate ($Co(NO_3)_2{\cdot}6H_2O$, 99.0 wt.%), aluminum nitrate nonahydrate ($Al(NO_3)_3{\cdot}9H_2O$, and iron (II) chloride tetrahydrate ($FeCl_2{\cdot}4H_2O$, 99.0+wt.%) were utilized to prepare synthetic solutions. Sodium sulfide ($Na_2S{\cdot}9H_2O$, >98 wt.%) was used as a precipitant in sulfide precipitation tests. Nitric acid ($HNO_3$, 67 wt.%) and sodium hydroxide (NaOH, ≥ 97.0 wt. %) were utilized as pH regulators. All the chemicals were purchased from Thermo Fisher Scientific, USA. All solutions were prepared with Type I deionized water with a resistivity of 18.2 MΩ·cm at 25 °C.

**Methods:** Sulfide precipitation tests were performed on the synthetic solution containing Mn, Co, Ni, and Zn. Aluminum and iron were added at predetermined concentrations. The tests were performed at room temperature, and during the tests, suspension pH was maintained constant by frequently adding diluted $HNO_3$ and/or NaOH solutions. A sodium sulfide solution of 1 M was prepared and used as the precipitant. Representative samples were collected at different time intervals from the start of the precipitation process up to a total reaction time of 60 min. The solid and liquid fractions of the samples were separated by centrifuging, and the liquid fractions were analyzed using ICP-MS. Sulfur to metal molar ratio ($k$) was calculated based on the concentration and the molar weight of the elements (i.e., Al, Fe, Mn, Co, Ni, and Zn) existing in the feed solution, following the equations below:

$$\boldsymbol{k = \frac{n_s}{n_m}}$$

$$\boldsymbol{n_s = C_s \times V_s}$$

$$n_m = \sum \frac{C_i \times V_f}{1000 \times M_i}$$

where $n_s$ and $n_m$ represent the moles of sulfur and valuable elements, respectively; $C_S$ the concentration of sodium sulfide solution (1 M); $V_S$ the volume (L) of sodium sulfide solution; $C_i$ the concentration (mg/L) of the elements in the feed solution ($i$ = Al, Fe, Mn, Co, Ni, and Zn); $V_f$ the volume (L) of feed solution; $M_i$ is the molar mass (g/mol) of the valuable elements. Precipitation recovery of the elements was calculated based on elemental concentrations of the feed solution and the supernatants generated from the tests, with considering the dilution effect caused by HCl and/or NaOH additions and representative sample collections.

Results and Discussion

1. The influence of pH on the sulfide precipitation

The sulfide precipitation tests were performed on the synthetic solution, and the recoveries of Mn, Co, Ni, and Zn as a function of pH and reaction time are presented in Figure 3.8.97. As shown in the figures, rapid precipitation kinetics were observed for all the elements, with maximum recoveries being obtained within the first 5 min of reaction. However, with the prolongation of the reaction time, the recovery of all the elements decreased to varying degrees. The recovery of Mn was significantly decreased with prolonging the reacting time. To efficiently separate Mn from other elements, it is essential to control the aging time of the sulfide precipitation process in 10 minutes. To avoid the formation of hydroxide precipitates, the tests were conducted at low pH values (i.e., 2, 3, 4, and 5). As presented in Figure 3.8.97(a), around 4% of Mn was recovered at pH 2 after reacting for 5 min, and only small increases in the recovery were observed when the pH increased to 3. However, when the pH increased to 4 and 5, after reacting for 5 min, the recovery of Mn increased to 10% and 15%, respectively. Figure 3.8.97(b) shows that pH imposed minor effects on Zn recovery, and 100% of the Zn was precipitated at all the four pH values after reacting for 5 min. Figure 3.8.97(c) and (d) show that at pH 2, the highest recoveries of Co and Ni were 60% and 80%, respectively, and the recovery was significantly improved to around 90% when raising the pH to 4. Around 100% of Co and Mn could be recovered at pH 5.

The impact of sulfur to metal molar ratio ($k$) was evaluated to optimize the selectivity of sulfide precipitation. The equations used to calculate $k$ were described in the Methods section. The recoveries of Mn, Zn, Co, and Ni as a function of $k$ and reaction time are presented in Figure 3.8.98. As shown in Figure 3.8.98(a), the recovery of Mn was only around 15% when $k$ was 0.5 and 1, and the recovery increased to around 35% and 40%, respectively, when $k$ was enhanced to 1.5 and 2. Compared with Mn, the recovery of Co and Ni shown in Figure 3.8.98(c) and (d) were much higher, and $k$ considerably affected the recovery. Around 40% of Co and Ni were precipitated with a $k$ of 0.5, while the recovery of Co and Ni increased to 90% when $k$ was enhanced to 1. Further increasing $k$ to 1.5 and 2 improved the recovery of Co and Ni to 100%. Compared with Mn, Co, and Ni, the recovery of Zn was not greatly influenced by $k$, with around 100% recovery being achieved even with the lowest $k$ of 0.5 (Figure 3.8.98 (b)).

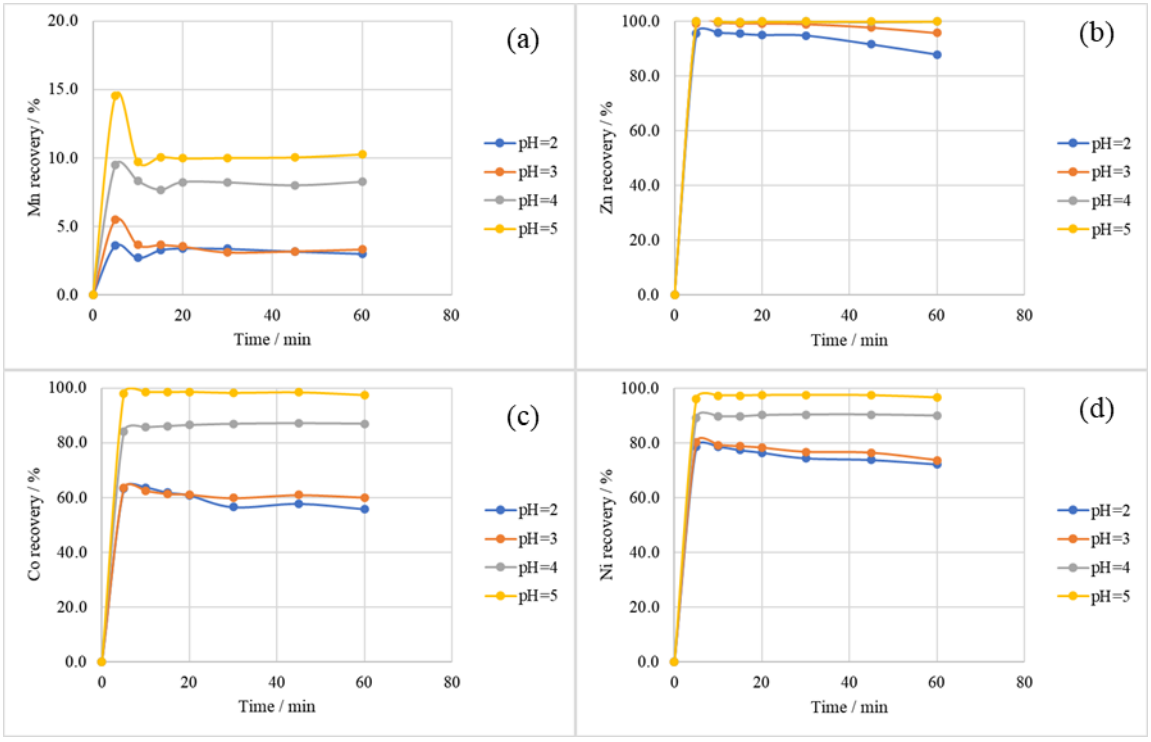


Figure 3.8.97. Effects of pH on the recovery of (a) Mn, (b) Zn, (c) Co, and (d) Ni (k = 1).

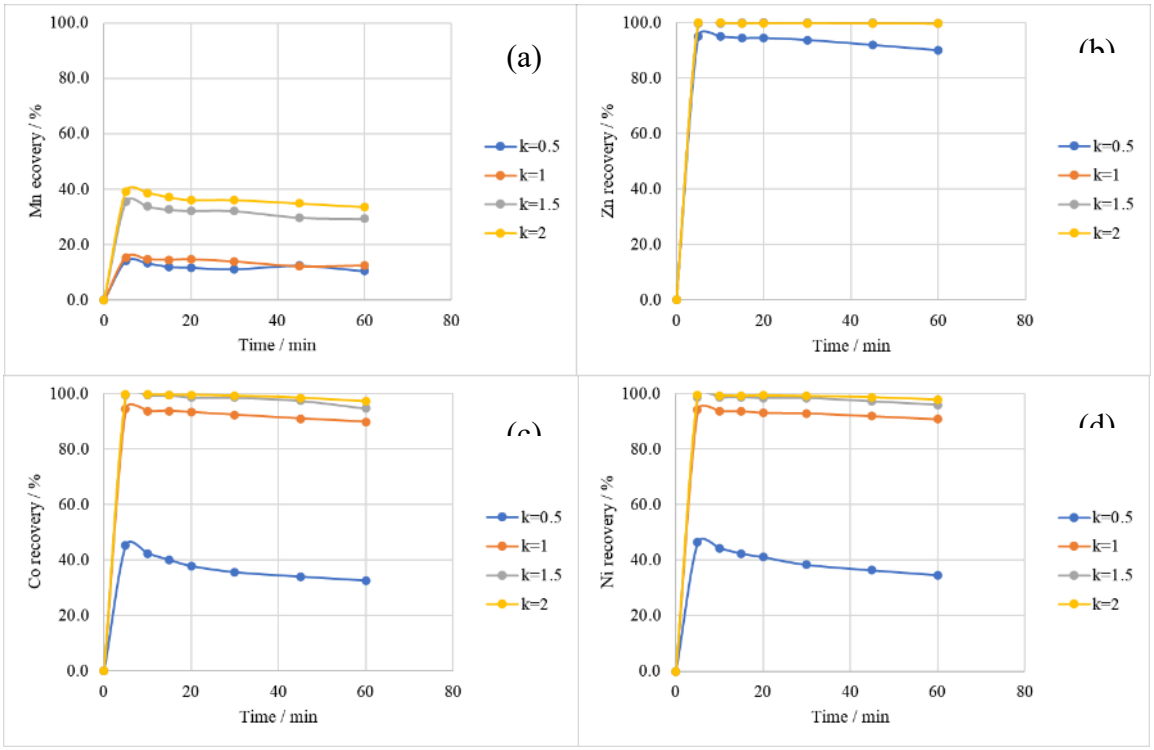



Figure 3.8.98. Effects of the molar ratio of sulfur to metal (k) on the recovery of (a) Mn, (b) Zn, (c) Co and (d) Ni (pH=4.00±0.05).

2. The influence of contaminant elements on sulfide precipitation

The effect of aluminum on the recoveries of manganese (Mn), zinc (Zn), cobalt (Co), and nickel (Ni) in sulfide precipitation was investigated. The synthetic solutions containing different concentrations of aluminum were prepared, and the sulfur to metal molar ratio (*k*) was calculated based on the concentration and the molar weight of the valuable elements (i.e., Mn, Co, Ni, and Zn) existing in the synthetic solution. The recoveries of the elements are shown in Figure 3.8.99. Aluminum was barely recovered when the initial pH of the prepared solution was 1.5. However, when preparing the solution with initial pH of 4, aluminum was recovered, and the recoveries decreased with increasing the concentration of aluminum. The recovery of the Mn was much lower than other valuable elements, and it decreased with increasing aluminum content. When the initial pH of the synthetic solution was 1.5 and contained 3,000 ppm aluminum, the recovery of Mn decreased to 0%. For the Zn, it can be seen that the recovery of Zn was not influenced by increasing the content of aluminum which was still 100% even when 3,000 ppm of aluminum was in the solution. Among the valuable elements, the recovery of cobalt was less affected by aluminum content. The recovery of Co was around 90% for both the solutions with initial pH of 1.5 and 4, and it was decreased to 80% and 70%, respectively when 3,000 ppm aluminum was in two synthetic solutions. The initial pH and aluminum content significantly influenced the recovery of Ni. When 0 ppm aluminum was in the solutions, the recovery of Ni was 70% and 90%, respectively, in two solutions, which indicated that initial pH of 4.0 would benefit the recovery of Ni. With the increasing concentration of aluminum, the recovery of Ni decreased significantly. Only around 40% and 30% of the Ni were recovered when 3,000 ppm of aluminum presented in the solutions.

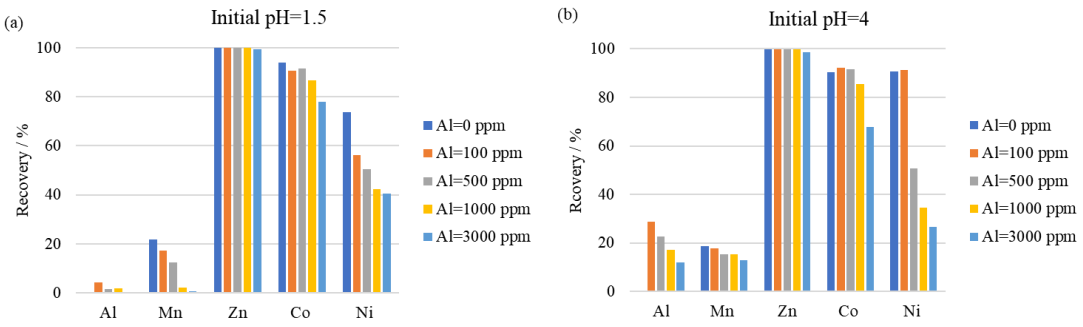



Figure 3.8.99. Effect of aluminum on the recoveries of valuable elements in sulfide precipitation (a) with initial pH 1.5; (b) with initial pH 4 ($k$=1 was calculated based on the concentration of valuable metals, Mn=1,500 ppm, Zn=2,000 ppm, Co=100 ppm, Ni=200 ppm; aging pH=4; aging time =10 min)

The effect of aluminum on the recoveries of the valuable elements was also investigated with increasing the sulfur content. The sulfur to metal molar ratio ($k$) was calculated based on the concentration and the molar weight of all the elements, including aluminum, in the synthetic solution. The recoveries of the elements are shown in Figure 3.8.100. For the solutions with initial pH of 1.5 and 4, the recovery of aluminum was around 45% when 3,000 ppm aluminum was added in the solution. Due to the excess sulfur, the recovery of manganese was increased with increasing the aluminum content. In the synthetic solution with initial pH of 1.5 and containing 3,000 ppm aluminum (Figure 3.8.100(a)), the recovery of Mn was around 50%. For Zn and Co, the recoveries for them maintained at 100% with increasing the aluminum content. In the synthetic solution with initial pH of 1.5, the recovery of Ni was increased with the increase of sulfur.

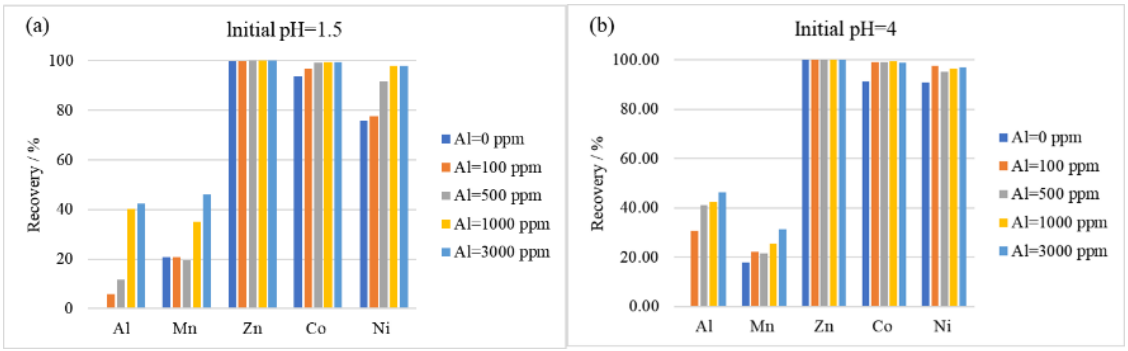



Figure 3.8.100. Effect of aluminum on the recoveries of valuable elements in sulfide precipitation (a) with initial pH 1.5; (b) with initial pH 4 ($k$=1 was calculated based on the concentration of all metals, Mn=1,500 ppm, Zn=2, 000 ppm, Co=100 ppm, Ni=200 ppm, Al=0, 100, 500, 1,000, 3,000 ppm; aging pH=4; aging time =10 min)

Iron was added in the synthetic solutions together with aluminum, and the results were shown in Figure 3.8.101. The solution was only prepared with an initial pH of 1.5 since iron could precipitate at pH 4. As shown in Figure 3.8.101(a), when aluminum was added in the solution (dark blue bar), the recovery of Ni was decreased significantly (74% to 46%), and the recovery of Co was also decreased from 95% to 75%. However, compared with aluminum, the existence of iron barely influenced the recovery of the valuable elements. The recovery of Mn, Zn, Co, and Ni remained the same when adding iron in the solution. When excessive sulfur was added to the solution (Figure 3.8.101(b)), the recovery of aluminum kept increase with increasing the sulfur and iron content. When 3,000 ppm aluminum and 1,000 ppm iron were in the solution, the recovery for these two elements was 60% and 90%, respectively, which is harmful to

producing high purity valuable element products. The recovery of Mn was also increased to 40% when excessive sulfur was added to the solution.

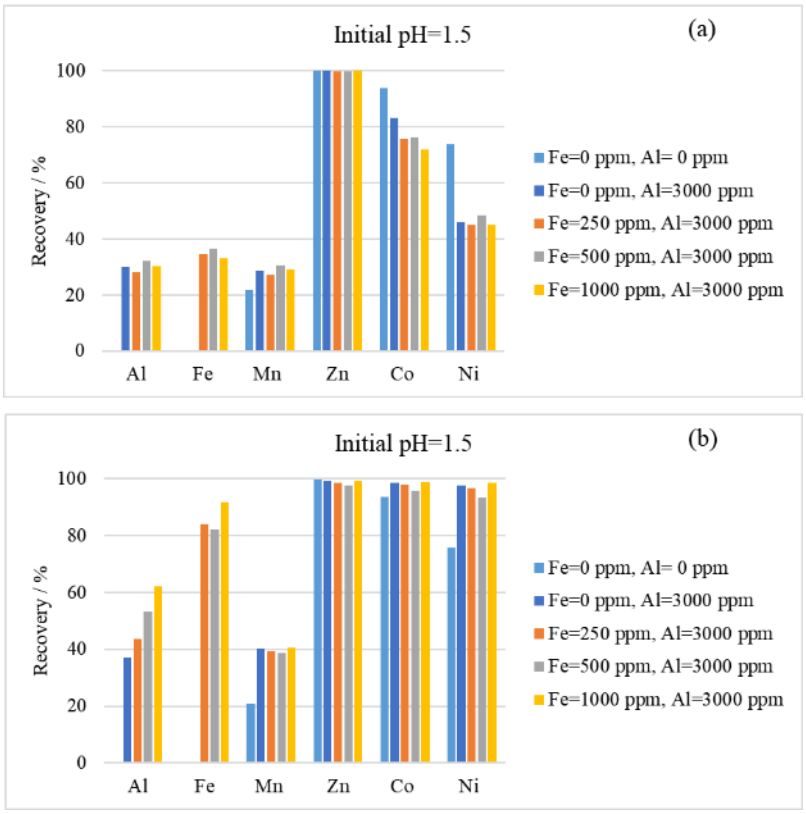



Figure 3.8.101. Effect of aluminum and iron on the recoveries of valuable elements in sulfide precipitation (a) *k* was calculated based on the concentration of valuable metals (b) *k* was calculated based on the concentration of all metals (Mn=1,500 ppm, Zn=2, 000 ppm, Co=100 ppm, Ni=200 ppm, Al=3,000 ppm, Fe=0, 250, 500, 1,000 ppm; initial pH 1.5, aging pH=4; aging time =10 min).

## 3.9 ENVIRONMENTAL MONITORING AND MANAGEMENT (TASK 11)

The pilot plant facility has a well-documented and effective environmental control plan. In addition, many of the new process circuitry components were specifically selected to mitigate existing environmental hazards and improve the environmental outcomes of the REE recovery process. For example, the integration of pre-leach roasting was expected to raise the target pH needed for effective REE leaching. Higher leaching pH typically leads to reduced contamination in the process water, particularly contamination from radionuclides such as uranium and thorium. Despite these potential benefits, an updated assessment of the environmental impacts, both real and perceived, was necessary to ensure appropriate controls and/or mitigation strategies.

To address this requirement, University of Kentucky researchers at the pilot plant worked with environmental specialists from the host site to conduct an initial environmental analysis prior to pilot-scale testing. The results of this analysis were used to update the environmental monitoring and control plan at the facility. Waste products from the laboratory-scale evaluations (i.e. Tasks 6 – 10) were evaluated to determine the appropriate control strategies, and modifications to the pilot plant will be made as needed. Continuous environmental monitoring was conducted at the pilot facility throughout the duration of the project.

An amendment environmental control plan was established on the basis of the environmental control plan of the existing pilot plant facility, with integrated actions addressed on the new circuit components.

It is noted that a detailed study of environmentally sensitive elements and the process stream where they were concentrated was conducted as part of the pilot plant study. The findings are reported in the later sections of this report associated with Tasks 15 and 17.

### *1.1.12 Air pollutants control and management*

The existing pilot plant is equipped with a well-designed central ventilation system that exhausts the mists or vapors generated as part of the hydrometallurgical extraction system. The central ventilation system is powered by a 16” Duct Inline Mixed Flow Centrifugal Fan (4,274 CFM) to exhaust any possible acid vapor, water vapor, and volatile organic compounds (VOCs) from the leaching and solvent extraction circuit. Air vents were installed on top of the leaching tanks and over the solvent extraction system. In the new circuit, the ventilation system will be extended to the bio-oxidation reaction tank to capture any possible gasturous emission. Since calcite will be used in the new circuit for neutralization purpose, possible carbon dioxide emission is accounted. Potential air emission sources and control plan is shown in Table 3.9.1.

Upon the installation of a 6” tube furnace roaster in 2019, a Donaldson TD486 dust collection system was installed outside of the pilot plant facility to handle the dust produced during crushing and roasting process. Air quality monitors were installed both in the plant area and the lab area to monitor the air quality including total VOCs, PM2.5, PM10, $CO_2$, etc. Carbon monoxide detector was installed in the plant area to monitor any CO emission and trigger alarm at critical concentration. Respirators with combo organic vapor, acid vapor, and P100 particulate filter cartridges have been used for personal protection.

Table 3.9.1.Air emission sources and control plan.

| Process | Source | Type | Actions |
|---|---|---|---|
| Biooxidation | Carbon Dioxide | Gas | Ventilation |
| Leaching | Water Vapor | Vapor | Ventilation |
| Solvent Extraction | Hydrochloric Acid | Vapor | Ventilation |
| Redissolution | Hydrochloric Acid | Vapor | Ventilation |
| Solvent Extraction | Orfom | Vapor | Ventilation |
| Crushing | Dust | Powder | Dust Collector |
| Roasting | Dust | Powder | Dust Collector |
| Neutralization | Carbon Dioxide | Gas | Ventilation |

### *1.1.13 Waste disposal management*

The projected waste streams in the current pilot plant hydrometallurgy circuit includes three solid waste streams and one liquid waste stream as shown in **Error! Reference source not found.**. Water samples were collected periodically and analyzed using an ICP-OES located on site as well as an independent commercial lab. Water pH and conductivity measurements of the wastewater were conducted. Solid samples were collected and analyzed for toxic elements. The data is reported in later sections of this report.

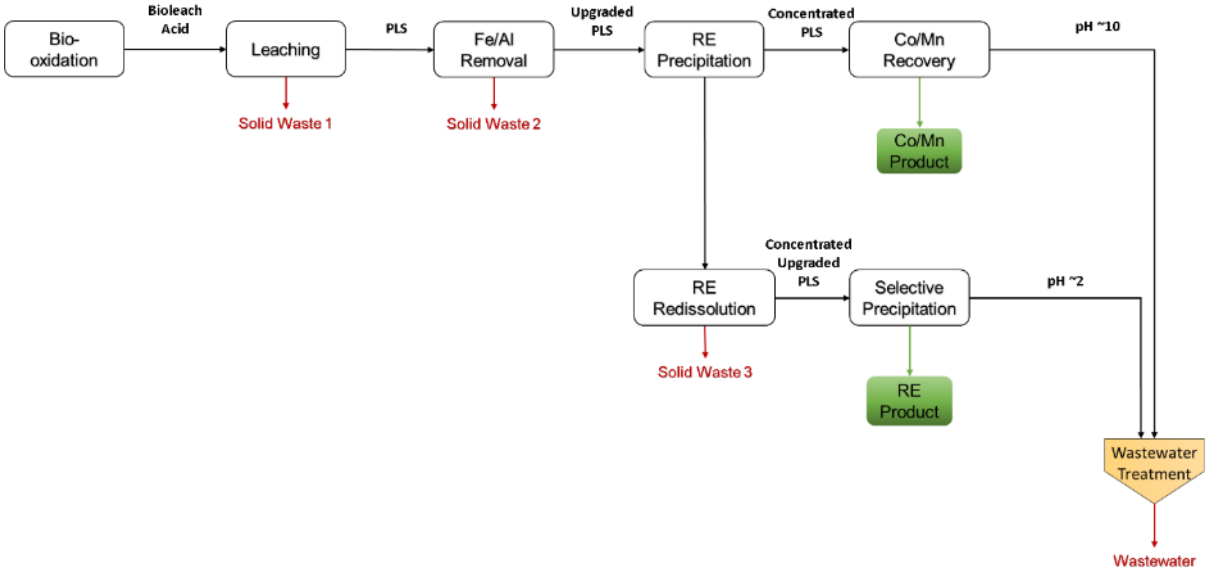



Figure 3.9.1.Waste disposal sample point and monitoring.

**Non-chemical process related water:** The wastewater from housekeeping and washdown purposes was discharged to an on-site water pond. A floor drain and sump was used to capture wash water and pumped out of the plant to 5000 gallon tanks and transported to an off-site permitted facility. Any potential spills of chemicals during operation was pumped to the wastewater treatment tank. Water quality in the water pond nearby the pilot plant facility was monitored periodically for any abnormality and toxication. During the June to August 2020, when the pilot plant was active, water samples collected from the water pond showed no sign of environmental issues as shown in Table 3.9.2.

Table 3.9.2. Monitoring of the toxic elements in surrounding water pond at the pilot plant.

| Sample Location | Date | As ppm | Be ppm | Cd ppm | Cr ppm | Cu ppm | Ni ppm | Pb ppm | Se ppm | Tl ppm | Zn ppm | Th ppm |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Pond1 | 6/23/2020 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.005 | < 0.001 | < 0.016 |
| Pond1 | 6/23/2020 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.008 | < 0.001 | < 0.005 | < 0.001 | < 0.001 |
| Pond2 | 6/23/2020 | < 0.001 | < 0.001 | < 0.001 | 0.006 | 0.019 | 0.023 | < 0.001 | < 0.001 | 0.003 | 0.026 | < 0.020 |
| Pond2 | 6/23/2020 | < 0.002 | < 0.001 | < 0.001 | 0.006 | 0.013 | < 0.020 | < 0.001 | < 0.001 | < 0.009 | 0.026 | < 0.001 |
| Pond1 | 7/8/2020 | < 0.001 | < 0.001 | < 0.001 | < 0.000 | < 0.001 | < 0.002 | < 0.001 | 0.039 | < 0.005 | < 0.001 | < 0.001 |
| Pond2 | 7/10/2020 | < 0.006 | < 0.001 | < 0.001 | 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.004 | < 0.001 | < 0.001 |
| Pond1 | 7/24/2020 | < 0.001 | < 0.001 | < 0.000 | < 0.001 | < 0.001 | < 0.002 | < 0.001 | < 0.001 | < 0.003 | < 0.001 | < 0.004 |
| Pond1 | 7/24/2020 | < 0.001 | < 0.001 | < 0.000 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.002 | < 0.001 | < 0.008 |
| Pond2 | 7/24/2020 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.002 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.005 | < 0.015 |
| Pond2 | 7/24/2020 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.000 | < 0.002 | < 0.015 |
| Pond1 | 7/31/2020 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.003 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 |
| Pond2 | 7/31/2020 | < 0.008 | < 0.001 | < 0.001 | < 0.001 | < 0.002 | < 0.001 | < 0.016 | < 0.006 | < 0.001 | < 0.001 | < 0.002 |
| Pond1 | 8/10/2020 | < 0.006 | < 0.013 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.002 | < 0.001 | < 0.001 | < 0.001 | < 0.001 |
| Pond2 | 8/10/2020 | < 0.001 | < 0.013 | < 0.001 | < 0.001 | < 0.001 | < 0.002 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 |
| Pond1 | 8/18/2020 | < 0.005 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.003 | < 0.001 | < 0.001 |
| Pond2 | 8/18/2020 | < 0.002 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.002 | < 0.001 | < 0.006 | < 0.012 | < 0.001 | < 0.013 |

**Chemical process related water:** The acidic wastewater stream generated in the pilot plant circuit was neutralized using sodium hydroxide to adjust the pH to 8-9 in a waste treatment tank. Any precipitates occurred during neutralization was captured in a pressure filter and discarded with the solid waste material. The filtrate was stored in two 5000-gallon tank then transported to the waste water pond at the Alliance's site for further treatment.

In the current modified circuit, the major processing stream that was sent to the treatment tank was the effluent after Co/Mn precipitation at pH~10, and a very small portion of the oxalic precipitation barren solution from the RE oxalate precipitation process (pH~2). The ratio of the pH10 and the pH2 stream was around 8:1 to 10:1. The typical concentration of the wastewater after pH adjustment at value of 10 obtained from lab experiments on the bioleached PLS and the AMD are shown in Table 3.9.3. The wastewater monitored during the period of July to September, 2020, that was hauled to the mine site for further treatment, is shown in Table 3.9.4.

Table 3.9.3. Concentration of the toxic elements in the pH 10 filtrate obtained in the preliminary lab tests.

| Test | | Date | As | Be | Cd | Cr | Cu | Ni | Pb | Se | Tl | Zn | Th |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | ppm | ppm | ppm | ppm | ppm | ppm | ppm | ppm | ppm | ppm | ppm |
| pH Filtrate | 10 | Bioleach PLS | <0.001 | <0.001 | <0.001 | <0.001 | 0.03 | <0.001 | 0.02 | <0.001 | <0.001 | <0.001 | <0.001 |
| pH Filtrate | 10 | Bioleach PLS | 0.12 | <0.001 | <0.001 | <0.001 | 0.01 | 0.01 | 0.37 | <0.001 | 0.02 | <0.001 | <0.001 |
| pH Filtrate | 10 | Bioleach PLS | <0.001 | <0.001 | <0.001 | <0.001 | 0.06 | 0.02 | 0.04 | 0.48 | <0.001 | 0.05 | <0.001 |
| pH Filtrate | 10 | AMD | NA | NA | NA | <0.001 | 0.06 | 0.07 | 0.02 | NA | NA | 0.36 | 0.008 |

Table 3.9.4. Monitoring of the toxic elements in the rough treated waste water at the pilot plant.

| Test | Date | As | Be | Cd | Cr | Cu | Ni | Pb | Se | Tl | Zn | Th |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | ppm | ppm | ppm | ppm | ppm | ppm | ppm | ppm | ppm | ppm | ppm |
| Wastewater | 2020/7/8 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.005 | < 0.001 | < 0.026 | < 0.001 | < 0.001 | < 0.001 |
| Wastewater | 2020/7/10 | < 0.001 | < 0.001 | < 0.001 | 0.002 | < 0.001 | < 0.001 | < 0.008 | < 0.001 | < 0.001 | 0.006 | < 0.010 |
| Wastewater | 2020/7/24 | < 0.001 | < 0.001 | 0.005 | 0.028 | 6.395 | 4.029 | < 0.030 | 0.325 | < 0.001 | 5.675 | 0.015 |
| Wastewater | 2020/7/24 | < 0.001 | < 0.001 | 0.005 | 0.03 | 6.377 | 3.974 | 0.055 | 0.308 | < 0.001 | 5.633 | < 0.019 |
| Wastewater | 2020/8/10 | < 0.054 | < 0.021 | 0.007 | < 0.001 | 0.186 | 0.676 | 0.038 | 0.108 | < 0.001 | 0.412 | < 0.001 |
| Wastewater | 2020/9/16 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | 0.019 | < 0.003 | < 0.001 | < 0.001 | < 0.001 | < 0.021 | < 0.010 |
| Wastewater | 2020/9/16 | < 0.020 | < 0.001 | 0.008 | 0.084 | 0.107 | 0.285 | 0.076 | 0.063 | < 0.001 | 1.563 | 0.042 |
| Wastewater | 2020/9/16 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.001 | < 0.004 | < 0.001 | < 0.009 | < 0.001 | < 0.001 | < 0.001 |
| Wastewater | 2020/9/16 | < 0.001 | < 0.001 | < 0.001 | 0.323 | 0.192 | 0.459 | < 0.011 | < 0.073 | < 0.001 | 0.478 | < 0.008 |
| Wastewater | 2020/9/18 | < 0.014 | < 0.013 | < 0.001 | < 0.001 | 0.073 | < 0.002 | 0.048 | < 0.002 | < 0.001 | 0.225 | < 0.005 |
| Wastewater | 2020/9/23 | < 0.001 | < 0.001 | < 0.001 | 0.008 | 0.092 | 0.018 | < 0.001 | < 0.001 | < 0.023 | < 0.036 | < 0.001 |

*Note that <0.001 ppm was used when the lower detection limit is less than 0.001 ppm. The Lower Limit of Quantification (LLOQ) was established using a series of multielement CRM standard solution at concenrtaiton of 0 µg/mL, 0.001 µg/mL, 0.005 µg/mL, 0.01 µg/mL, 0.05 µg/mL, and 0.1 µg/mL. For each aforementioned concentration level, a series of 20 replicate samples were prepared and analyzed. The results were used to establish the LLOQ, or the minimum detection level in which the concentration can be accurately determined for each element respectively. The criteria for this determination are that 95% of the results yield a value within 10% of the expected concentration and that the variance within the same at that concentration level have less than 5% variance within the sampled population.*

An independent commercial laboratory (MICROBAC) collected wastewater produced from the pilot plant on November 19, 2021. The results reported by the commercial laboratory are provided in Figure 1.1.2. The analysis noted low concentrations of environmentally sensitive elements and we were not required to implement any corrective actions by the host company based on these results.

**Analytical Testing Parameters**

| | | | |
|---|---|---|---|
| **Client Sample ID:** | Dotiki 3 | | |
| **Sample Matrix:** | Aqueous | **Collected By:** | CUSTOMER |
| **Lab Sample ID:** | L1K0999-01 | **Collection Date:** | 11/19/2021 12:04 |

Analyses Performed by: Microbac Laboratories, Inc. - Chicagoland

| Metals Total by ICPMS | Result | Limit(s) | MDL | RL | Units | Note | Prepared | Analyzed | Analyst |
|---|---|---|---|---|---|---|---|---|---|
| **Method: EPA 200.8, Rv. 5.4 (1994)** | | | | | | | | | |
| Silver | <0.050 | | 0.0026 | 0.050 | mg/L | D1 | 11/24/21 1212 | 12/01/21 1437 | RPL |

Analyses Performed by: Microbac Laboratories, Inc., Evansville

| Field Parameters | Result | Limit(s) | MDL | RL | Units | Note | Prepared | Analyzed | Analyst |
|---|---|---|---|---|---|---|---|---|---|
| **Method: SM 2550 B-2010** | | | | | | | | | |
| Temperature | 12.4 | | | | °C | | 11/19/21 1204 | 11/19/21 1453 | SHB |
| **Method: SM 4500-H+ B-2011** | | | | | | | | | |
| pH | 9.23 | | 1.00 | 1.00 | S.U. | | 11/19/21 1204 | 11/19/21 1453 | SHB |

Analyses Performed by: Microbac Laboratories, Inc., Louisville

| Inorganics Total | Result | Limit(s) | MDL | RL | Units | Note | Prepared | Analyzed | Analyst |
|---|---|---|---|---|---|---|---|---|---|
| **Method: EPA 420.4, Rv. 1 (1993)** | | | | | | | | | |
| Total Phenols | <0.050 | | 0.016 | 0.050 | mg/L | | 11/26/21 0704 | 11/26/21 1523 | EPL |
| **Method: SM 2510 B-2011** | | | | | | | | | |
| Conductivity at 25 C | 51000 | | 2.0 | 2.0 | umhos/cm | | 11/22/21 0733 | 11/22/21 0937 | CPC |
| **Method: SM 4500-CN⁻ E-2011** | | | **Method Notes: P1** | | | | | | |
| Cyanide | <0.010 | | 0.0035 | 0.010 | mg/L | | 11/24/21 0948 | 11/24/21 1440 | EPL |
| **Method: USGS I-3765-85** | | | | | | | | | |
| Solids, Total Suspended | 163 | | 5 | 5 | mg/L | | 11/22/21 1018 | 11/22/21 1354 | RXG |

| Metals Total by ICPMS | Result | Limit(s) | MDL | RL | Units | Note | Prepared | Analyzed | Analyst |
|---|---|---|---|---|---|---|---|---|---|
| **Method: EPA 200.8, Rv. 5.4 (1994)** | | | | | | | | | |
| Antimony | <0.50 | | 0.22 | 0.50 | ug/L | | 11/23/21 1056 | 12/03/21 1930 | JSW |
| Arsenic | 1.3 | | 0.27 | 0.50 | ug/L | | 11/23/21 1056 | 12/03/21 1930 | JSW |
| Beryllium | <1.0 | | 0.53 | 1.0 | ug/L | C3 | 11/23/21 1056 | 12/03/21 1930 | JSW |
| Cadmium | 1.9 | | 0.28 | 0.50 | ug/L | | 11/23/21 1056 | 12/03/21 1930 | JSW |
| Chromium | 3.7 | | 0.22 | 0.50 | ug/L | | 11/23/21 1056 | 12/03/21 1930 | JSW |
| Copper | <5.0 | | 1.1 | 5.0 | ug/L | | 11/23/21 1056 | 12/03/21 1930 | JSW |
| Iron | 260 | | 24 | 50 | ug/L | | 11/23/21 1056 | 12/03/21 1930 | JSW |
| Lead | <0.50 | | 0.33 | 0.50 | ug/L | | 11/23/21 1056 | 12/03/21 1930 | JSW |
| Nickel | 26 | | 0.69 | 1.0 | ug/L | B1 | 11/23/21 1056 | 12/03/21 1930 | JSW |
| Selenium | 25 | | 0.45 | 0.50 | ug/L | | 11/23/21 1056 | 12/03/21 1930 | JSW |
| Thallium | 1.1 | | 0.28 | 0.50 | ug/L | | 11/23/21 1056 | 12/03/21 1930 | JSW |
| Zinc | 19 | | 6.2 | 10 | ug/L | | 11/23/21 1056 | 12/03/21 1930 | JSW |
| Manganese | 550 | | 0.19 | 0.50 | ug/L | | 11/23/21 1056 | 12/03/21 1930 | JSW |

Microbac Laboratories, Inc.

**3323 Gilmore Industrial Blvd | Louisville, KY 40213 | 502.962.6400 p | www.microbac.com**

Figure 3.9.2 Wastewater analysis performed and reported by an independent commercial laboratory.

**Solid waste disposal:** The solids generated by the high-pressure filters will be collected in 55-gallon drums, which will then be mixed with lime and placed in a cement mixer and transported to the MSHA-permitted coarse refuse storage area at the coal company's coal preparation plant. The typical concentration of the toxic elements in the solid waste stream that will be reported to the solid disposal stream as shown in Table 3.9.5. TCLP tests will be conducted on representative samples taken from the solid waste material periodically to monitoring the mobility of the toxic elements prior to transport to the coal preparation plant. Additionally, the excessive amount of calcite generated as a byproduct of the pyrite concentration circuit will be a neutralization and diluent material be added to the solids waste material to encapsulate any environmentally sensitive elements into the fixed solids.

Table 3.9.5. Concentration of the toxic elements in the solid waste obtained in the preliminary lab tests.

| Sample | As | Be | Cd | Cr | Cu | Ni | Pb | Se | Tl | Zn | Th |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | ppm | ppm | ppm | ppm | ppm | ppm | ppm | ppm | ppm | ppm | ppm |
| Bioleach Solid Residue | 45.2 | 0.0 | 11.1 | 90.7 | 57.1 | 30.8 | 186.9 | 25.0 | 0.0 | 53.2 | 17.7 |
| Bioleach Solid Residue | 44.2 | 0.0 | 11.9 | 94.0 | 53.1 | 35.9 | 189.1 | 27.6 | 0.0 | 53.5 | 18.2 |
| Fe/Al Precipitation Cake (Calcite) | 41.5 | 0.0 | 28.4 | 15.5 | 68.5 | 38.0 | 97.7 | 0.0 | 0.0 | 969.8 | 9.0 |

### *1.1.14 Environmental Study*

Solid residuals produced from the leaching processing and the various precipitation steps have an acidity that could continue reactions harmful to the environment. A concept studied was to mix calcite enriched during pyrite recovery with the solid waste produced by the pilot plant to neutralize the material prior to waste storage. The reduced acidity would cause less rare earth elements, critical materials, and environmentally hazardous elements to be leached from the waste as the pH increased. As such, a systematic study was designed to study the impact of calcite addition on the leaching characteristic of leaching solid residues obtained from various plant operations and the results were compared to a control sample (no calcite).

The setup used in the study is shown in Figure 3.9.3. The cylinders were 30-cm tall x 16.5-cm diameter with a roughly 16.5 x 16.5 square cm of netting (Figure 3.9.4) at the bottom to impede finer material from falling out of the nine 2.5-mm holes in the bottom. Fifteen cylinders with netting made to receive the material were utilized. Four different waste samples were studied materials; leached material from the Pilot Plant's Heap Leach, Leached Shakespeare (SHXP) from its third Plant test, an un-leached WK-13 column charge from the Leach Column test, and leached WK-13 (WKLSR) from the Pilot Plant's final WK-13 test, were gathered in standard sized 30-cm x 16.5-cm concrete test cylinders arrayed inside University of Kentucky's Dotiki Pilot Plant.



Figure 3.9.3 Complete test setup. In back three rows left to right: Heap, SHXP, and WKY13 materials. Rear to front: 0, 5, 10, 20 mass% calcite additions. In foreground is the WKLSR test material, right to left is 0, 10, and 20 mass% calcite additions

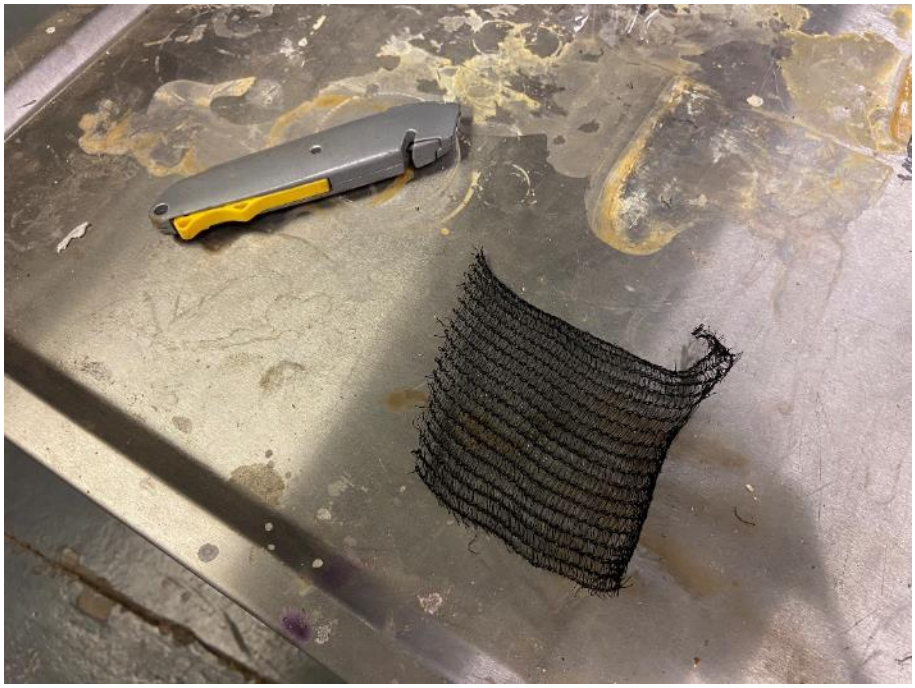


Figure 3.9.4 An example of the netting placed at the bottom of the test cylinders prior to loading

Cylinder material was gathered from its corresponding sources. Heap material was shoveled out of four corners of the Pilot Plant's Heap Leach into one five-gallon bucket. SHXP material was shoveled out of the #3 LSR waste barrel from Pilot Plant test SHXP-C4 into another 13-liter bucket. WK-13 was drawn from 7.7-kgs of solids and 8lbs of fines mixed by shovel then transferred into yet another bucket. The calcite to be used for mixing was drawn from the Monyo filter overflow barrel in the Pyrite Extraction Circuit of the Pilot Plant. This was also gathered in a fourth five-gallon bucket. On June 1st, three additional cylinders with WK-13 LSR material were added to the test, and these were collected from the #3 WK-13 LSR barrel using the same method as the SHXP material was collected. The calcite for this test was drawn from calcite that was separated using a spiral instead of the Monyo filter in the Pilot Plant's pyrite circuit.

The mixtures were made in the following manner. For the 0 mass% calcite mixtures, the control cylinders were placed on a lab scale that was then zeroed. Next, 4.5 kgs of material was loaded directly into the cylinder. For the mixtures, a different, clean bucket was used as a mixing basin. The bucket was placed on the lab scale, zeroed, and then calcite was added in the weight corresponding with its percent mixture (e.g., 5% calcite would be .23 kg of calcite for a 4.5 kg total material mixture in a cylinder). Once the calcite weight was reached, the remaining material was added. After adding 4.5 kgs of total material, the bucket was removed from the scale and the material within was kneaded together by hand into a homogeneous mass. This mass was then loaded into the test cylinder, which was in turn placed into a disposable aluminum baking sheet that was a minimum of 76 mm deep with its plastic lid turned inside out onto its floor, in contact with the bottom of the cylinder, creating a layer of plastic over the aluminum.

For the rain event simulator, a 13-liter bucket had a hole cut out of its bottom in which a 25-mm bulkhead fitting was placed and caulked. Attached to this bulkhead was a 25-mm valve connected to a small shower head. In this bucket 2.5L of de-ionized water was measured via

graduated cylinder. Due to the bulkhead at the bottom of the bucket not being flush, 500mL of additional water had to be added to ensure that 2L of de-ionized water flows out of the showerhead into the cylinder.

The shower bucket was hung from a stand with the handle approximately 122-cm from the ground, and the shower head 56-cm from the ground. One at a time, the cylinders are placed under the rain simulator and its valve is opened just wide enough to ensure that the spray covers the whole surface of the cylinder material. Flow is monitored and regulated to prevent overflow. This can vary by material. Fluid will flow out from the bottom of the test cylinder into the collection basin. Once 2L of water has fallen let the sample sit for 12 hours before getting a 50ml digestion tube and collecting the drainage in the basin. The 50ml sample is then microfiltered, probed for pH and Eh, and then used to create dilutions for ICP examinations. The rain events are one material a week on a rotation. Heap material was rained on the first week, SHXP the next, then WKY13 the week after that, and finally the WKLSR after that, then starting over again with the Heap material the following week.

**Results:** Initially, the hypothesis that calcite admixture raised pH was correct. pH increased with calcite mass%. Over the course of the simulations, four for the original three materials and two for the WKLSR, pH in calcite blends increased for a period but tapered off and began to decrease. The decrease more significant the less calcite mass% present, and vice versa. Eh readings were more erratic, although SHXP and WKY13 materials showed a generally increasing trend, while the Heap material had a declining trend outside of its control test. Not enough information is yet available to report Eh trends for WKLSR.

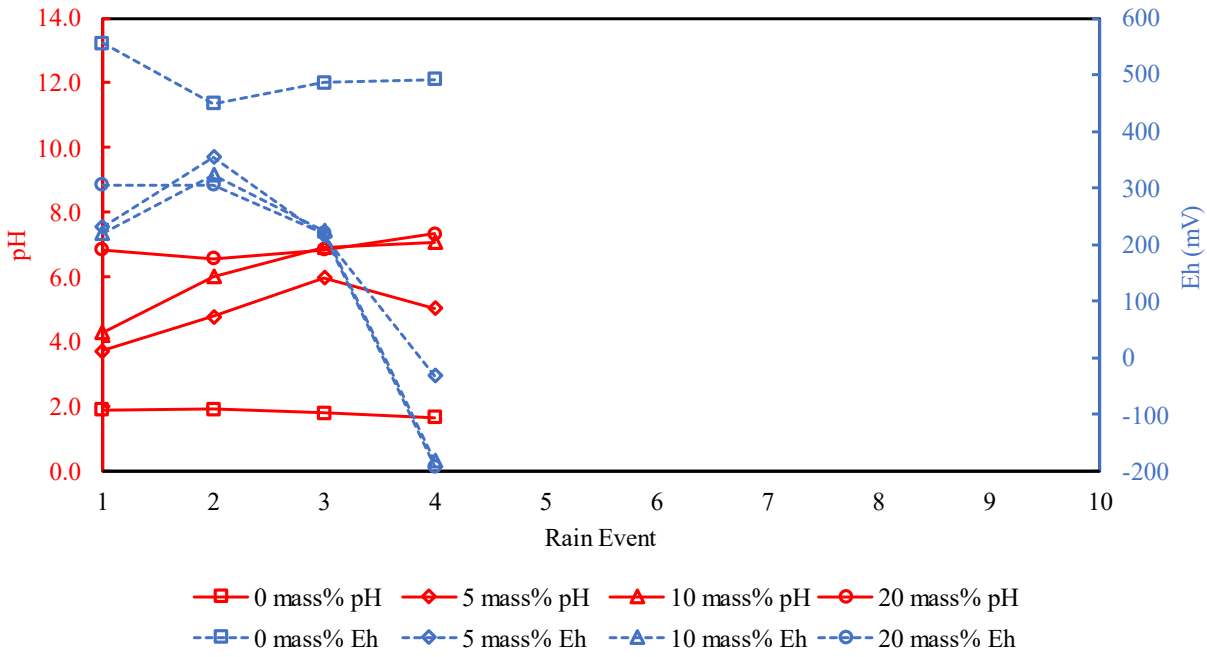



Figure 3.9.5 Influence of various concentrations on the pH and Eh readings for Heap material

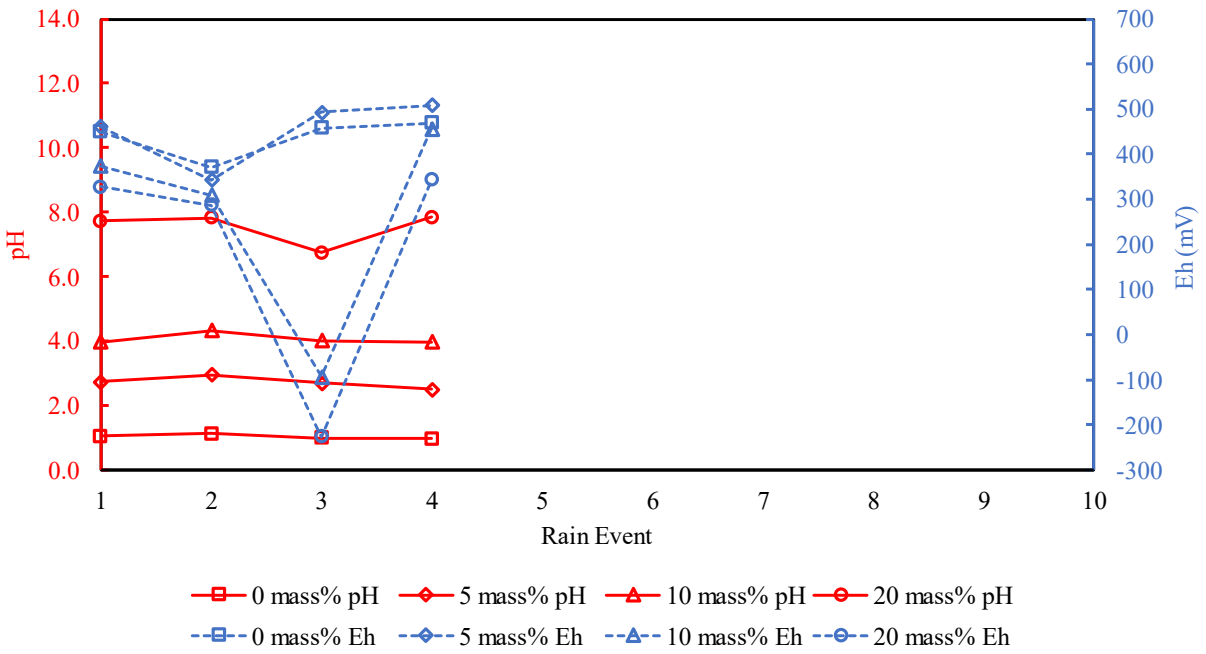



Figure 3.9.6 Influence of various concentrations on the pH and Eh readings for SHXP material

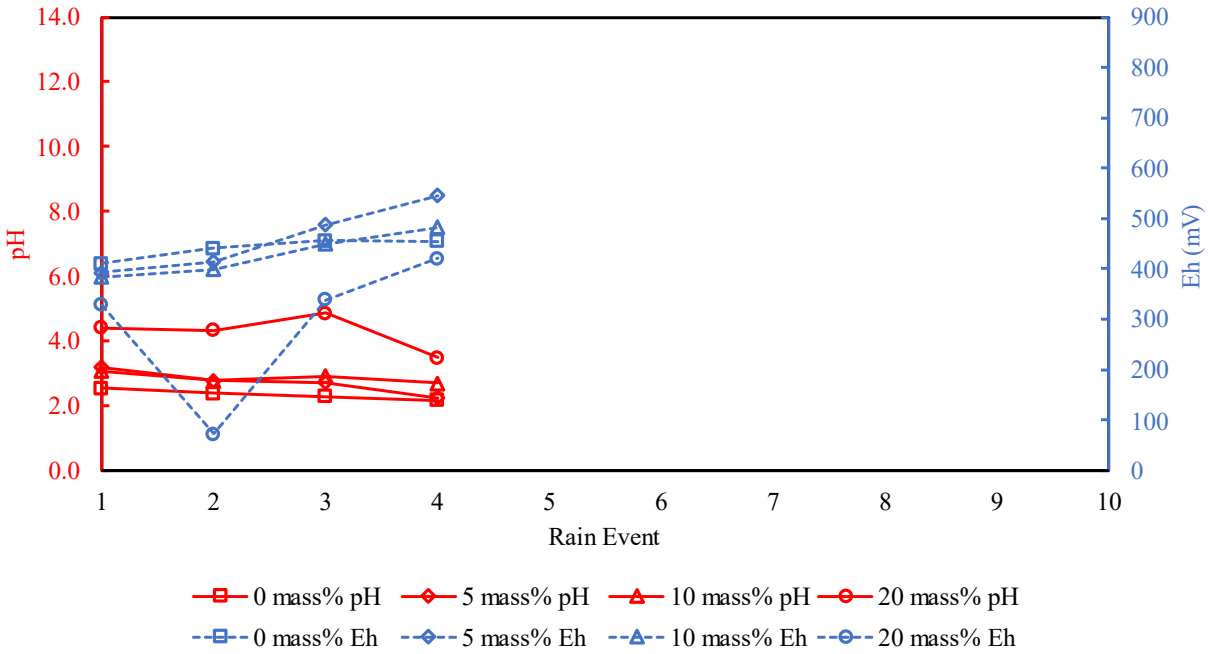



Figure 3.9.7 Influence of various concentrations on the pH and Eh readings for WK-13 material

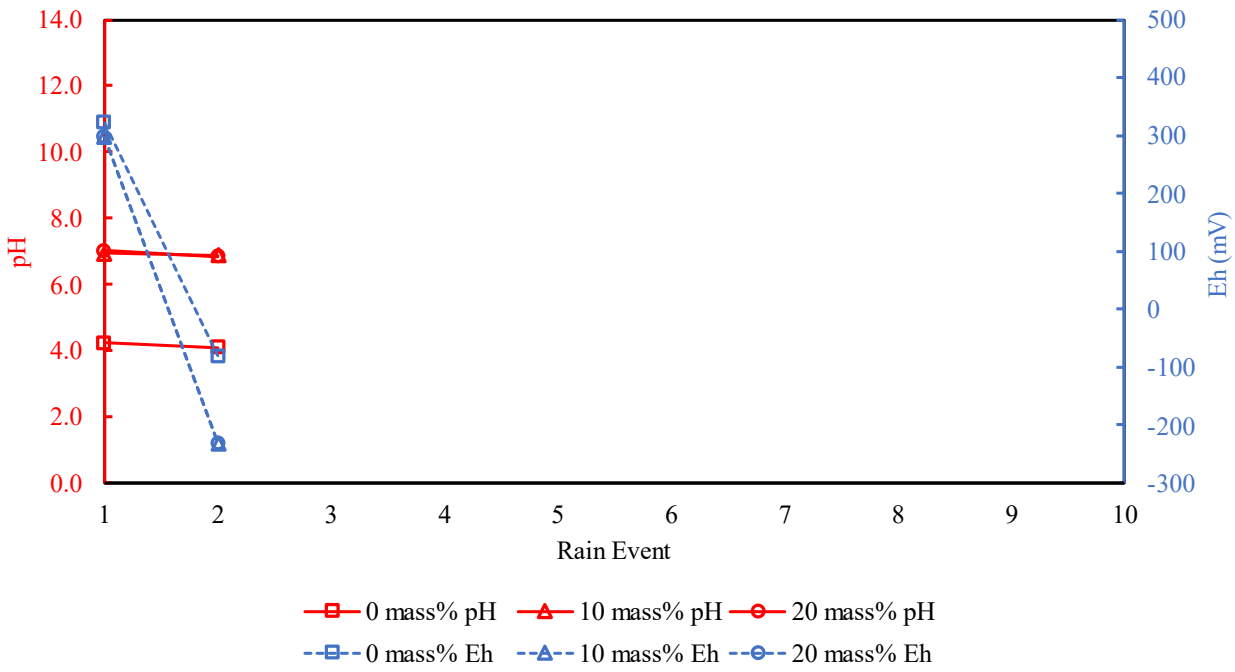



Figure 3.9.8 Influence of various concentrations on the pH and Eh readings for WK-13 LSR material

Calcite's effect on material leaching was in line with the hypothesis, but as pH's decreased, leaching did not increase as expected. Heap and SHXP materials still show declines in TREEs (Total Rare Earth Elements) concentrations despite a decline in pH. WKY13 material is so far the only material that has seen its TREE concentrations increase as its pH decreases.

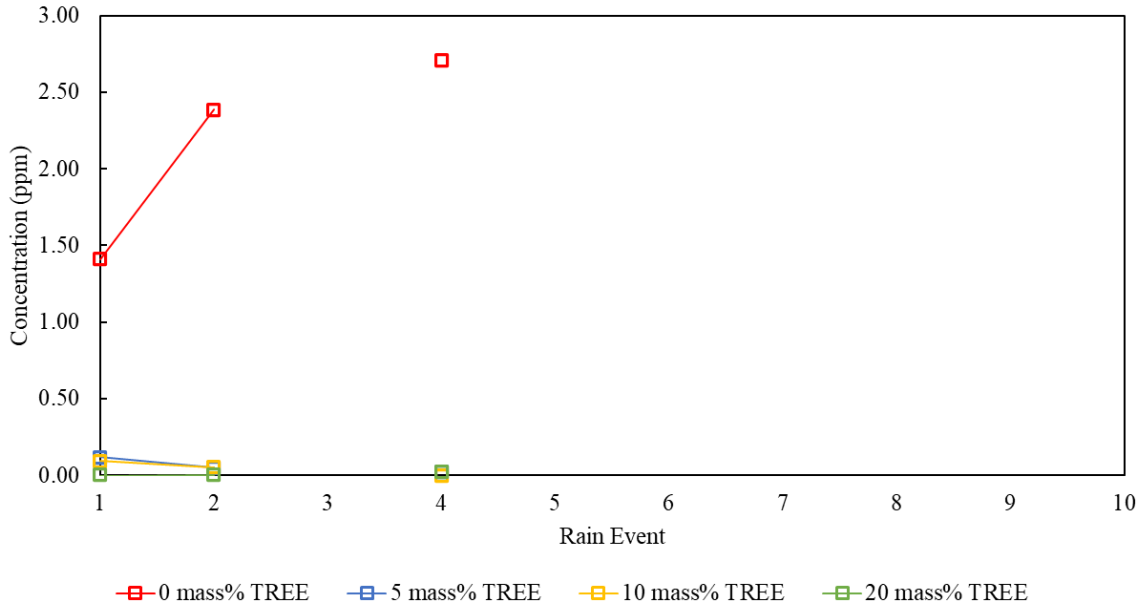



Figure 3.9.9 Leaching behavior of rare earth elements from heap leach material as a function of various calcite concentrations

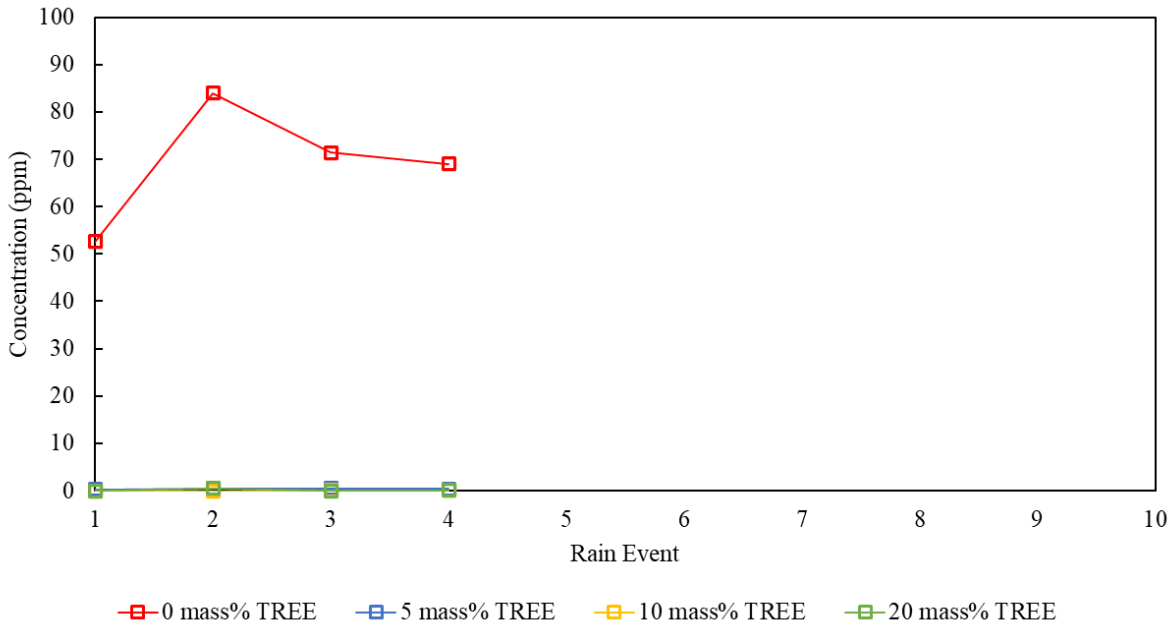



Figure 3.9.10 Leaching behavior of rare earth elements from SHXP material as a function of various calcite concentrations

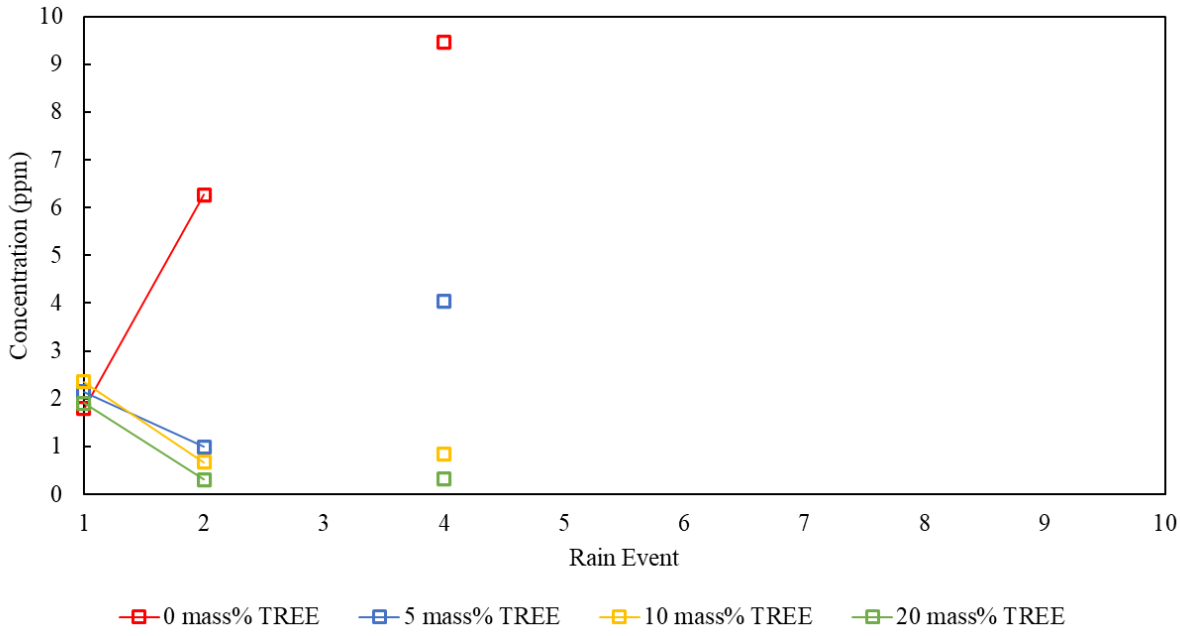



Figure 3.9.11 Leaching behavior of rare earth elements from untreated WK-13 material as a function of various calcite concentrations

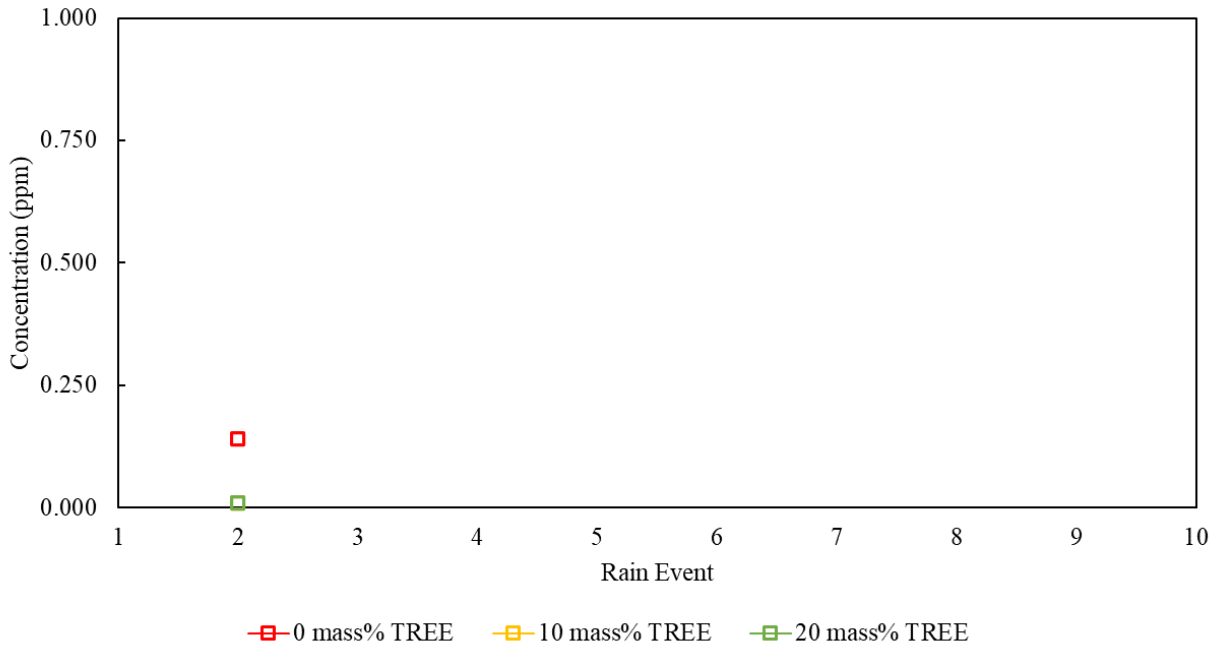



Figure 3.9.12 Leaching behavior of rare earth elements from WK-13 LSR material as a function of various calcite concentrations

Concentrations of environmentally hazardous elements were also surveyed across all tested materials. Arsenic, lead, mercury, and selenium are commonly cited examples of pollutants from mine runoff. Calcite also significantly impeded the leaching of these elements.

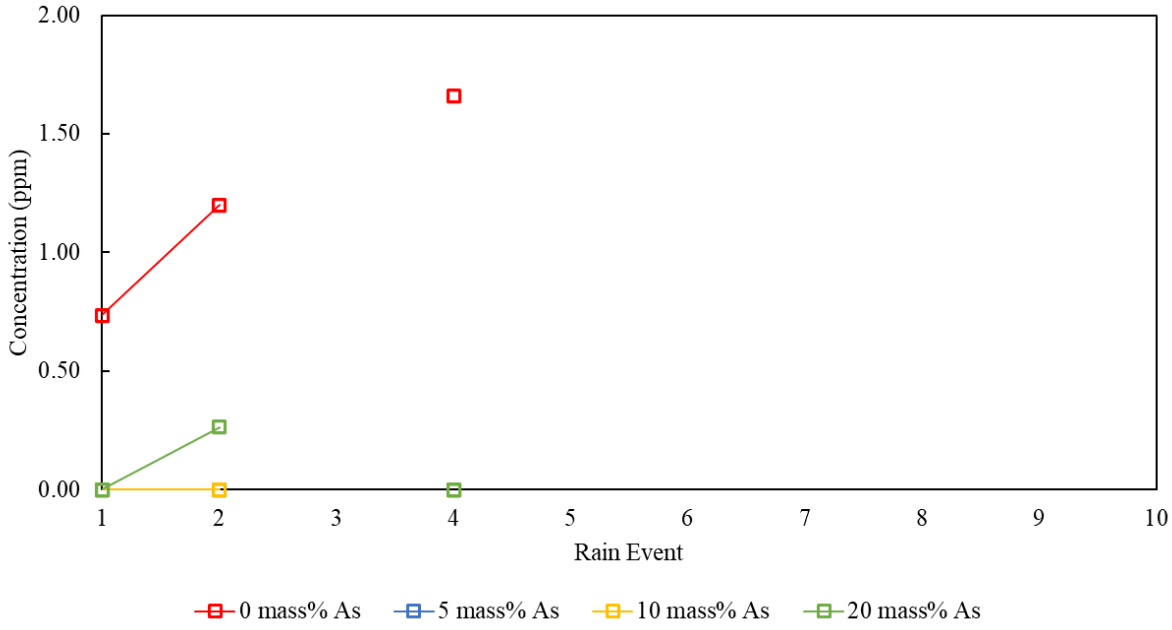



Figure 3.9.13 Leaching behavior of As from heap leach material as a function of various calcite concentrations

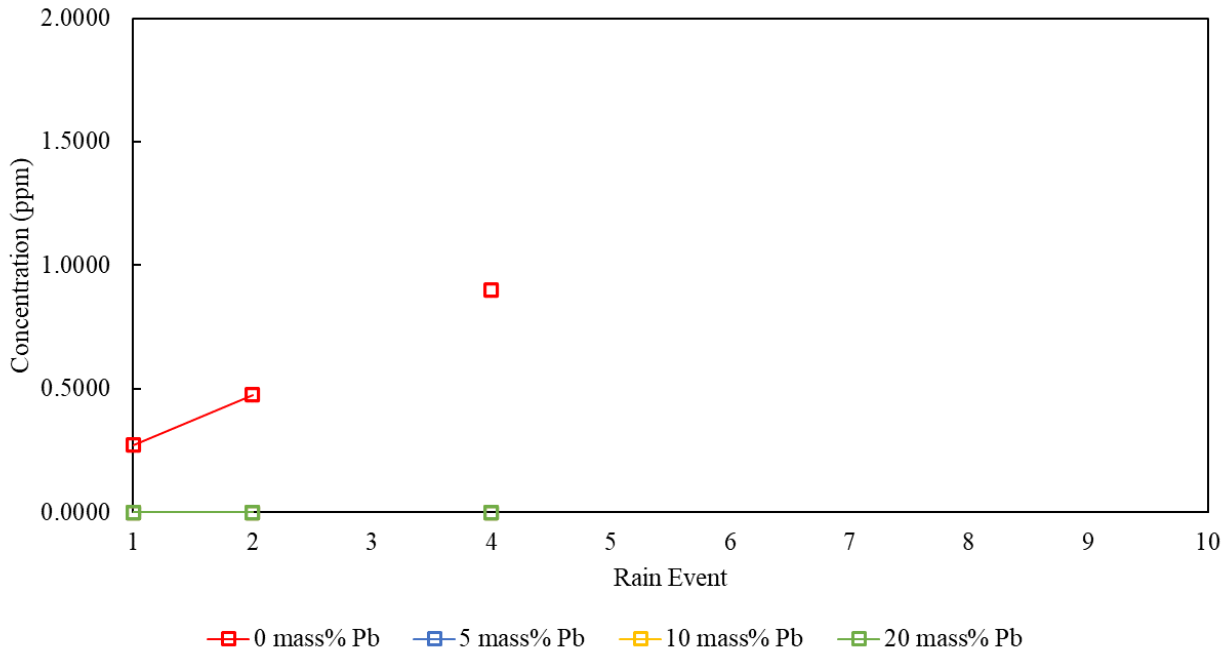



Figure 3.9.14 Leaching behavior of Pb from heap leach material as a function of various calcite concentrations

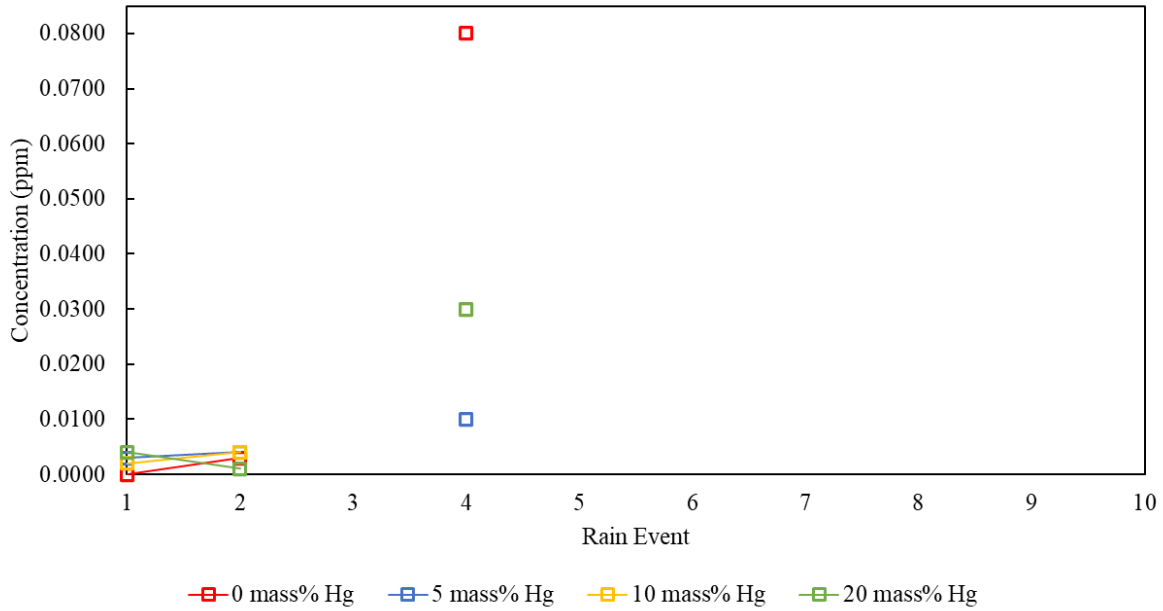



Figure 3.9.15 Leaching behavior of Hg from heap leach material as a function of various calcite concentrations

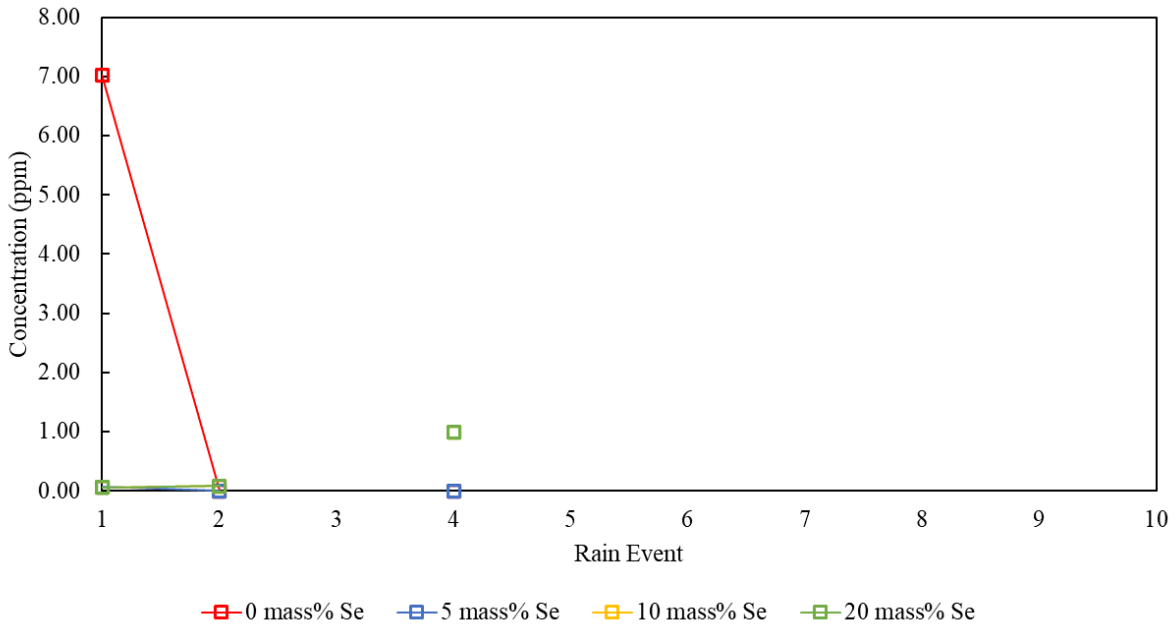



Figure 3.9.16 Leaching behavior of Se from heap leach material as a function of various calcite concentrations

Other than the possibly erroneous reading for lead concentration for the 20 mass%, the control test had the only detectable concentrations of arsenic and lead. Mercury concentrations began below .01 ppm but later saw concentrations increase, with the non-calcite blended control having the most significant increase. Selenium concentrations started at 7 ppm for the control but fell to undetectable levels after. Only in the fourth rain event was another concentration detected in the Heap 20 mass%.

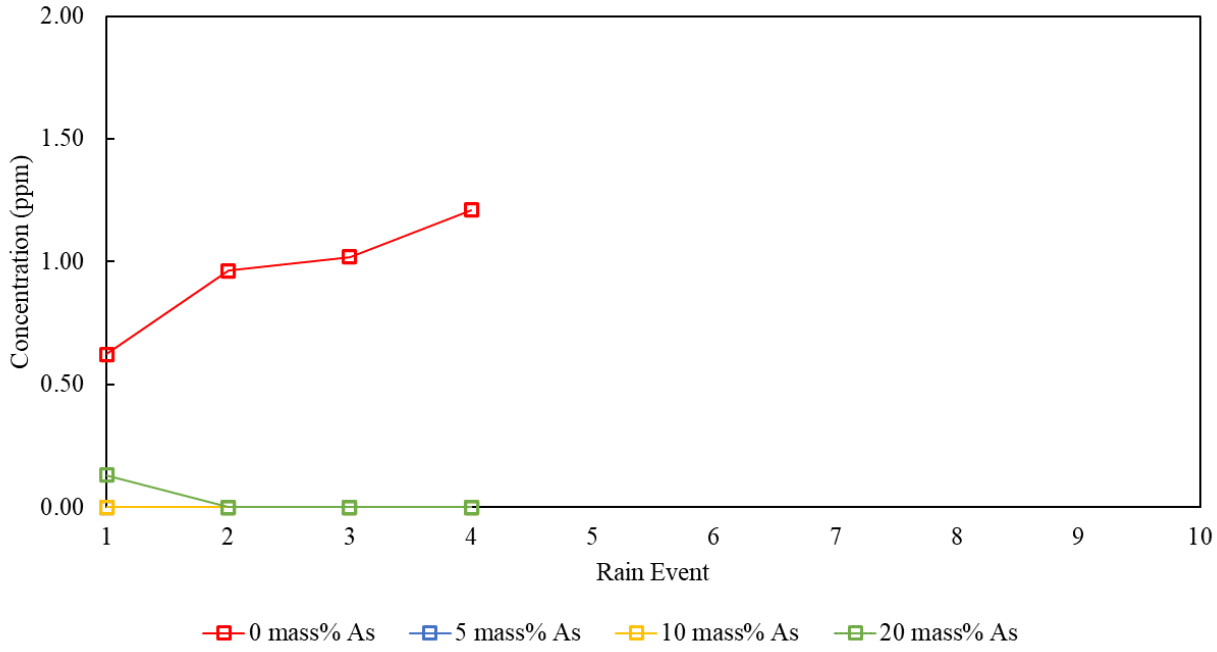



Figure 3.9.17 Leaching behavior of As from SHXP material as a function of various calcite concentrations

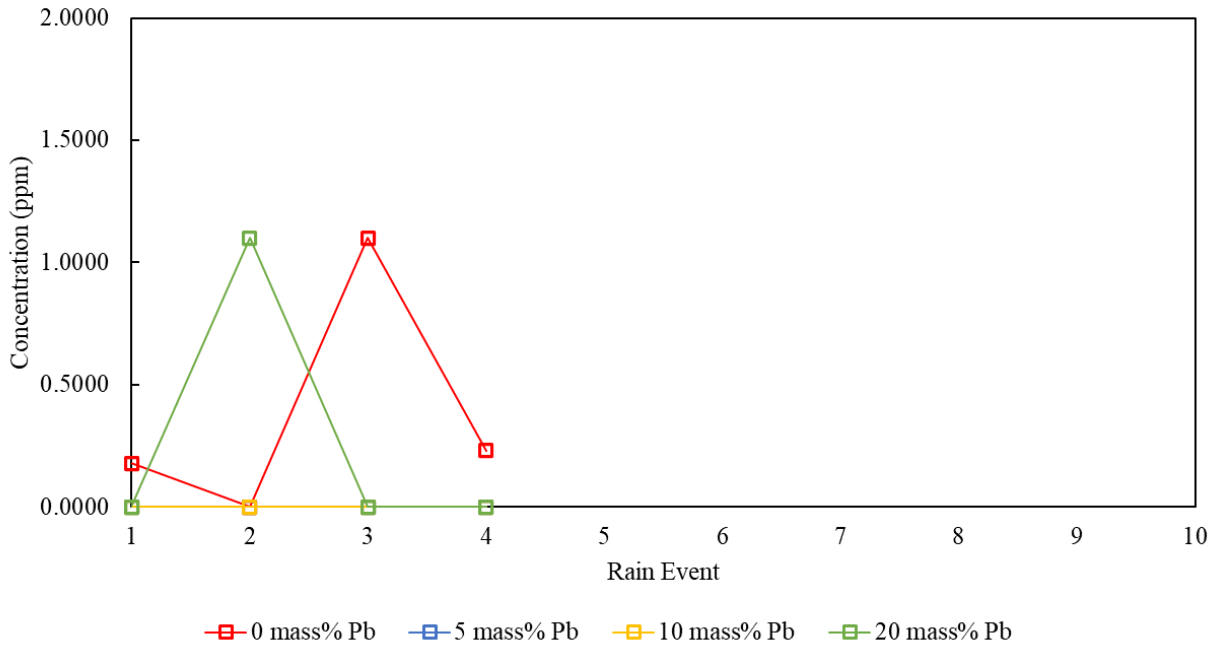



Figure 3.9.18 Leaching behavior of Pb from SHXP material as a function of various calcite concentrations

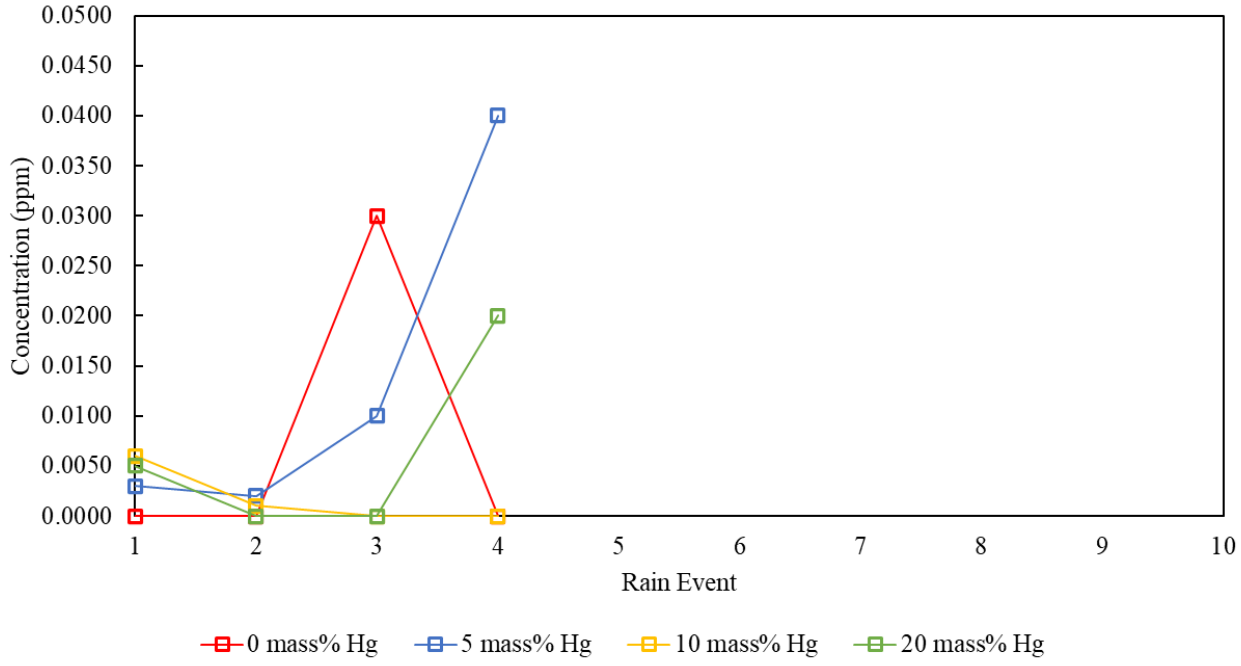



Figure 3.9.19 Leaching behavior of Hg from SHXP material as a function of various calcite concentrations

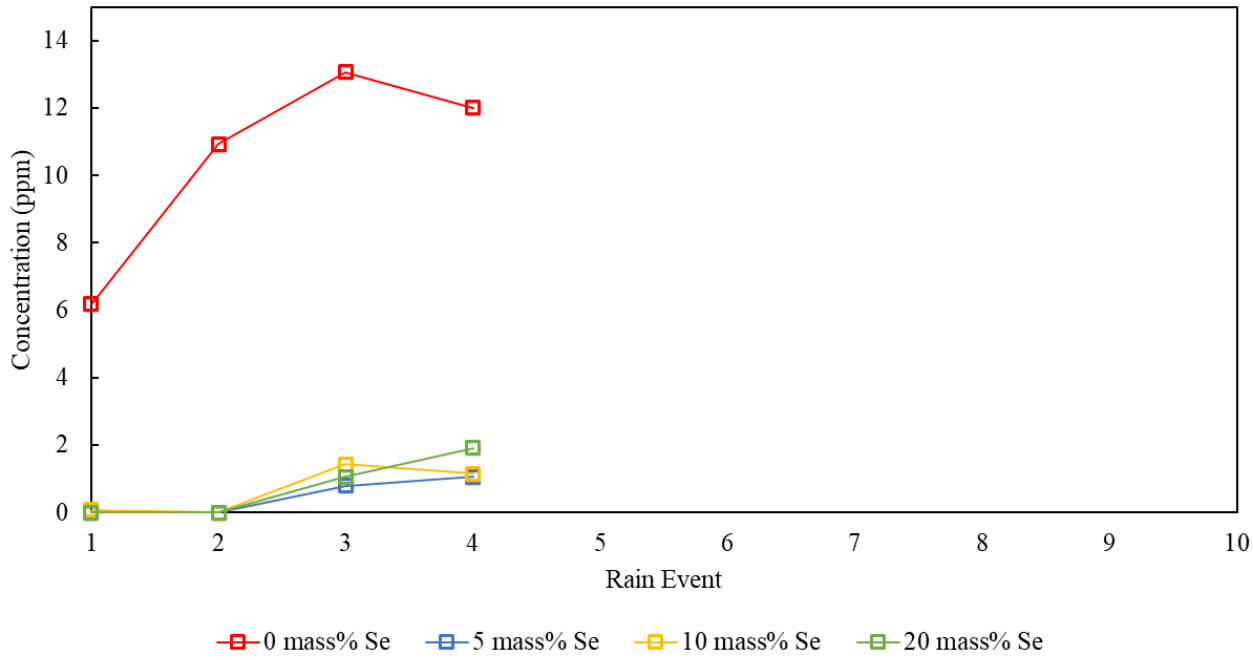



Figure 3.9.20 Leaching behavior of Se from SHXP material as a function of various calcite concentrations

SHXP's leaching does not appear to be completely correlated with calcite mass%. While the control test does have significantly higher concentrations of arsenic and selenium than calcite blends, the trend is not the same with lead and mercury. The 1.1 ppm reading for the 20 mass% may be an error from the ICP-OES, if so it would be in line with the arsenic and selenium observations, as every other calcite blend test reads lead concentrations below detectable limits. Mercury concentrations are the outlier in this test, as the calcite blends all observed higher concentrations than the control test initially. The control test only registered a detectable concentration in the third rain simulation, whereas the 5 mass% had detectable concentrations in every event and have been rising since the second.

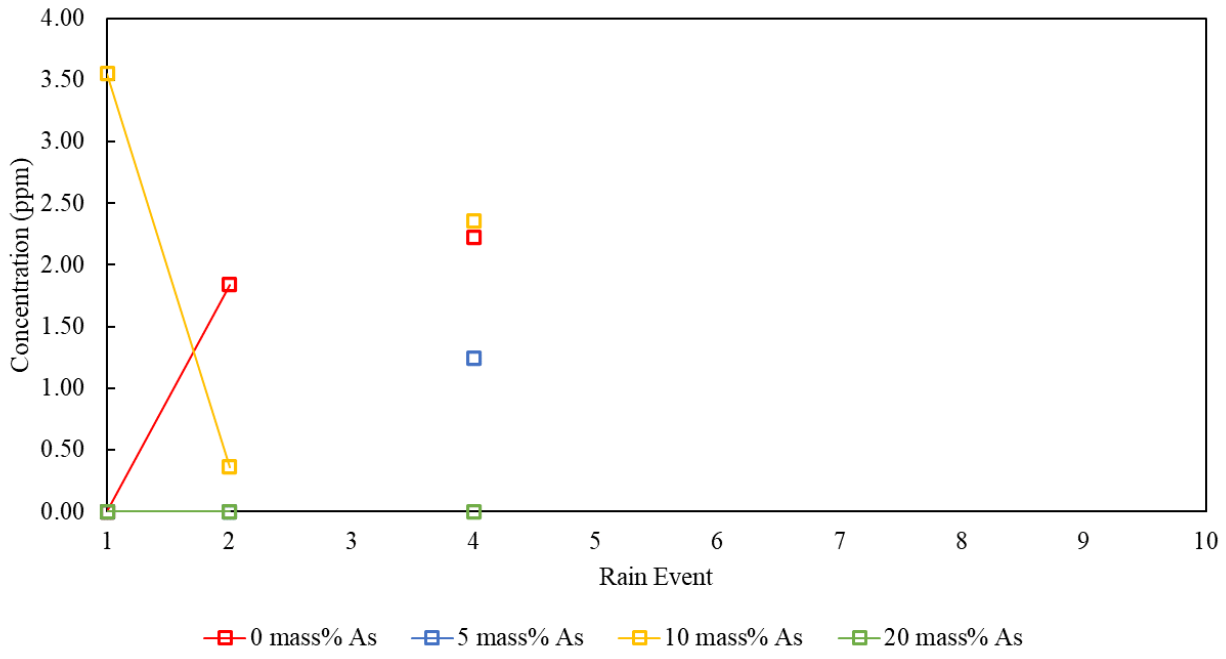



Figure 3.9.21 Leaching behavior of As from untreated WK-13 material as a function of various calcite concentrations

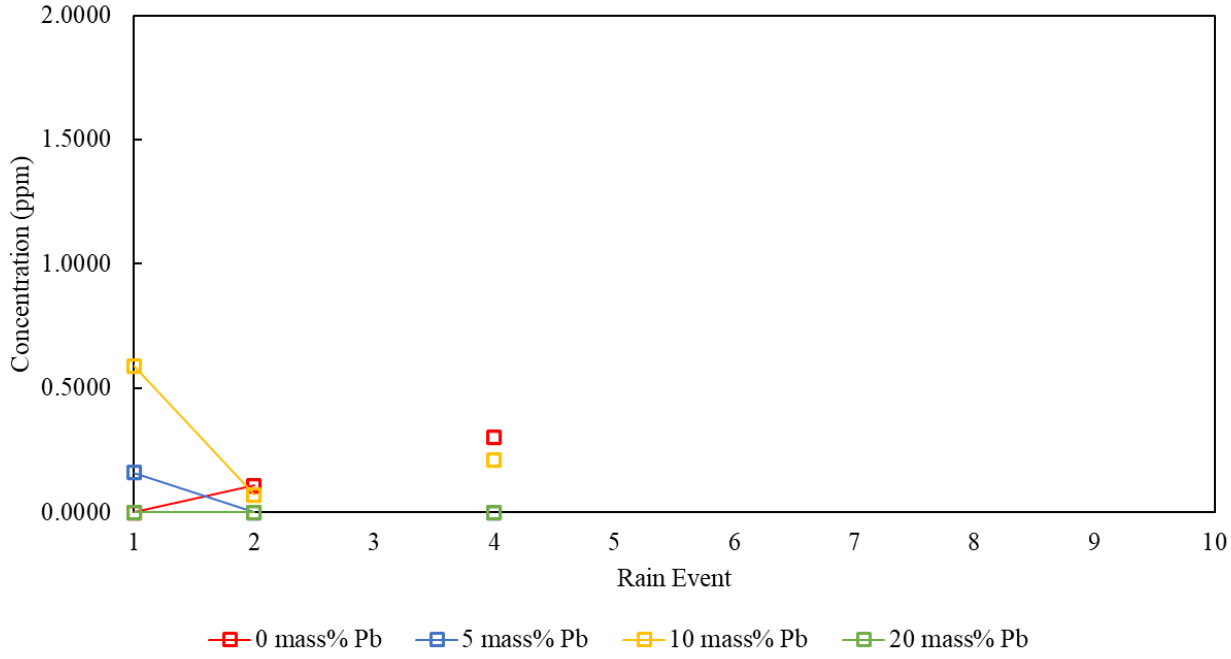



Figure 3.9.22 Leaching behavior of Pb from untreated WK-13 material as a function of various calcite concentrations

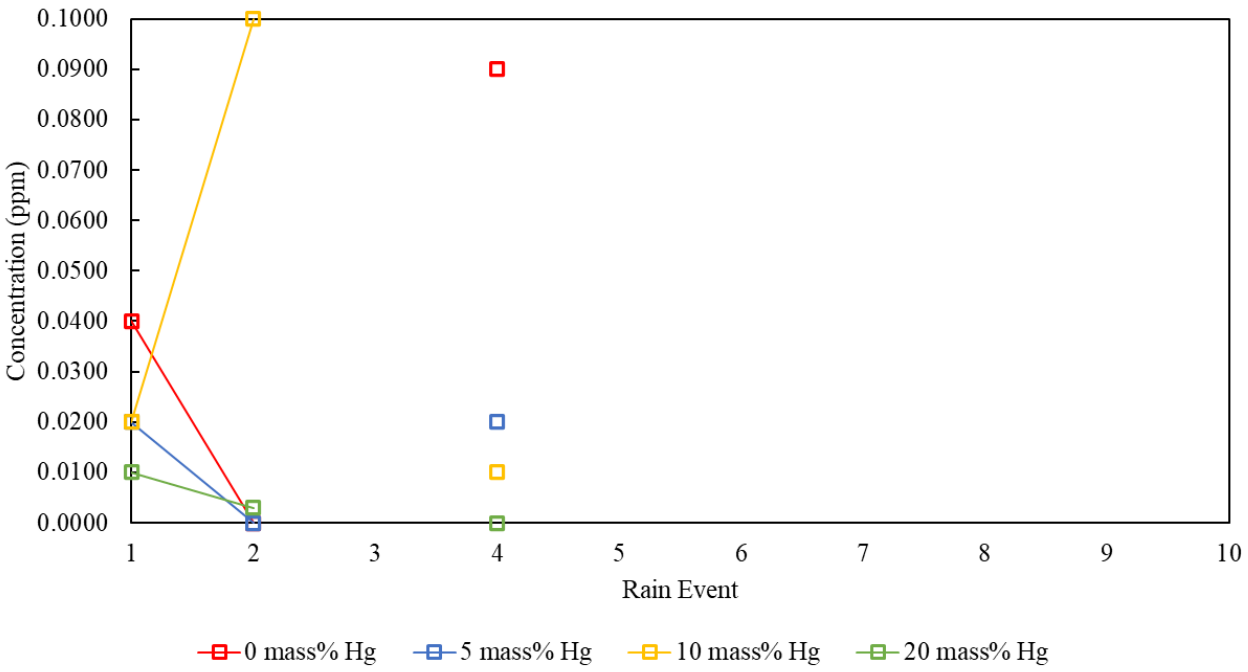



Figure 3.9.23 Leaching behavior of Hg from untreated WK-13 material as a function of various calcite concentrations

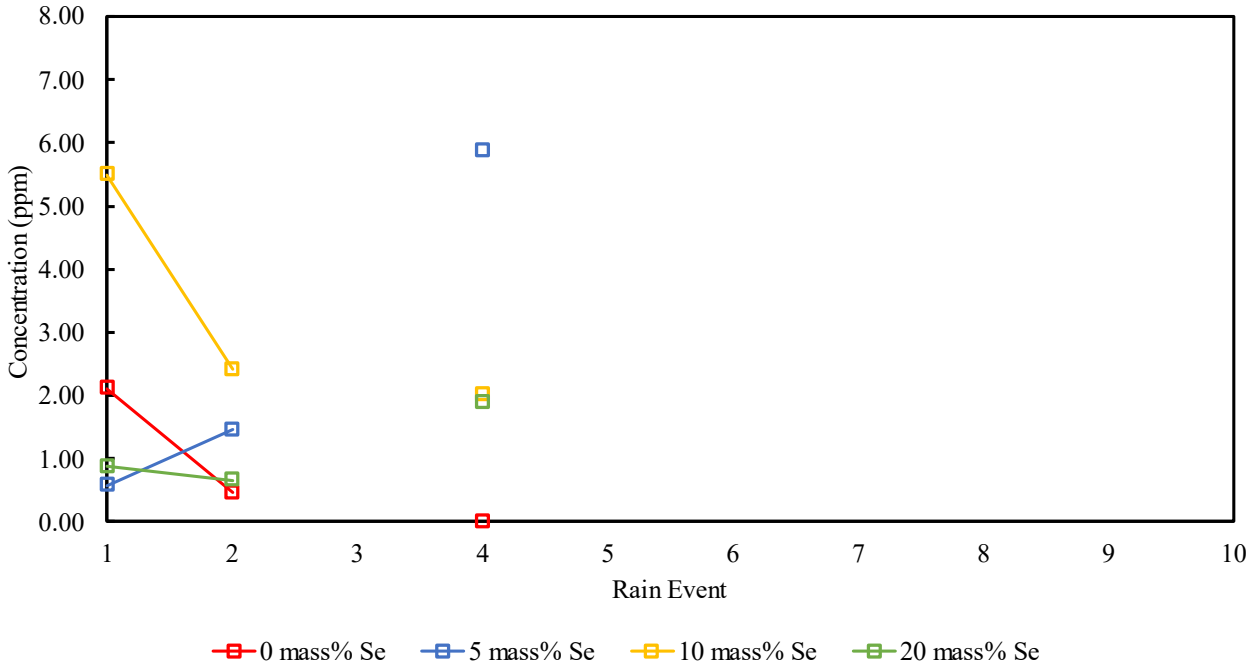



Figure 3.9.24 Leaching behavior of Se from untreated WK-13 material as a function of various calcite concentrations

As the only un-leached material used in this experiment, WKY13 concentrations are much more erratic and currently impossible to be explained without more data. Of note is the 10 mass% having the highest initial concentrations in all surveyed elements but mercury, and the seemingly random distribution of concentrations in the fourth rain event.

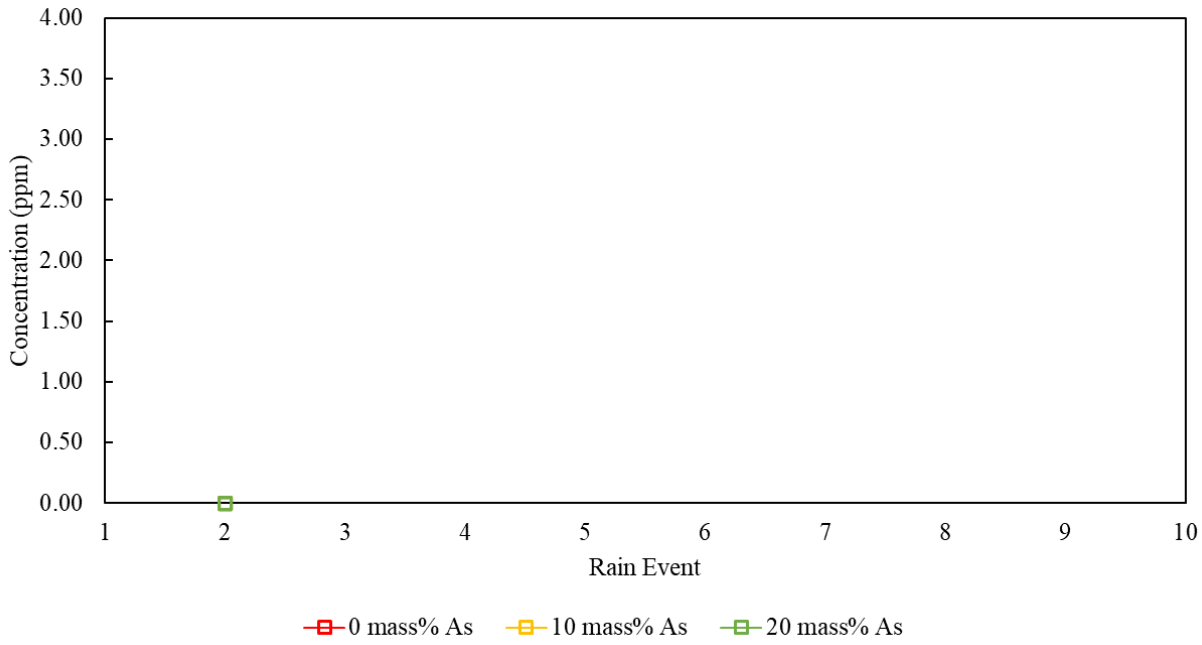



Figure 3.9.25 Leaching behavior of As from untreated WK-13 LSR material as a function of various calcite concentrations.

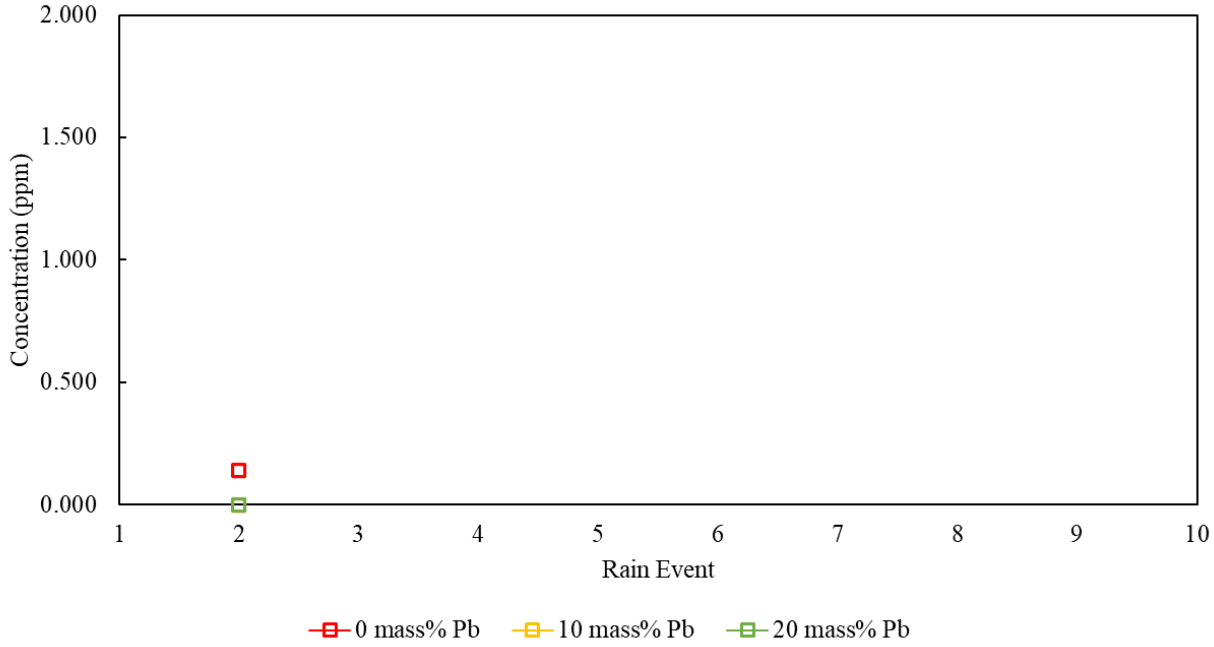



Figure 3.9.26 Leaching behavior of Pb from untreated WK-13 LSR material as a function of various calcite concentrations.

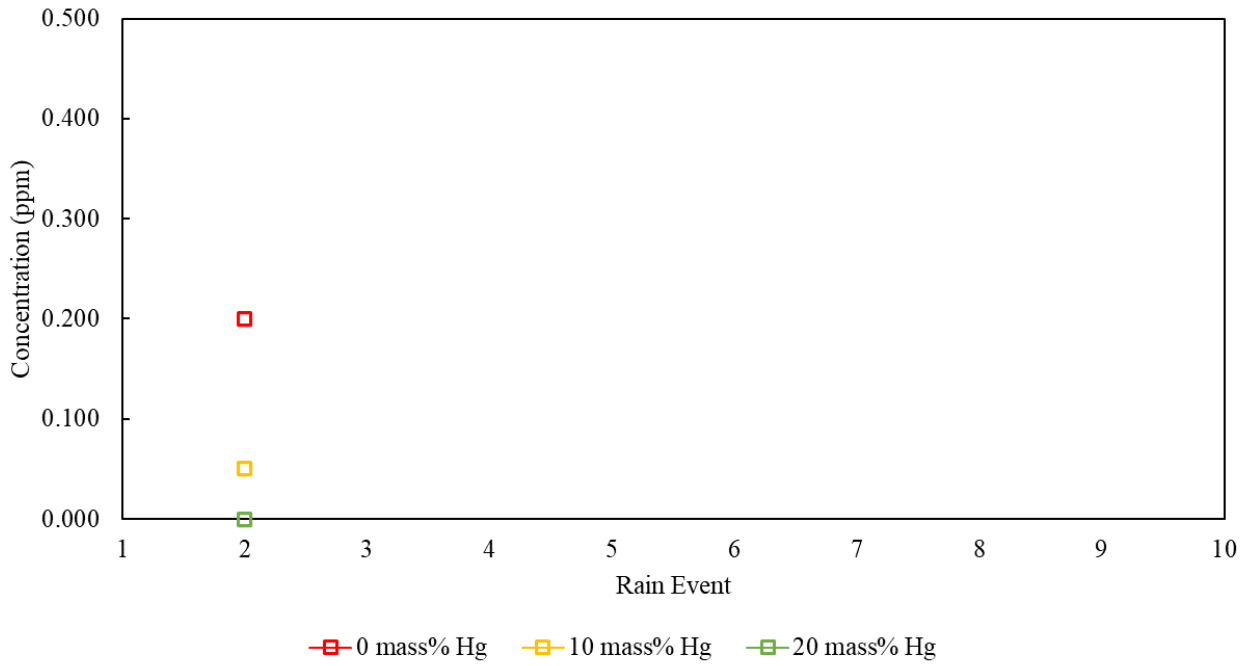



Figure 3.9.27 Leaching behavior of Hg from untreated WK-13 LSR material as a function of various calcite concentrations.

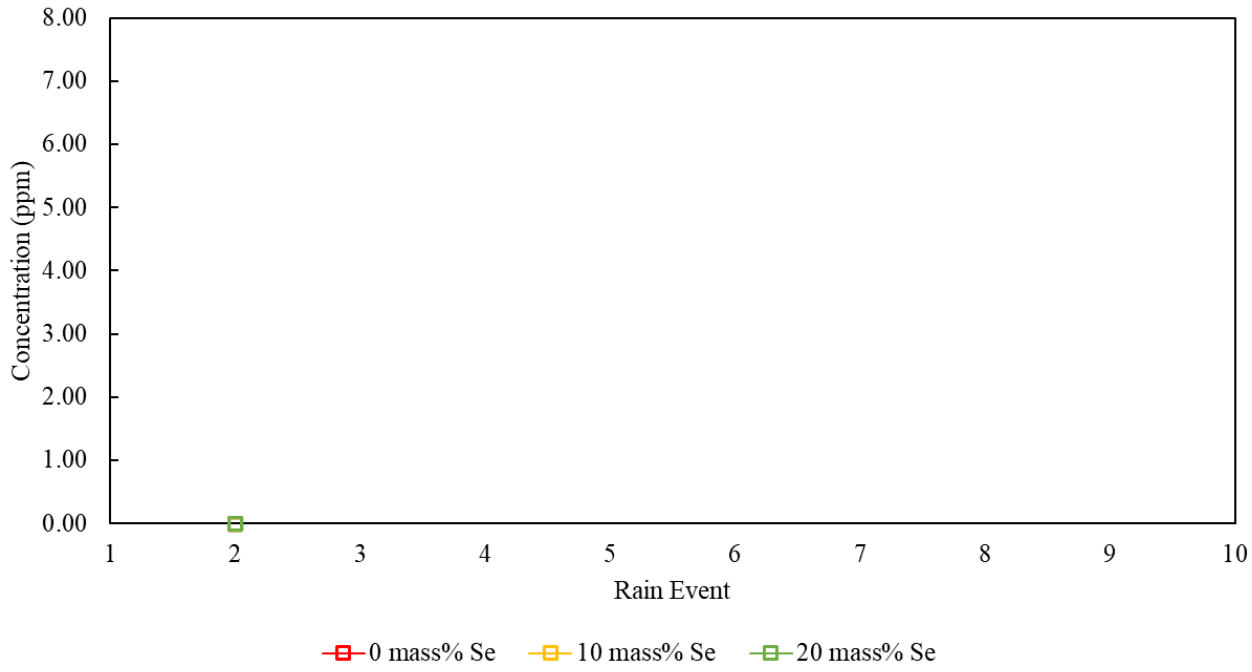



Figure 3.9.28 Leaching behavior of Se from untreated WK-13 LSR material as a function of various calcite concentrations.

WKLSR material has so far been observed to have much lower concentrations of the surveyed elements than the rest of the materials. Arsenic and selenium are undetectable for all WKLSR tests thus far. Only the control test has a detectable amount of lead, and the 10 mass% only appears.

**Conclusions:** The presence of calcite provided a significant effect on raising the pH in all of the waste residues studied. However, the effect is diminished over time as calcite is dissolved

and eliminated from the material. Calcite significantly retards leaching of rare earth elements from all the tested materials. Its effect on pollutants are less clear. In the heap leach material, calcite does appear to reduce the leaching of the selected contaminants over the first few rain simulations. SHXP sees calcite suppress leaching of arsenic, possibly lead, and selenium. WKY13, as the material that was not put through a leaching process before being used as a test material, has more inconclusive results in regard to elemental leaching. TREE data showed that calcite does inhibit the leaching of rare earth elements, but pollutants are not nearly as affected by the presence of calcite with the 10 mass% test achieving higher pollutant concentrations than both the control test and the 20 mass% tests in some rain events. In the concentrations that were detectable, WKLSR also showed a reduction in leaching due to the presence of calcite. More testing is needed on this material before final conclusions can be drawn. TREEs appeared to be the elements most significantly affected by the presence of calcite in all materials. Leached mercury did not appear to be as significantly affected by calcite presence in a material. Overall, calcite does appear to retard acidity and leaching for a limited amount of rainfall overtime but the effective is eventually negated by the dissolution of the added calcite.

### *1.1.15 Radioactive material monitoring and management*

Over the last 3 years, the University of Kentucky pilot plant has thoroughly investigated and developed a separation technique to remove radioactive and hazardous elements such as thorium (Th) and uranium (U) in any possible stream of the pilot scale process. In addition, a Geiger counter was installed in the laboratory area for radioactivity monitoring. There was no sign of accumulated radioactive material existing in any of the processing stream. Continuous effort was taken throughout the project to monitor the radionuclides in the waste disposal streams. Details about where these elements report in the pilot plant are provided in later sections of this report.

## 3.10 PILOT PLANT UPGRADES AND MODIFICATION (TASK 12)

The existing plant circuitry was modified and upgraded to include the optimal unit operations and circuit configurations identified in the initial laboratory testing programs described in Tasks 6 – 10. To the largest extent possible, existing equipment was re-purposed and reused; however, significant modifications was required to properly integrate the roasting, bio-oxidation, and critical material recovery components. Detailed engineering drawings were prepared that specified the size, capacity, layout, and materials of construction for new process equipment. From these specifications, competitive bid packages for major purchases of equipment, materials, fabricated components, and services necessary to complete the installation of the proposed circuitry were prepared and solicited. The bid packages were reviewed, and appropriate vendors were selected on the basis of cost, availability and suitability.

Fabrication and construction activities were performed as major equipment was delivered to the pilot plant facility. These activities included both internal fabrication work conducted within the shops/facilities operated by the project partners as well as off-site fabrication work carried out by equipment vendors and contract service providers. Lastly, all units were assembled and installed into the existing pilot plant. In addition to the actual process equipment, the work included installation of piping, electrical wiring, and control systems needed for plant operations.

### *3.10.1 Plant Upgrade Management*

To facilitate timely completion of the pilot plant upgrades a Gantt chart was developed as shown in Figure 3.10.1, which provides the general goals and expectations concerning the timeline of events required for complete set up of the Rare Earth Elements Recovery Plant. This was designed in such a way that a setback or delay in one aspect of the plan does not negatively impact the other sectors, and could, perhaps, even lead to completion of other tasks ahead of schedule. The project schedule is outlined in the categories listed in Table below.

Table 3.10.1. Project area.

| Area |
|---|
| Design Documents Finalization |
| Procurement |
| Belt Filter |
| Bio- Reactors |
| Mn Process |
| Co Process |
| Roaster |
| Conveyors |
| Pumps |
| Ventilation |
| Controls |
| Air |
| Power |
| Layout and Construction |
| Final Construction |
| Pilot Plant Water Test Shakedown |
| Pilot Plant Inspection |

Broadly speaking, the detailed engineering and procurement took longer than allotted. A belt filter was sourced and purchased in the first quarter of 2021. Similarly, the Bio-reactor processes were designed, and purchased in the first quarter of 2021. The Mn and Co processes were addressed by the team with two potential options detailed in following sections of this document. Solvent extraction process equipment was modified as necessary to support flowsheet development. The largest two single risks associated with the project were the delivery of the belt filter and finalizing the roaster. For the belt filter, the time taken to procure it and then have FLSmidth perform the review of the belt filter took longer than anticipated. The roaster was also in an interesting spot. In the first quarter of 2021, the small-scale roaster was providing “build ahead” capacity to provide a stockpile of material available upon process start up. The reason for this approach was that the TGA experienced significant operation issues due to an equipment failure that delayed the project by about two months. The primary decision points remained between low and high temperature roasting, and the use of additives, which fundamentally changed the design nature of the required roaster. At this point, a buffer stock was built to allow this decision point to be further analyzed.

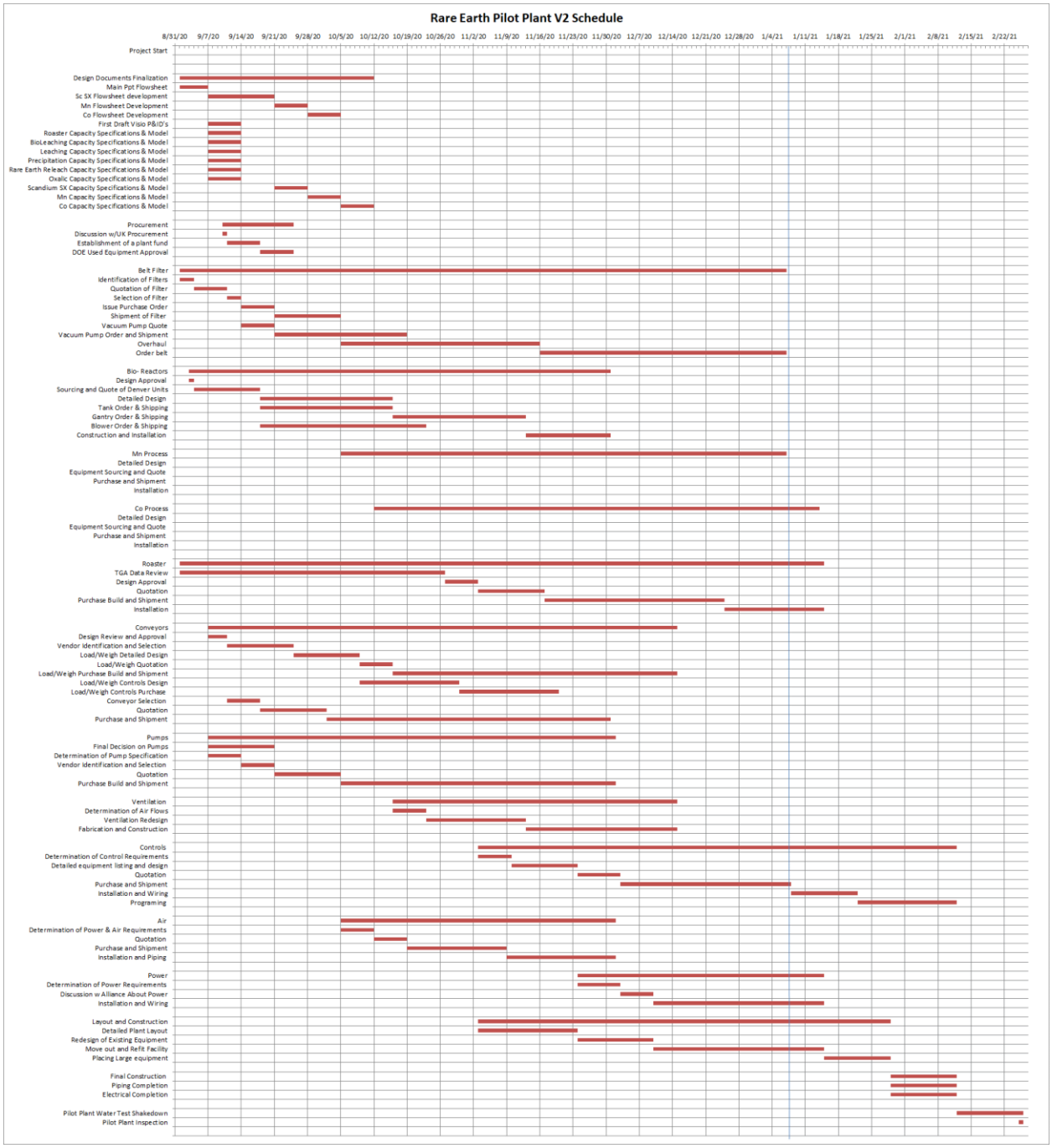



Figure 3.10.1.Gantt chart to track progress.

### *3.10.2 Flowsheet Development*

To achieve the desired outcomes of the project, the following flowsheet was conceptualized and revised as experimental data became available, which materially impacted the functionality of the process. The flowsheet is shown in Figure 3.10.2 and Figure 3.10.3. The general principle of operation was as follows. Feed material was preliminarily processed through an x-ray sorter to upgrade the material for processing in the plant. Further size reduction occurred via jaw and hammer mills to achieve a top-size of approximately 2mm. This material was then fed to a roaster to further liberate the material by combusting the available carbon and promoting the recovery of REEs. To produce acid for the circuit, two bio-oxidation reactors were utilized each containing a Denver flotation mechanism to provide simultaneous aeration and agitation.

Recirculation pumps and cone bottoms were also utilized to prevent sanding in the tank bottoms. The bio-oxidation tanks were fed a mixture of pyrite, water and small amounts of nutrients to promote bacteria growth. The outflow from the bio-oxidation tanks was combined with the feed from the roaster in the leaching stage.

In leaching, two possible scenarios existed. The first being termed “long leaching” occurred over a period of several hours, where material was leached and the contained calcite began to provide a measure of self-neutralization, which promotes iron precipitation. This is desirable from a processing standpoint, however REEs were lost as part of the precipitation, causing a significant tradeoff. To counter this tradeoff, a “short leaching” solution was proposed where the lixiviant was contacted with the material for a period of approximately 50 minutes and separated. This prevented the co-precipitation of REEs as influenced by the iron/calcite system which occurred in the longer leaching scenario. As shown in Figure 3.10.2, additional neutralization occurred with the addition of calcite. To reduce the filtration burden on the belt filter, a thickener was utilized. The underflow of the thickener was dewatered via the belt filter where the solids were discharged to tails. The filtrate and thickener overflow were transferred to a two-stage solvent extraction loading phase where scandium was selectively loaded. Scandium stripping from the organic phase was achieved through two stages. Scandium was recovered from the strip solution by raising the pH until a scandium precipitation occurred.

REE recovery was achieved by taking the scandium barren solution and raising the pH until REE precipitation occurred. The REE cake was then recovered via filter press and washed with a strong NaOH solution to remove excess aluminum. The purpose of the aluminum removal was to promote significant oxalic acid cost savings later in the circuit. The output of the alkaline aluminum rinse could be used as a reagent in prior neutralization stages. The REE cake was recovered via filter press for further processing. The filtrate from the REE precipitation stage was further neutralized to produce a cobalt and manganese precipitate in addition to a multiplicity of contaminants. The recovery of cobalt and manganese is discussed further in Figure 3.10.3. To complete the recovery REEs in the process the REE cake was mixed in a high shear mixing tank and transferred to a re-leach where the REEs were resolubilized with HCl, rejecting many contaminants remaining in the insoluble fraction which were recovered via filter press and discarded to tails. The final recovery of REEs occurred with oxalic acid precipitation in a series of three small precipitation vessels. The REE oxalate could subsequently be roasted to generate high quality REOs.

To recover cobalt and manganese, two processes were proposed. A complicated and simplified version. The difference being the complicated process and simplified version was experimentally evaluated. For the complicated process, the initial precipitate was re-slurried and mixed with an oxidative solution of $Na_2S_2O_8$ (sodium persulfate) to increase the oxidation state of cobalt and manganese, which have multiple oxidation states as opposed to the contaminants. The valance change was utilized in the following stage where contaminates were resolubilized selectively, with the cobalt and manganese remaining in the solid phase. Following oxidation, utilizing an HCL lixiviant, the contaminants were resolubilized with the cobalt and manganese being recovered via filter press. Following the selective leaching, the remaining cake is mixed with a reducing leaching solution (Hydroxylamine hydrochloride) so that the cobalt may be selectively precipitated as a sulfide by the addition of sodium sulfide. The manganese is then recovered by pH manipulation to create a manganese rich precipitate. The second, simplified version, of this process omitted the oxidation and reduction stages of the more complicated process. In this embodiment, only releach and precipitation steps remained. The described flowsheets provided in Figure 3.10.2 and Figure 3.10.3 present what we believe to be the most likely and cost-effective flowsheet based on current experimental data.

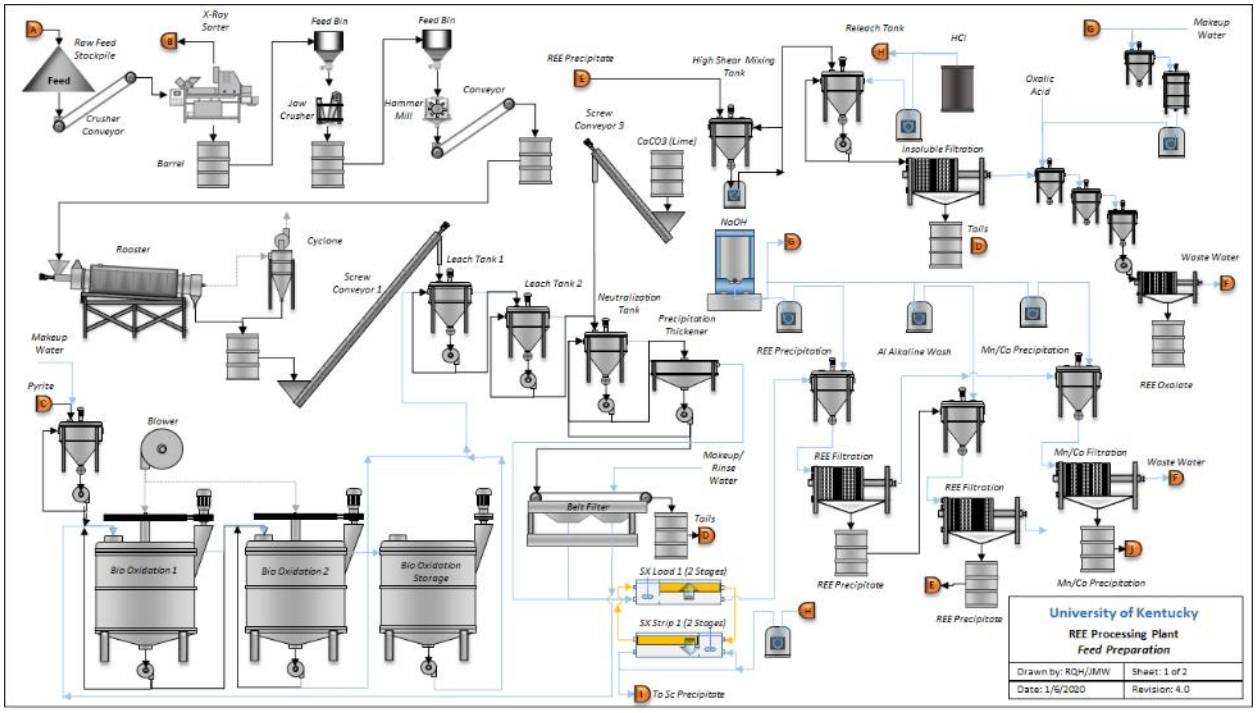



Figure 3.10.2. Pilot process flowsheet showing material processing, roasting, acid generation via bio-oxidation, leaching, various precipitation and REE recovery via oxalic acid precipitation.

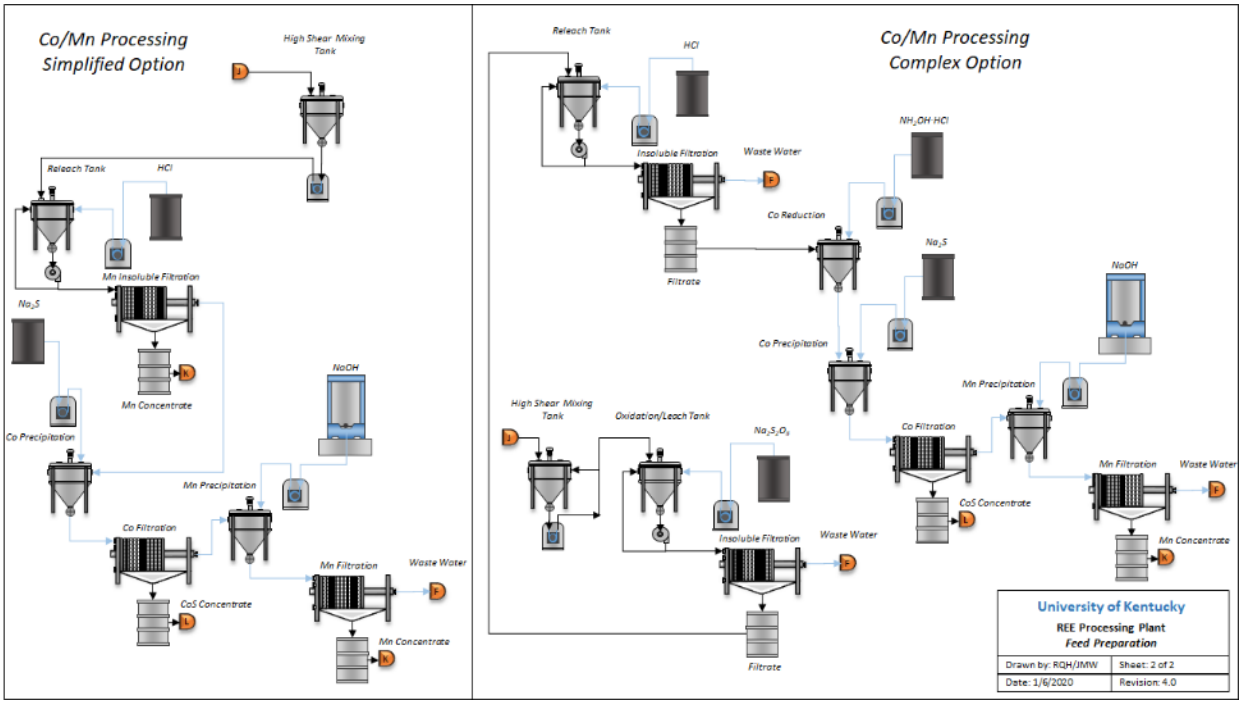



Figure 3.10.3. Pilot process showing the cobalt and manganese portions showing two processing options.

### *3.10.3 Plant Design*

A primary focus for the pilot plant upgrade was the modification of existing equipment and the development of the floorplan. Before introducing this addition, the constraints for the arrangement should be noted. First, as far as practical, the vessels to be used for leaching needed to remain in their current locations. Movements of tanks was expensive and time consuming due to the wiring and plumbing associated with each tank. Due to the high feed rates in the pilot plant from the initial version, fork truck access with a barrel mover would have been required for the belt filter spent effluent, screw feeder hopper, calcite slurry makeup tank and pyrite slurry makeup tank. Because of this feed rate, reducing barrel movement was significant. A system involving rolling stands for the barrels was considered for a point of time. Eventually it was determined that this would prevent the ability to run lines to across the floor (as is necessary for heavy slurries) and would inhibit flexibility of the system, so this idea was discarded. With fork truck access, it was still possible to run lines across the floor with the speed bumps. The bioreactor tanks would also need to be able to access the current ventilation system. This was possible at a distance but would be made easier if the bioreactor was near the ventilation header. It would also be ideal if there was access to the two drains in case of spills.

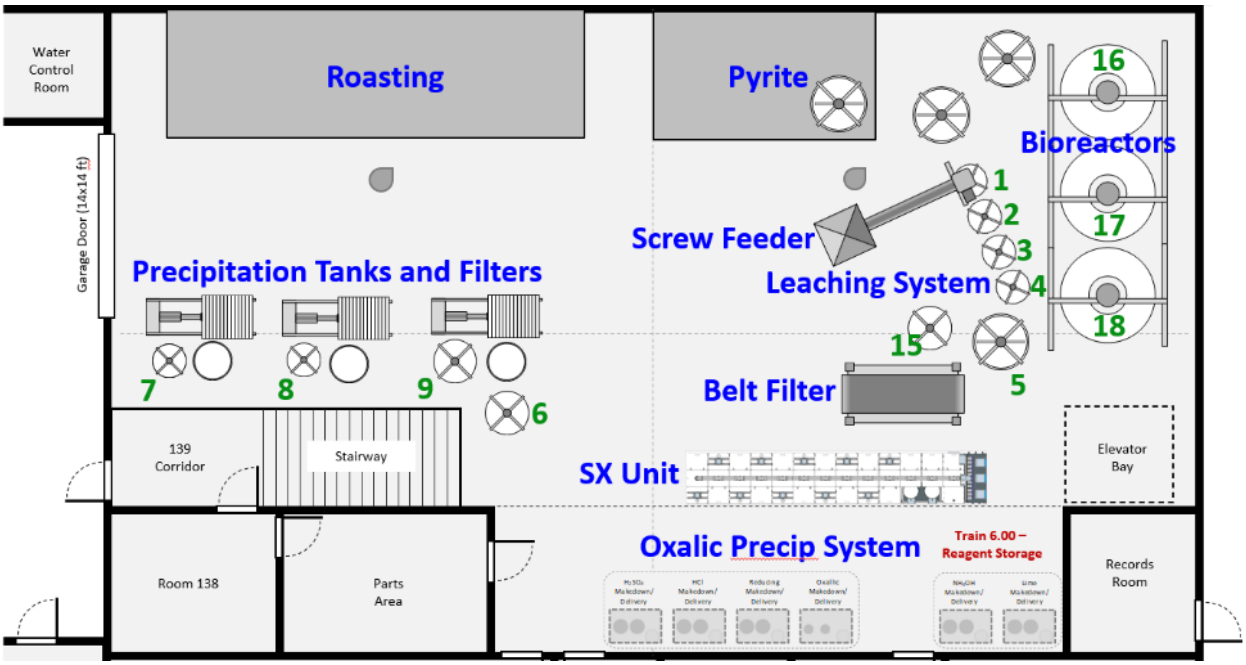



Figure 3.10.4. Project Floorplan with areas in Blue and Tank numbers in Green

The design shown above in Figure 3.10.4 includes all these requirements. The text in blue indicates the section of the process each area is in. The text in green shows the tank numbers which correspond to the tank numbers in the piping and instrumentation diagrams which are discussed in detail further below. The bioreactor tanks (16,17 and 18) are shown on the far-right wall and are underneath the ventilation header. To the left of the bioreactor tanks are the leaching vessels (1-4) and the screw conveyor. These are in the same general position that they are located currently, though they have been moved leftward. The leaching effluent goes into a thickener (5), which is shown directly below the leaching vessels. To the left of the leaching vessels is the pyrite slurry makeup tank (15), which is the feed to the bioreactors. Below is the belt filter, which will be used to remove the solids from the thickener underflow. Above the

leaching vessels is a hot acid makeup tank to feed leaching. The roasting system and pyrite system are not going to be altered, and as such, are not shown in much detail for floorplan changes. The three plate-and-frame filters which will run continuously are set up linearly as shown below the roasting section. For each filter press, there is a precipitation tank (7,8,9) to control residence time as well as an overflow tank which will be continuously pumped out of dry. The overflow tanks can also be allowed to fill up and be filtered all at once. Below the rightmost filter unit is the calcite slurry makeup tank (6), which will be used to neutralize the acid to initiate precipitation. The location and orientation of the Oxalic precipitation circuit will remain the same. As it turns out, the flowrate used to run the previous project batch-wise is very similar to the flowrate to be used in the new continuous process.

**Piping and Instrumentation Diagrams:** The following section will discuss the Piping and Instrumentation Diagrams with updates on the status of the required equipment. A legend for the system is to the right in Figure 3.10.5. This shows the different types of pumps, filters, lines, and other equipment as well as the corresponding items on the P&IDs.

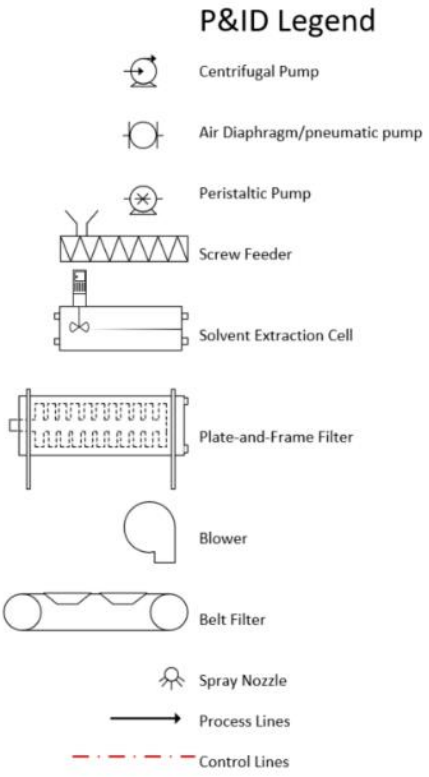



Figure 3.10.5. P&ID legend

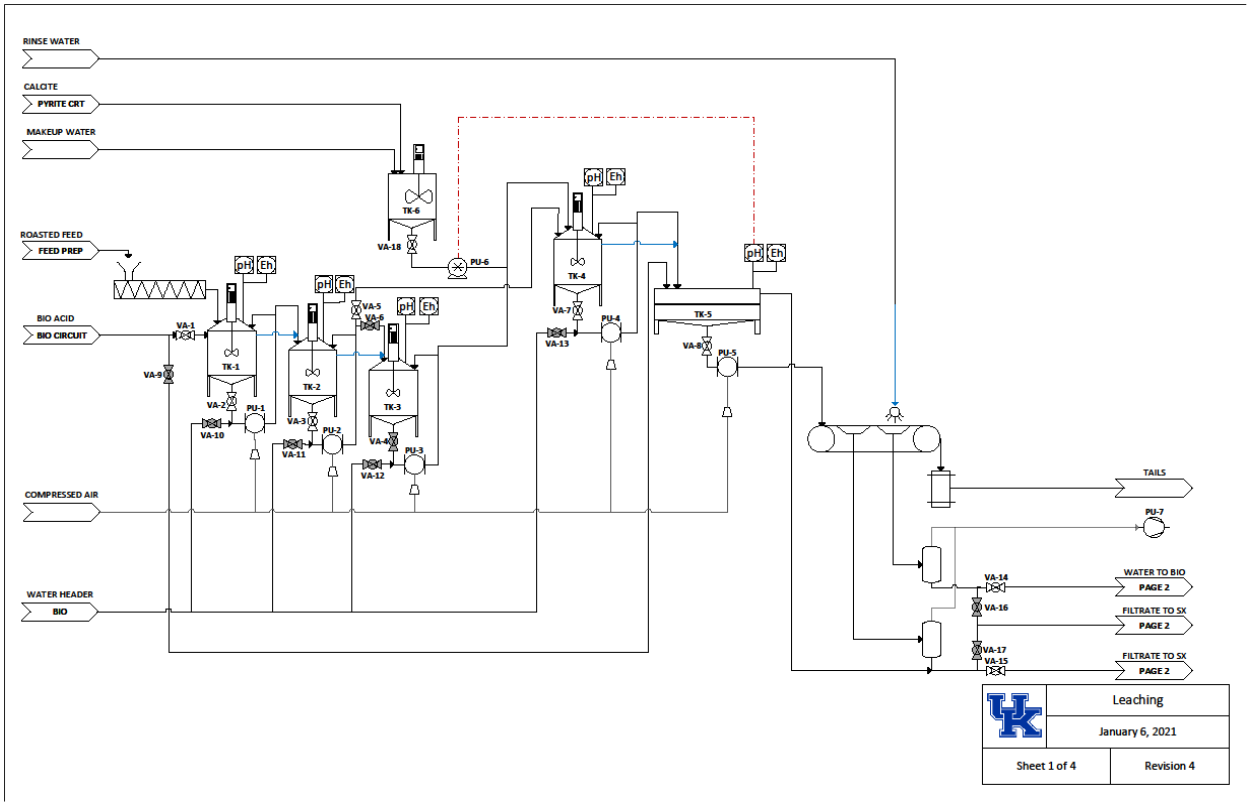



Figure 3.10.6. Leaching Piping and Instrumentation Diagram

The first P&ID, shown above in Figure 3.10.6, shows in detail the system of leaching and removing the solids and iron precipitation that has occurred. The roasted feed material was fed into the first leaching vessel at 250 lbs/hr. This screw feeder was already on site and was suitable for this feed rate. The bio acid was also fed into the first leaching tank at around 2 gallons per minute with a peristaltic pump (currently available). These flowrates equated to around 20% solid content in the leaching system. The leachate and solid was pumped with air diaphragm pumps through a series (probably 2 but no more than 4) of 100-gallon leaching tanks to increase the residence time for leaching. Due to the high solid content in this system, there was some concern for plugging lines. To prevent this, lines were sized properly to ensure sufficiently high line velocity to prevent settling. To correct plugging if it did occur, water flush lines were available connecting to the bottoms of the tanks. They were able to flush out both the bottoms of the tanks as well as the pumps and subsequent lines. The diaphragm pumps had variable air valves to control the pressure of air sent to the pumps. pH and ORP probes were in each of the leaching tanks to measure these variables. These were necessary mainly as a safeguard, as our iron precipitation curve depends on pH. The last leaching vessel effluent was sent to a 400-gallon thickener settling tank. This thickener also had a pH and ORP sensor. The overflow of this tank was pure leachate without any solids and was ready to be sent to the Solvent Extraction and precipitation steps. The underflow of this tank was sent to a small belt filter, at which the leachate was filtered off, the solids were rinsed, and the rinse water was used to feed the bioreactor. The fork truck with barrel mover attachment was used to transport the barrel of spent solids to waste.

As far as equipment was concerned for this section, the leaching vessels, thickener, and screw feeder are all pieces of equipment from the previous project that suited the needs. The pumps, pH and ORP sensors and all of the PLC system components were ordered through online vendors. The compressed air system was upgraded to include capacity for all new diaphragm pumps and filter aeration.

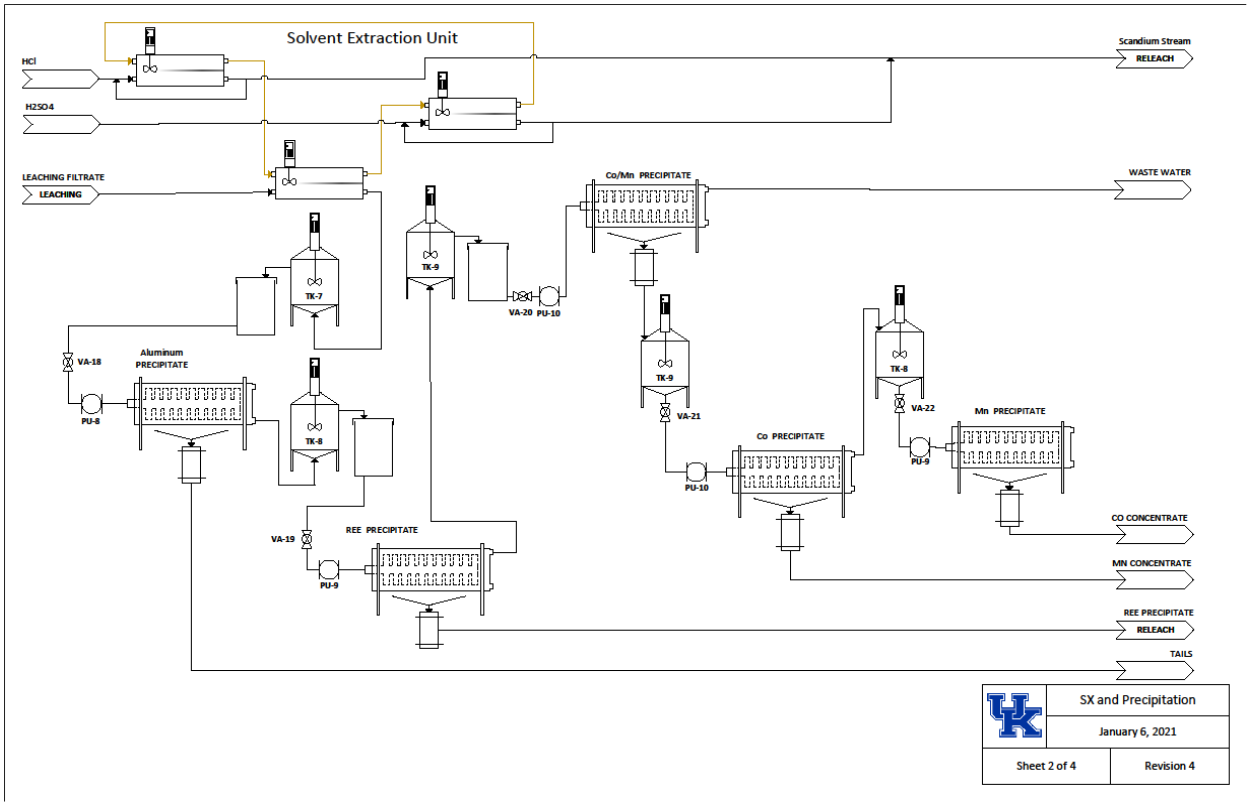



Figure 3.10.7. Solvent Extraction and Precipitation Piping and Instrumentation Diagram

For the Solvent Extraction and Precipitation Piping and Instrumentation diagram, shown above in Figure 3.10.7, the following system was used. The leachate was transferred directly into solvent extraction, where there were loading, scrubbing, and stripping phases. The specific number of stages in each phase were selected based on the bench-scale operations. All solvent extraction equipment was already on site and provided flexibility to add and remove stages. The primary purpose of the solvent extraction unit was to remove scandium from the leachate and produce a scandium-rich acid stream to be precipitated by pH manipulation. The extractant used was recirculated throughout the system and was not replaced or replenished which prevented the REE loss. The leachate was then moved into aluminum precipitation, where the pH was raised through the addition of calcite or 4M NaOH. The tank overflowed into a holding tank and was subsequently filtered. The aluminum cake was cleaned and sent to waste. The filtrate was then sent to a tank to start REE precipitation. Again, the pH of the incoming liquid was raised to match that of an ideal REE precipitation curve. After this step, the Cobalt/Manganese step proceeded in the same manner as the previous two precipitation steps. The filtrate from Co/Mn precipitation was sent to wastewater. The Co/Mn cake was redissolved using an acid solution, and then separated using the same precipitation process outlines in the steps above.

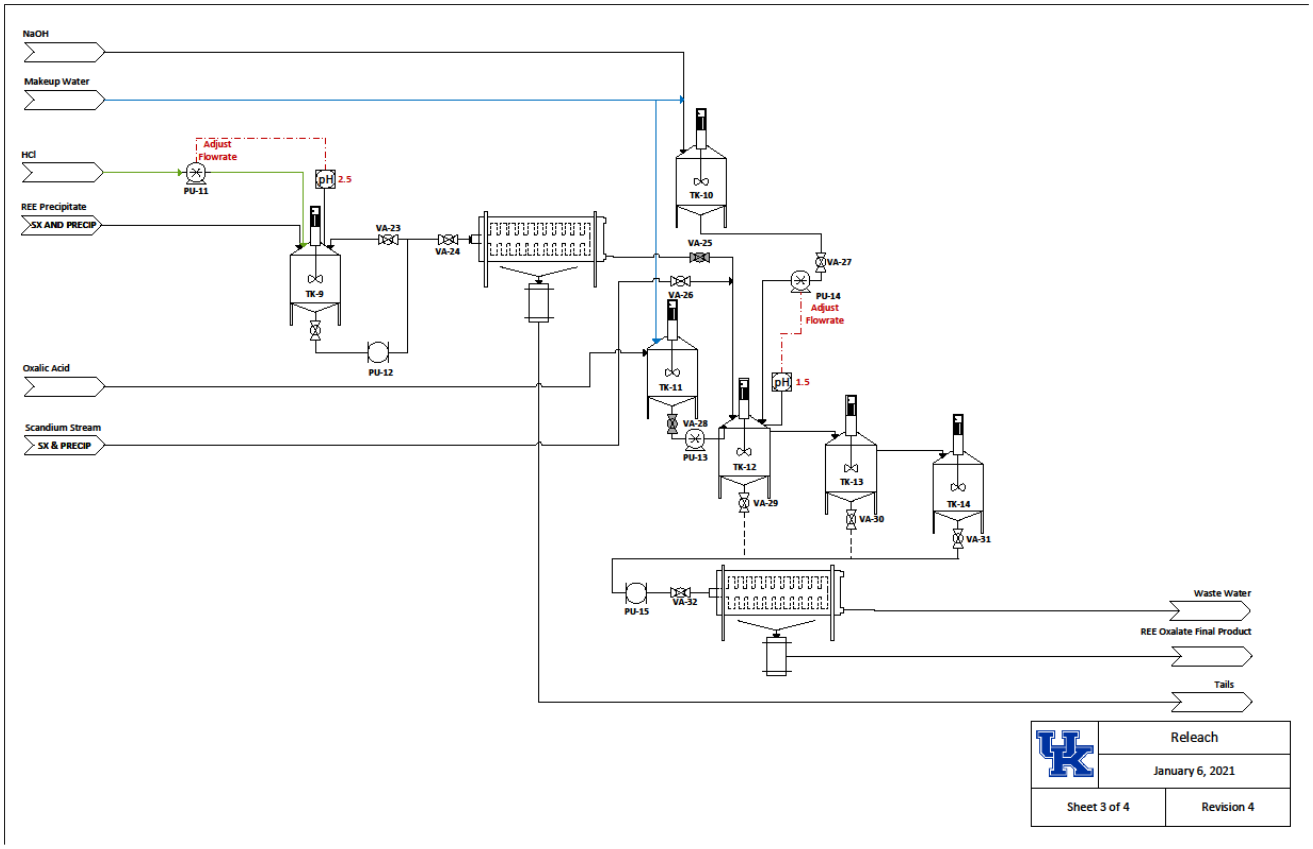



Figure 3.10.8. Releach and Oxalic Precipitation Piping and Instrumentation Diagram

The system regarding re-leach and oxalic precipitation is shown in Figure 3.10.8. For this project it was unchanged and did not require any purchases or modifications. This was used continuously for scandium precipitation as well as precipitation for REEs on a batch scale. Oxalic acid must be added to the solution to create RE Oxalates, which precipitated very predictably. Given these facts, it was not prudent to extensively modify this system.

### *3.10.4 Bioreactor Design*

Efforts in detailed flowsheet engineering and design continued throughout the period. One major design consideration assessed during the prior quarter was the reactor selection for the bio-oxidation units to be used in the acid generation circuit. In addition to the long retention times needed for acid generation, these reactors have other unique and competing constraints that preclude the use of standard agitators and tank configurations. For example, bacteria benefit will benefit from uniform air distribution throughout the tank as well as a low mixer tip speed. Alternatively, the bank must also be able to adequately suspend the coarse pyrite particles to ensure high reactivity of the bacteria with the particulate phase.

Given these challenges, the engineering team opted to use froth flotation cells as the agitating mechanism in the bio-oxidation reactors. An example of a flotation cell mechanism is shown in Figure 3.10.9. Left: Example of force air flotation mechanism. Right: Schematic of flotation mechanisms in a typical flotation tank. By design, flotation cells are employed to meet the simultaneous objectives of air dispersion and solid particle dispersion. A forced air flotation cell employs an external blower to disperse air through a bubble generation mechanism. The combined action of the rotor and the bubble generation ensure superior air dispersion throughout the cell volume. Flotation cells can also be equipped with independent control of rotor tip speed and air flow rate to optimize the hydrodynamic conditions for the intended duty.

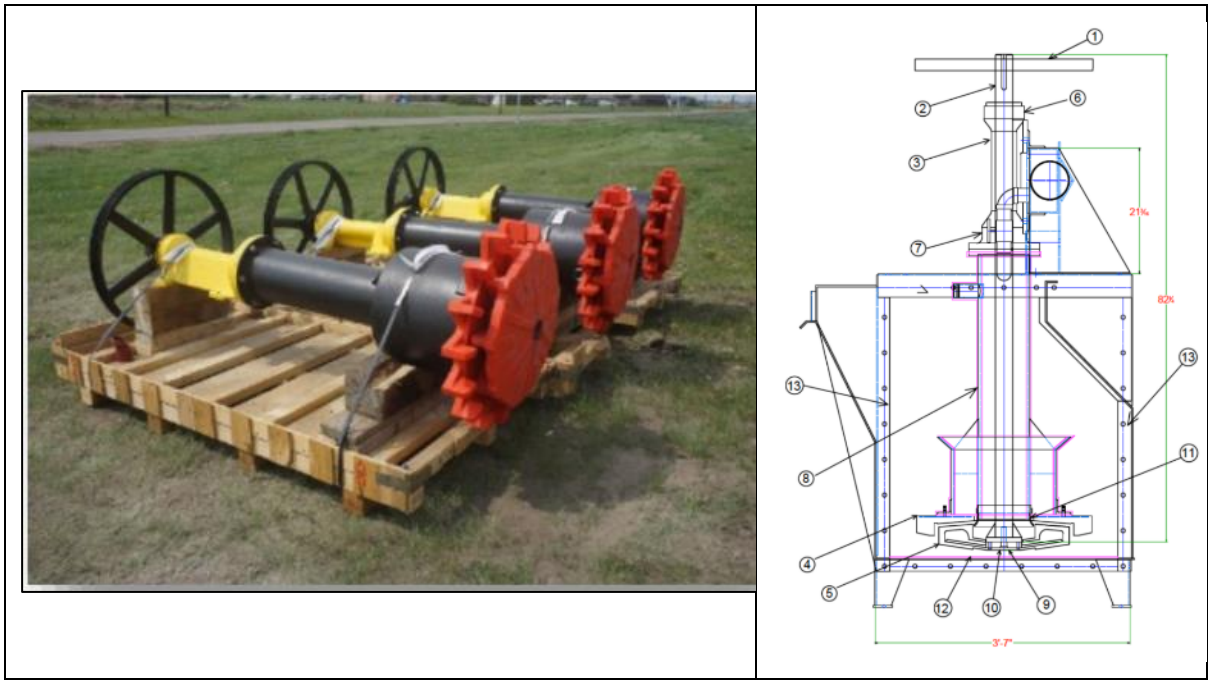



Figure 3.10.9. Left: Example of force air flotation mechanism. Right: Schematic of flotation mechanisms in a typical flotation tank.

The design of the flotation mechanism is usually dictated by several dimensionless scaling parameters that balance the rotor diameter, the rotational speed, the gas flow rate, and the overall power draw. A list of common dimensionless numbers used in flotation cell design is shown in Table 3.10.2. Using these dimensionless numbers and other calculated, a design spreadsheet was generated and used to determine the optimal equipment configuration. Table 3.10.3 shows the output from the current design revision. Red values in the table represent inputs, while the black values represent calculated outputs.

Table 3.10.2. Common Dimensionless numbers used in Flotation Machine Design (after Noble, 2012)

| Name | Expression | Description | Range |
|---|---|---|---|
| Reynolds Number, $Re$ | $\frac{\rho N D^2}{\mu}$ | Ratio of inertial force to viscous force | $(1-7)\cdot 10^6$ |
| Froude Number, $Fr$ | $\frac{DN^2}{g}$ | Ratio of inertial force to gravitational force | $0.5-5$ |
| Power Number, $P_N$ | $\frac{P}{\rho N^3 D^5}$ | Ratio of resistance force to inertial force | $0.5-5$ |
| Aeration Number, $N_A$ | $\frac{Q_a}{ND^3}$ | Ratio of gas flow to pumping capacity | $0.01-0.2$ |
| Weber Number, $We$ | $\frac{\rho N^2 D^3}{\sigma}$ | Ratio of inertial force to surface tension force | |

Legend: $\rho$ = fluid density, $N$ = rotor's rotational speed, $D$ = rotor diameter, $\mu$ = fluid viscosity, $g$ = gravitational acceleration, $P$ = power input, $Q_a$ = airflow rate, and $\sigma$ = surface tension

During the design process, emphasis was placed on selecting a rotor diameter that provided adequate power dissipation needed to suspend solids (ideally 5 to 10 HP/1000 gallon) at a low tip speed (<4.5 m/s). Other significant design factors include: a tank diameter to rotor diameter of less than 3.0 to ensure off bottom solid suspension and an aeration number less than 0.2 to prevent overgassing or flooding the rotor. Lastly, a total tank volume of greater than 2000 gallons is required to ensure adequate retention time. All the range values and thresholds for the critical factors are based on the PI's prior experience in flotation machine design (see Noble, 2012). Using this information, an ideal rotor diameter of 36 to 48 inches was selected to best balance the required technical criteria and the overall equipment cost.

Table 3.10.3. Flotation Machine Design Criteria

| Parameter | Value | Unit |
|---|---|---|
| *Tank Geometry* | | |
| Rotor Diameter | 36 | inch |
| Tank Diameter | 96 | inch |
| Liquid Height | 80 | inch |
| Active Tank Volume Ratio | 80% | v/v |
| Active Tank Volume | 463247 | in^3 |
| *unit conversion* | 268.1 | ft^3 |
| *unit conversion* | 2005 | gal |
| *unit conversion* | 7.59 | m^3 |
| Tank Area | 7238 | in^2 |
| *unit conversion* | 50.3 | ft^2 |
| *Pulp Properties* | | |
| Dry Solid Density | 5 | g/cc |
| % Solids | 15% | kg/kg |
| Water dyamic viscocity | 0.0008 | Pa s |
| Slurry Density | 1.13 | g/cc |
| Volumetric % Solids | 3.4% | v/v |
| $\mu_p$ / $\mu_f$ | 1.10 | Pa s / Pa s |
| Pulp Dynamic Viscodity | 0.00084318 | Pa s |
| *Aeration Properties* | | |
| Air Flow Rate | 100 | cfm / 1000 ft^3 of tank |
| Air Flow | 26.81 | cfm |
| Superficial Gas Velocity | 0.53 | ft/min |
| *unit conversion* | 0.27 | cm/s |
| *Power Properties* | | |
| Desired Tip Speed | 4.5 | m/s |
| Rotational Speed | 94.0 | RPM |
| *unit conversion* | 1.6 | RPS, Hz |
| Ungassed Power Number | 4.5 | HP/HP |
| P_gas/P_ungas | 90% | HP/HP |
| Ungassed Shaft Power | 16.7874 | HP |
| *unit conversion* | 12.518 | kW |
| Gassed Shaft Power | 15.109 | HP |
| *unit conversion* | 11.267 | kW |
| Dissipated Power | 7.5340 | HP/1000 gallon |
| *unit conversion* | 1.4841 | kW / m^3 |
| *Dimensionless Analysis* | | |
| D_tank/D_rotor | 2.67 | -- |
| Foude Number | 0.229 | -- |
| Reynolds Number | 1.06E+08 | -- |
| Aeration Number | 0.011 | -- |

Figure 3.10.10 contains the bioreactor P&ID describes the process by which pyrite was converted into sulfuric acid to be used for leaching to reduce cost. Pyrite was added by the barrel into a 250-gallon slurry tank. Water was added to keep the solid percentage around 50%, which was dictated by pumping limits. The resulting slurry was transferred into the first bio reactor at a low flowrate. Each of the first two bioreactors consisted of a 3000-gallon tank with a Denver floatation unit secured in the inside. The floatation unit was used to mix and aerate the tank. The bacteria will require oxygen for their reaction, so a process of aeration was necessary. These Denver-style units were modified from the original design: a longer shaft and rubber coated to be acid resistant. These units are shown in Figure 3.10.11. This type of mechanism was used because it could both suspend the solids and introduce air into the reactor. The shaft had a hollow outer portion and a solid inner portion, with a space for air to be forced through in between. The rotor section had an inner impeller with a stationary outer portion. As

the impellor spins, air is forced between the slats and bubbles up through the fluid. The mechanisms were oversized for this application because running at a high tip speed could destroy the bacteria.

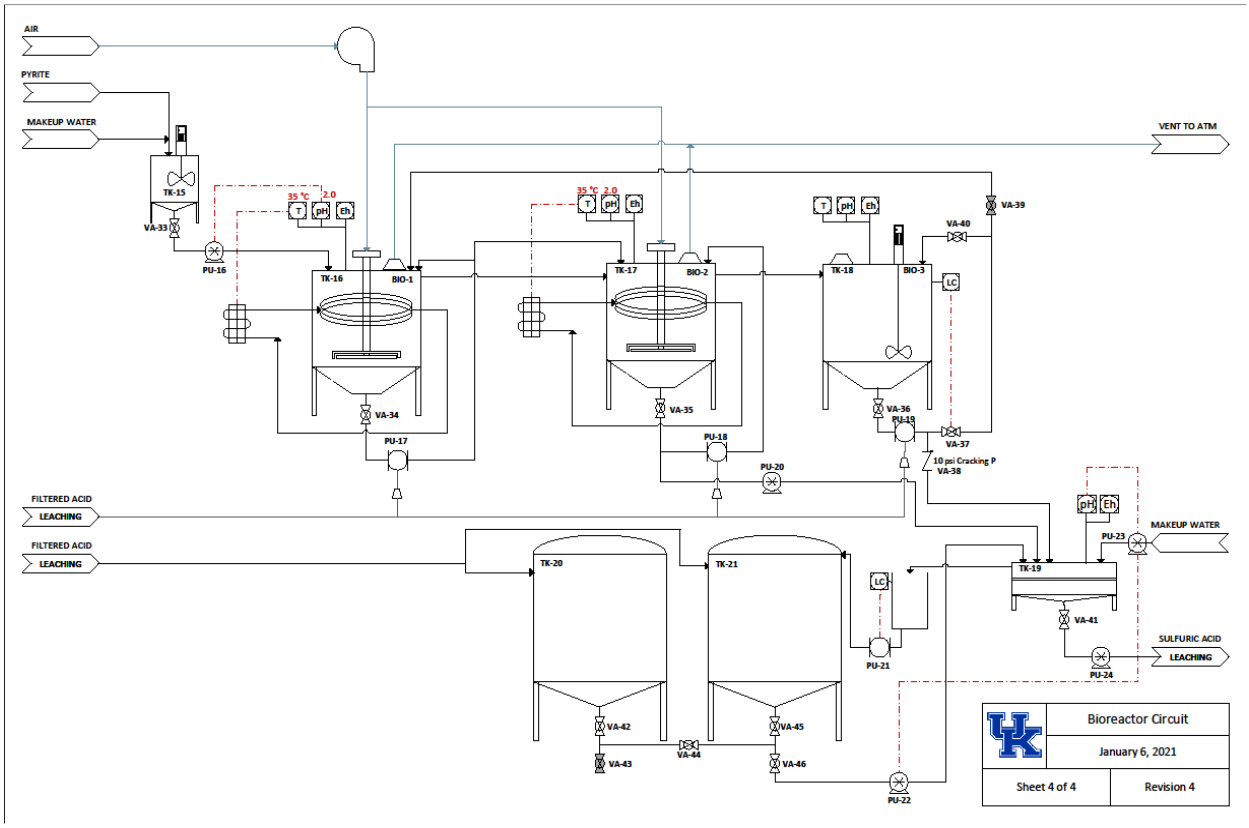
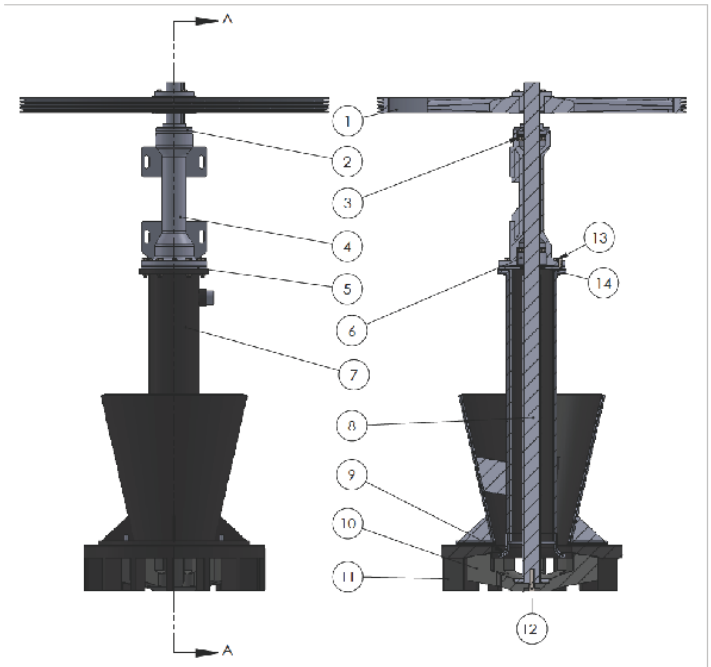



Figure 3.10.10. The Bioreactor Circuit Piping and Instrumentation Diagram




| Number | Part Number | Part |
|---|---|---|
| 1 | DAIDR24-01 | Pulley |
| 2 | DAIDR24-02 | Top cap for bearing |
| 3 | DAIDR24-03 | Bearing |
| 4 | DAIDR24-04 | Bearing housing |
| 5 | DAIDR24-05 | Lower bearing cap |
| 6 | DAIDR24-06 | Locking collar |
| 7 | DAIDR24-07 | Rubber coated stand pipe |
| 8 | DAIDR24-08 | Mechanism shaft |
| 9 | DAIDR24-09 | Rubber covered bell |
| 10 | DAIDR24-10 | Impeller |
| 11 | DAIDR24-11 | Diffusor |
| 12 | DAIDR24-12 | Fastener screw |
| 13 | DAIDR24-13 | Screw |
| 14 | DAIDR24-14 | Seals |

Denver Style DR24 mechanism parts list

D'ANGELO INTERNATIONAL

D'Angelo International, LLC

/ 15 2019

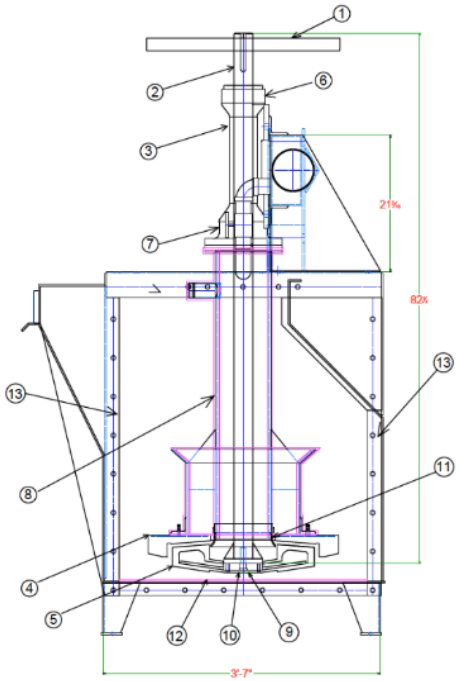



Figure 3.10.11. Diagram of Denver Mechanism(left). Denver Mechanism Design (right).

A blower was used to inject air into the system with the Denver units and baffles were installed on the inside of the tank to aid mixing and contact. There was also a cooling system to two bioreactors since the synthesis of sulfuric acid produces heat. A 1-inch stainless steel run of

pipe was run around the inside of the tanks with cooled water running through it. This was in lieu of a standard heat exchanger, since there was concern a heat exchanger would plug up too often for this situation's needs. The cooling pipe was attached to a water chiller for each unit, which cooled the system. It was calculated that around 40 feet of pipe would be required for each tank, so twice around was plenty. The temperature in the bioreactors was maintained at 35 degrees C. The bioreactors contained pH, ORP, and temperature sensors. The first two bioreactors were identical and run in series. There was an overflow line from the first to the second to safeguard against spillage. The second bioreactor was overflow into a third 3000-gallon tank. The third 3000-gallon tank was to be used as an aquarium to hold a bacteria-rich solution in case all of the bacteria in the bioreactor are killed. This tank required a mixer and a level control to prevent overflow. The level control caused a valve to divert flow from recycling to the settling tank. There were also two 5000 gallon holding tanks for storage. Since there were suspended solids in the bioreactor and it would have been a bad idea to have solids build up in the storage tank, the suspended solids must be removed using a thickener. It was possible to run the thickener underflow into the belt filter when leaching was not running, but due to the low flowrates, it was sufficient to plug the underflow and allow solids to settle and clean it out intermittently. The thickener overflow was to be collected in an overflow tank and will be continuously pumped out using a peristaltic pump. A spare diaphragm pump was also installed for this purpose for emergency situations where the level builds up. This was not expected to occur but is theoretically possible, so prevention measures needed to be established. When leaching was running, there was a second 400-gallon thickener which would be used to makeup hot acid for leaching at the proper pH. Makeup water would be added to control the pH. Addition of stored acid was added to ensure the proper flowrate. It was also possible to control the pH of the tank directly from the stored acid addition (pump 22) and match the proper flowrate from the addition of makeup water. To decide between these two options, it was best to control the pH based on the flow to which pH was most sensitive and match the proper flowrate (currently 2 gpm) with the other. For example, if an additional 0.1 gpm of water changed the pH more than an additional 0.1 gpm of acid, then pH would be controlled from water addition. It was expected that water addition would be more influential than acid addition, which is why it is shown that pH is being controlled from the makeup water. The tank overflow would run to an overflow tank where it would be pumped out using a peristaltic pump. These two controls working together would produce an acid at the correct pH with suitable flowrate.

Much of the equipment for the bioreactor system was new and required special orders. The Denver units themselves needed to be specially ordered from D'Angelo International. They required longer shafts than standard and an acid resistant rubber coating. There was some difficulty with the University of Kentucky regarding placing the order. The University requires all purchases over $40,000 to go through a month-long bidding process which was compounded by the fact that these were custom units. This cost nearly two months in the order of the Denver units. Due to all of this, ordering was delayed, and the units arrived mid-February 2021.

### *3.10.5 Summary of Construction*

- Everything in the old plant was unbolted and set to its new location. Due to the back order of the bio reactors which were set in the back of the plant, shown in the floor plan, some of the equipment could not be placed immediately. The precipitation circuit, solvent extraction circuit and oxalic acid circuit were among the first circuits to be completed and set in place because of the plant arrangement.

- A logistic procurement of scale up and essential equipment took place. Previously used equipment was assessed, rebuilt and put back into operation. Everything that was deemed ineffective due to ware or damage was procured. The scale up required more equipment such has tanks, tank stands, heat exchanger, PLC sensors, flow meters, various pumps and plumbing supplies. These items were secured as well.
- Compressed air lines were installed against the walls around the pilot plant in order to provide air to all pneumatic equipment and made accessible for potential optimization changes.
- A bioreactor containment area was designed large enough to contain the total volume of all three reactors. The containment is constructed of cement barriers. A single 60-mil polyethylene liner was bolted into the cement barriers and laid up the walls of the barriers and cover the floor.
- Custom Denver mechanisms and stands were outsourced to our design specifications and installed with the bioreactors in the containment area. Motors, belts, air blower and VFDs completed the construction of the bioreactors.
- The previous ventilation system was reconstructed to make space for the bioreactors. The ventilation fan was place outside of the pilot plant and the ventilation was piped around the backside of the bioreactors.
- Waste tanks were placed into the new and more secured containment area. A new PVC plumbing system from the pilot plant to the new containment area was established.
- Acid storage tanks were also placed into the new and more secured containment area. A new PVC plumbing system from the pilot plant to the new containment area was established as well.
- There were significant changes to the flow of material and new equipment introduced into the P&ID which called for extensive plumbing requirements and reassessment of pipeline sizing and procurement.
- The PLCs and redesign of the floor plan introduced a vast electrical demand. There was substantial wiring required for each sensor, motor and various equipment. The data set delivered from the PLC will allow real time data collection for analysis and optimization. The PLC will also provide safety alarms that are both audible and visual.
- The new SCADA system software will be used for storing and combining batch and continuous historical data which will then be used to analyze various processes for optimization, identify critical relationships between systems and produce figures and automated reports. The SCADA system is to be accessible remotely which will limit lag associated with multiple teams wanting access to data sets.

### *1.1.16 Plant Shakedown Activities*

Plant shakedown activities included performing a detailed systems safety analysis and associated health and safety training, required to ensure proper startup, operation and shutdown of the modified pilot-scale facility. The shakedown was completed under the supervision of the DOE on April 27, 2021

An FMEA report was completed by breaking down each processing unit and identifying every potential safety hazard and concern. The report included potential startup, operation and shutdown issues. Potential safety hazards and concerns were identified by assessing past issues within the pilot plant which included past incident reports and taking the severity of each potential safety hazard and concern is scaled on a 1-10 basis with 10 being the worst possible outcome. The level of detention was also addressed in the FMEA and is scaled on a 1-10 basis with 10 being extremely hard to detect past operators. Course of actions to reduce and eliminate

issues were addressed within the FMEA. The FMEA also lists the persons responsible for recommended remedial action and a due by date. These documents were provided in a separate report.

### *1.1.17 Data collection integration*

The pilot plant automation had a minimum of 34 data collection channels collected continuously with an additional 29 inputs totaling 63 data points. Further, assays collected had a minimum of 20 elements being reported at different times to when the process was running. This created a vast amount of data as the processes run continuously, which was unmanageable by manual means. Due to the extensive nature of the data collection at the pilot plant a data management, curation and analysis software system was needed to be effective in operating and managing the pilot plant. Lessons learned from the operation of the previous pilot plant indicated that data collection and curation were a significant operational issue and hampered the effectiveness of the researchers by not receiving actionable data in a timely manner. To solve this issue OSI soft software was implemented. OSI soft is a data management and analysis software that specializes in managing and curating large data sets from disparate sources such as PLC, user entered, and csv files for process plants. It is compatible with the Allan Bradely PLC system and was chosen over Factory Talk (Allan Bradely solutions) due to its ability to add time stamped data in from other sources like the ICP. It collects data, organizes data, displays data, and analyzes data in way that appears better than other software suits such as, AspenONE Engineering, Factory Talk, Wonderware. What is key in OSI is something called "Asset Framework" and "Event Frames" which defines the hierarchy of the data which makes it essential to define and organize data into context. It appears to be the superior information management system between the Control Layer (PLC) and Business layer (SAP) for the process industry.

### *1.1.18 Controls and Data Acquisition*

**Control Theory:** The scaled pilot plant was first conceptualized, then a flow sheet was developed and finally a P&ID was created. Once those tasks were completed the control theory was introduced. First, plant operation and environmental safety was considered based on the P&ID. Every essential control variable to maintaining specified values and effective high yield recovery of REOs was identified and developed around each individual unit. The table below shows key safety and production variables.

Table 3.10.4. Summary of key safety and production variables.

| Control Variable | Essential to Safety | Critical to Production |
|---|---|---|
| Temperature | X | X |
| pH | X | X |
| Fluid Level in Tanks | X | X |
| Airline Pressure | X | X |
| Pumps | | X |
| Motors | | X |
| Percent Solids | | X |
| ORP | | X |
| Residence Time | | X |
| Flow Rates | | X |
| Water Make Up | X | X |
| Acid Make Up | X | X |
| Base Make Up | X | X |

**PLC setup**

PLC input channels are used to monitor plant devices such as pH sensors, ultrasonic sensors and other switches. The selection of input module type results from the various types of input devices used at the pilot plant. Some devices respond to either ON/OFF digital inputs whilst others respond to analog signals, which represent process conditions via current or voltage signals. The outputs control devices such as pumps, motors and acid/base/water makeups. In operation, the output channel converts signals into either analog (4-20mA) or digital signals. Table shown below shows a summary list of the input and output control channels list for the plant process control system. Also included as a representation is Tank 1 showing what are typical channels for a leach take.

Table 3.10.5. REE Plant Equipment and I/O List.

| Devices | Digital Input | Relay Output | Analog Input | Analog Output |
|---|---|---|---|---|
| **Total I/O Points Needed** | **12** | **28** | **74** | **18** |
| Tank 1 | | | | |
| PU-1 (Pneumatic Pump) | | x | | |
| PH Sensor | | | x | |
| EH Sensor | | | x | |
| Temperature Sensor | | | x | |
| Level Sensor (Float) | x | | x | |

**HMI (Human/Machine/Interface)-GUI (Graphical User Interface) Design**

The graphics user interface (GUI) of the pilot plant was displayed on the PLC's human machine interface (HMI). The design of HMI was achieved with a computer based FactoryTalk View software by AB Rockwell Automation. The HMI was mounted on the PLC enclosure box and is made available to the operator on the plant floor. Different display screens (shown in Figure 3.10.12) were created along with the ladder logic programming of the various parts of the plant system. The HMI main screen presented the operator (user) with an overview of each processing unit within the pilot plant. The user was able to scroll through screens within the HMI and find displays for the critical process variables in real time. A brief status and description was also provided. The HMI was also used as a monitoring tool for present and past data of analog and digital types. These trend graphs were designed and programmed using the FactoryTalk View SE software. Based on operators need; the HMI trend displayed control performance at different times. The control system allowed various process variables to be maintained within specified limits in order for the plant to operate smoothly. Routine process monitoring would ensure that system performance satisfied the operating objectives of the plant.




Figure 3.10.12. Depiction of various PLC input and control screens.

## 3.11 FEEDSTOCK COLLECTION, PREPARATION, AND CHARACTERIZATION (TASK 13)

### *3.11.1 Feedstock Collection*

West Kentucky No. 13 and Fire Clay coarse refuse material was received for initial testing purposes. Approximately 20 tons of Fire Clay material was delivered to the DXRT sorter site from which three 55-gallon drums of sample was taken for characterization purposes. Likewise, a large bulk sample from the West Kentucky No. 13 coarse refuse was collected and density fractionation to determine the potential for upgrading the REE content by density fractionation.

### *3.11.2 Feedstock Characterization*

The West Kentucky No. 13 samples were cut at 3 specific gravity (S.G) cut-points (1.80, 2.00, and 2.20 S.G.) resulting in 4 density fractions (1.80 S.G float, 1.80 x 2.00 S.G., 2.00 x 2.20 S.G., and 2.20 S.G. sink). The Fire Clay coarse refuse material was fractionated at 4 S.G. cut points (1.60, 1.80, 2.00 and 2.20 S.G.) resulting in 5 density fractions (1.60 S.G float, 1.60 x 1.80 S.G, 1.80 x 2.00 S.G., 2.00 x 2.20 S.G., and 2.20 S.G. sink). The sink float analysis for both the samples was performed by SGS North America Inc.

The mass yield for fractions less than a 2.20 S.G. was significantly small for both the materials, 4.9% for West Kentucky No. 13 material, and 12.5% for Fireclay material (Figure 3.11.1). The low mass yield for lighter fractions is indicative of the high density cut point and the cleaning efficiency of the process employed in the processing plant. The density cut point for the coal preparation plant processing for West Kentucky No. 13 was 1.70 and the cut point for Fireclay material was 1.60.

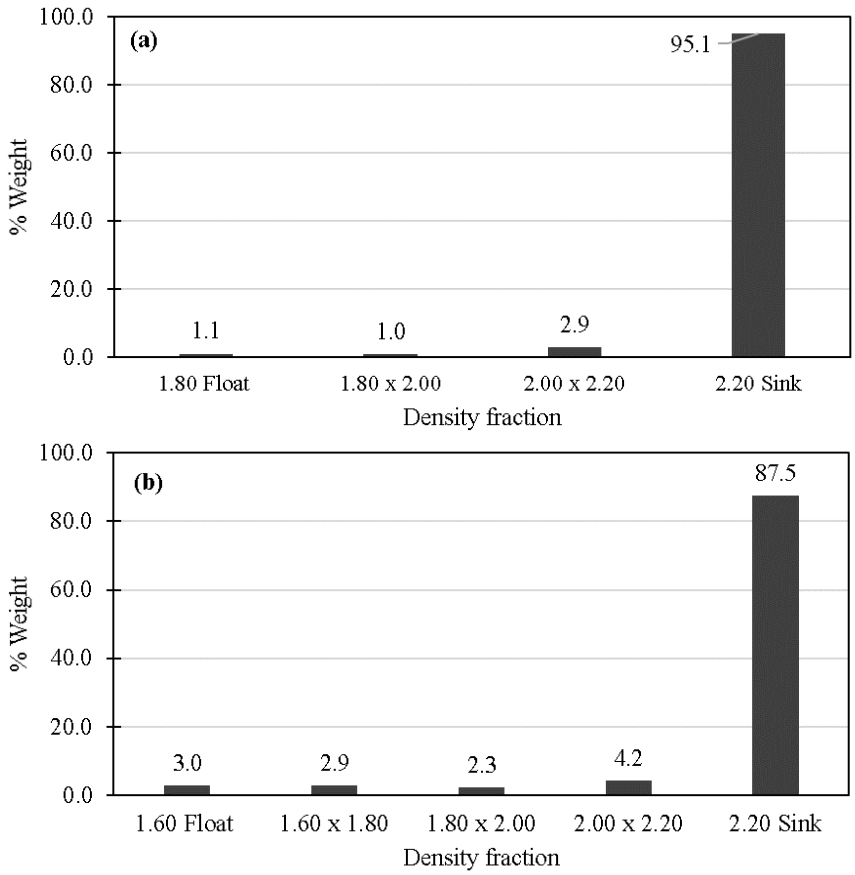



Figure 3.11.1. The weight distribution of the (a) West Kentucky No. 13 and (b) Fire Clay coarse refuse samples in different density fractions.

Proximate analysis was performed on each density fraction of both the samples using a thermogravimetric analyzer, TGA-701. Based on the results of the proximate analysis, the composition of the materials was estimated. The composite samples of both the materials were inherently dry with very low moisture content. The West Kentucky No. 13 composite sample had 1.2% moisture while the Fire Clay composite sample had 1.1 % moisture. The composite sample of West Kentucky No. 13 had 88.8% ash material with 10% carbon content while the composite sample of Fireclay had 85.6 % ash material and 13.3% carbon content as shown in Figure 3.11.2.

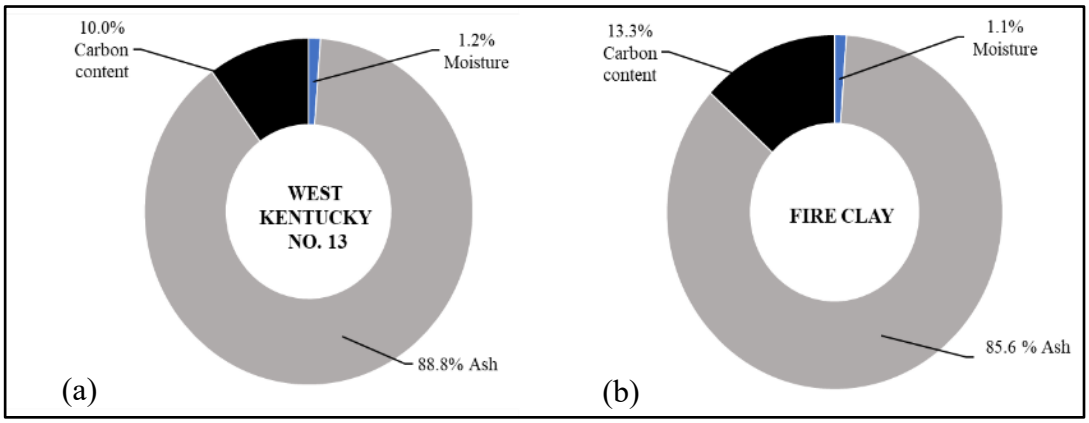


Figure 3.11.2. Composition of the composite samples of a.) West Kentucky no. 13 samples and b.) Fire Clay coarse refuse samples.

A more detailed analysis of the proximate analysis performed on individual density fractions revealed that there were significant differences in the composition of individual density fractions (Figure 3.11.3). The lowest density fraction (1.8 S.G. float) of West Kentucky No. 13 material contained 75.0% combustible content while the highest density fraction (2.2 S.G. sink) was comprised of 8.3% combustible content. Similarly, for the Fire Clay material the lowest density fraction was comprised of 70.1% combustible content while the highest density material contained around 8.7% carbon content. The 2.2 S.G. sink fraction for both the sample had similar composition and moisture content despite having different mass yields.

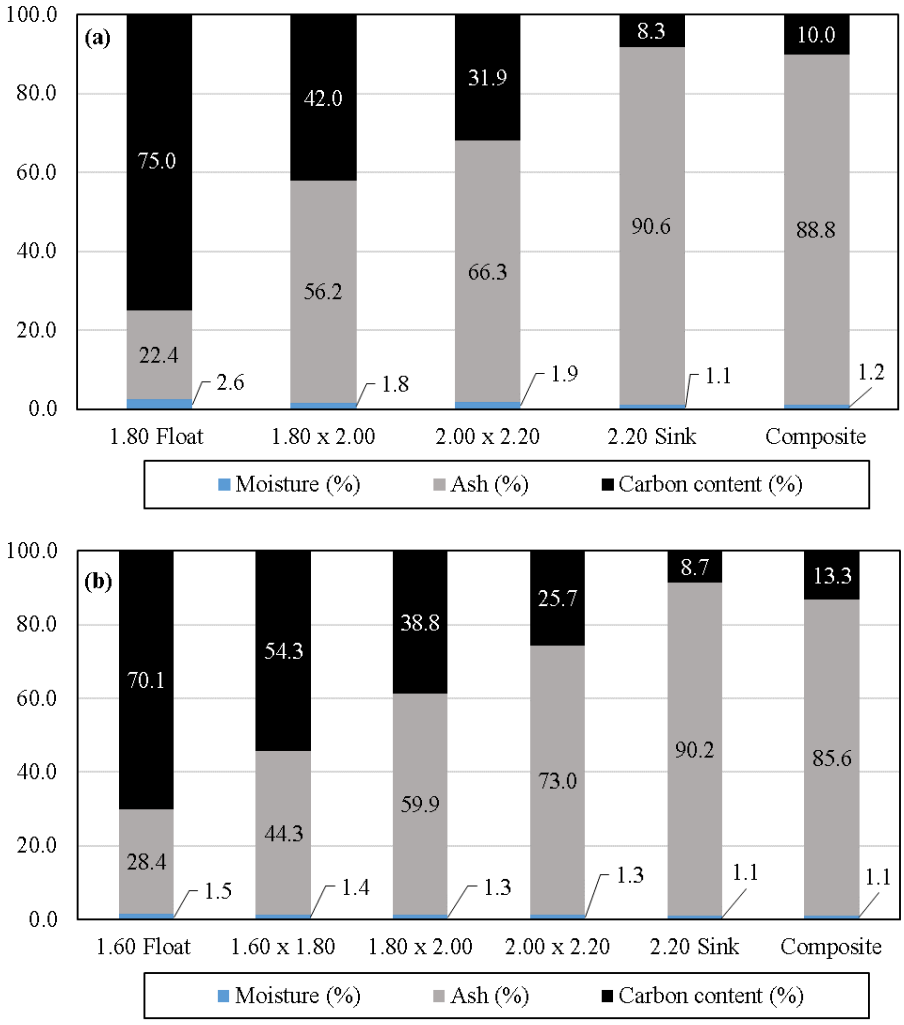



Figure 3.11.3. Composition of different density fractions of (a) West Kentucky No. 13 and (b) Fire Clay coarse refuse samples.

The sulfur analysis of the different density fractions was carried out using the LECO sulfur analyzer. Total sulfur in the ultimate analysis is the measured weight percent of sulfur in the coal. It is determined by ASTM methods D5373-08, or D4239-02 (American Society for Testing and Materials, 2013, p. 628–636, 556–561, respectively). In the analyzer, the sample is combusted at a temperature of 1,370°C in an oxygen atmosphere. The oxygen reacts with sulfur to form sulfur dioxide gas. The gas passes through an infrared absorption detector in the analyzer, which measures the concentration of sulfur.

The results (Figure 3.11.4) show that the composite sample of West Kentucky No. 13 contains significantly higher sulfur content (1.09 %) as compared to Fire Clay composite sample (0.28 %). This was expected as West Kentucky No. 13 has been shown in previous studies to contain higher concentration of pyrite which results in the higher sulfur content of the sample. For both the materials, the sulfur content in the 2.20 S.G. sink is lower than other density fractions. The density fraction 2.00 X 2.20 S.G. of West Kentucky No. 13 had the highest sulfur concentration at 6.24%. For the Fire Clay material, the sulfur content in all the density fractions lighter than 2.20 S.G. sink had similar sulfur concentrations, while the 2.20 S.G. sink had significantly lower sulfur content.

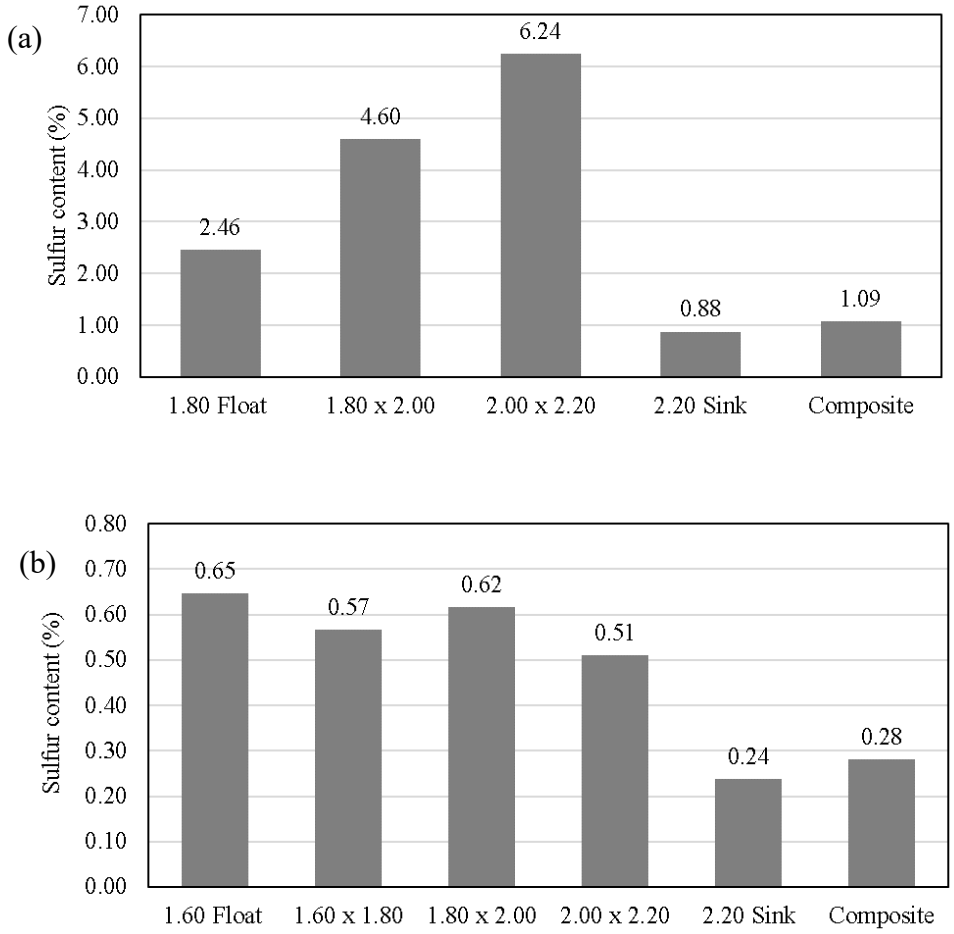



Figure 3.11.4. Sulfur concentrations of different density fractions and the composite samples of (a) West Kentucky No. 13 material and (b) Fire Clay

The different density fractions were subjected to X-Ray Fluorescence (XRF) analysis to measure the concentration of the major elements present in the samples (Table 3.11.1 and Table 3.11.2). The samples were roasted at 750°C for two hours in a muffle furnace to remove the combustible material. The results of the XRF analysis showed that $SiO_2$, $Al_2O_3$, and $Fe_2O_3$ were the major components in the ash obtained from the West Kentucky No. 13 sample. Combined, the three components represented more than 93% of the total weight of the sample. The $Fe_2O_3$ contribution, which is indicative of the pyrite concentration was highest in the 2.00 X 2.20 S.G. fraction. The high iron content along with high sulfur content indicates a high pyritic concentration in this density fraction.

The majority of the Fire Clay sample consisted of SiO2, Al2O3 and Fe2O3 and K2O. The four elements together comprised more than 97% of the total weight of the samples. It was

noteworthy that the iron content in the Fire Clay samples is significantly lower than that in the West Kentucky samples. This observation is in line with the fact that the pyrite content in the Fire Clay material is known to be lower as compared to West Kentucky No. 13 material.

Table 3.11.1. XRF analysis of different density fractions of West Kentucky No. 13 samples on ash basis

| Sample | $SiO_2$ (%) | $Al_2O_3$ (%) | $Fe_2O_3$ (%) | CaO (%) | MgO (%) | MnO (%) | $Na_2O$ (%) |
|---|---|---|---|---|---|---|---|
| 1.80 Float | 54.71 | 25.84 | 12.71 | 1.53 | 0.90 | 0.02 | 0.07 |
| 1.80 x 2.00 | 54.71 | 25.92 | 12.77 | 1.51 | 0.90 | 0.02 | 0.06 |
| 2.00 x 2.20 | 52.41 | 26.21 | 15.36 | 0.83 | 1.01 | 0.02 | 0.05 |
| 2.20 Sink | 62.38 | 24.44 | 6.91 | 0.49 | 1.48 | 0.08 | 0.18 |
| Sample | $K_2O$ (%) | $P_2O_5$ (%) | $TiO_2$ (%) | BaO (%) | SrO (%) | $SO_3$ (%) | Total (%) |
| 1.80 Float | 2.72 | 0.21 | 1.37 | 0.06 | 0.05 | 0.98 | 101.16 |
| 1.80 x 2.00 | 2.73 | 0.21 | 1.36 | 0.07 | 0.05 | 0.49 | 100.80 |
| 2.00 x 2.20 | 2.88 | 0.40 | 1.07 | 0.06 | 0.10 | 0.50 | 100.89 |
| 2.20 Sink | 3.53 | 0.17 | 1.05 | 0.07 | 0.04 | 0.33 | 101.15 |

Table 3.11.2. XRF analysis of different density fractions of Fire Clay samples on ash basis.

| Sample | $SiO_2$ (%) | $Al_2O_3$ (%) | $Fe_2O_3$ (%) | CaO (%) | MgO (%) | MnO (%) | $Na_2O$ (%) |
|---|---|---|---|---|---|---|---|
| 1.60 Float | 66.56 | 26.30 | 2.54 | 0.76 | 0.71 | 0.01 | 0.00 |
| 1.60 x 1.80 | 66.38 | 25.19 | 2.71 | 1.42 | 0.78 | 0.01 | 0.00 |
| 1.80 x 2.00 | 66.97 | 26.41 | 2.95 | 0.57 | 0.76 | 0.01 | 0.00 |
| 2.00 x 2.20 | 67.95 | 26.89 | 2.97 | 0.33 | 0.76 | 0.01 | 0.00 |
| 2.20 Sink | 62.62 | 27.20 | 5.33 | 0.35 | 1.41 | 0.05 | 0.16 |
| Sample | $K_2O$ (%) | $P_2O_5$ (%) | $TiO_2$ (%) | BaO (%) | SrO (%) | $SO_3$ (%) | Total (%) |
| 1.60 Float | 2.66 | 0.13 | 1.96 | 0.08 | 0.04 | 0.32 | 102.07 |
| 1.60 x 1.80 | 2.92 | 0.09 | 1.70 | 0.08 | 0.03 | 0.52 | 101.83 |
| 1.80 x 2.00 | 2.65 | 0.14 | 1.58 | 0.08 | 0.04 | 0.24 | 102.40 |
| 2.00 x 2.20 | 2.28 | 0.15 | 1.44 | 0.08 | 0.04 | 0.10 | 103.00 |
| 2.20 Sink | 3.19 | 0.10 | 1.13 | 0.08 | 0.03 | 0.18 | 101.83 |

The trace elements e.g. the REEs, cannot be detected by XRF analytical technique as they are below the detection limit of the technique. For measuring the concentration of these trace elements, Inductively Coupled Plasma Spectroscopy Optical Emission Spectroscopy (ICP-OES) technique was used. The representative samples were digested by the methodology described in the ASTM D6357-11 procedure. The standard solid sample supplied by the National Institute of Standards (NIST) was used as a quality control standard for each batch of solids analyzed. The ICP analysis was carried out by KGS.

The REE concentrations of the different density fractions for West Kentucky No. 13 and Fire Clay material are shown in Figure 3.11.5 and Figure 3.11.6. The West Kentucky No. 13 composite sample contained 297.5 ppm TREEs while Fire Clay composite sample contained 365.5 ppm TREE. The light REE (LREE) dominated the overall concentration of REEs for both the materials. Lanthanum, cerium, and neodymium contributing more than 70 % of the

total REEs present in both the materials. The lighter density fractions for both West Kentucky No. 13 and Fire Clay had significantly higher REE concentration than the heaviest 2.20 S.G. sink fraction.

A weighted average of the density fractions lighter than 2.20 had an REE concentration of 506.2 ppm for West Kentucky No. 13 and 727.7 ppm for Fire Clay material. This indicates that REEs can be significantly concentrated using density separation. However, it is important to note that the mass yield of the density fractions lighter than 2.20 S.G. is very low. The mass yield of density fractions lighter than 2.20 is 4.9% for Dotiki and 12.5% for Fireclay material. This would correspond to an REE recovery of 1.70% for West Kentucky No.13 and 1.99% for the Fire Clay coarse refuse material even if a density separation was made at 2.20 S.G. Therefore, it will not be economically feasible to concentrate the REEs using density separation, even though the REE concentration in the lighter material is significantly higher than the heavier fraction.

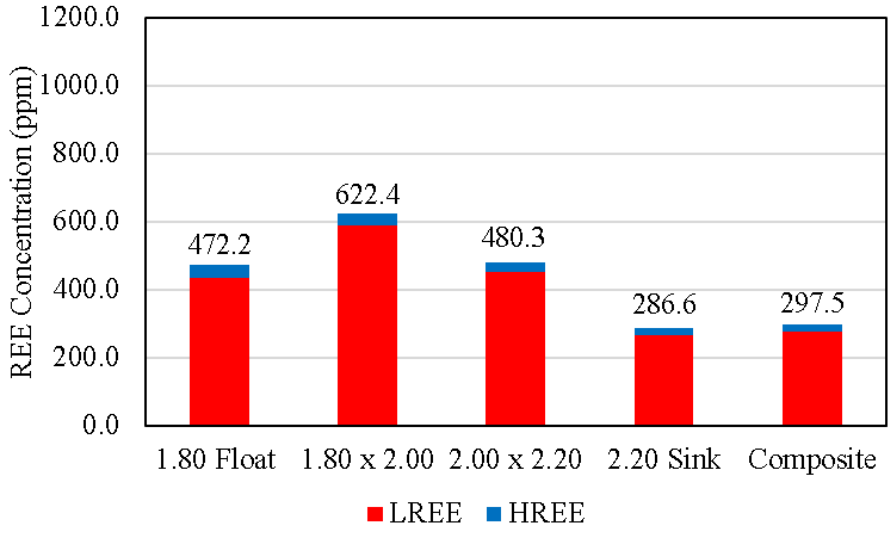



Figure 3.11.5. REE concentration for Western Kentucky No. 13 coarse refuse samples based on density fractions.

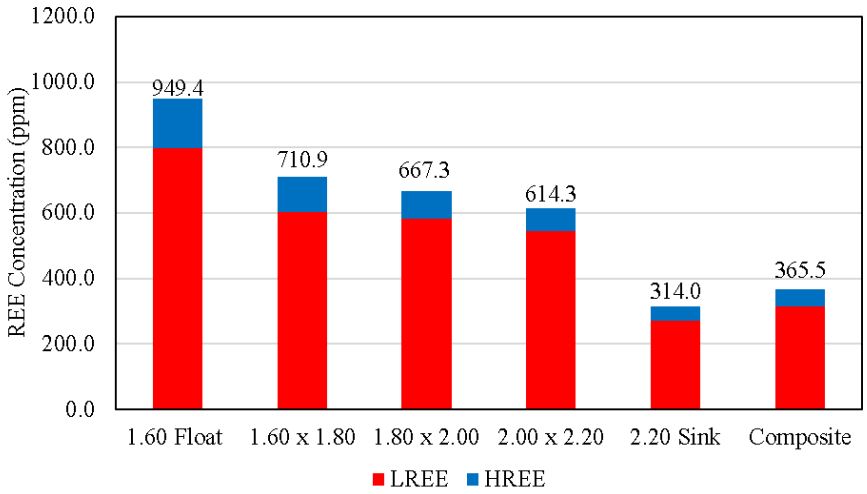



Figure 3.11.6. REE concentrations in each density fraction of the Fire Clay coarse refuse sample on a dry, ash basis.

Table 3.11.3. REE concentrations (ppm) by ICP-OES in each density fraction of West Kentucky No. 13 coarse refuse on a dry, ash-basis.

| Density Fraction | Sc | La | Ce | Pr | Nd | Sm | Eu | Gd | LREE |
|---|---|---|---|---|---|---|---|---|---|
| 1.80 Float | 28.3 | 79.0 | 186.8 | 20.4 | 78.4 | 25.8 | 3.1 | 11.4 | 433.1 |
| 1.80 x 2.00 | 25.2 | 107.8 | 262.8 | 28.8 | 112.2 | 32.0 | 4.3 | 14.2 | 587.2 |
| 2.00 x 2.20 | 21.4 | 80.3 | 194.6 | 22.2 | 85.2 | 31.3 | 3.5 | 12.0 | 450.5 |
| 2.20 Sink | 17.2 | 51.7 | 107.3 | 12.8 | 50.5 | 15.2 | 1.9 | 8.2 | 264.8 |

| Density Fraction | Tb | Dy | Ho | Er | Tm | Yb | Lu | Y | HREE |
|---|---|---|---|---|---|---|---|---|---|
| 1.80 Float | 2.0 | 9.1 | 4.0 | 7.5 | 1.5 | 6.7 | 1.7 | 6.7 | 39.1 |
| 1.80 x 2.00 | 1.0 | 8.3 | 4.1 | 6.8 | 1.2 | 6.0 | 1.9 | 6.0 | 35.2 |
| 2.00 x 2.20 | 1.3 | 6.4 | 2.6 | 5.8 | 0.9 | 5.3 | 2.1 | 5.3 | 29.8 |
| 2.20 Sink | 0.8 | 4.8 | 2.6 | 5.1 | 0.7 | 3.6 | 0.7 | 3.6 | 21.8 |

Table 3.11.4. REE concentrations (ppm) by ICP-OES of different density fractions of Fire Clay samples.

| Density Fraction | Sc | La | Ce | Pr | Nd | Sm | Eu | Gd | LREE |
|---|---|---|---|---|---|---|---|---|---|
| 1.60 Float | 32.7 | 154.8 | 382.7 | 36.4 | 145.5 | 28.5 | 3.3 | 14.2 | 798.2 |
| 1.60 x 1.80 | 27.9 | 116.4 | 285.5 | 27.9 | 111.6 | 21.9 | 2.8 | 8.7 | 602.6 |
| 1.80 x 2.00 | 22.1 | 113.7 | 280.6 | 27.0 | 105.8 | 21.9 | 2.9 | 9.2 | 583.2 |
| 2.00 x 2.20 | 19.3 | 106.0 | 264.1 | 25.6 | 97.5 | 20.6 | 2.9 | 9.3 | 545.4 |
| 2.20 Sink | 18.0 | 53.5 | 118.2 | 13.6 | 49.8 | 10.8 | 1.8 | 4.6 | 270.3 |

| Density Fraction | Tb | Dy | Ho | Er | Tm | Yb | Lu | Y | HREE |
|---|---|---|---|---|---|---|---|---|---|
| 1.60 Float | 0.8 | 23.6 | 4.9 | 10.6 | 4.1 | 9.8 | 0.8 | 96.6 | 151.2 |
| 1.60 x 1.80 | 0.8 | 16.9 | 3.6 | 9.1 | 3.0 | 7.1 | 0.4 | 67.3 | 108.2 |
| 1.80 x 2.00 | 0.5 | 13.8 | 3.2 | 7.8 | 2.7 | 5.5 | 0.3 | 50.2 | 84.1 |
| 2.00 x 2.20 | 0.4 | 11.7 | 2.7 | 7.3 | 2.3 | 4.5 | 0.1 | 39.9 | 68.9 |
| 2.20 Sink | 0.7 | 6.1 | 1.3 | 5.3 | 1.3 | 3.5 | 0.4 | 25.1 | 43.7 |

### *3.11.3 Feedstock Preparation*

Approximately 20-ton of Dotiki coarse refuse material was collected and delivered to the Dotiki coal preparation site where the X-ray sorter is located. The as received fresh material was characterized at the SGS lab in Henderson, KY. The material was screened at different size and the total REEs concentration in plus 2", 2"x1", 1"x1/2", and minus ½" was 232 ppm, 256 ppm, 294 ppm, and 287 ppm, respectively. During natural weathering, the swelling clay material within the inner partings of coal particle and the shell material dehydrated and hydrated alternatively, which caused the particle to shrink and expend. The coal and shell particle showed a tendency to break down during regular material handling. A representative sample of the weathered material was collected in two of the 5-gallon buckets from the weathered pile and screened at 1-1/2" and 3/8" to assess the particle size change and the TREE migration. The particle size distribution before and after weathering is plotted in Figure 3.11.7,

where 81% of material was passed through 1.5 inch screen after weathering whereas 78% of particle was smaller than 2 inch in a fresh pile. The TREE analysis for the weathered material is shown in Table 3.11.5. Due to the breakage of the lighter density fraction material, the REEs bearing material breaks down and transports to the smaller size fraction, resulting in an enrichment of TREEs in the finer size material. The TREEs on a whole mass basis for the Plus 1-1/2", 1-1/2"x3/8", and minus 3/8" material is 237 ppm, 303 ppm, and 307 ppm, respectively. The individual element distribution of REEs in the three size fractions is as shown in Figure 3.11.8.

Leaching characteristic analysis were conducted on the three size fractions material before and after roasting. The material was split using a raffle splitter and pulverized to a top particle size of 80 mesh (177 microns). Then a split of each pulverized material was roasted at 600 °C for 2 hours in a muffle furnace. The leaching experiments were carried out using 1.2M trace metal sulfuric acid (Fisher Scientific) at 75°C temperature with 10 g/L solid concentration. Sample were collected at different time interval to understand the leaching kinetics. As shown in Figure 3.11.9, the leaching recovery of TREEs at 2 hours before roasting is 23%, 17% and 15% for plus 1-1/2", 1-1/2"x3/8", and minus 3/8" material, respectively. After roasting, the recovery increased to 61%, 60%, and 62%, respectively. The results show that roasting improved the leaching characteristic by transferring mineral bond RE to acid soluble form of RE. Even thought the leaching characteristics are similar for all three size fractions, the recovery of Fe plays an important role. The Fe content in the plus 1-1/2", 1-1/2"x3/8", and minus 3/8" material after roasting is 12.9%, 4.9%, and 3.4%, respectively. The dissolved solid by weight for the plus 1-1/2" roasted material was 30% by weight. About 66% of Fe was leached into solution from the plus 1-1/2" roasted material which can harm the downstream purification process. The plus 1-1/2" material retained on the screen deck is mostly the hard rock material from the pile. Three signature composition of the weathered pile were virtually identified and hand-picked form the pile before processing. The coal and shell material were very fragile and easily fragmented during screening whereas the hard rock material was held together and remained on the screen. The XRD analysis shows that the hard rock material contains significant amount of siderite, which is an iron carbonate ore. Therefore, by ejecting the Dotiki coarse refuse material at 1-1/2", the majority of iron was removed from feeding into the leaching circuit and TREEs were enriched.

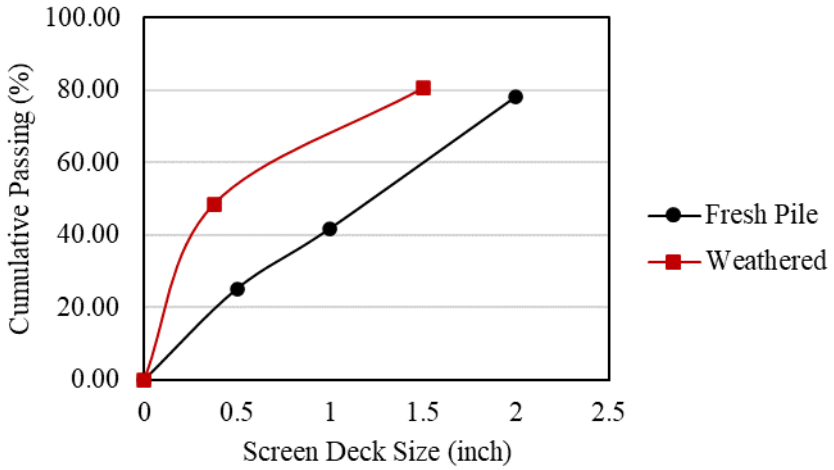



Figure 3.11.7. Screening analysis of Dotiki coarse refuse material before and after weathering.

Table 3.11.5. TREE concentration in different size fraction of weathered Dotiki coarse refuse.

| Size Fraction | Wt. (%) | % Ash, ad | Total REE (ppm) | |
|---|---|---|---|---|
| | | | Ash Basis | Whole Basis |
| Plus 1-1/2" | 19.5 | 86.4 | 273.9 | 236.8 |
| 1-1/2" x 3/8" | 32.1 | 87.2 | 347.6 | 303.0 |
| Minus 3/8" | 48.5 | 87.5 | 350.2 | 306.6 |
| Total | 100 | 87.2 | 334.7 | 291.8 |

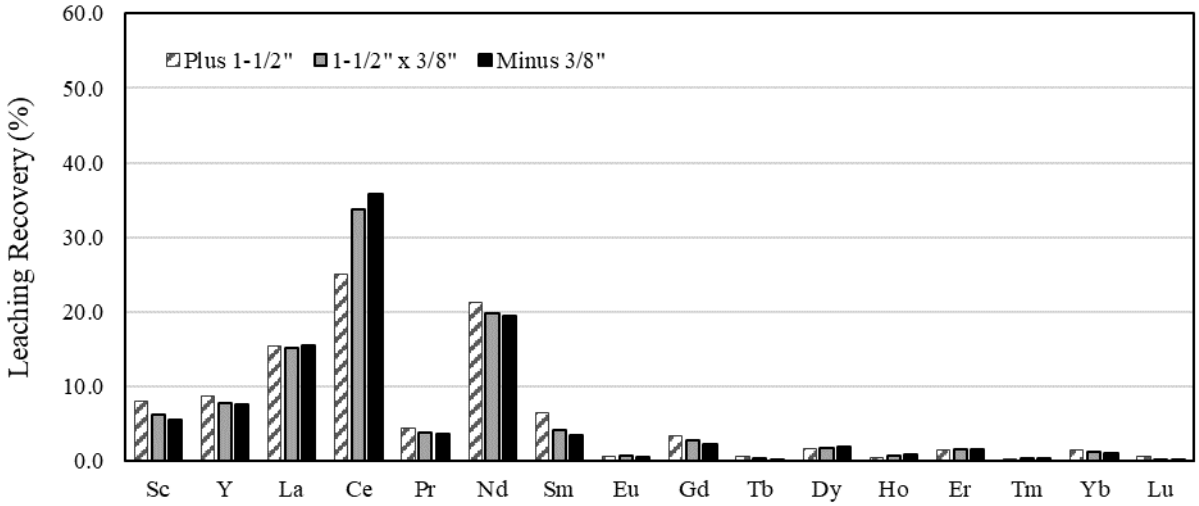



Figure 3.11.8. Individual rare earth element distribution in three size fractions of weathered material.

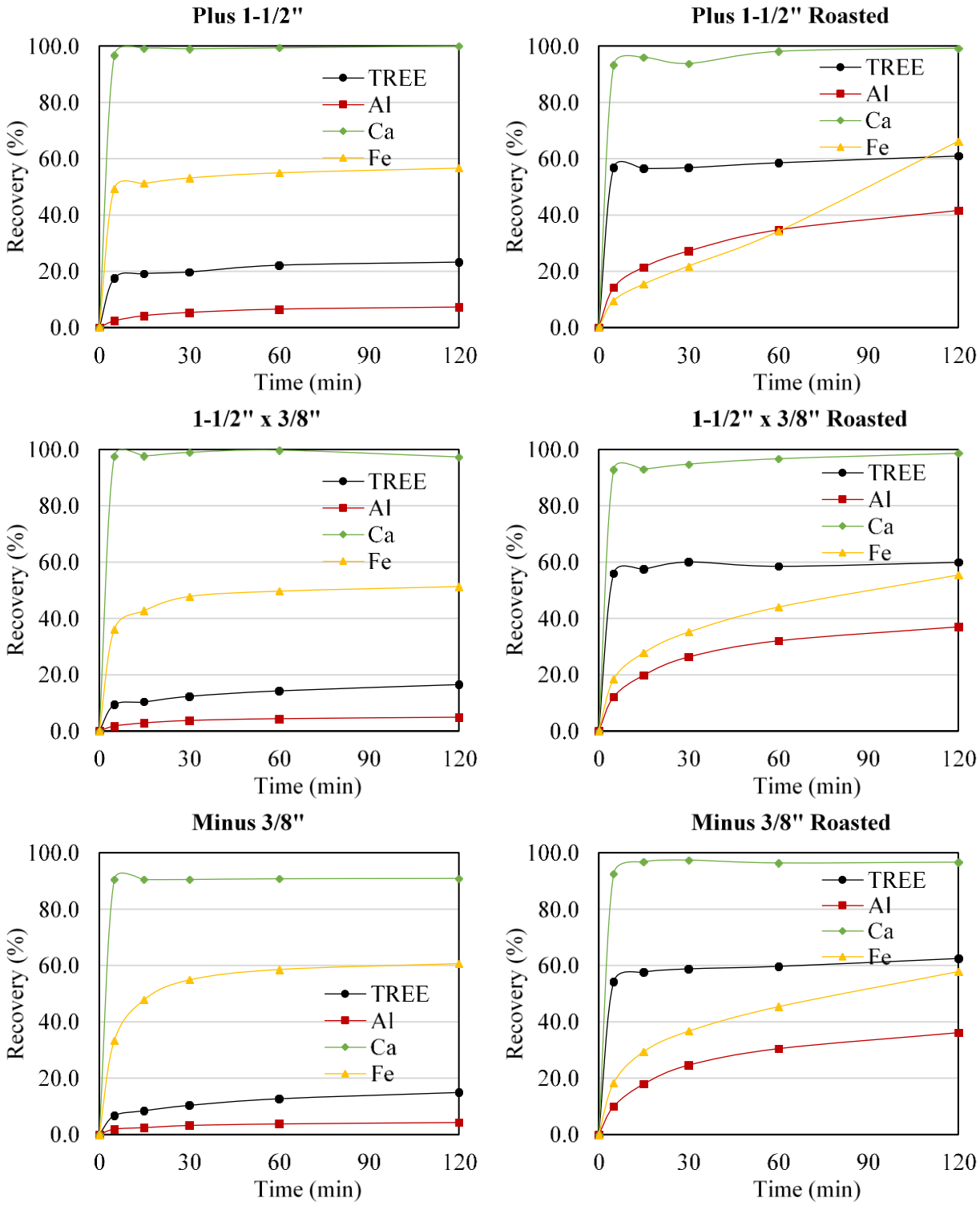



Figure 3.11.9. Leaching characteristics of different size Dotiki coarse refuse material with and without roasting.

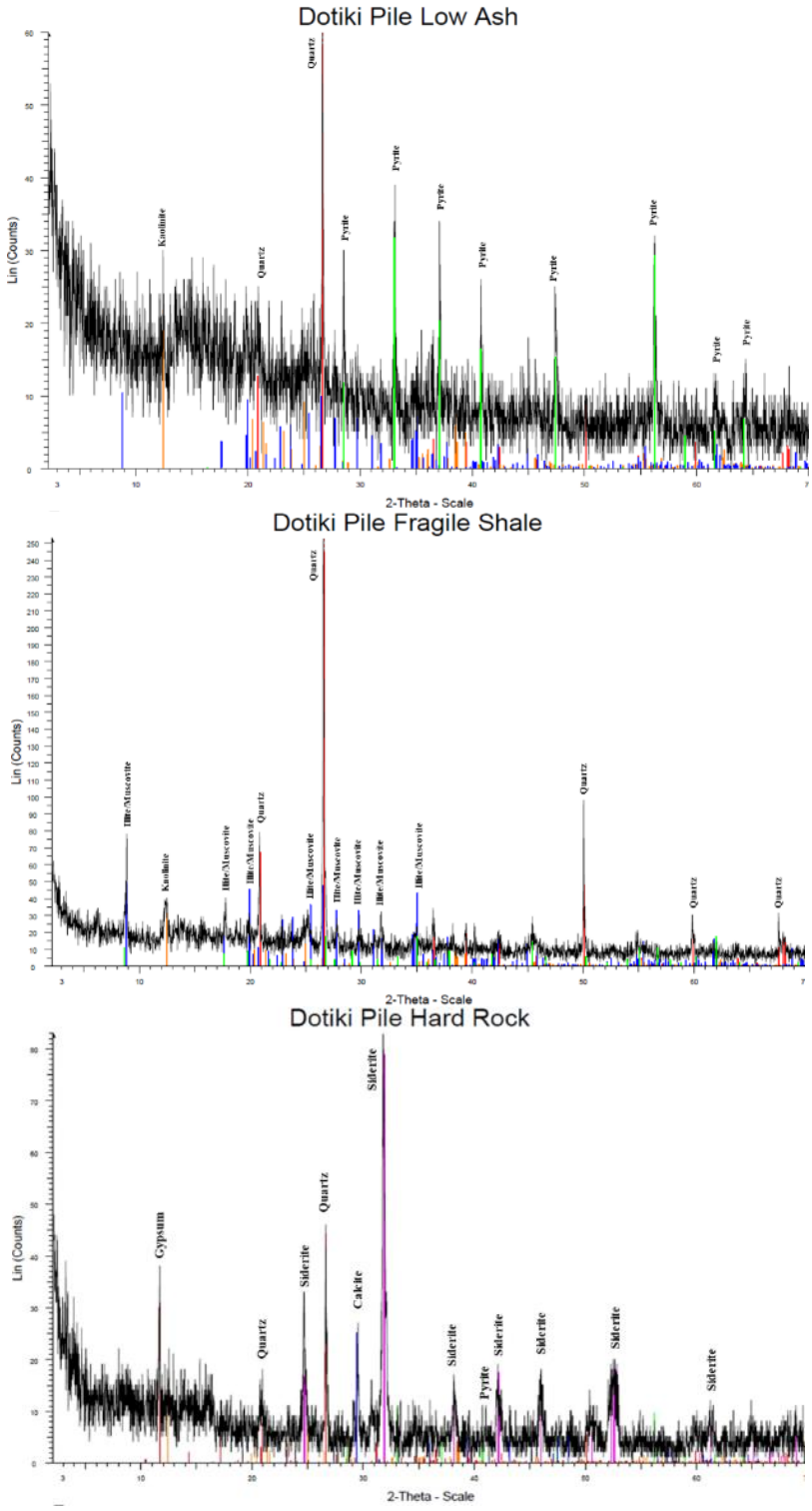



Figure 3.11.10. XRD analysis of three signature material visually identified from the weathered Dotiki coarse refuse pile.

### *1.1.19 Sequential Extraction*

The sequential extraction tests utilized a four-step extraction sequence to identify the ion exchangeable, carbonate, metal oxide, acid soluble, and residual REE phases. The current testing program focused on the as-received density fractionated material, and future tests will be conducted on samples subjected to roasting pretreatment. For all tests, the final residual

assay is still pending analysis, and as a result, the experimental data is reported herein as concentration of REE extracted, rather than total recovery. Figure  and Figure  show the total REE extracted from each density fraction for both samples, while Figure  and Figure  show the same data on an element-by-element basis. In most cases, the acid soluble fraction grossly exceeded that of other fractions with the lone exception being the West Kentucky No. 13, 2.2 sink material, where the metal oxide fraction was predominant.  In comparing the two samples, the Fire Clay material typically produced a higher overall concentration TREE extracted, which may be due to the higher overall feed concentration (these results will be confirmed once the total assays are complete.  With respect to the distribution between the faces, the Fire Clay tended to have a much higher content of the difficult to extract species (i.e. acid soluble) when compared to the West Kentucky No. 13. This trend was primarily driven by Ce, which was a major constituent of the acid soluble component in all cases. By contrast, the extraction of the easier to extract species (i.e. ion exchangeable, carbonate, and metal oxide) was primarily driven by yttrium. Overall, the same element-by-element trends were observed in all samples, with the notable exception being the West KY No. 13, 2.2 sink material.

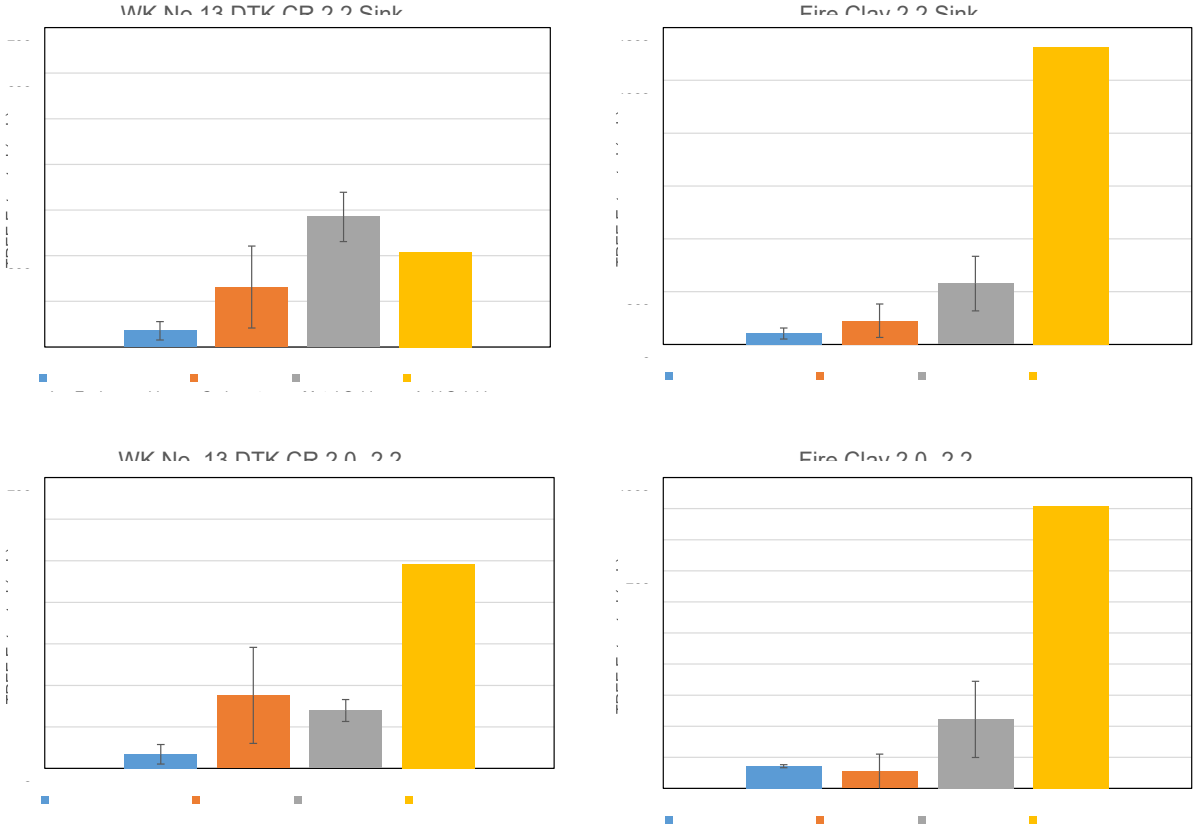


Figure 3.11.11 TREE sequential extraction data for higher density fractions from the West KY No. 13 and Fire Clay coarse refuse samples.

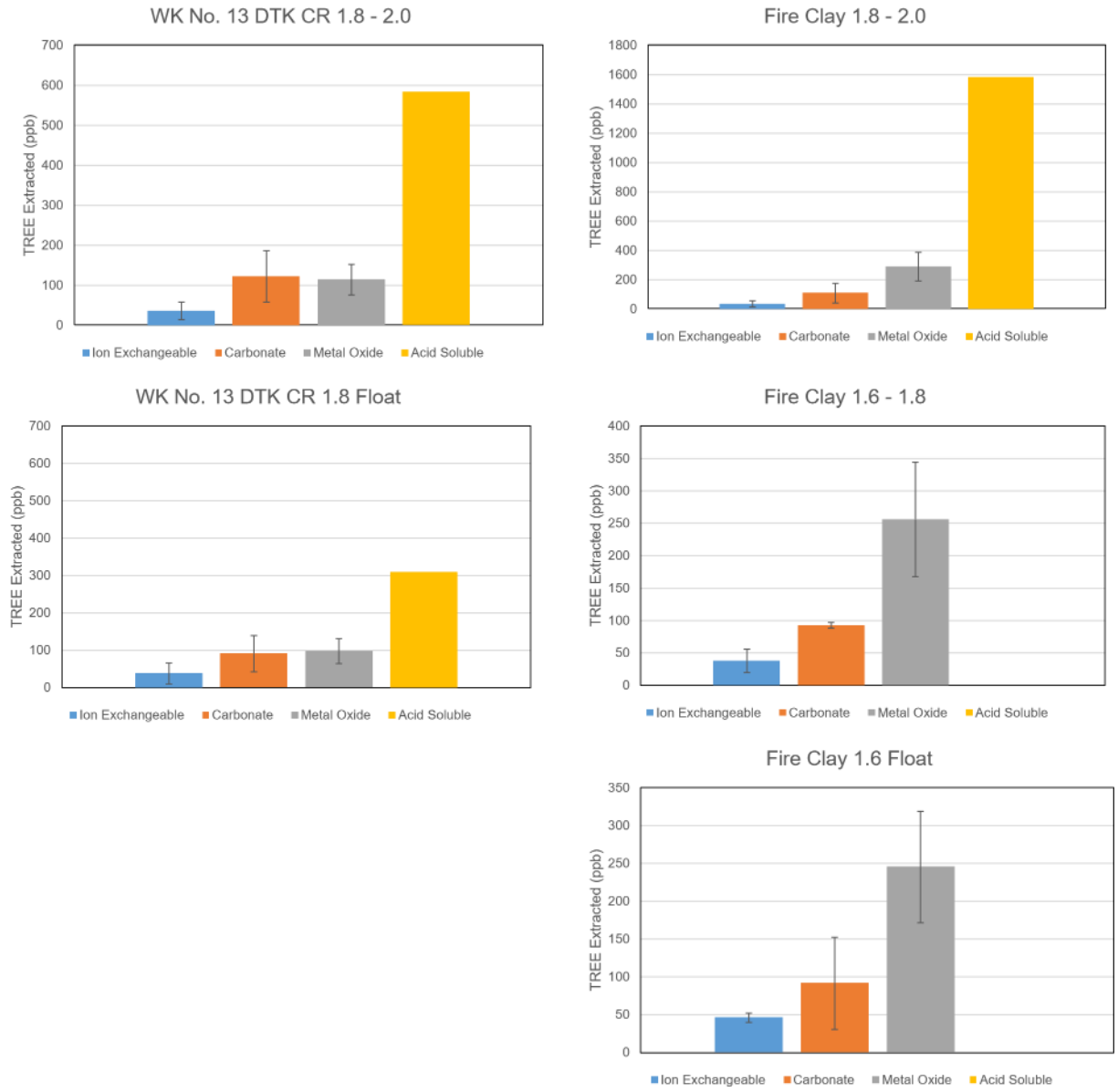



Figure 3.11.12 TREE sequential extraction data for lower density fractions from the West KY No. 13 and Fire Clay coarse refuse samples.

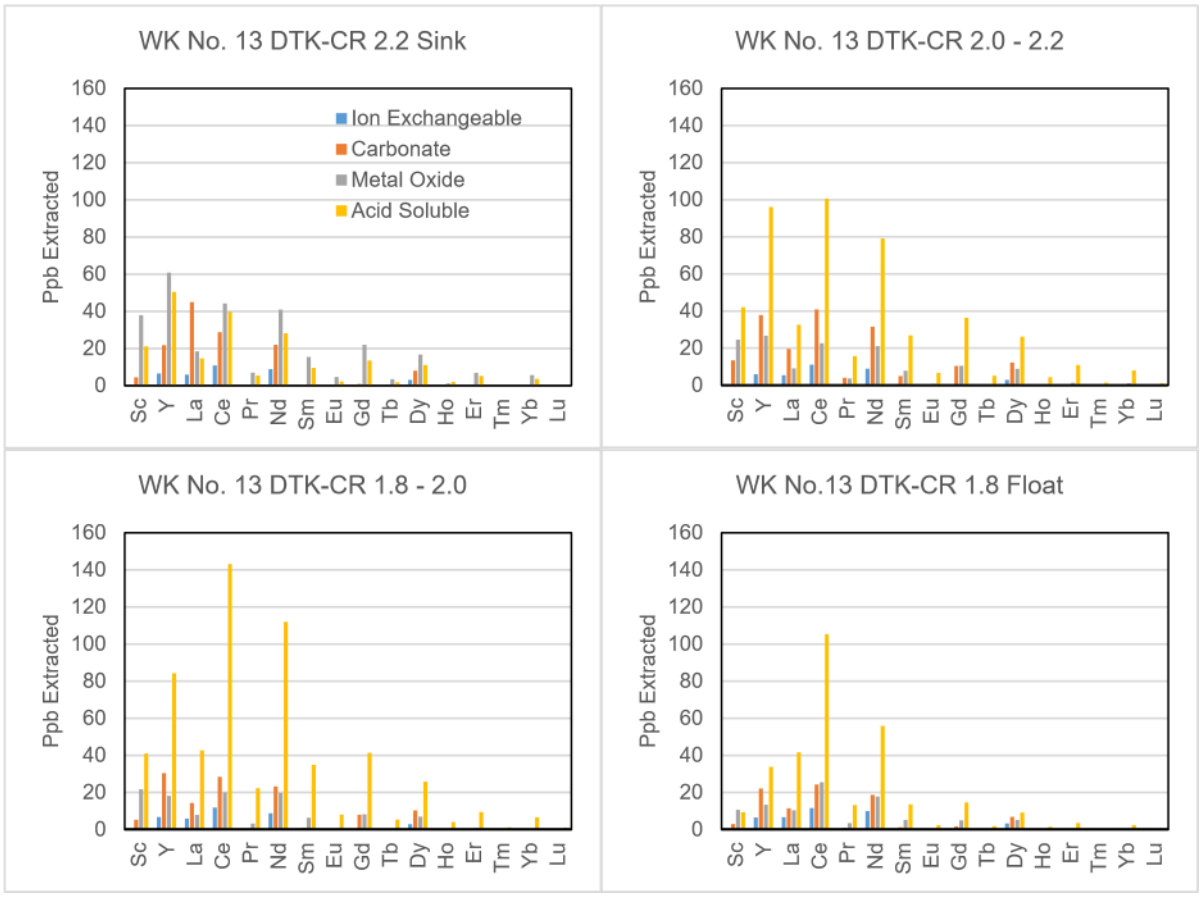



Figure 3.11.13 Element-by-element sequential extraction data for the West KY No. 13 samples.

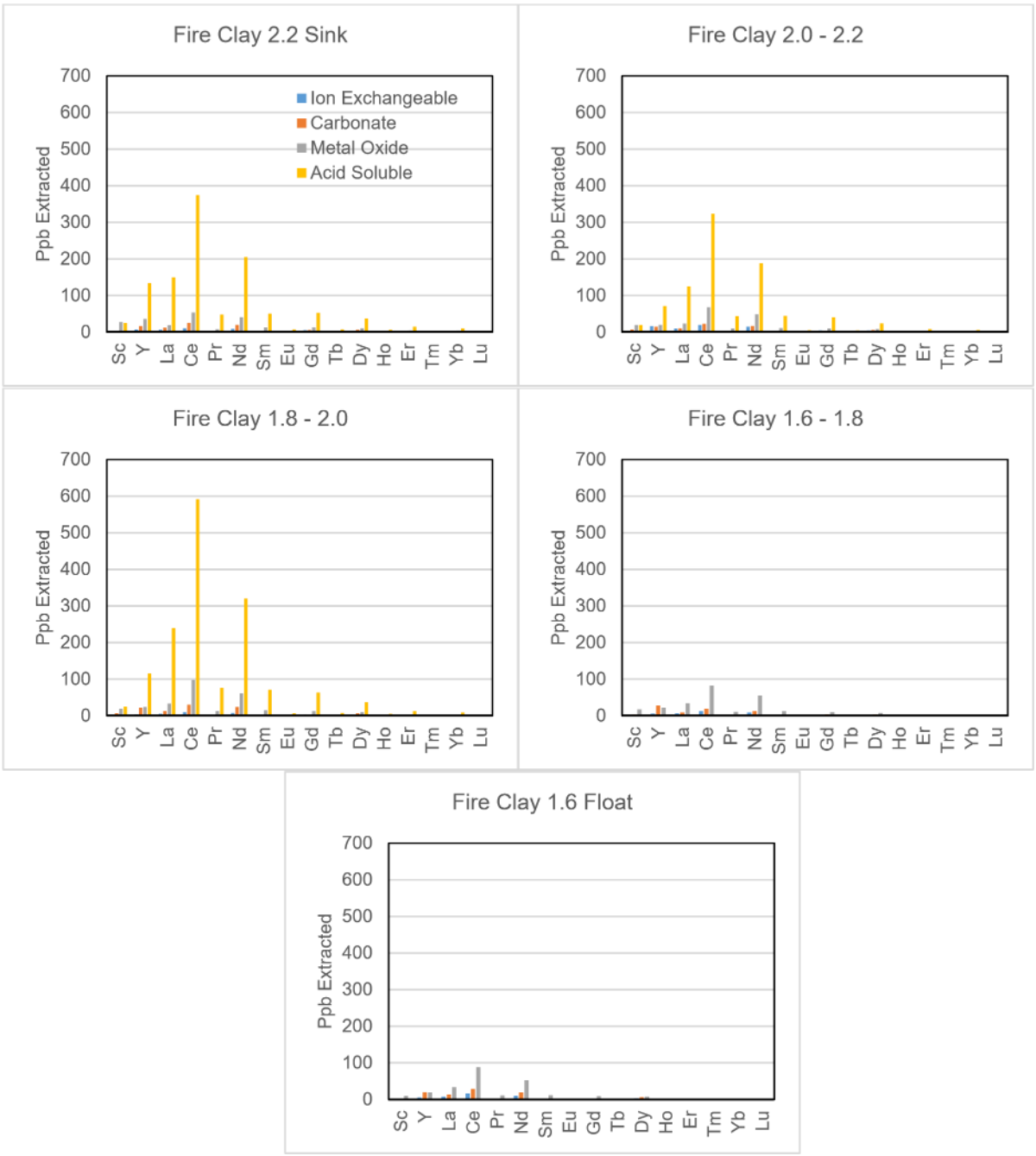



Figure 3.11.14 Element-by-element sequential extraction data for the West KY No. 13 samples.

## 3.12 PILOT PLANT OPERATION AND CONTINUOUS IMPROVEMENT (TASKS 15 AND 17)

Continuous pilot plant tests were performed on primary (Task 15) and secondary sources (Task 17) to fully evaluate the performance of all primary unit operations, particularly focusing on the new operations introduced in this project. Key performance indicators (KPIs) were developed for each unit operation including element-by-element recovery values, REE and critical material purity in output streams, unit capacity, chemical and power consumption, and other consumable costs.

As part of the pilot scale demonstration of rare earth production, eleven pilot scale tests were conducted during the project duration. Results obtained from each test were analyzed and the findings were applied to the next pilot scale experiment with an aim to maximize the recoveries and grades of products. The testing was conducted using three distinct feedstocks classified as primary, secondary and heap leach PLS. The primary feedstock was Baker (West Kentucky No. 9) seam coarse refuse material and the secondary feedstock was Shakespeare lignite material. Additionally, heap leach PLS generated from the leaching of a 2000-ton Baker seam coarse refuse heap was also processed in the facility for the production of rare earth oxide and metal sulfide. Five different circuit arrangements across eleven pilot scale experiments were tested in this project. The results and learnings from each circuit are presented in sequential order.

### *3.12.1 Process Development*

The concentrations of the rare earth element and other critical metals such as cobalt, nickel, manganese and zinc in the primary and secondary feedstocks are significantly lower than the contaminants. Because the recovery of REEs is also lower than other elements due to their association with difficult-to-leach minerals, the PLS generated through conventional leaching or heap leaching is dilute with respect to REEs. As such, contrary to the typical processing of leachate generated from primary REE-bearing minerals, the solution cannot be directly processed for selective rare earth element precipitation. Since the solution volume generated through leaching is thousands of gallons, the chemical cost associated with such a process will be extremely high and no selectivity for REE precipitation is anticipated. Therefore, the dilute PLS is typically processed through a series of selective precipitation stages to generate a concentrated feedstock for further processing. The RE enriched cake generated through this approach has a significantly lower mass and can produce a concentrated PLS upon redissolution which can be subsequently processed to recovery rare earth elements and other critical metals.

To identify the optimum pH points for different precipitation stages, bench-scale precipitation experiments were performed on the PLS (Figure 3.12.1). The results indicated that the Fe precipitation from the solution started at pH 2.5 and accelerated exponentially until pH 3.5, where approximately 92% of the iron was removed from the solution. Therefore, pH range of 3.2-3.5 could be effectively used to remove iron from the PLS rich in iron. It should be noted here that the iron precipitation stage was not required for the Shakespeare material because of relatively lower iron contamination levels. The aluminum precipitation started at pH 3.5 and increased quickly to 84% at pH 4.5. It is interesting to note nearly identical precipitation behavior of aluminum and scandium. Because TREE precipitation also started at pH 4.5., this set point was selected to minimize the REE recovery loss to the aluminum cake. Furthermore, the PLS was sent directly to the solvent extraction circuit before any staged precipitation step to avoid loss of scandium with aluminum. Rare earth precipitation was complete at pH 6.5 and other critical metals also started precipitating at this pH point. Finally, it was found that nearly 100% removal of the critical metals could be accomplished at pH 9.0.

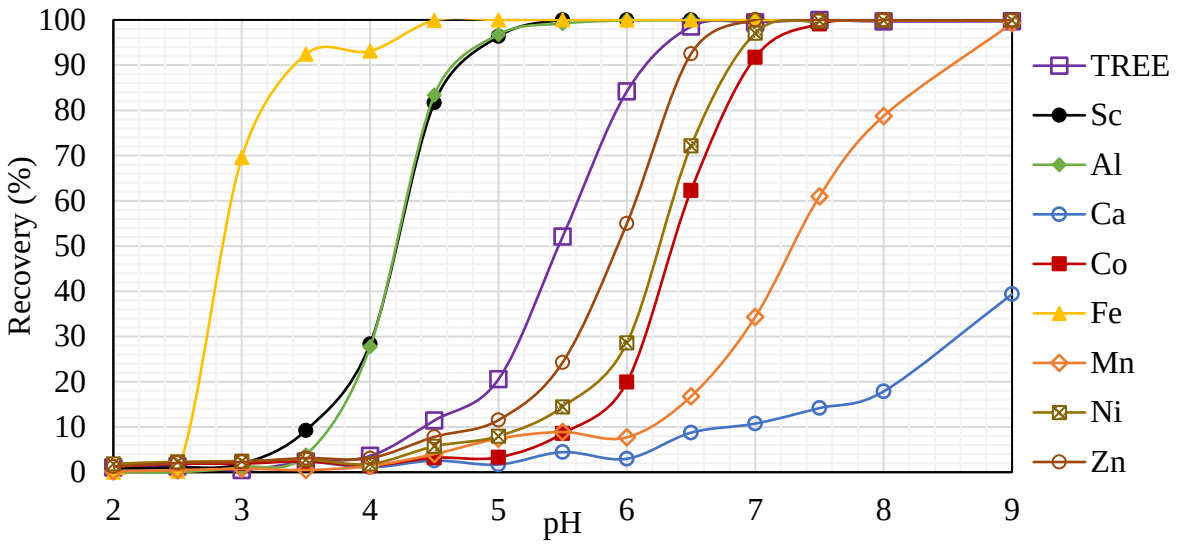



Figure 3.12.1 Precipitation behavior of different elements in the PLS using 4M NaOH as a pH modifier.

*3.12.2 Circuit Arrangment-1*

**Bench Scale Experiment:** Shakespeare (SHXP) material had low levels of contaminants, therefore, based on the results shown in Figure 3.12.2, it was postulated that the aluminum raffinate could be directly processed with oxalic acid precipitation. The benefits of this reduction in stages were a lower requirement of tanks and filters as well as less manual labor for operators because fewer filters needed to be scraped and chemical costs were reduced because no sodium hydroxide was used for precipitation. To determine if this system could be implemented in the plant, this circuit was tested first in the lab.

As previously established from bench scale experiments, leaching was performed with 0.5 M sulfuric acid for 2 hours at room temperature. Following the leaching, two aluminum precipitation tests were performed at pH 4.2 and 4.5 to determine the pH setpoints for the plant stage. It was determined that pH 4.5 was the ideal setpoint for the pilot plant due to the higher removal of contaminants without compromising rare earth element content (Figure 3.12.2). The raffinate obtained from the aluminum precipitation stages at pH 4.2 and 4.5 was used to determine the oxalic acid dosage to ensure efficient REE precipitation. Oxalic tests were performed with doses of 2 ml, 4 ml, and 6 ml of oxalic acid per 50 mL of feed. Results showed that with a lower concentration of Al in the solution, the precipitation efficiency of RE-oxalate increased significantly due to the availability of oxalic acid in the solution (Figure 3.12.3). The bench scale data showed promising results, therefore, the SHXP-C2 test was carried out using leaching, solvent extraction, aluminum precipitation, oxalate precipitation and sulfide precipitation, as shown in Figure 3.12.4.

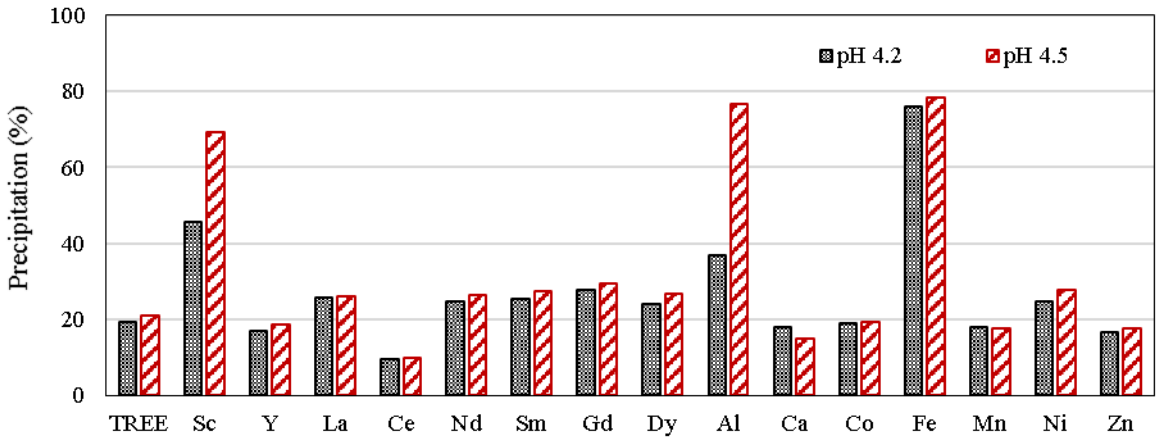



Figure 3.12.2. Comparison of Al removal at pH 4.2 and pH 4.5.

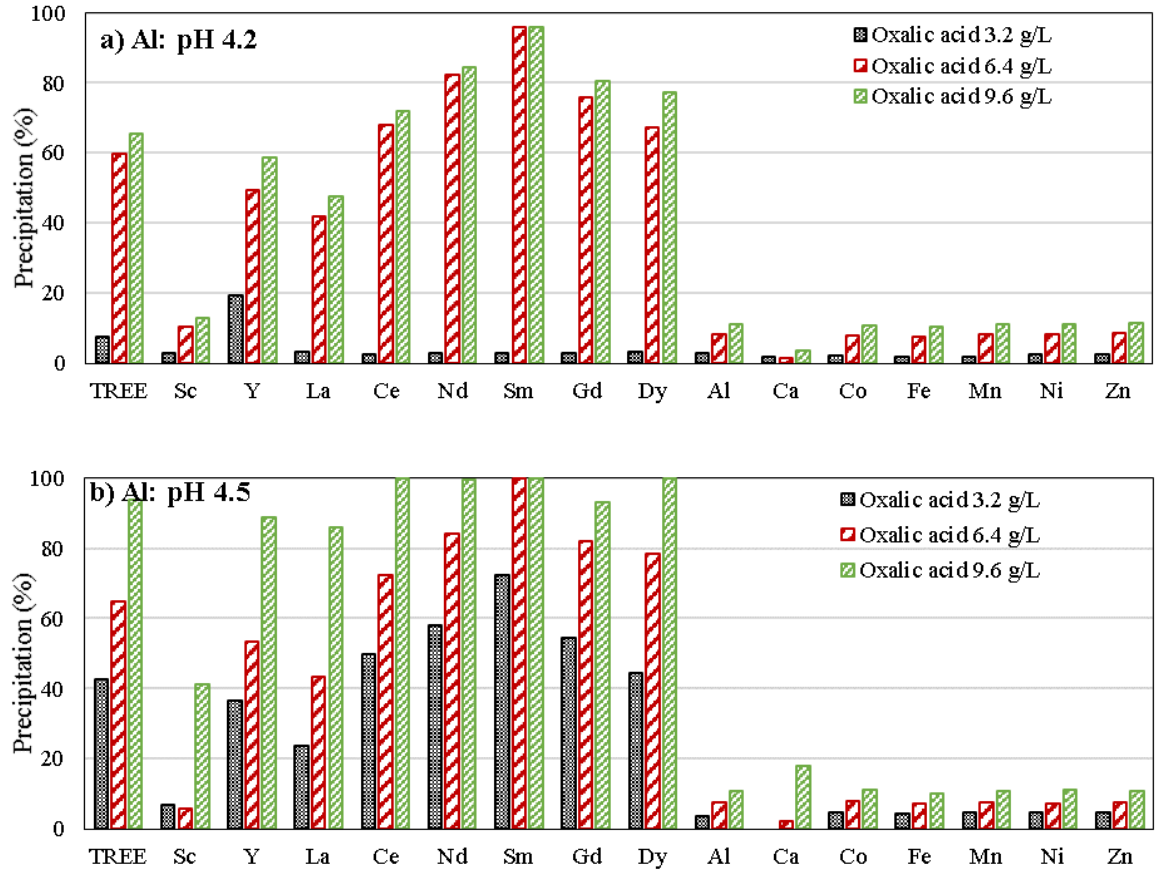



Figure 3.12.3. Precipitation efficiency of RE-oxalate at various dosage of oxalic acid after Al precipitation at (a) pH 4.2 and (b) pH 4.5.

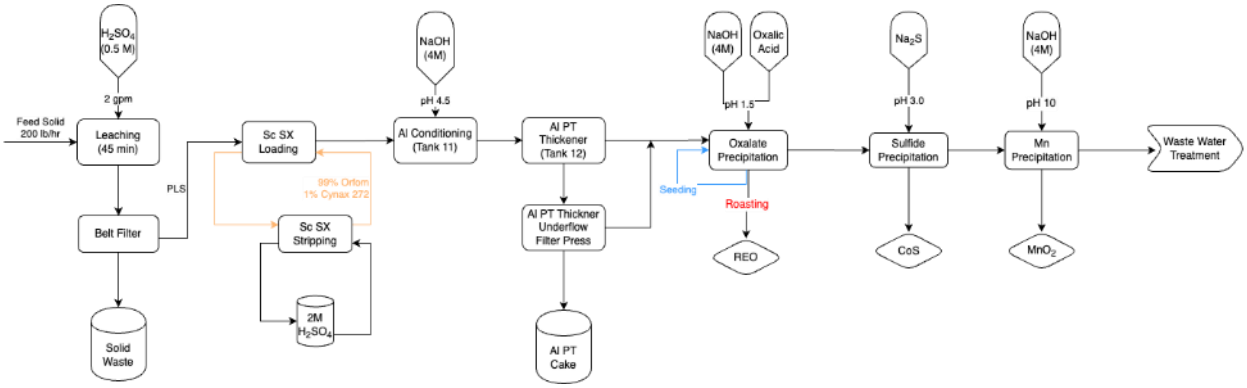



Figure 3.12.4. Circuit arrangment 1 for the pilot scale experiment involving leaching and several precipitation stages for REE and CM recovery.

**Pilot Scale Experiment:** The test started with leaching at ambient temperature and a 20% S/L (%w/v) ratio. Shakespeare material was fed in the 90-gallon mixing tank at 200 lb/hr using a screw feeder and 0.5 M sulfuric acid solution was added at 2gpm. Due to the changing densities among primary and seconday feedstocks, the screw feeder was calibrated prior to the experiment to ensure a consistent and accurate feeding rate. The solids and liquid flow rates were designed to provide approximately 45 minutes of residence time in the mixing tank. REE leaching recovery was approximately 60%, whereas Co, Mn, Ni, and Zn recoveries were 71%, 67%, 64% and 67%, respectively (Figure 3.12.5). Higher recoveries of critical metals compared to REEs suggested that critical metals were in the form more amenable to acid leaching in the current conditions. The recovery of contaminants such as Al, Ca, and Fe was 20%, 27% and 37%, respectively. Since there was significantly higher calcium in the feedstock compared to the Fe and Al, even a small fraction of calcium recovery would translate into a considerably high solution concentration. A relatively lower recovery of aluminum was likely due to its primary association with clays such as kaolinite and illite, which are resistant to chemical attack without prior treatment. The leaching slurry was dewatered using a belt filter and the PLS was further processed in the solvent extraction stage to recover scandium. In solvent extraction stage, 1% (v/v) Cyanex-272 was used as an extractant and 99% (v/v) Orform as a diluent. It was found that the Sc recovery was only 15 percent.

The PLS was further treated in the aluminum conditioning tank to remove aluminum and iron from the solution. As determined by the bench-scale experiment presented previously, the solution pH was increased to 4.5 using 4M NaOH. The mixing tank contained pH and Eh probes connected to the PLC through RSLogix software. The PLC program was designed to control the pH of the mixing tank by varying the peristaltic pump speed, which changed the base addition in the tank. The solution pH data collected over the test duration indicated a fairly good control of the solution pH (Figure 3.12.6). The residence in the aluminum conditioning tank was also 45 minutes, based on the designed flow rates. The slurry from this stage was mixed with a 0.1% flocculant solution and discharged into a 400-gallon thickener. The addition of a flocculant was necessary to increase the settling rate of the precipitates. The underflow from the thickener was filtered using a filter press operated at 7000-psi external pressure. Filtrate from the dewatering stage was combined with the thickener overflow and used for oxalic acid precipitation. The aluminum precipitation stage recovery for Al and Fe was 70% and 92%, respectively, indicating high contaminant removal efficiency. Additionally, there was a minimal recovery of the rare earth elements and other critical metals which decreased the loss of these elements to the aluminum cake.

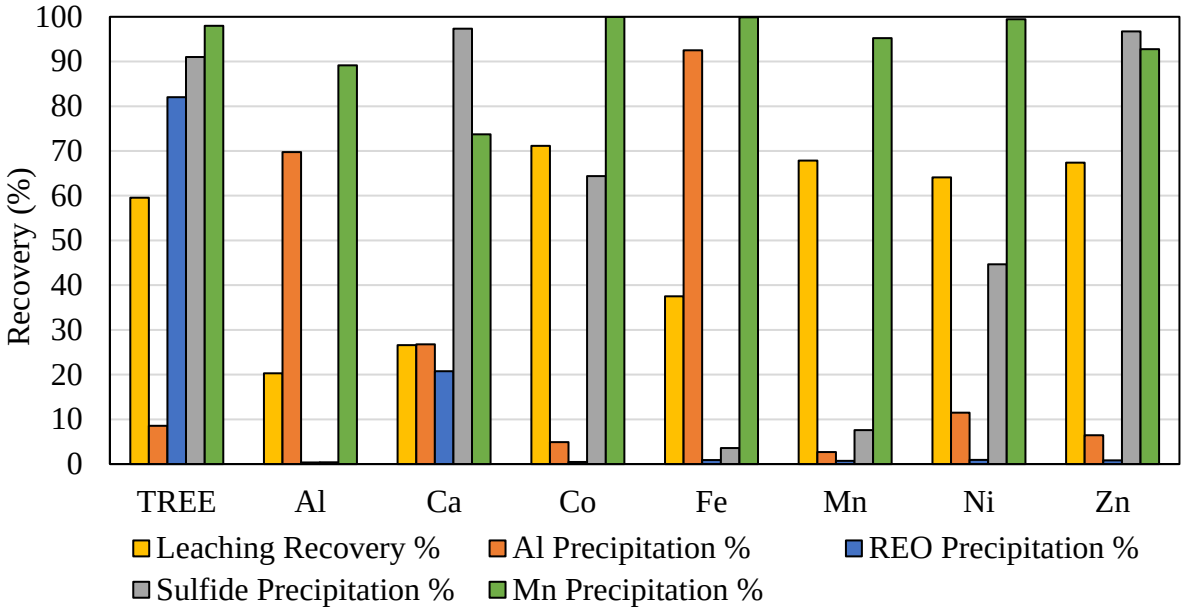



Figure 3.12.5 Recovery of various elements during the leaching and precipitation stages in the test.

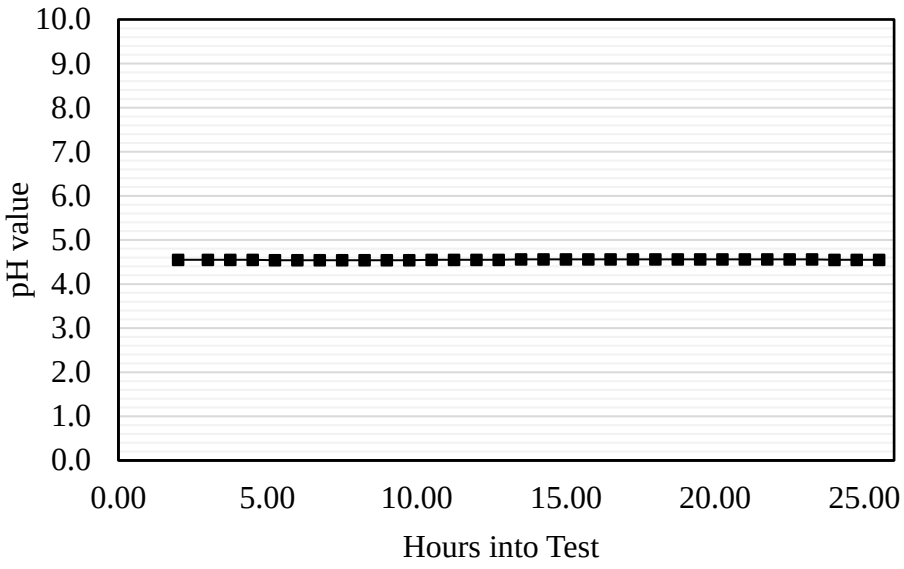



Figure 3.12.6 Solution pH during the aluminum precipitation stage of the test.

The aluminum stage raffinate was directly treated with oxalic acid to selectively precipitate rare earth elements. The optimum dosage for the oxalic acid was selected based on the bench-scale experiments conducted on the aluminum raffinate generated from the pilot-scale test. All the experiments were conducted on the same pH of 1.5 to maximize the purity of the rare earth oxides but different oxalic acid dosages, typically ranging from 2-8ml per 50 ml of the feed solution to maximize the recovery. The chemical dosages from the bench-scale experiments were scaled up to provide an estimated amount of oxalic acid, and 4M NaOH required to treat all the PLS generated during the test. During the pilot-scale test, the base pump speed in the oxalic acid precipitation circuit was also controlled through the PLC program receiving feedback from an online pH probe to achieve a pH setpoint of 1.5. The results showed that the REE recovery was more than 80%, with virtually no recovery of Al, Fe and other critical metals (Figure 3.12.5). However, there was approximately 20% recovery of calcium. As the solution had a high content, 20 percent precipitation efficiency would result in considerable

contamination of the production. A more detailed discussion on this is provided later in this report.

Raffinate from the oxalic acid precipitation stage was treated with sodium sulfide at pH 3.0 to produce a concentrated cobalt-nickel and zinc cake. The optimal chemical dosages for this stage were identified in the same manner as the oxalic acid experiments stage. Furthermore, the pilot-scale experiment was also conducted in the same circuit as the oxalic acid stage with a different pH setpoint and precipitating agent. It was found that the TREE precipitation efficiency in the sulfide precipitation stage was more than 90 percent. This was likely a result of the increased availability of oxalate anions at elevated pH levels. Cobalt, Manganese, Nickel and Zinc recoveries were 64%, 4%, 8%, 45%, and 97%, respectively. The precipitation efficiencies for cobalt and nickel were significantly lower than zinc, indicating that Zn preferentially consumed sodium sulfide. A significant difference in the precipitation efficiency was also likely a result of higher contaminant content relative to the critical metals, which may have impacted their recovery. Surprisingly, calcium recovery in this stage was more than 95 percent, which was likely due to its precipitation as calcium oxalate at elevated pH levels. The metal sulfide precipitation stage filtrate was used for Manganese precipitation at pH 9.0 using a 4M NaOH solution. The results revealed that the recoveries of REEs, Al, Co, Mn, Ni, Zn and Fe were above 90 percent, which was an anticipated outcome. The solution from this stage was considered wastewater.

As mentioned previously, leaching only recovered about 60% of REEs, and most of the contaminant content was rejected to the solid residue cake (Figure 3.12.7). The overall recovery data demonstrated that the aluminum precipitation stage removed most of the leached Fe and Al from the solution, with less than 10 percent content going to the downstream processes. The RE-Oxide precipitation stage recovered approximately 85 percent of the REEs in the aluminum precipitation stage. As discussed previously, the hypothesis for the circuit arrangement presented in Figure 3.12.4 was that the direct treatment of aluminum precipitation stage raffinate for REE precipitation would provide benefits with respect to CAPEX, OPEX and assist in maximizing the REE recovery by minimizing the REE losses in different stages. It was found that the REE recovery attained from this pilot scale arrangement was one of the highest obtained in this project due to lack of losses in any of the typical precipitation and redissolution stages. However, while the recovery was high, the purity of the RE oxide was severely impacted due to the precipitation of calcium. The results shown in Figure 3.12.8 indicated that calcium oxide content accounted for more than 70 percent content in the RE-oxide cake. Aside from decreasing the purity, high calcium content in the solution increases the oxalic acid dosage required to ensure high precipitation of rare earth elements, thereby increasing the cost of the precipitation process. Similarly, the remaining calcium from the oxalic acid precipitation stage precipitated during the metal sulfide precipitation stage due to the increased availability of oxalate anions in the solution. The calcium oxide content in the final cake was more than 80 percent (Figure 3.12.8). It is likely that elevated calcium content resulted in the lower overall recovery of critical metals in the metal sulfide cake resulting in a high loss of Ni and Co to the next stage (Figure 3.12.7). The manganese oxide recovered the remaining the critical metals, however, increasing the solution pH to 9.0 precipitated the aluminum and iron in the solution resulting in reduced purity of the manganese cake. This precipitation behavior likely followed the results presented discussed previously.

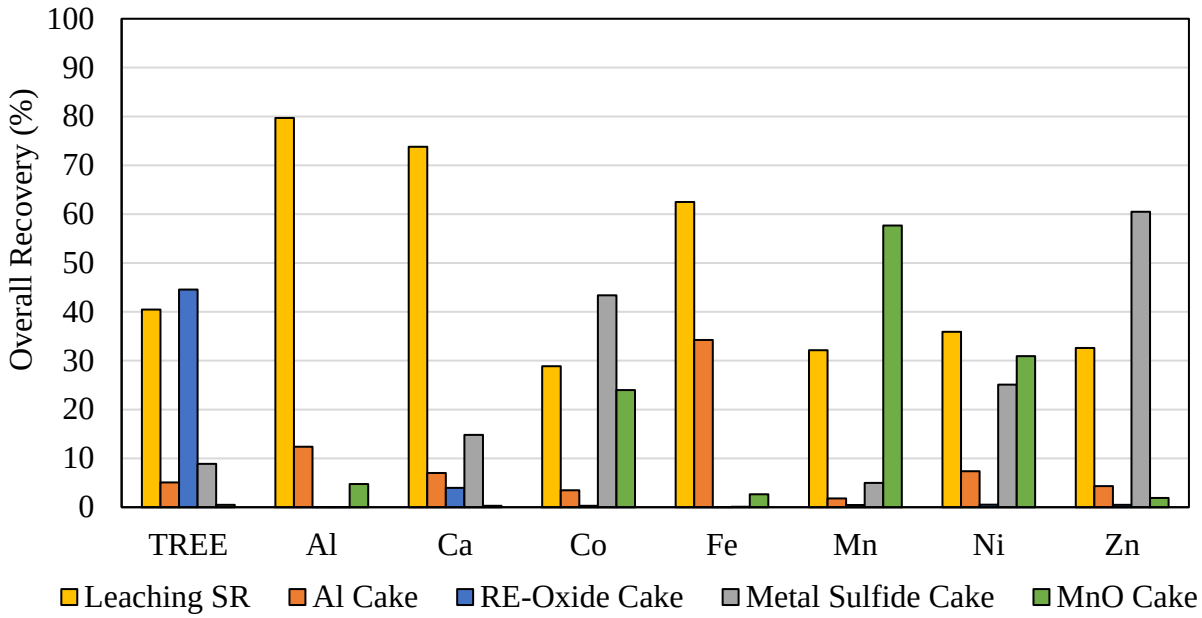



Figure 3.12.7 Overall distribution of various elements of interest in different solids generated during the test.

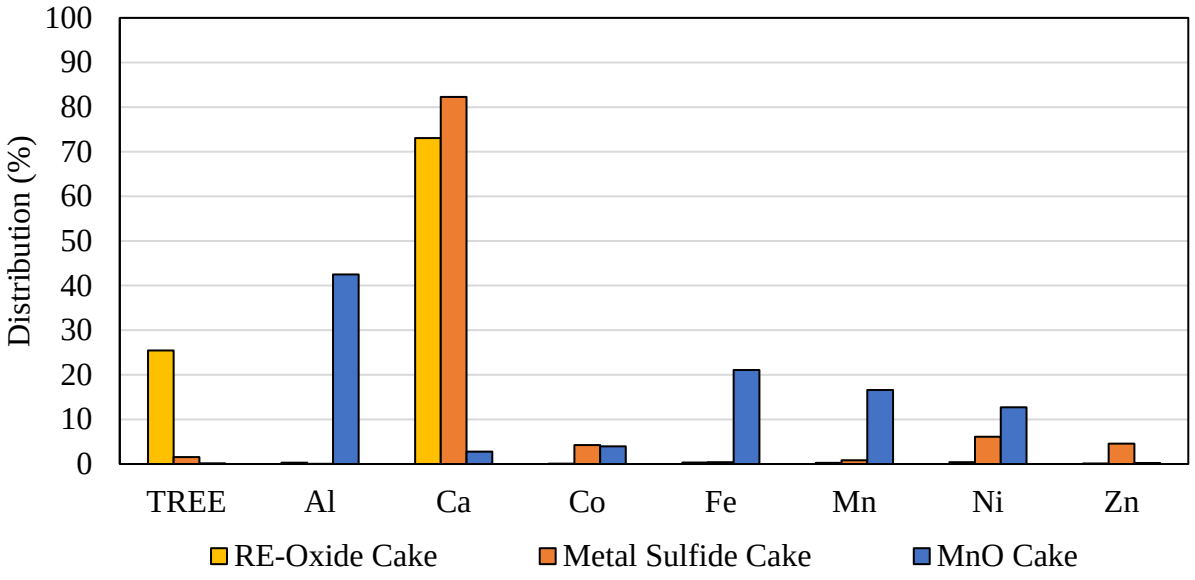



Figure 3.12.8 Distribution of elements in the final rare earth oxide, metal sulfide, and manganese oxide cakes generated in the test.

*1.1.20 Circuit Arrangement-2*

The first pilot experiment revealed that using the process flow presented in Figure 3.12.4 impacted the final product purity due to the precipitation of calcium oxalate. As such, a new circuit flow was designed with an aim to maximize both the purity and recovery of the REEs and other critical metals in the final products (Figure 3.12.9). The first half of the process was identical to the previous circuit arrangement where the material was leached and the scandium was removed using solvent extraction followed by Fe/Al precipitation at 3.2/4.5. As most calcium is rejected at pH 9.0 (Figure 3.12.1), the new process will generate enriched REE and CM cakes by treating the PLS at pH 6.5 and 9.0, respectively. The concentrated cakes will be

leached using HCl at pH 1.5 to produce an enriched REE/CM PLS, which can subsequently be processed using oxalic acid and sodium sulfide precipitation to generate RE-Oxalate and Metal sulfide cakes, respectively.

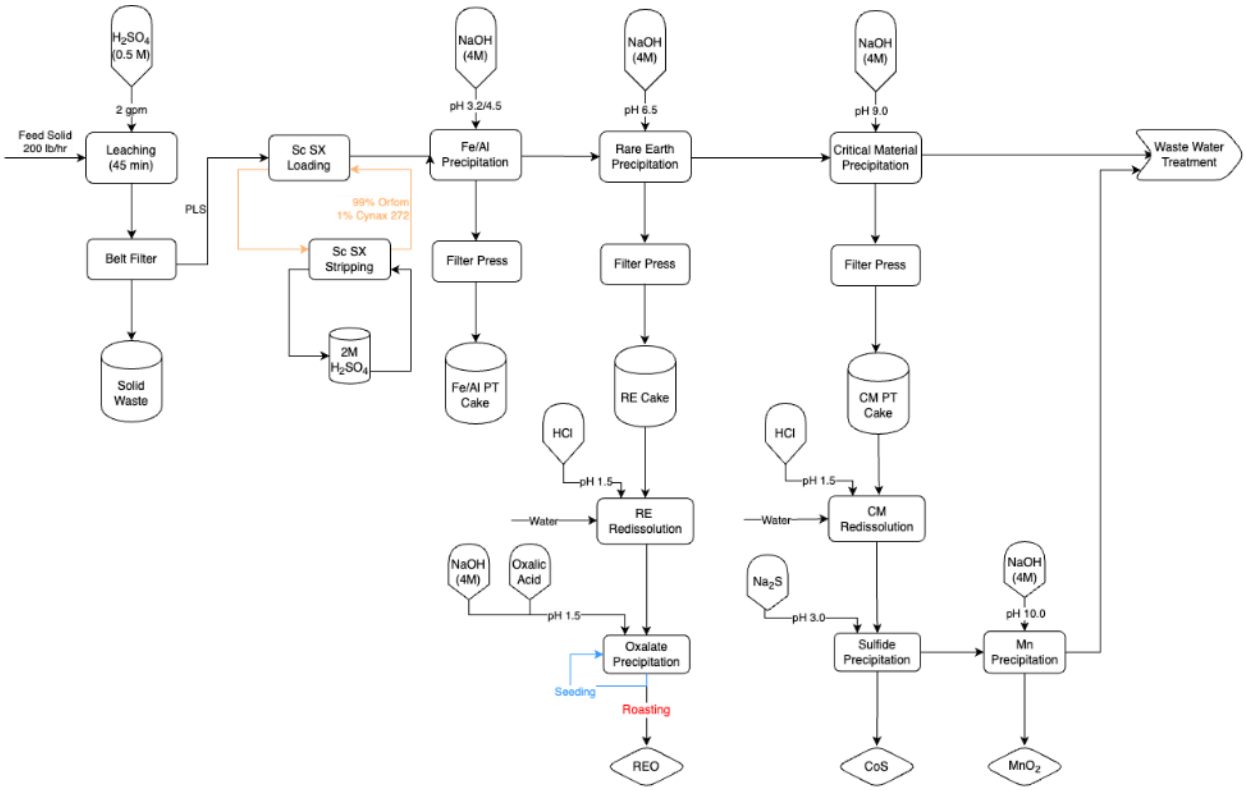



Figure 3.12.9 Modified circuit arrangement for the pilot scale experiment to produce REEs and CM cakes 2.

**<u>Pilot Experiment</u>:**

The modified process flowsheet presented in Figure 3.12.9 was tested on a PLS generated from the leaching of a 2000-ton Baker seam coarse refuse. However, there were small changes to the circuit, particular to the treatment of PLS generated from the leaching of Baker seam coarse refuse. It is evident from the PLS content shown in Table 3.12.1 that there were significantly higher concentrations of contaminants than REEs and other critical metals. Since the heap leach PLS was rich in iron, especially ferric iron, there was a need for a separate iron precipitation stage to avoid the REE loss to the aluminum cake due to their tendency to adsorb on the surface of iron precipitates (Li et al., 2022). Additionally, as the ferric iron is preferentially loaded on the organic phase in solvent extraction, the scandium removal stage was bypassed when treating the heap leach PLS. Ferrous ions in the solution are also problematic due to an overlap in the precipitation pH of REEs and $Fe^{+2}$ ions. Therefore, an iron oxidation step is essential to improve the REE precipitation selectivity. Flowsheet for the pilot scale test is shown in Figure 3.12.10.

Table 3.12.1 Concentration of various elements of interest in the heap leach PLS used for the pilot scale experiment.

| TREE | Al | Ca | Co | Fe | Mn | Ni | Zn |
|---|---|---|---|---|---|---|---|
| 32.91 | 2703.09 | 402.09 | 17.37 | 4289.32 | 188.94 | 60.41 | 53.17 |

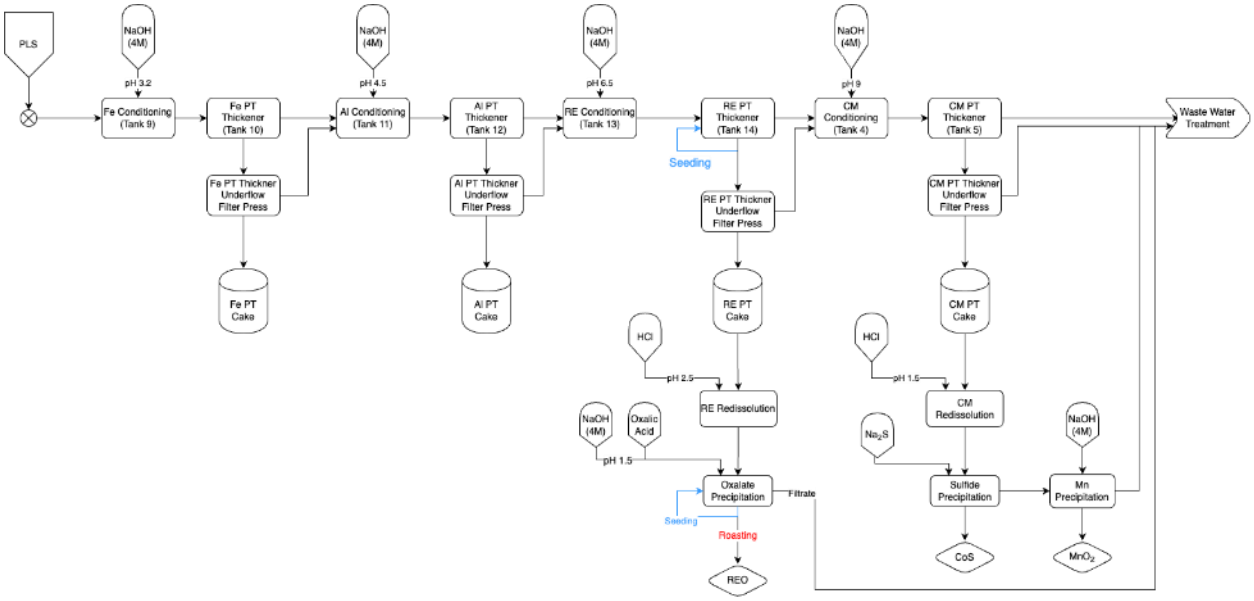



Figure 3.12.10 Circuit schematics for the processing of PLS following circuit arrangement-2.

For the pilot scale experiment, heap leach PLS was directly added to the iron conditioning tank at 2 gpm and the solution pH was raised to 3.2 by adding the 4M NaOH. The pH was measured using an online probe connected to the PLC system controlling the peristaltic pump speed attached to the base tank, providing control over the solution acidity. The pH control system was identical in all the precipitation stages during the test. Slurry from the conditioning tank was mixed with 0.1% flocculant solution at predetermined dosages using an inline mixer and discharged into a 400-gallon thickener. Subsequently, underflow from the thickener was dewatered using a plate filter press operated at 7000-psi external pressure. For each filter batch, approximately 15 lbs. of perlite, used as a filter aid, was mixed with 35 gallons of water and pushed through the filter in a close loop until the discharge was clear. Following the perlite addition, the thickener underflow was pumped through the filter and discharge was monitored continuously. The filter operation was stopped upon observation of particles in the solution. After each filling cycle, compressed air was passed through the filter plates for 30 minutes to remove any retained moisture from the cakes. The filtration process was the same for all precipitation stages shown in Figure 3.12.11.

Results indicated that the iron precipitation stage was highly efficient in removing the iron contamination from the PLS (Figure 3.12.11). The Fe and Al recovery was 89 and 19 percent, respectively. Additionally, there was a minuscule loss of REEs and other critical metals such as Co, Mn, Ni and Zn to the iron precipitation cake, likely due to the entrainment. The pH data from the iron precipitation stage over the entire test run is shown in Figure 3.12.12. The overflow from the iron thickener was combined with the raffinate from the iron filter and subsequently used for the aluminum precipitation stage, which was performed in the same manner as the iron precipitation but at a different pH operating point, i.e., 4.5. The Al and Fe recovery in the aluminum precipitation stage was 85 and 90 percent, respectively. However, the REE recovery in the Al removal stage was 25%, along with a loss of Co, Ni, Mn, and Zn, which may be a result of pH fluctuations during the experiment (Figure 3.12.12). The filtrate from the Al removal stage was further processed at pH 6.5 for REE precipitation. The results demonstrated that more than 90 percent of the REEs in the solution were insolubilized in the RE precipitation stage. Interestingly, the recovery of Co, Mn, Ni and Zn was more than 80 percent, resulting in significant losses of critical metals to the RE cake. Aside from REEs and

CM, recovery of Al and Fe was above 90 percent, which was an anticipated outcome considering the precipitation behavior of these elements. Finally, the CM precipitation stage recovered the remaining REEs and CMs from the solution. Significant pH changes during RE and CM precipitation stages may have resulted in some inefficiencies during the precipitation (Figure 3.12.12).

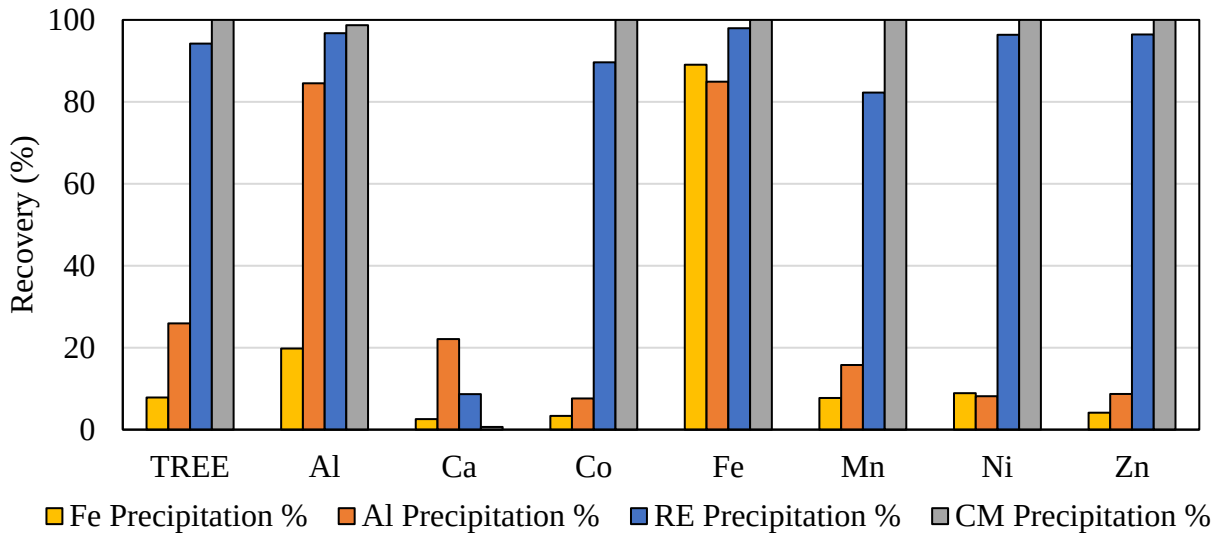



Figure 3.12.11 Percent recovery of various elements of interest across multiple precipitation stages during the pilot scale test.

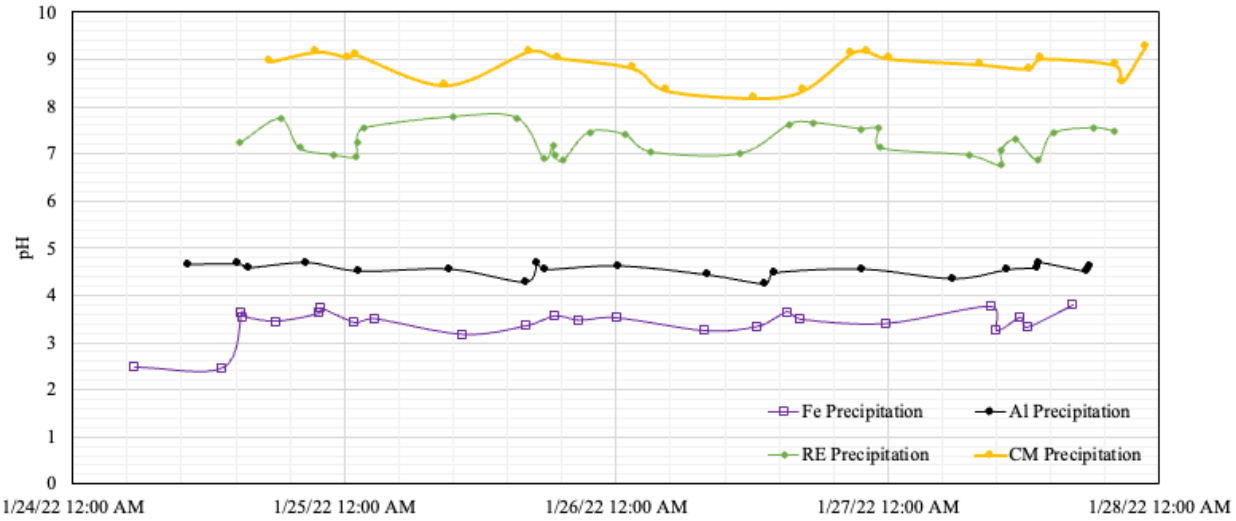



Figure 3.12.12 pH control in different precipitation stages as a function of time during the pilot scale experiment.

The RE and CM precipitation stage aimed to produce enriched cakes that can be subsequently leached to produce a concentrated PLS for oxalic acid and sodium sulfide precipitation. As shown in table below, RE and CM cake had substantially high concentrations of REEs and other critical metals. A relatively lower RE, Co, Ni, and Zn content in the CM cake relative to the RE cake was likely due to their precipitation in the RE precipitation stage. Subsequently, the RE and CM cakes were leached at 10 and 20 % S/L (w/v), respectively, and pH 1.5 using HCl as a pH modifier. The resulting solution from the leaching of these enriched cakes had elevated contents of all elements relative to the initial concentrations shown in Table 3.12.1. However, it should also be mentioned that the RE and CM cakes had only 22% and 19% solids content. Therefore, although relatively high solution concentrations were obtained from the leaching, there was a significant dilution effect on the elemental content.

Table 3.12.2 Elemental content in the RE and CM precipitation cake (dry, whole mass basis, ppm) and corresponding PLS concentration (ppm) from the leaching of the cakes.

| | **Solid Concentrations** | | | | |
|---|---|---|---|---|---|
| | TREE | Co | Mn | Ni | Zn |
| **RE Cake** | 4536.8 | 2977.7 | 25564.6 | 10436.7 | 9621.8 |
| **CM Cake** | 1801.3 | 2565.0 | 41547.9 | 2925.6 | 2636.2 |
| | PLS Concentrations | | | | |
| **RE PLS** | 88.1 | 65.7 | 593.1 | 246.8 | 228.0 |
| **CM PLS** | 40.6 | 58.6 | 954.9 | 65.8 | 50.4 |

Bench scale experiments were conducted to determine the optimum oxalic acid and sodium sulfide dosages to ensure high precipitation efficiencies for REEs and CMs. The identified concentrations were scaled up to the total PLS content for the pilot scale experiment. Similar to the previous stages, pH of the reaction was controlled at 1.5 and 3.0 through a pH probe connected to the PLC system for REE and CM precipitation, respectively. It was determined that the REE precipitation efficiency was approximately 90 percent, whereas calcium precipitation was only 3.2 percent (Figure 3.12.13). Recoveries of all other elements were less than 0.5 percent, indicating high selectivity of oxalic acid. Similarly, the sulfide precipitation stage recovered approximately all the leached Co, Ni and Zn content whereas the remaining

elemental concentrations was insolubilized at pH 9.0 during Mn precipitation. Significant recovery of contaminants such as Al and Fe in the Mn precipitation stage was likely due to their precipitation at pH above 4.5.

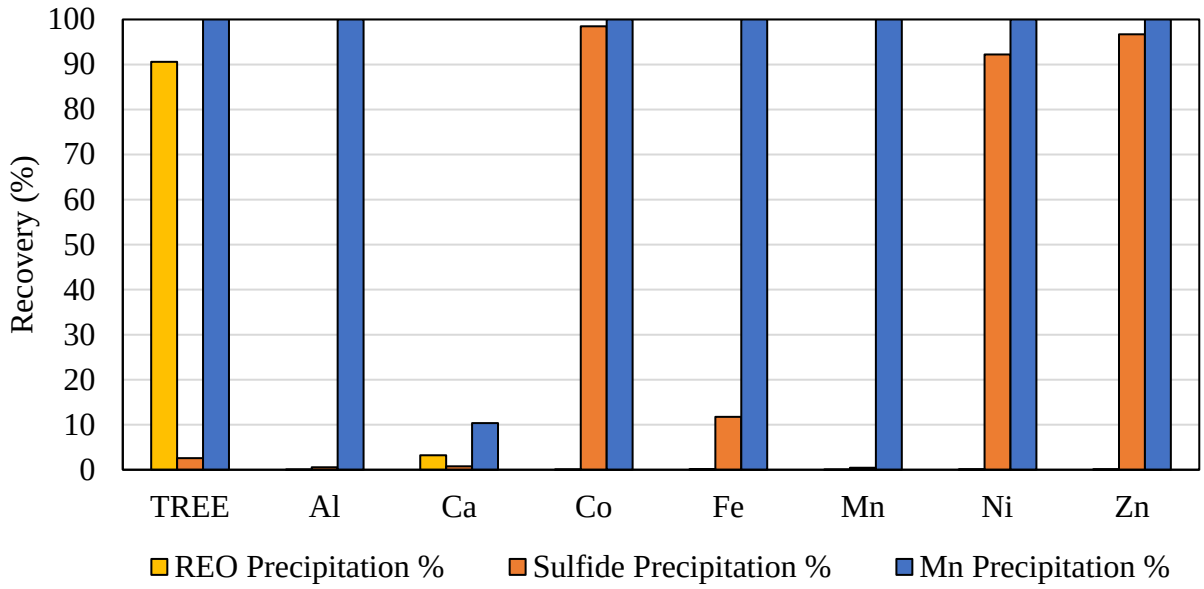



Figure 3.12.13 Percent recovery of various elements of interest in the REE, sulfide and manganese precipitation stages during the pilot scale test.

The overall recovery data presented in Figure 3.12.14 indicated that approximately 87 and 20 percent of the total iron and aluminum, respectively were removed during the iron precipitation stage. Similarly, the aluminum precipitation stage removed 68 percent of the total aluminum and 12 percent of the iron. Overall, aluminum and iron precipitation stages removed more than 85 percent of iron and aluminum, demonstrating high contaminant removal efficiency. It is interesting to note that approximately 24 percent of the REEs were also lost to the aluminum cake which may be due to the pH oscillation in the experiment. The RE precipitation stage recovered 64 percent of the REEs, and 80%, 64%, 81%, and 85% of Co, Mn, Ni, and Zn, respectively. As mentioned previously, this circuit arrangement aimed to maximize the recovery and purity of the final products. However, it is evident that there was a significant loss of critical metals to the rare earth cake (Figure 3.12.14). Therefore, only a portion of the total Co, Mn, Ni and Zn were recovered in the CM Cake. A relatively higher Mn content in the CM cake was due to its high pH precipitation pH compared to other critical metals. As such, it can be concluded that the loss of CMs in the RE cake makes this circuit arrangement ineffective in achieving the goal of maximum recovery. As such, the additional cost of CAPEX and OPEX incurred by different precipitation, subsequent redissolution stages and processing for the production of final products cannot be justified with a significant loss in CM recovery. An alternative approach to limit the CM precipitation may be to perform precipitation pH at 6.0 instead of 6.5. However, this would result in a loss of rare earth elements to the critical metal cakes.

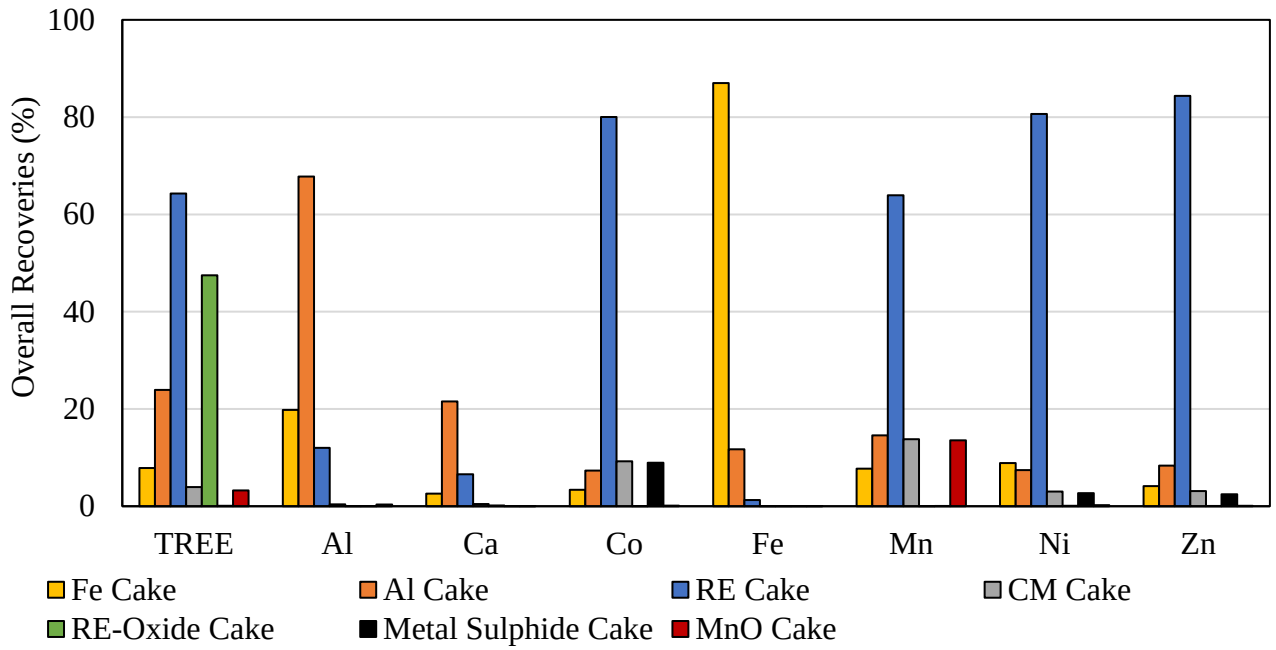



Figure 3.12.14 Overall recoveries of various elements of interest in the different cakes generated in the pilot scale experiment.

The overall RE Oxide recovery was 47 percent with a minimal recovery of any other contaminant (Figure 3.12.14). Therefore, the purity of the RE-oxide precipitate was significantly high (Figure 3.12.15). As the calcium precipitation was minimal at pH 6.5, the oxalic acid precipitation generated a 94 percent pure rare earth oxide product with only 4 percent calcium which precipitated as calcium oxalate. Even though most critical metals were lost to the REE cake, the metal sulfide product was still high purity due to the lack of precipitation of other contaminants. Because calcium in the previous circuit arrangement precipitated as calcium oxalate, bypassing the oxalic acid stage did not cause any precipitation of calcium due to its insolubility at pH 3.0. The metal sulfide product contained 32%, 34% and 27% content of cobalt, Ni and Zn, respectively, accounting for more than 90 percent of the content in the final product. There was some precipitation of iron, which may be due to the formation of iron sulfates during sodium sulfide precipitation. Finally, manganese was insolubilized at pH 9.0 and the resulting cake contained approximately 70% Mn-oxide. The primary contaminant in this precipitation stage was aluminum because of its precipitation above pH 3.5.

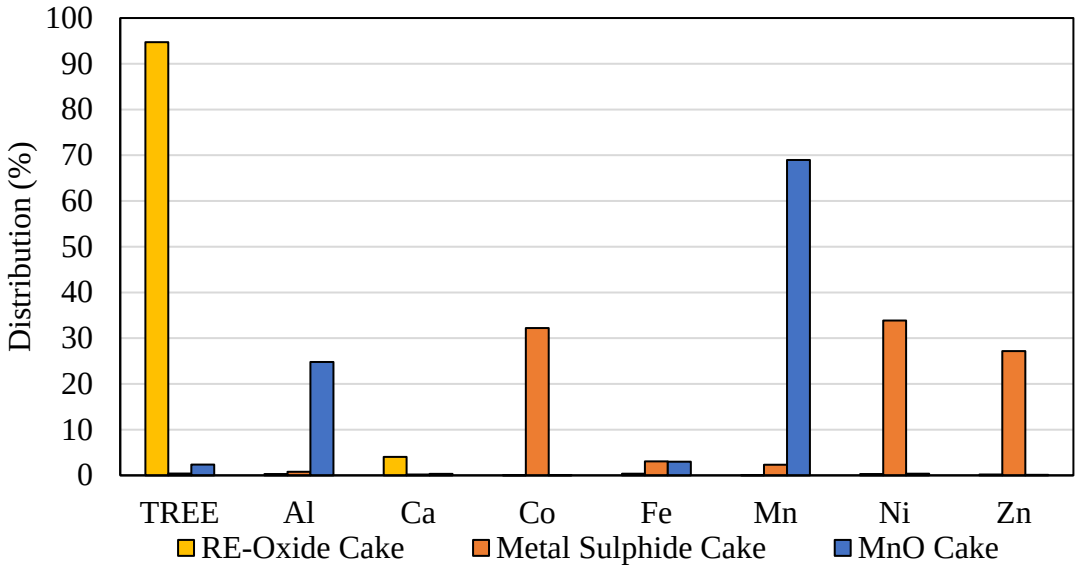



Figure 3.12.15 Elemental distribution in the RE-Oxide, Metal sulfide and Mn cakes generated from the pilot scale treatment of PLS.

### *1.1.21 Circuit Arrangement-3*

The second circuit arrangement presented previously resulted in high purity RE and metal oxide products. However, there was a significant loss of critical metals to the RE cake. As such, modifications were made to the previous circuit arrangement to maximize the CM recovery. Overall, the new circuit flow was similar to the 2$^{nd}$ circuit (Figure 3.12.9), where the REEs are recovered using leaching and then passed through several precipitation stages to remove contaminants and finally recover concentrated REE and CM cakes (Figure 3.12.16). Subsequently, the cakes are leached at pH 1.5 using HCl for about an hour and used for oxalic acid and sodium sulfide precipitation. However, in this arrangement, raffinate from the oxalic acid precipitation was not discarded as wastewater but mixed with the critical metal PLS to ensure a high recovery of critical metals.

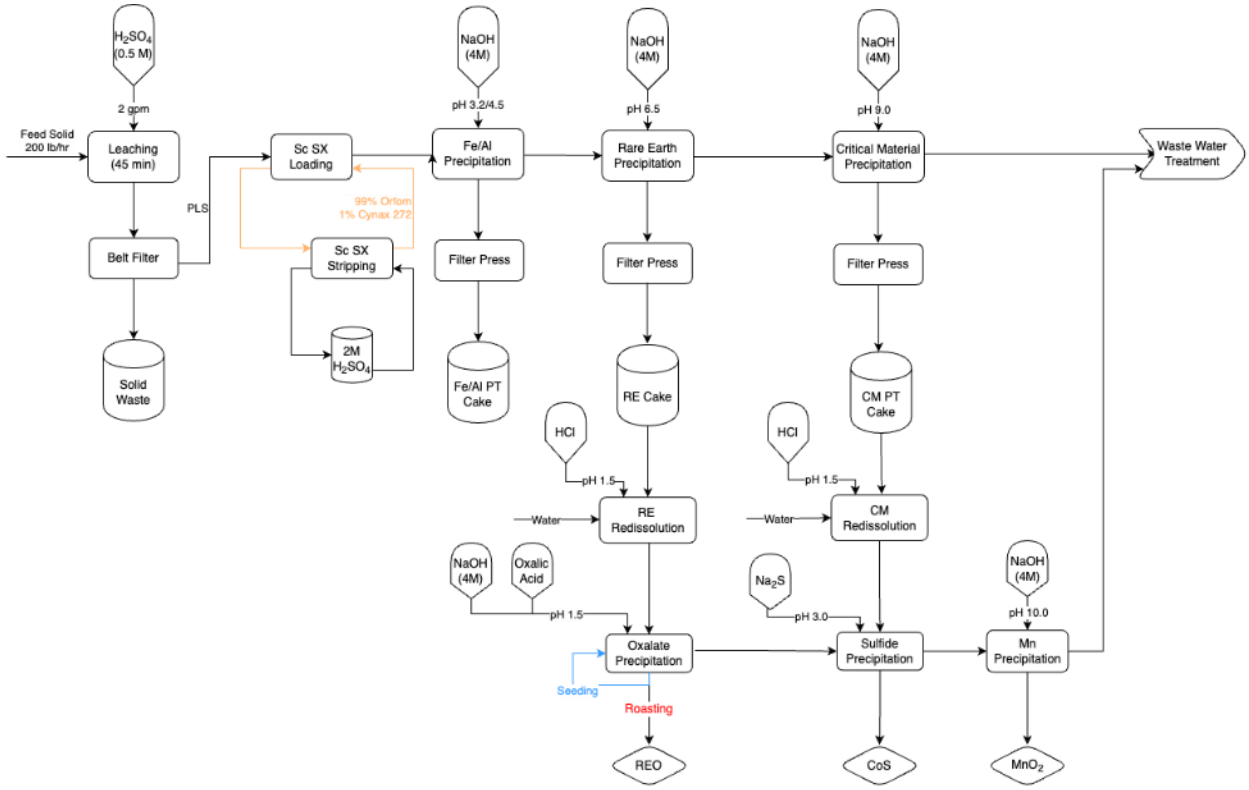



Figure 3.12.16 Modified circuit flow based on the circuit arrangement-2 to increase the CM recovery.

**Pilot Scale Experiment:** Impact of the new circuit arrangement presented in Figure 3.12.16 on the recovery and purity of the final products was tested using Shakespeare material. As mentioned previously in 0, Shakespeare material had significantly lower contaminant concentrations. Therefore, the iron precipitation stage was bypassed and leachate was processed at pH 4.5 to remove iron and aluminum. The process conditions, equipment such as mixing tanks, thickeners and filter presses, and process controls were the same as discussed previously (0, 0). Therefore, details on their operations are not presented in this section.

For this experiment, leaching was performed using a 20% S/L (%w/v) ratio at ambient temperature for 45 minutes. The results presented in Figure 3.12.17 showed that the REE recovery was 57 percent whereas recovery of Co, Mn, Ni and Zn was 66%, 60%, 64%, and 68%, respectively. The recovery of contaminants such as Al, Ca and Fe was 25%, 28%, and 29%, respectively, suggesting higher selectivity towards elements of interest. However, it should be noted that the concentrations of contaminants were orders of magnitude higher than the REEs and CMs. Therefore, even a small percentage recovery will result in elevated concentration in PLS (Table 3.12.3). Leaching performed under the same conditions in the previous test using Shakespeare material (Figure 3.12.5) reported similar recovery values for REEs and CMs. A small deviation in the percentage values was likely due to the variation in the feed material. Overall, the results demonstrate excellent repeatability in the leaching data for pilot-scale experiments. The solvent extraction stage removed 3% scandium and over 11% iron from the pregnant leachate solution.

Table 3.12.3 Elemental concentrations in the PLS generated from the leaching during the pilot scale experiment.

| | TREE | Al | Ca | Co | Fe | Mn | Ni | Zn |
|---|---|---|---|---|---|---|---|---|

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| **PLS** | 47.5 | 393.0 | 979.0 | 29.9 | 629.1 | 39.8 | 81.0 | 25.2 |

The aluminum precipitation stage removed 72 and 98 percent of aluminum and iron, respectively, demonstrating high contaminant removal efficiency (Figure 3.12.17). However, there was a 16 percent loss of REEs along with some other critical metals. Since the pH of the aluminum raffinate was, on average, 4.3 during the test, instead of the 4.5 set point, insolubilization of REEs was likely not the primary reason for the REE loss in this stage. It was determined that the PLS feeding the aluminum precipitation stage contained 701.1 ppm of iron, whereas only 392 ppm of Al. As aluminum can neutralize the surface charge on the iron precipitates, insufficient aluminum ions can cause REEs adsorption on the surface of the iron precipitates. As such, the REE loss was likely a combination of co-precipitation and entrainment during the aluminum precipitation stage. Furthermore, due to the lower operating pH, aluminum ions were carried over to the RE and CM precipitation stages, which would likely result in contamination and increased chemical consumption during the further processing of the solution.

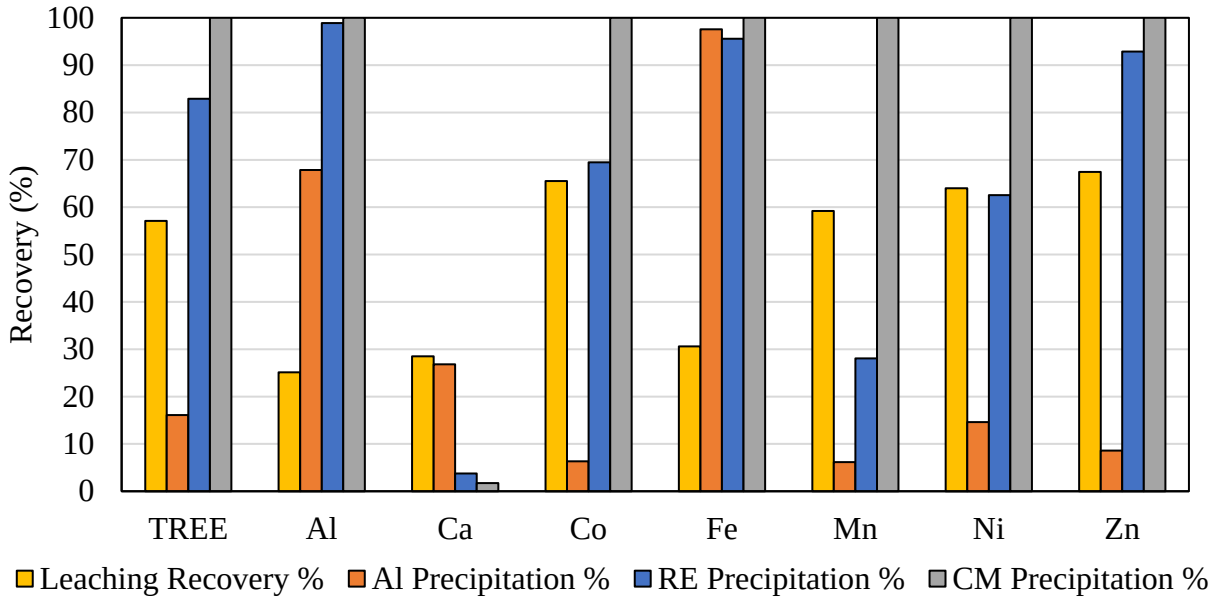



Figure 3.12.17 Elemental recovery in different precipitation stages during the pilot scale experiment.

The RE precipitation stage recovered 83% of the REEs in the feed to the circuit, whereas Co, Ni, Mn and Zn precipitation efficiency was 70%, 28%, 63%, and 93%, respectively (Figure 3.12.17). Relatively lower precipitation efficiency compared to the results presented in the previous circuit arrangement (Figure 3.12.11) was due to the lower operating pH during the experiment. As the average pH over the entire test run was 5.9, instead of the desired setpoint of 6.5, lower precipitation efficiencies were an expected outcome. The recovery of contaminants such as Al and Fe was above 95 percent due to their insolubilization in the operating pH. The raffinate from this stage was further processed at pH 9.0 for CM precipitation. Recovery for all the elements except calcium was 100 percent in this stage. The enriched RE and CM cakes were leached at pH 1.5 using HCl as a pH modifier for approximately 1 hour. The solid concentration in RE and CM leaching was 10% and 20% S/L (%w/v), respectively. It should be noted here that the RE leaching was on a whole mass weight basis, whereas CM leaching was on a dry weight basis. The resultant PLS solution had elevated concentrations of REEs and CMs compared to the leachate solution (Table 3.12.3, Table 3.12.4).

Table 3.12.4 Elemental concentration in the RE and CM cake (dry, whole mass basis, ppm) and the PLS content (ppm) generated from their leaching.

| | Solid Concentration | | | | |
|---|---|---|---|---|---|
| | TREE | Co | Mn | Ni | Zn |
| **RE Cake** | 18329.0 | 10763.3 | 5648.4 | 23914.9 | 11872.7 |
| **CM Cake** | 5248.3 | 6583.7 | 20724.1 | 19962.2 | 1265.5 |
| | PLS Concentration | | | | |
| **RE PLS** | 292.4 | 169.5 | 77.6 | 295.5 | 188.2 |
| **CM PLS** | 402.2 | 504.8 | 1047.1 | 1530.7 | 93.6 |

The enriched PLS was processed for oxalic acid precipitation at pH 1.5 in the same manner as the previous circuits. REE recovery during the oxalic acid precipitation was 80 percent. The primary contaminant in this stage was calcium due to its relatively higher precipitation efficiency than other elements. Calcium recovery was 4 percent, whereas all the other elements had a precipitation efficiency less than 1 percent. The oxalic acid precipitation raffinate was mixed with the PLS generated from the leaching of CM cake and further treated with sodium sulfide at pH 3.0 to precipitate Co, Ni and Zn. The optimum chemical dosages in this stage were also identified based on the bench-scale data. It was determined that Co, Ni and Zn precipitation efficiency was above 90 percent and manganese recovery was only 5 percent. Furthermore, iron and aluminum recovery was 67% and 2%, respectively. Significantly higher Fe precipitation was likely due to its precipitation above pH 2.0. Finally, manganese precipitation stage at pH 9.0 recovered the remaining REEs, Al, Co, Fe, Mn, Ni and Zn from the solution.

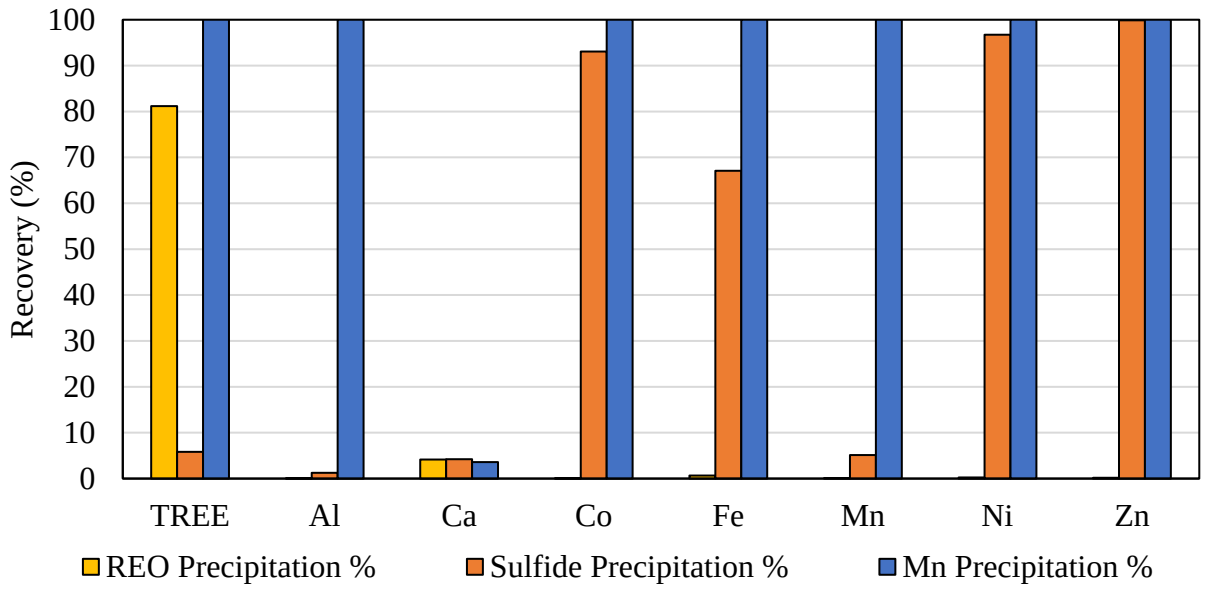



Figure 3.12.18 Recovery of different elements in the RE-Oxide, sulfide and manganese precipitation stages during the pilot scale experiment.

The overall recovery data presented in Figure 3.12.19 indicated that more than 70 percent of the contaminant content was rejected to the leaching solid residue and elevated recoveries for REEs and CM were obtained. The results suggested REEs and CMs were easily leachable compared to the contaminants. The aluminum precipitation rejected virtually all of the iron with only 0.8 percent iron going to the downstream process. However, due to the lower than the optimum pH for the aluminum precipitation stage, only 17 percent of the total aluminum

was removed and 8 percent precipitated with the RE-CM cake. Since Al content was significantly higher than REEs, 8 percent aluminum content would translate into a considerable chemical cost increase in the oxalic acid precipitation stage. Similarly, as the RE precipitation pH was 5.9 instead of the designed 6.5, approximately 8 percent of the total REEs were carried over to the CM cake. Even with the lower operating pH in the RE precipitation stage, nearly 43%, 15%, 34%, and 57% of Co, Mn, Ni, and Zn were lost to the RE cake. The RE-Oxide stage in this process only recovered 29% of the total REEs in the system. Lower REE precipitation efficiency in REE removal stage coupled with inefficiencies associated with the RE cake leaching and subsequent oxalic acid precipitation resulted in a significant loss of REEs. However, despite the losses, the oxalic acid cake generated from this test had high purity. The RE-Oxide content in the oxalic acid was 95%, with approximately 3% calcium contamination (Figure 3.12.19).

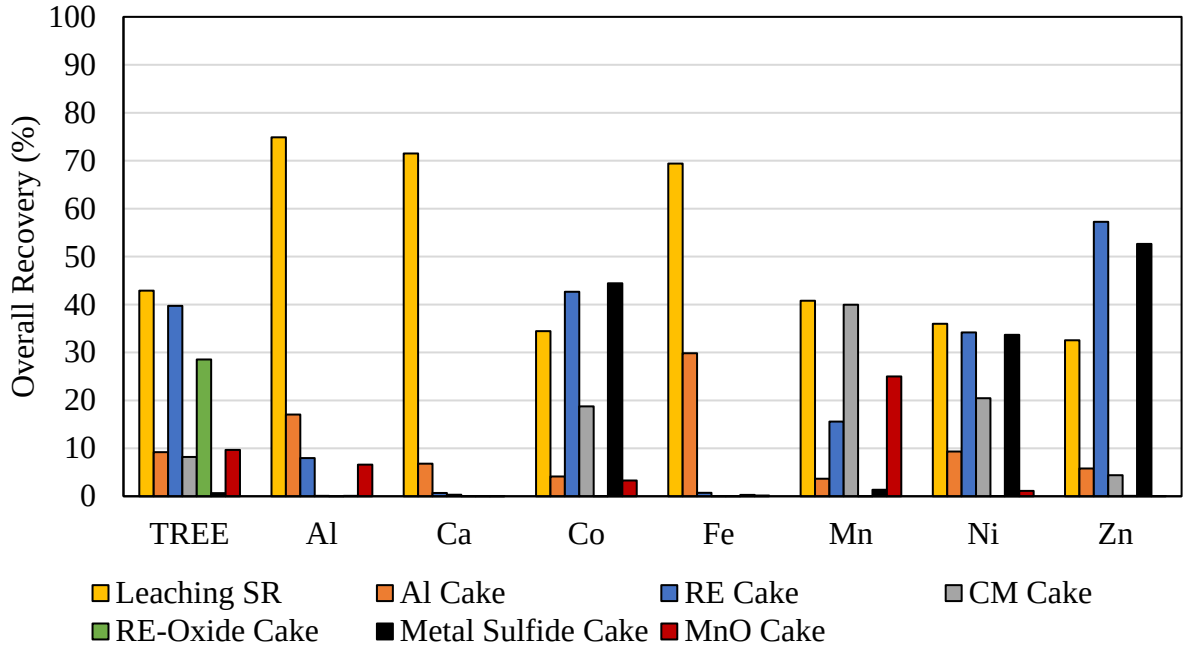



Figure 3.12.19 Overall recovery of various elements of interest in solid cakes generated during the pilot scale test.

As the oxalic acid raffinate was mixed with the CM leached solution, the overall recovery of the critical metals was significantly higher than the previous test (Figure 3.12.14, Figure 3.12.19). The overall recovery of Co, Ni and Zn in the metal sulfide cake was 44%, 34%, and 53%, respectively. The Co, Ni and Zn distribution in the metal sulfide cake was 22%, 46%, and 21%, respectively, accounting for approximately 90 percent of the total content in the cake. Higher Ni content in the final cake relative to cobalt and zinc was due to its elevated initial concentrations in the leachate (Table 3.12.4). The primary contaminant in the metal sulfide cake was iron. The manganese precipitation stage recovered the remaining REEs and critical metals along with Fe and Al in the solution. As the oxalic acid raffinate was recombined with the CM leachate, the REEs lost to the CM cake precipitated with the Mn cake, accounting for 10 percent of the total REEs in the feed. The distribution data presented in Figure 3.12.20 revealed that the manganese cake contained more than 70 percent aluminum, with only 12 percent manganese. Comparatively, the manganese oxide product generated in the previous circuit comprised of approximately 70 percent of manganese. This drastic shift in the distribution was due to the aluminum bypassed during the Al precipitation stage. As the concentration of Al was significantly higher than manganese (Table 3.12.3), even with 100

percent process efficiency after the aluminum precipitation stage would have produced the cake, which was predominantly aluminum oxide. This finding reiterates the need for strict process control during the experiments to provide unnecessary losses and carryovers to the next stage.

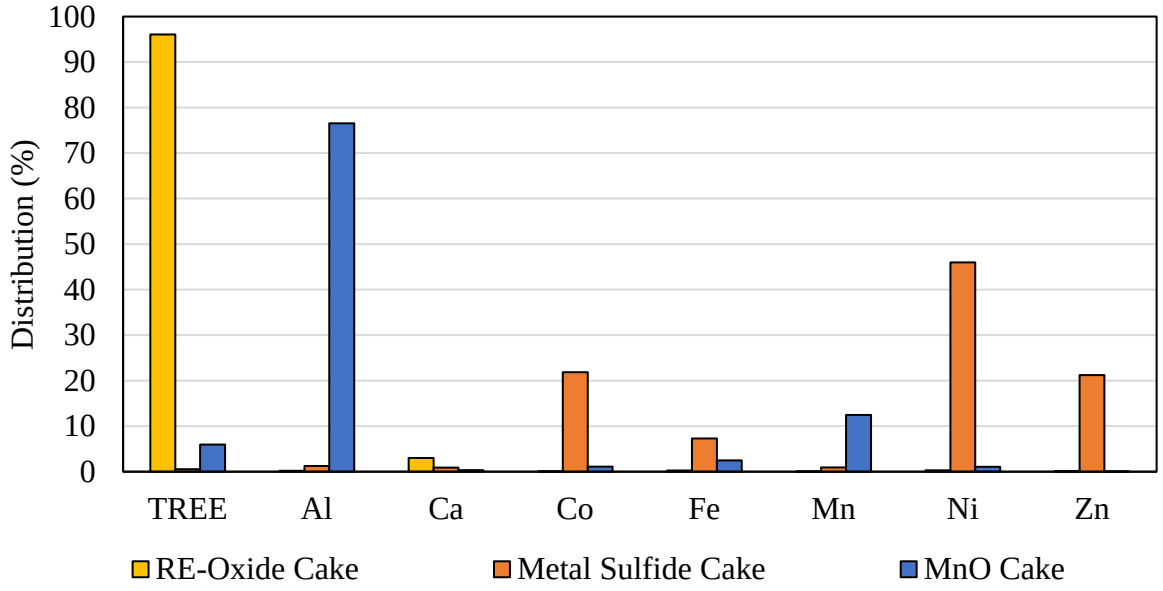



Figure 3.12.20 Percent distribution of different elements in the final products generated during the pilot scale test.

Overall, circuit arrangement 3 proved beneficial for the recovery of critical metals lost in the previous circuit arrangement. However, the process inefficiencies associated with this test run resulted in significant elemental losses during different stages of the experiment. For instance, lower pH setpoints in Al and RE precipitation stages resulted in elemental carryover to the next process stages. Therefore, if this circuit arrangement is adopted, strict control over the process pH would be required to ensure high precipitation efficiencies and lower losses. Furthermore, as the REE loss to the CM cake was eminent during this circuit arrangement, one approach would be mixing leached PLS from RE and CM cakes to ensure minimal loss of REE recovery.

### *1.1.22 Circuit Arrangement-4*

The process flowsheet tested in section 1.1.21 revealed significant REE losses to the CM cake even with perfect process controls. Similarly, a significant loss of CMs to RE cakes was also inevitable due to the overlapping precipitation behavior of CMs and REEs. As such, it was concluded that mixing the PLS generated from the leaching of RE and CM cakes would ensure minimal elemental losses. It was postulated that if mixing the redissolved solution is crucial to maximizing recovery, instead of two different precipitation and redissolution stages, the circuit could be abbreviated to a single precipitation stage at pH 9.0 followed by a single redissolution stage. Since separate processing of RE and CM cakes incurred additional costs in terms of CAPEX and OPEX, single precipitation at pH 9.0 and subsequent leaching would decrease the processing cost while ensuring minimal losses of REEs and CMs. The modified process schematic is shown in Figure 3.12.21. Similar to previous flowsheets, the process starts with leaching followed by solvent extraction and aluminum precipitation. It should be noted here that leaching could be conducted using either 0.5 M sulfuric acid or bio acid of the equivalent sulfuric acid molarity. In the new circuit flow, raffinate from the aluminum precipitation stage

would be treated at pH 9.0 to recover both REEs and CMs, followed by the cake redissolution using HCl.

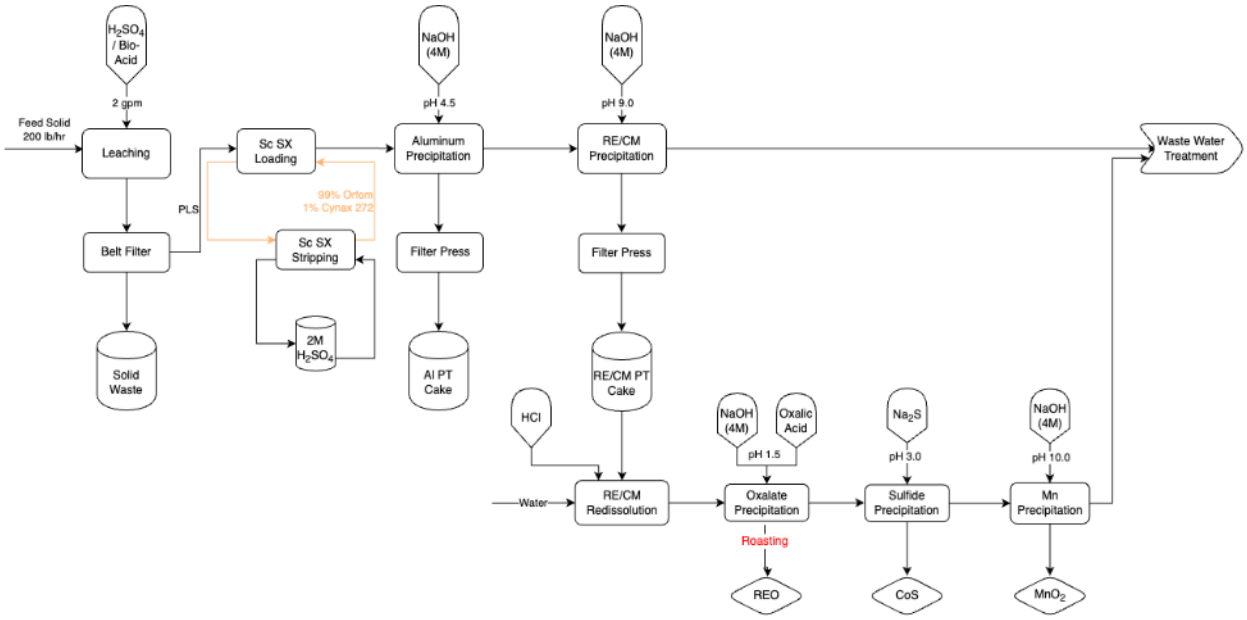



Figure 3.12.21 Circuit arrangement 4 for the production of RE-Oxide, Metal Sulfide and Manganese oxides.

**<u>Pilot Scale Experiment-PLS</u>:**

The new circuit arrangement was first tested on the heap leach PLS. Similar to the previously discussed heap leach process, the PLS was first treated for iron removal at pH 3.0. In this flowsheet, solvent extraction was used to remove Sc from the PLS after the iron precipitation stage. Subsequently, the solution pH was raised to 4.5 for Al precipitation followed by rare earth and critical metal precipitation at pH 9.0. The enriched RE and CM cakes were redissolved at pH 2.5 instead of 1.5 to provide higher selectivity toward REE and CM dissolution. The PLS generated from the leaching was processed with oxalic acid, sodium sulfide and sodium hydroxide precipitation in succession to produce RE-oxide, metal sulfide, and manganese-hydroxide products. As the process controls and equipment were the same as in previous experiments, their operations are not presented in this section.

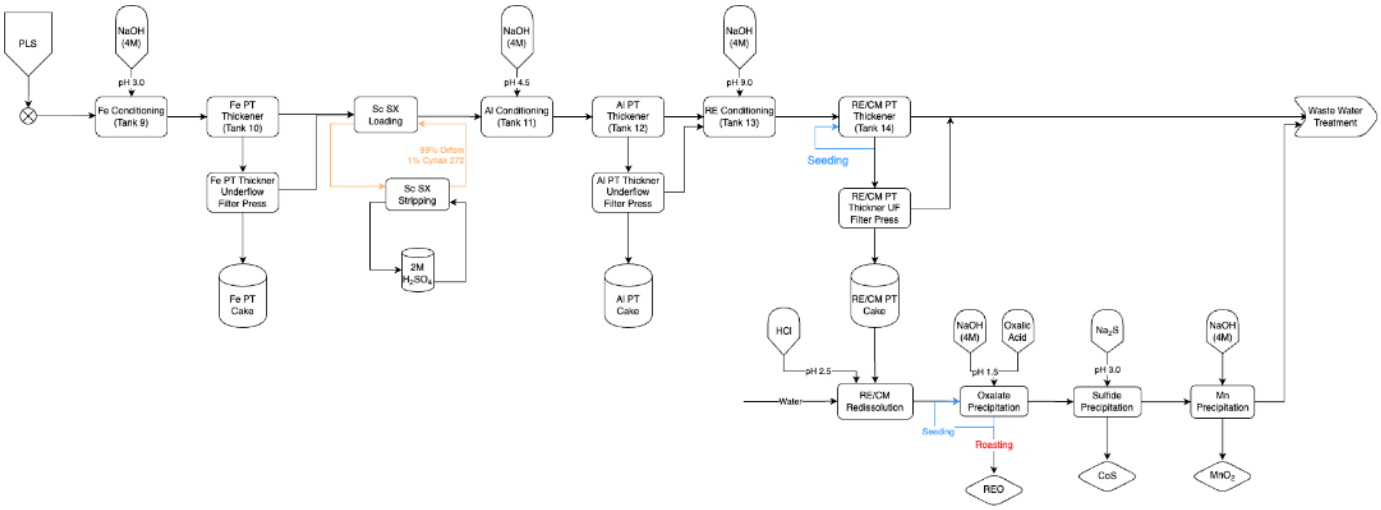



Figure 3.12.22 Process flowsheet based on circuit arrangement-4 for the treatment of heap leach PLS to recover REEs and CMs.

Elemental content in the heap leach PLS used in this pilot scale test is shown in Table 3.12.5. It is evident that iron concentration in the PLS was significantly higher than the rest of the elements. The REE content was only 12.82 ppm, whereas Co, Mn, Ni and Zn concentrations were 6, 68, 20, and 21 ppm, respectively. These concentrations are significantly lower than the heap leach PLS shown in Table 3.12.3. The iron precipitation stage at pH 3.0 removed approximately 87 percent of the iron in the feed PLS, indicating high precipitation efficiency (Figure 3.12.23). There was a small loss of other elements with the iron cake, likely due to the entrainment, as all the elements except iron are soluble in the operating pH range. The raffinate was subsequently passed through the solvent extraction circuit for selective removal of scandium. It was found that scandium extraction efficiency during this test was 78 percent, along with 18 percent iron loading onto the organic phase. Subsequently, the PLS was treated for aluminum removal at pH 4.5. The results showed 81 percent aluminum and 90 percent iron removal, with a minimal loss of REEs and CMs during this stage (Figure 3.12.23). The results shown here clearly indicate the significance of the iron removal step. As the iron content after the Fe precipitation stage was just 270 ppm, no significant loss of REEs was observed. Finally, the raffinate was further processed at pH 9.0 to recover all the REEs and CMs in the solution (Figure 3.12.23).

Table 3.12.5 Concentrations (ppm) of various elements of interest in the heap leach PLS used for the pilot scale test.

| | **TREE** | **Al** | **Ca** | **Co** | **Fe** | **Mn** | **Ni** | **Zn** |
|---|---|---|---|---|---|---|---|---|
| **PLS Content** | 12.82 | 994.74 | 295.75 | 6.07 | 2074.12 | 67.87 | 19.88 | 20.68 |

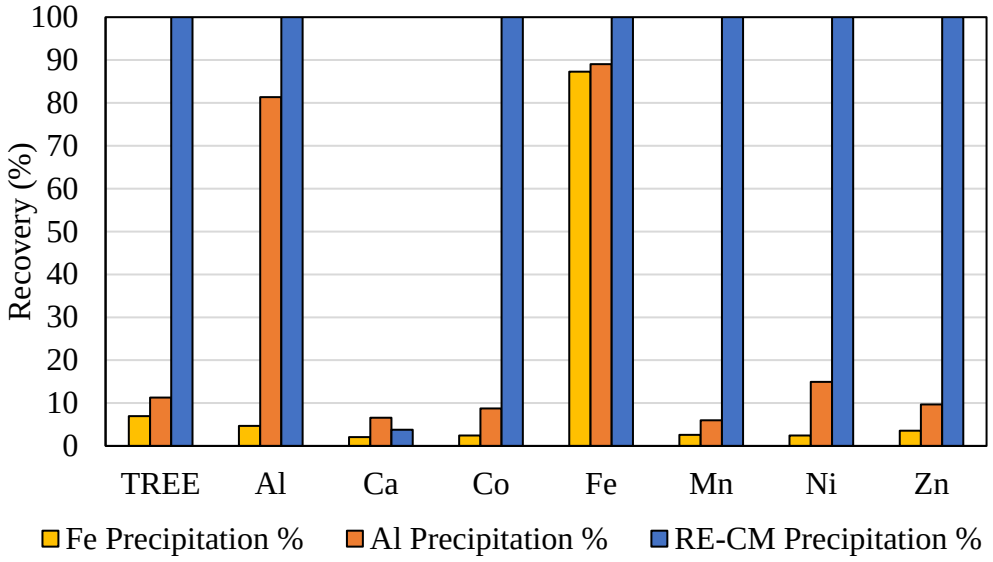



Figure 3.12.23 Recovery of various elements across different precipitation stages during the pilot scale experiment.

As in the previous circuits, enriched RECM cake was leached with HCl at 10% S/L (w/v, whole mass basis) for approximately an hour to generate a concentrated RE-CM leachate solution. The concentrations in the RE-CM cake and the resultant leachate solution are shown in Table 3.12.6. It is evident the concentrations of all the elements have increased roughly five folds relative to the initial feed concentration data shown in Table 3.12.5. The slurry was filtered and a leachate sample was collected and processed at bench-scale to identify the optimum chemical dosages for oxalic acid and sodium sulfide precipitation stages. REE recovery during the pilot scale experiment was 76 percent, whereas calcium recovery was 5 percent (Figure 3.12.24).

Recovery of all other elements was less than 1 percent. The sodium sulfide precipitation efficiency for Co, Ni and Zn was 68%, 57%, and 85%, respectively (Figure 3.12.24). Lower precipitation efficiencies in the oxalic acid and sodium sulfide precipitation stages would result in REE and CM losses to the following precipitation stage. Finally, the manganese precipitation step recovered 100 percent REE and CM content in the solution (Figure 3.12.24).

Table 3.12.6 Elemental content (ppm) in the RECM cake (dry, whole mass basis) and PLS solution generated through leaching.

| | **TREE** | **Co** | **Mn** | **Ni** | **Zn** |
|---|---|---|---|---|---|
| **RE-CM Cake** | 2796.9 | 2242.5 | 26060.8 | 6915.8 | 7553.2 |
| **RE-CM PLS** | 67.6 | 34.6 | 320.6 | 99.2 | 116.2 |

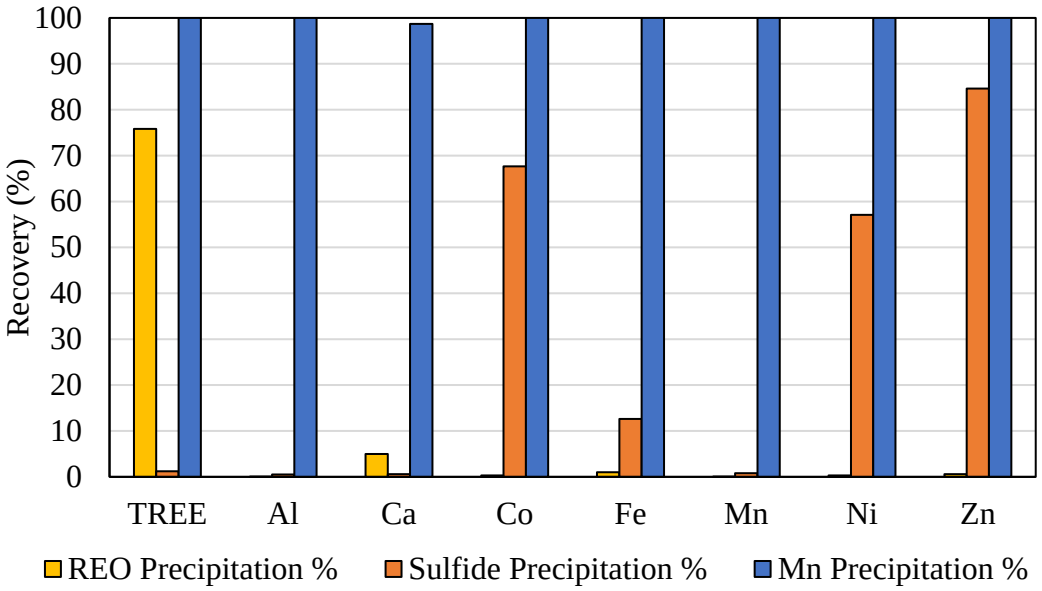



Figure 3.12.24 Precipitation efficiency of different elements in oxalic acid, metal sulfide and manganese precipitation stages during the pilot scale experiment.

The overall data demonstrated that a combined 97 and 83 percent of the total iron and aluminum was removed during the aluminum and iron precipitation stages (Figure 3.12.25). However, there was also a loss of REEs and CM in these stages, which was unavoidable owing to the trace concentrations of these elements. All the remaining content of REEs and CM precipitated at pH 9.0. Since there was a bypass of Al and Fe from Al precipitation stage, approximately 18% Al and 1% Fe precipitated in the RE-CM cake (Figure 3.12.25). Similar to the previous tests, since their initial concentrations were significantly higher than REEs and CMs, a small percentage will result in significantly higher contamination in the downstream processes. The RE-Oxide precipitation stage recovered 56 percent of the total REEs, which was the highest recovery obtained in this project. As the REE precipitation efficiency was only 76% (Figure 3.12.24), 18% of the REEs were lost to the manganese product. Granted that the oxalic acid precipitation recovered >90 percent of the REEs, an overall recovery as high as 70 percent could have been accomplished. As such, it is crucial to determine the chemical dosages with utmost care. The REE product obtained from this experiment contained approximately 90 percent RE-Oxides with Ca and Fe as the primary contaminants (Figure 3.12.26).

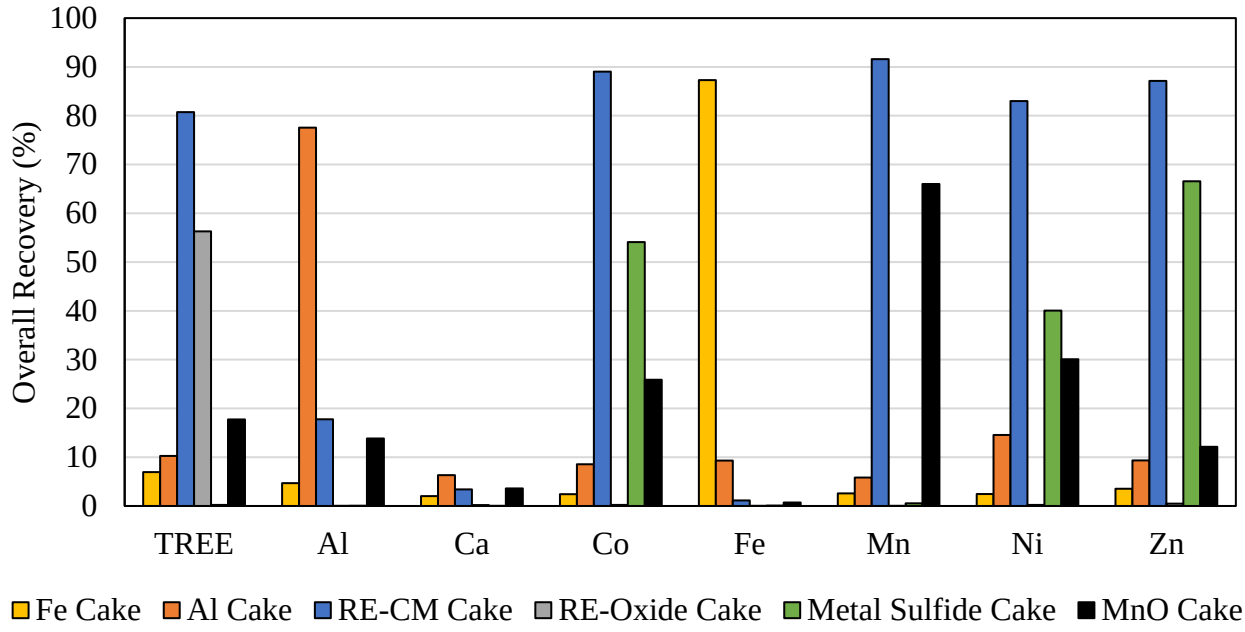



Figure 3.12.25 Overall recoveries in the solid cakes generated during the pilot scale testing of the heap leach PLS.

The metal sulfide cake recovered 54%, 40%, and 66% of total Co, Ni and Zn, respectively (Figure 3.12.25). Similar to the RE-Oxide cake, although the overall recovery values obtained during this test were higher than all the previous tests, lower recovery in the metal sulfide precipitation stage resulted in the carryover of these elements to the Mn precipitation stage. It is apparent based on the results that 26%, 30% and 12% of the total Co, Ni and Zn content reported to the Mn cake (Figure 3.12.25). However, despite the losses, CM product purity was still close to 90 percent (Figure 3.12.26). The metal sulfide product contained 50% Zn, 29% Ni and 12 % Co with approximately 8 percent Fe content. The overall recovery of manganese and aluminum in the manganese cake was 66 and 14 percent, respectively. Even though manganese recovery was 5-6 folds higher than aluminum recovery, manganese product primarily consisted of Al. The Mn content in the final product was 21 percent whereas the Al content was 64 percent. This trend is similar to the previous tests and re-emphasizes the significance of contaminant removal to generate a high-purity product. Other contaminants in this stage were calcium, iron, nickel and cobalt.

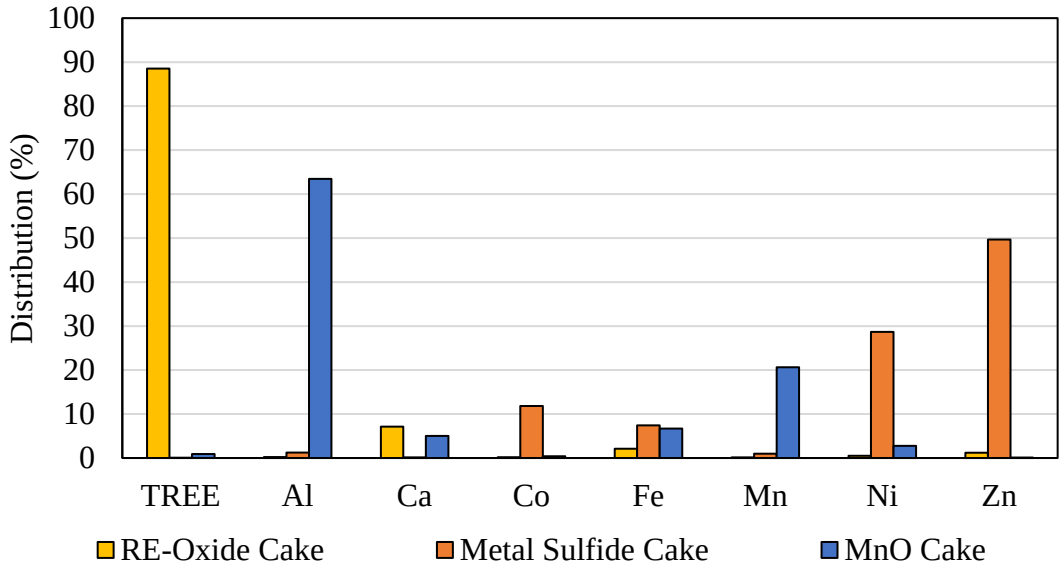



Figure 3.12.26 Elemental distribution in the final products generated from the treatment of PLS.

**Pilot Scale Experiment-Shakespeare**

Two pilot-scale tests using circuit arrangement-4 were also conducted on Shakespeare material. In the first experiment, 0.5 M sulfuric acid was used as a lixiviant, whereas in the second test, bio acid with 0.5 sulfuric acid molarity was employed for the leaching operation. As the bio-acid contained significant iron content, the solvent extraction stage was skipped for the second test. Furthermore, due to the high overall contaminant content in the bio acid, the RE-CM cake was leached at pH 2.5 instead of 1.5 to provide higher leaching selectivity towards REEs and CMs. The remaining circuit arrangement and process conditions were the same between the two tests. The process flowsheet for both tests is shown in Figure 3.12.21. The results for both tests are discussed below.

**Sulfuric acid:** As mentioned previously, leaching in this test was conducted using 0.5 M sulfuric acid. Other leaching conditions such as solid-to-liquid ratio, temperature and time, were the same as previous experiments. Results showed that the REE leaching recovery during this test was 54 percent, whereas recovery of other critical metals such as Co, Mn, Ni, and Zn was 70%, 72%, 68%, and 69%, respectively (Figure 3.12.27). REE recovery in this test was 3 percent less than the previous Shakespeare test, while the recovery of CM, on average, was higher than in the last test. This was likely due to the change in feed composition. The REE content in the feed used for this test decreased, whereas the CM content increased relative to the previous test feed. The leaching recovery for contaminants such as Al, Ca and Fe was only 18%, 25%, and 32%, respectively (Figure 3.12.27). The concentrations of different elements of interest in the leachate are shown below. Following the leaching step, scandium was removed from the PLS using solvent extraction. It was found that the scandium and iron loading efficiencies in the solvent extraction stage were 25 and 2 percent, respectively.

Table 3.12.7 Elemental concentration (ppm) in the PLS generated from the leaching of Shakespeare material in the pilot scale test.

| | TREE | Al | Ca | Co | Fe | Mn | Ni | Zn |
|---|---|---|---|---|---|---|---|---|
| **Leaching-PLS** | 37.9 | 399.3 | 803.9 | 29.7 | 692.3 | 52.8 | 81.2 | 22.3 |

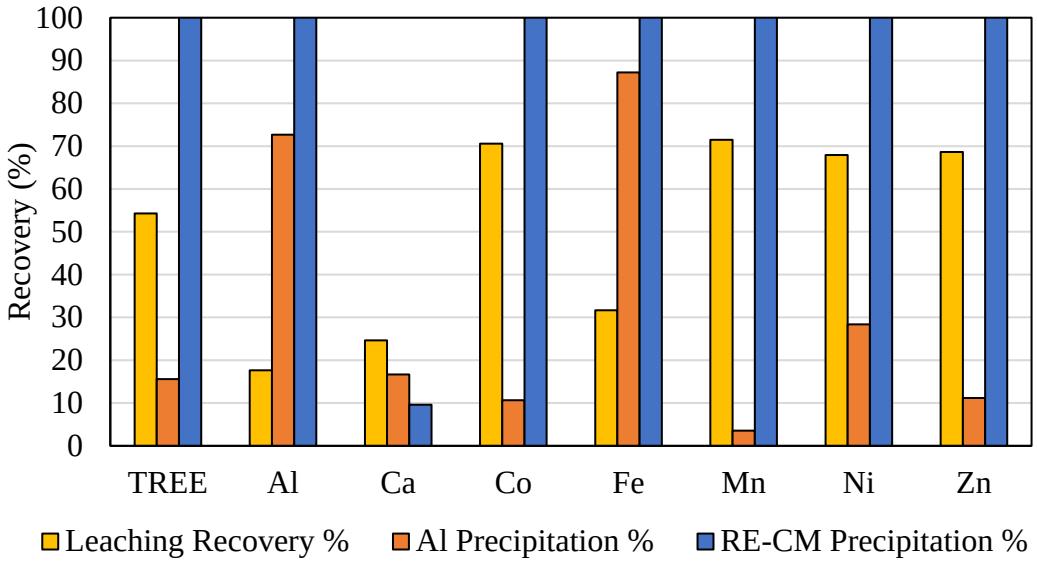



Figure 3.12.27 Recovery of various elements across different precipitation stages during the pilot scale test.

The Al precipitation stage removed 73 percent Al and 87 percent iron (Figure 3.12.27). As observed in the previous circuits, there was some loss of REEs and CMs during the aluminum precipitation stage, possibly due to entrainment. The aluminum precipitation stage raffinate was further processed at pH 9.0 to recover the remaining REEs and CMs, along with the aluminum and iron in the system. The concentrations of rare earth elements and critical metals are shown in Table 3.12.8. It is interesting to note that nickel content in the cake was significantly higher than REEs, Zn, and Co. This was simply due to the higher initial concentration of the Ni content in the feed material. The PLS generated from the leaching of RE-CM cake contained 95 ppm REE, 78 ppm Co, 142 ppm Mn, 166 Ni, and 58 ppm Zn content. Overall, the concentrations of all the elements were roughly 2.5x the initial concentrations. As mentioned previously, since the RE-CM cake contained only 20% solid content, the dilution caused by the water in the cake resulted in significantly lower elemental content in the PLS.

Table 3.12.8 Elemental concentration (ppm) in the RE-CM cake (dry, whole mass basis) and PLS solution generated from its leaching during the pilot scale test.

| | **TREE** | **Co** | **Mn** | **Ni** | **Zn** |
|---|---|---|---|---|---|
| **RE-CM Cake** | 7617.8 | 6346.1 | 12178.8 | 13913.9 | 4742.0 |
| **RE-CM PLS** | 95.4 | 77.5 | 141.9 | 165.5 | 57.8 |

The optimum chemical dosages to ensure high precipitation of REEs and CMs were identified based on the bench-scale data. It was found that the REE recovery in the oxalic acid precipitation stage was 95 percent. In contrast, the recovery of other contaminants was minimal, demonstrating a higher selectivity of oxalic acid toward rare earth elements (Figure 3.12.28). The sulfide precipitation stage reported high precipitation efficiency for Co, Ni and Zn. It was determined that Co, Ni and Zn recoveries were 99%, 92% and 100%, respectively (Figure 3.12.28). Surprisingly, the recovery of REEs in the sulfide precipitation stage was 75 percent. Similarly, the calcium precipitation efficiency was 99 percent and iron recovery was 89 percent. This was likely a result of the excessive oxalic acid present in the system. High precipitation efficiency during the oxalic acid precipitation stage indicated sufficient oxalic acid was present in the solution. Raising the solution pH to 3.0 increased the availability of

oxalate anions, which resulted in the higher precipitation of REEs as well as other contaminants such as calcium and iron. An increase in the precipitation of contaminants has been reported to be due to the drop in selectivity of oxalic acid as a function of an increase in solution pH. These results differ from the previous PLS test results shown in Figure 3.12.24, likely due to the lower REE precipitation efficiency, which is symbolic of the insufficient availability of oxalate anions. Interestingly, manganese recovery in the sulfide precipitation stage was approximately 80 percent.

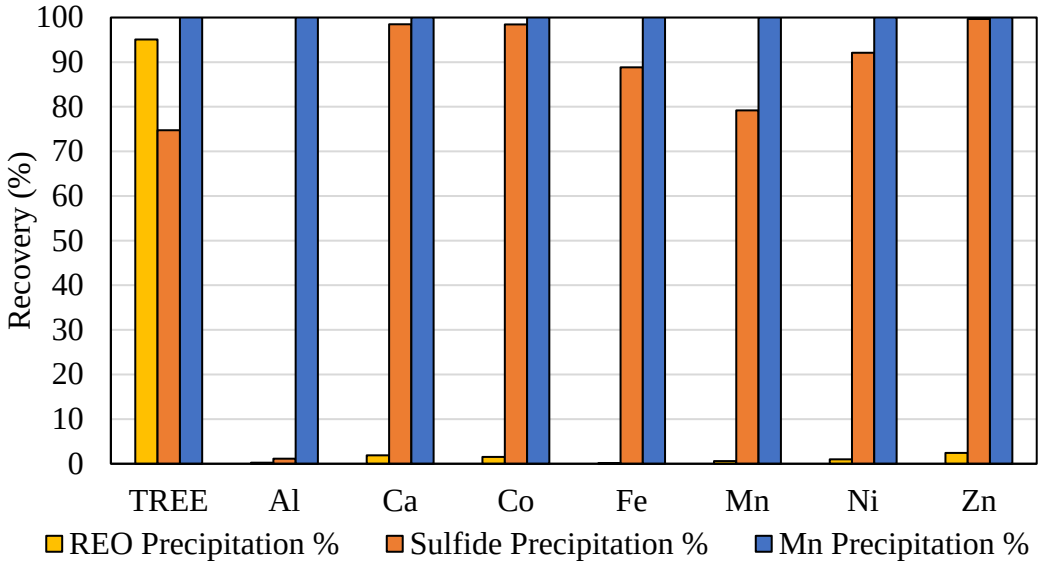



Figure 3.12.28 Recovery of various elements in the final product precipitation stages during the pilot scale test.

The overall recovery data presented in Figure 3.12.29 indicated high contaminant rejection and more selectivity toward REEs and CMs during the leaching stage, which was consistent with the previously reported findings. The aluminum precipitation stage removed 12% and 27% of the total aluminum and iron and less than 5% of the total content reported to the RE-CM cake. However, as discussed previously, since the total concentration was significantly higher than REEs and CMs combined, even a small percentage translated into elevated solution content. The RE-CM cake recovered 46% of total REEs while overall recovery of Co, Mn, Ni, and Zn was 63%, 69%, 49%, and 61%, respectively. Due to the losses in the redissolution stage, overall REE recovery in the RE-oxide recovery was only 36 percent, even with the 95 percent efficiency. The REE product generated in this test was high purity and contained 92% RE-Oxide content with calcium, cobalt and zinc as the primary contaminants (Figure 3.12.30).

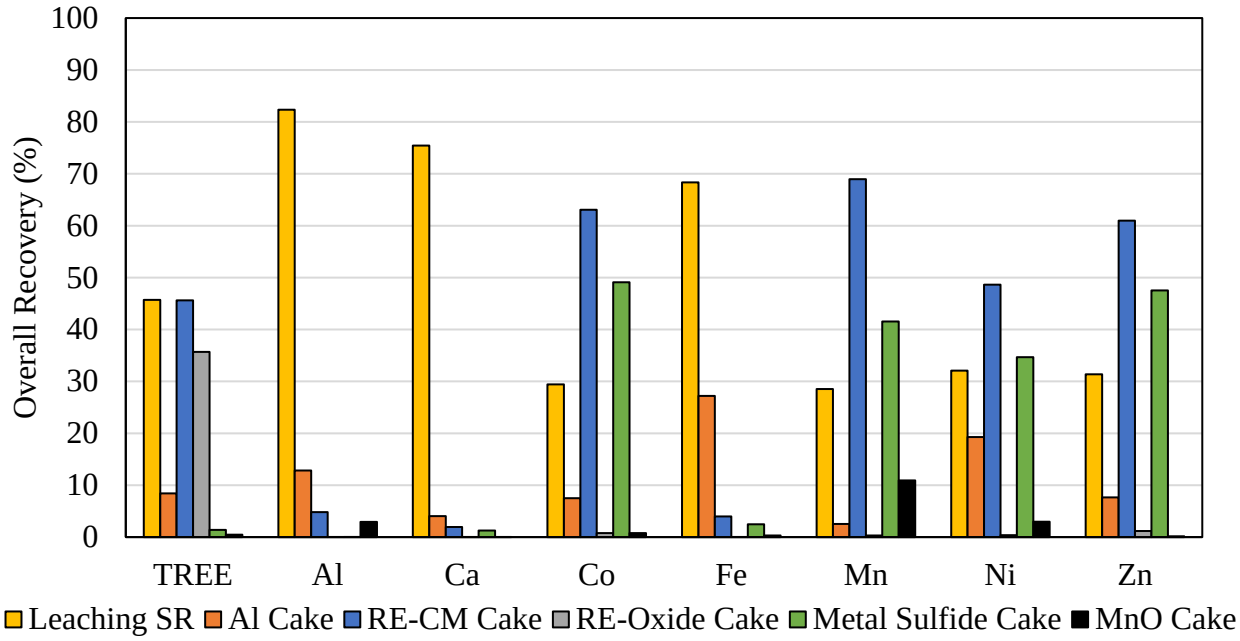



Figure 3.12.29 Overall recoveries of different elements in cakes generated during the pilot scale experiment.

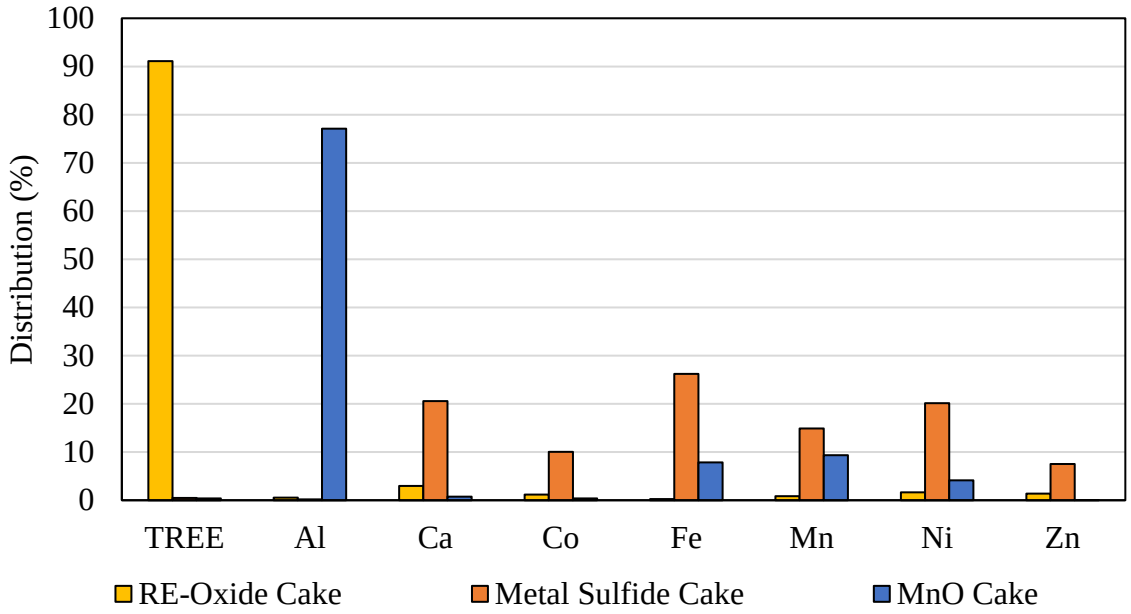



Figure 3.12.30 Elemental distribution in the final products generated in the pilot scale test.

The overall recovery of Co, Mn, Ni and Zn in the metal sulfide cake was 49%, 42%, 45%, and 48%, respectively (Figure 3.12.29). Due to the significantly higher content and precipitation efficiencies of other contaminants, the metal sulfide product generated in this test was low grade. Iron and calcium accounted for more than 45 percent of the total content in the cake (Figure 3.12.30). Similarly, manganese precipitation in the metal sulfide stage was another source of contamination, representing 15 percent of the total composition. As most of the Mn precipitated in the metal sulfide cake, the overall recovery of Mn in the final cake was only 11

percent. The MnO cake primarily consisted of aluminum, accounting for approximately 78 percent of the total mass. Other contaminants were iron and nickel, while the Mn content was only 9.3 percent.

**Bio-Acid:** A pilot scale experiment on Shakespeare material using bio-acid as a lixiviant instead of conventional sulfuric acid was also conducted following the circuit arrangement shown in Figure 3.12.21. To provide a direct comparison between the leaching efficiencies of the bio acid and sulfuric acid, the acid concentration was kept constant. Bio acid strength was determined through a molar equivalency test described in the previous section. It was determined that the bio-acid for the Shakespeare experiment contained less than 0.5M acid content. As such, industrial-grade sulfuric acid was added to the bio acid to achieve a 0.5M acid strength. The elemental content in the bio acid is shown in Table 3.12.9. It was found that the bio-acid contained significantly higher iron content due to the decomposition of pyrite. Iron content in the bio acid was 6029 ppm with appreciable concentrations of other critical metals such as manganese and zinc.

Table 3.12.9 Elemental content (ppm) in the bio-acid solution used for leaching in the pilot scale test.

| | TREE | Al | Ca | Co | Fe | Mn | Ni | Zn |
|---|---|---|---|---|---|---|---|---|
| **Feed** | 1.9 | 30.2 | 779.9 | 1.5 | 6093.8 | 40.0 | 5.7 | 70.5 |

The results showed that the leaching efficiency of REEs was significantly impacted when using bio-acid instead of conventional sulfuric acid as a lixiviant. The REE leaching recovery dropped from the 55-60 percent range in the previous experiments to only 47 percent in this test. Interestingly, the recovery of critical metals such as Co, Ni, and Mn was only marginally impacted. In fact, zinc recovery increased from 68% in the previous experiment to 76% in this test. This finding suggested easier leaching characteristics of critical metals relative to the rare earth elements. The iron and aluminum recovery in the leaching stage was 10% and 63%, respectively.

As the PLS contained considerable iron content, solvent extraction was bypassed in this experiment and the leachate was directly treated for aluminum precipitation. It was determined that Al and Fe recovery in the aluminum precipitation stage was 87% and 72%, respectively. It is interesting to note that the precipitation of aluminum is significantly higher than iron even though the latter starts precipitating at around a pH of 3.0. This finding implied that at least a portion of iron in the leachate solution was present as ferrous ions. Contrarily, the aluminum precipitation efficiency was significantly higher than the previous experiments. The average solution pH during the experiment was 4.0 instead of the designed 4.5. As such, elevated Al recovery was likely not due to their insolubilization at elevated pH levels. Since the solution had a considerable iron content, $Al^{+3}$ ions were likely adsorbed on the surface of iron precipitates to neutralize the surface charge. However, as the Al content was significantly lower than iron, REE ions tend to adsorb on the iron precipitate surfaces in the same manner as Al ions, which was likely the reason for approximately 50 percent REE loss during the Al precipitation stage even with a low operating pH of 4.0. As mentioned previously, solutions concentrated in iron content need to be treated separately in the iron precipitation stage. The results presented in this test clearly demonstrate the importance of such a stage to maximize REE recovery. Similar to REEs, there was an accompanying loss of other elements such as Co, Mn, Ni and Zn. However, the loss was not as prominent as REEs. Following the Al precipitation stage, remaining REEs, CMs, Al and Fe ions were recovered during the RE-CM precipitation stage. As the iron recovery was significantly lower in the Al precipitation stage, high iron

content was anticipated to create processing problems such as high chemical consumptions, lower precipitation efficiencies and elevated contamination of the final products.

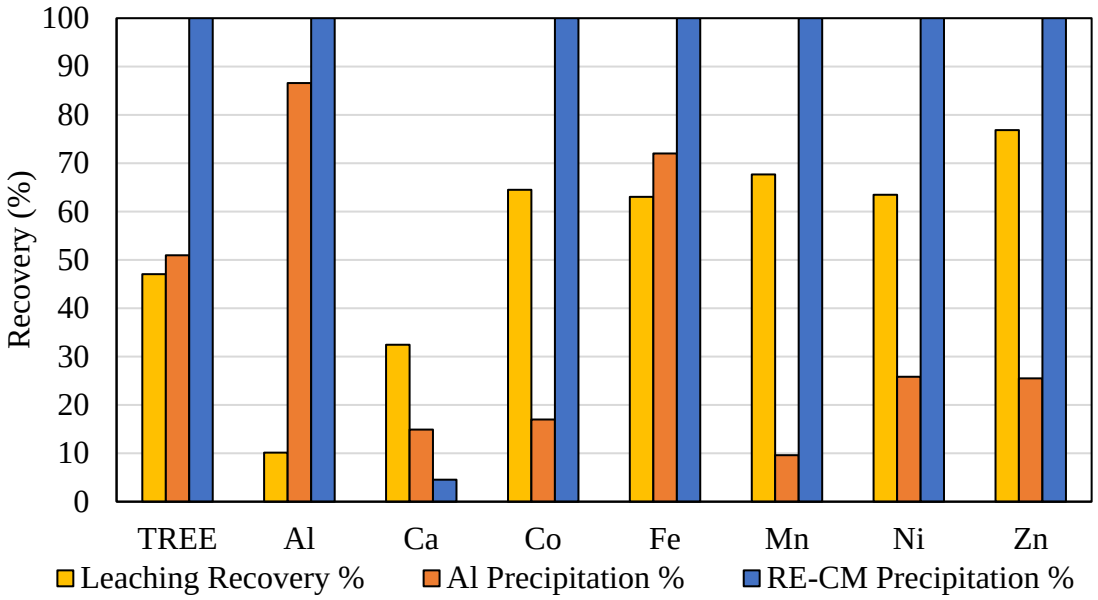



Figure 3.12.31 Recovery of various elements of interest in different precipitation stages during the pilot scale test.

The RE-CM cake generated in this experiment contained a substantially high iron content relative to other elements table below. As such, the redissolution/leaching was conducted at pH 2.5 instead of the conventional pH of 1.5 to provide a selectivity towards REEs and CM dissolution. However, even with only 43% leaching recovery of iron, the resulting PLS contained considerably higher concentration of iron compared to other elements. The iron content was 2318 ppm whereas REE content was only 41 ppm in the solution. As the bio-acid contained high concentrations of Zn and Mn (Table 3.12.9), the RE-CM cake and subsequent leachate contained high content of these elements compared to previous experiments (Table 3.12.8). A sample of the PLS was collected from the holding tank and bench-scale experiments were performed to identify the optimum oxalic acid and sodium sulfide dosages for this solution.

Table 3.12.10 Elemental content (ppm) in the RE-CM cake (dry, whole mass basis) and the PLS solution generated from the leaching of RE-CM cake.

| | **TREE** | **Al** | **Ca** | **Co** | **Fe** | **Mn** | **Ni** | **Zn** |
|---|---|---|---|---|---|---|---|---|
| **RE-CM Cake** | 2037.7 | 6021.6 | 3508.3 | 2592.4 | 245056.5 | 10250.1 | 6885.4 | 8566.9 |
| **RE-CM PLS** | 31.4 | 107.0 | 137.9 | 42.1 | 2317.7 | 172.5 | 92.9 | 127.5 |

The results revealed only 41 percent REE recovery during the oxalic acid precipitation stage whereas recovery of all other elements was <1 percent (Figure 3.12.32). Significantly lower REE precipitation efficiency compared to previous experiments was likely a result of the elevated iron content. However, as the same solution was used in the bench-scale experiments, it can be concluded that either the bench-scale study did not identify the correct dosage to ensure high precipitation efficiency or lower than optimal oxalic acid dosage was employed in the pilot-scale experiment. Subsequently, some of the REEs bypassing the oxalic acid precipitation stage insolubilized in the metal sulfide precipitation stage. Unlike the oxalic acid

precipitation stage, elevated critical metal recovery was observed in the sulfide precipitation stage. Co, Ni and Zn recovery was determined to be 90%, 93%, and 99%, respectively (Figure 3.12.32). However, there was 52 percent recovery of iron and 10.4% recovery of aluminum. Considering the elevated concentration of iron, such a high recovery would result in significant contamination of the sulfide product. Interestingly, contrary to the previous experiment, Mn recovery was only 9 percent. The filtrate from sulfide precipitation stage was treated for Mn precipitation where the remaining elemental content was insolubilized (Figure 3.12.32).

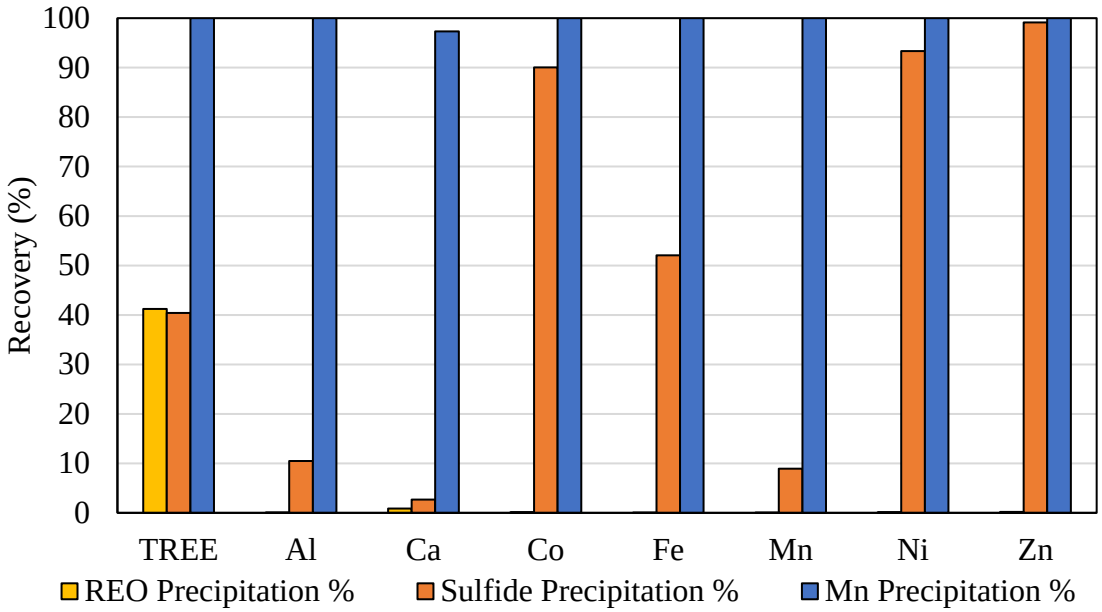



Figure 3.12.32 Recovery of different elements in oxalic acid, sulfide, and manganese precipitation stages during the pilot scale experiment.

As discussed previously, REE leaching efficiency in this experiment was lower than the previous tests as more than 50 percent of the REEs remained in the leaching solid residue. Interestingly, the leaching efficiency of CMs was not impacted as significantly when using bio-acid and their overall recovery was similar to the previous circuit arrangements. Furthermore, approximately 90 percent Al and 67 percent Ca were rejected to the solid residue cake. Comparatively, only 36 percent of the total iron reported to the solid residue, contributing to the elevated iron content in the PLS. The Al precipitation stage removed approximately 9 percent of the total aluminum, which eliminated roughly 99 % of the total Al content in the first two stages of the test. Unfortunately, only 46 percent of the total iron was removed in the aluminum precipitation stage. Even though a combined 83 percent of the total iron content was rejected in the first two stages, downstream processes were significantly impacted in terms of precipitation efficiency and purity due to the significantly higher concentration of iron in the system. Similarly, a combined 77 percent of the total REEs were removed in the leaching solid residue and Al cake, leaving only 23 percent for the downstream processing. Aside from REEs, the loss of Ni and Zn was relatively higher than other critical metals. However, as the recovery of Ni and Zn was also high, most of the elemental content still reported to the RE-CM cake. Overall, only 23 percent of the total REEs in the system were recovered in the RE-CM cake, indicating significant losses in the previous stages. Recovery of Co, Ni, Mn and Zn was 54%, 47%, 61%, and 57%, respectively. Iron recovery in the RE-CM stage was 17%, significantly higher than previous experiments where only 1-2 percent of the total iron content reported to the RE-CM precipitation.

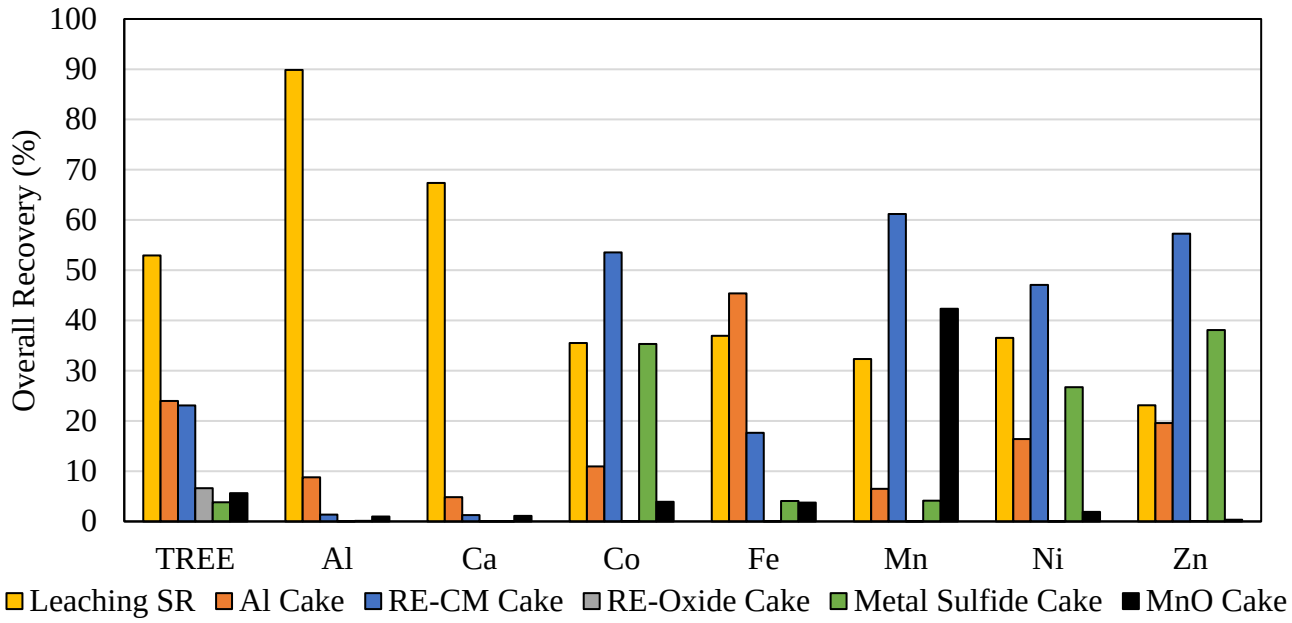



Figure 3.12.33 Overall recovery of different elements in the cakes generated in the pilot scale test.

Due to the lower precipitation efficiency in the oxalic acid precipitation stage, only 7 percent of the total REEs were recovered in the RE-Oxide cake (Figure 3.12.33). Despite the losses, the REE product still contained 85% RE-Oxide content (Figure 3.12.34). Contrary to the previous experiments where calcium was the primary contaminant, the RE-Oxide product generated in this test also contained higher iron content. The calcium and iron content in the RE-oxide was 8 and 4 percent, respectively (Figure 3.12.34). As the iron precipitation during oxalic acid treatment occurs in the ferrous form(Nawab et al., 2022a), elevated contamination supports the conclusion drawn previously that at least a portion of iron in the bio-acid was in ferrous form. It should be noted here that the ferrous ions can be converted to ferric ions during the leaching of RE-CM cake by increasing the mixing time of the reaction. This conversion of ferrous to ferric can be observed through the change in color of the slurry during the reaction.

In spite of high CM precipitation efficiency, the metal sulfide cake only recovered 35% Co, 26% Ni and 38% Zn due to their lower leaching efficiency in the redissolution stage (Figure 3.12.33). The metal sulfide product contained 81% percent iron with only 17 percent content of critical metals. Similarly, the Mn oxide cake was comprised of 73% Fe, 9% Ca, and 6% Al with only 10% Mn content. The results demonstrated that the iron content not removed during the earlier precipitation stages caused contamination in the metal sulfide and manganese precipitation stage whereas Al primarily contaminated the manganese product. As such, it is crucial to maximize the removal of both iron and aluminum in the contaminant removal stages to not only maximize the product purity but also to reduce the chemical consumption cost in the final precipitation stages.

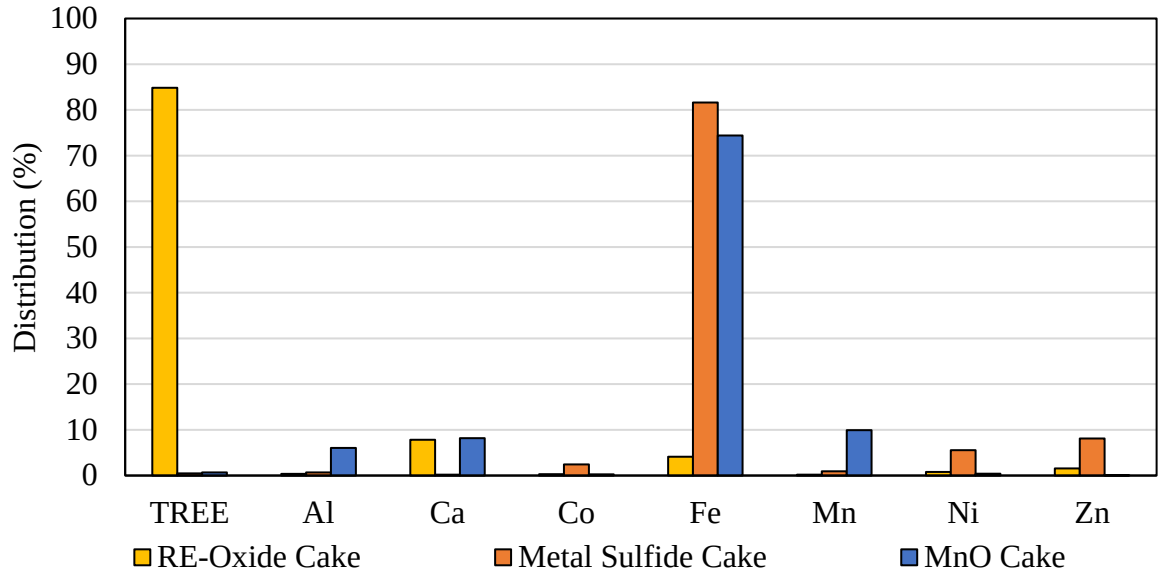



Figure 3.12.34 Elemental distribution in the final products produced in the pilot scale experiment.

### *3.12.3 Circuit Arrangement-5*

The experiments conducted using circuit arrangement-4 revealed contamination of metal sulfide products through calcium precipitation as calcium oxalate. Since the metal sulfide precipitation occurred at pH 3.0, raising the solution pH increased the oxalate anion content and reduced the selectivity of oxalic acid, resulting in calcium oxalate precipitation. As such, circuit arrangement 4 was amended to include a calcium precipitation stage prior to metal sulfide precipitation. Overall, the circuit arrangement was similar to the previous circuit with an addition of a calcium precipitation stage after oxalic acid precipitation. The new process schematic is shown in Figure 3.12.35.

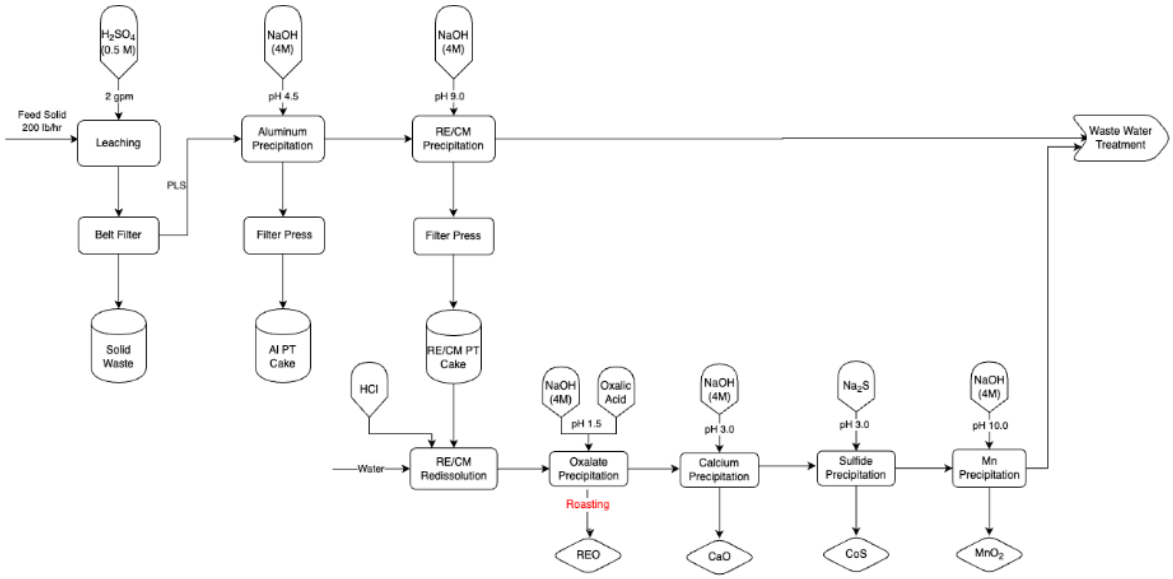



Figure 3.12.35 Process schematic for the circuit arrangemet-5 for the pilot scale testing.

**Pilot Scale Experiment:** The modified circuit arrangement was applied on the Baker seam coarse refuse material, which was the primary feedstock in this project. The refuse material was calcined at 600°C to decarbonize the coal, dehydroxylate the clays such as kaolinite and

illite and liberate the REE-bearing minerals which were subsequently decomposed into an acid soluble form. It was determined that the light REEs in the Baker seam material were primarily associated with the crandallite group minerals and monazite. Contrarily, heavy REEs were associated with xenotime and zircon(Ji, Li, & Zhang, 2022b). Thermal treatment at elevated temperatures can decompose the crandallite group minerals and accomplish elevated REE recoveries. However, monazite, xenotime and zircon have high thermal stability, therefore, thermal treatment is insufficient for their decomposition. As such, techniques such as acid baking and alkaline cracking are employed to ensure high decomposition efficiency of these minerals ((Demol et al., 2019; Jha et al., 2016; Sadri et al., 2017)). As the Baker seam coarse refuse material contained substantial REE content associated with difficult to leach minerals, the REE recovery, especially for HREEs, was anticipated to be significantly lower than the previous tests.

The Baker seam coarse refuse material was thermally treated in a tube roaster after mixing with perlite to aid the solid flow in the tube. As perlite is thermally stable and inert in leaching, no additional contamination was observed in leaching during the bench-scale experiments. The material was roasted, stored in 50-gallon barrels, and subsequently used for the test. The screw feeder was recalibrated for the Baker seam material to provide the desired solids flowrate. Four distinct pilot scale experiments were performed on the Baker seam coarse refuse material using different leaching conditions. All the tests were conducted using a 20% S/L (w/v) ratio at 75°C instead of ambient temperature conditions due to the better leaching characteristics of REEs at elevated temperatures. In the first two pilot experiments, 0.5M sulfuric acid was used as lixiviant whereas the residence time was 45 minutes and 5hrs., respectively. Longer leaching times were investigated to test the hypothesis of light REE precipitation through isomorphic substitution with gypsum. The last two experiments used bio-acid of equivalent 0.5M sulfuric acid strength. The residence time in the first and second bio acid experiments was 5 and 2.25 hrs., respectively. Higher leaching times were achieved by increasing the number of tanks and reducing the feed flowrates. Since the material was treated at 600°C, clay dehydroxylation and pyrite decomposition were anticipated to produce a PLS rich in iron and aluminum. As elevated iron concentrations are problematic in solvent extraction. Therefore, an iron precipitation stage was added to the circuit flowsheet for Fe removal before solvent extraction. The process flowsheet for the Baker seam experiments is shown in Figure 3.12.36.

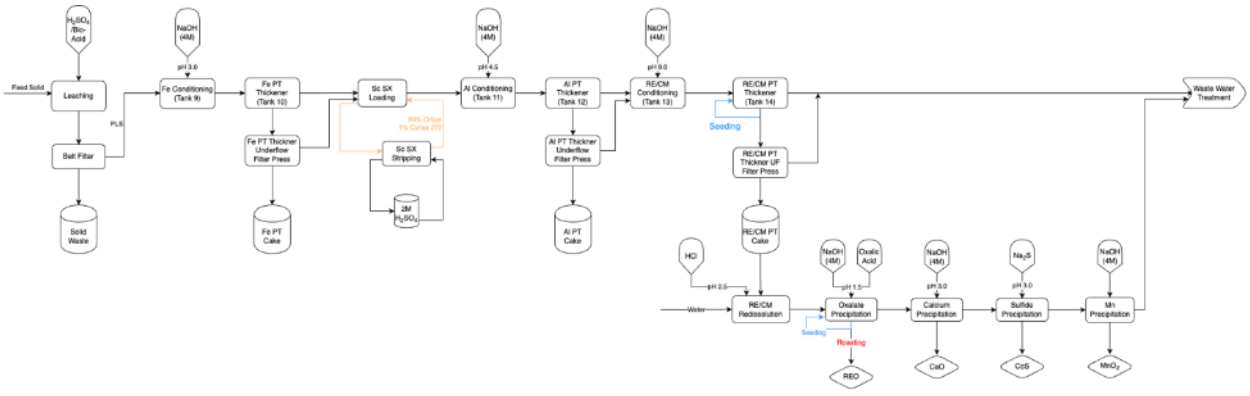



Figure 3.12.36 Circuit schematic employed for the treatment of Baker seam coarse refuse in the pilot scale experiments.

**<u>Baker seam Experiment Run-1 (WK-1)</u>:** The first pilot scale experiment was performed using the same leaching conditions as previous experiments. It was determined that REE recovery in the leaching stage was only 26 percent while Co, Mn, Ni and Zn recovery was 13%, 53%, 16%, and 26%, respectively (Figure 3.12.37). Elevated Mn recovery from the Baker seam material compared to other elements was likely the reason for elevated Mn content in the

PLS generated from the leaching of Baker seam coarse refuse material (Table 3.12.5). Interestingly, the recovery of contaminants such as Al, Ca and Fe was only 5%, 16% and 4%, respectively. Lower recovery of the REEs, CMs, and contaminants compared to the bench-scale studies reported previously was likely due to the particle size of the solids. The lab-scale studies were performed on the material ground to the top size of 180 µm whereas the baker seam material used in the pilot scale study was crushed to a top size of 1 mm. The elemental content in the resulting leachate solution is shown in Table 3.12.11. It is evident that the REE and CM content orders of magnitude lower than the Al, Fe and Ca content combined. Trace concentrations of Co, Ni and Zn in leachate demonstrated the lower concentrations of these elements in the Baker seam coarse refuse material compared to Shakespeare material.

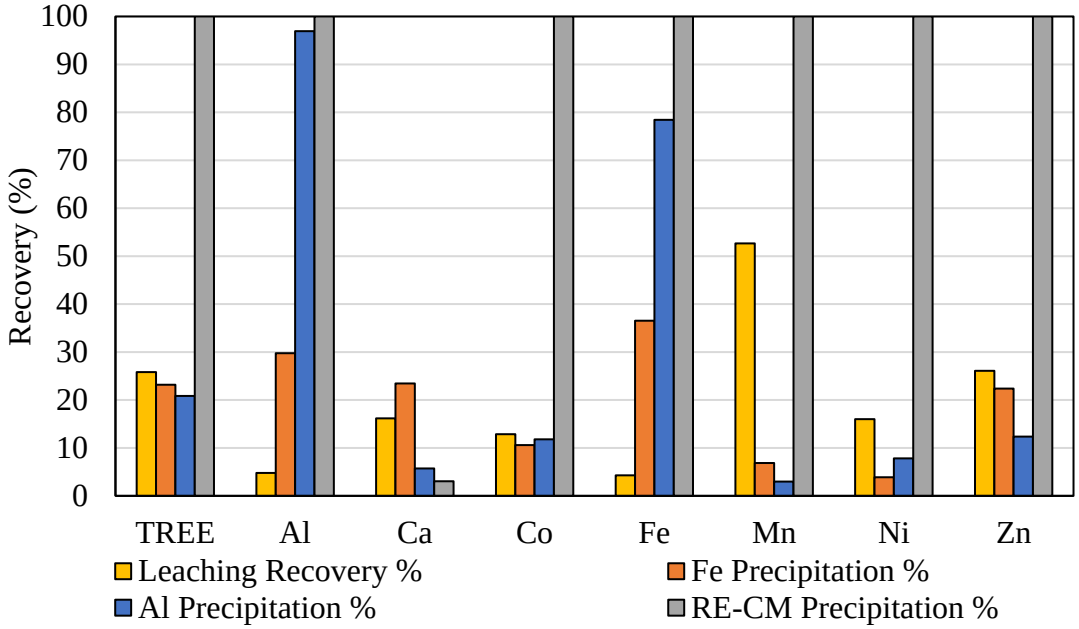



Figure 3.12.37 Percent recovery of various elements of interest in the different treatment stages during the pilot scale testing.

Table 3.12.11 Concentrations (ppm) of various elements in the PLS generated through leaching of PLS.

| | **TREE** | **Al** | **Ca** | **Co** | **Fe** | **Mn** | **Ni** | **Zn** |
|---|---|---|---|---|---|---|---|---|
| **Leaching-PLS** | 15.2 | 1093.3 | 681.7 | 0.6 | 385.9 | 20.6 | 1.7 | 6.1 |

The iron precipitation stage recovered 37% iron, 30% Al and 23% Ca (Figure 3.12.38). It is interesting to note that iron precipitation efficiency is significantly lower than the previous experiments. This finding suggested that a significant concentration of iron was present in the ferrous form, which precipitates in the circum-neutral to basic pH ranges. REE recovery in the iron removal stage was 23%, along with some losses of critical metals. The removal of REEs, CMs, and calcium in this stage was likely due to entrainment. Filtrate from the iron precipitation stage was sent to the solvent extraction circuit. It was determined that the Sc removal efficiency from the PLS in this test was 94 percent whereas only 0.3 percent Fe was loaded on the organic phase. As ferrous iron does not load onto the organic phase, this finding corroborated the previously drawn conclusion that iron primarily existed as ferrous ions. The PLS was subsequently treated for Al removal at pH 4.5, where the Al and Fe recovery was roughly 97 and 79 percent, respectively. Similar to the Fe precipitation stage, there were slight

losses of REEs and CMs in the Al precipitation stage. The remaining REEs, CM, Fe and Al content was recovered in the RE-CM cake with a small recovery of calcium. Elemental content in the RE-CM cake and the subsequent leachate is shown in Table 3.12.12. It is evident that the REE and CM content has been upgraded 3-6 folds, whereas the relative concentrations of the contaminants have dropped significantly. As the significant iron content was present as ferrous ions, the RE-CM cake contained elevated iron concentrations. Similarly, the Al bypassing the earlier precipitation stages was recovered in the RE-CM cake.

Table 3.12.12 Concentration (ppm) of various elements in the RE-CM cake (dry, whole mass basis) and PLS generated from the leaching of RE-CM cake.

| | **TREE** | **Al** | **Ca** | **Co** | **Fe** | **Mn** | **Ni** | **Zn** |
|---|---|---|---|---|---|---|---|---|
| **RE-CM Cake** | 7915.3 | 21799.8 | 8803.0 | 445.5 | 48432.1 | 17150.1 | 1351.1 | 3818.7 |
| **RE-CM PLS** | 61.2 | 175.7 | 99.9 | 3.5 | 209.9 | 105.8 | 10.4 | 22.6 |

The optimum dosage of oxalic acid and sodium sulfide was determined in the same manner as the previous experiments. It was found that the REE recovery in the oxalic acid precipitation stage was 96 percent, whereas the recovery of all other elements was less than 3 percent (Figure 3.12.38). As the recovery of contaminants was minuscule, high purity RE-Oxide product was anticipated in this experiment. The solution pH of the filtrate from the oxalic acid precipitation stage was raised to 3.0 to remove calcium present in the solution. As described previously, increasing the solution pH raises the oxalate anion availability in the solution and reduces the selectivity of oxalic acid, resulting in the precipitation of calcium oxalates. The calcium recovery in this stage was 74 percent whereas REE recovery was 64 percent. As most of the rare earth elements were precipitated in the oxalic acid precipitation stage, no significant losses to the calcium oxalate cake were anticipated. Subsequently, the raffinate was treated with sodium sulfide to precipitate Co, Ni and Zn at pH 3.0. It was found that zinc precipitation efficiency was over 95 percent. However, the recovery of Co and Ni was only 18 and 10 percent, respectively. It has been shown in the bench scale data that zinc preferentially consumes sulfide ions. As such, it is likely that elevated Zn precipitation efficiency was driven by its relatively higher content and its preferential consumption of sulfide ions. Due to the earlier calcium precipitation stage, Ca recovery in the sulfide precipitation stage was only 9 percent. Additionally, the iron and Al recovery in this stage was only 9 and 0.1 percent, respectively (Figure 3.12.38). Finally, the manganese precipitation stage recovered the remaining elemental content in the solution (Figure 3.12.38).

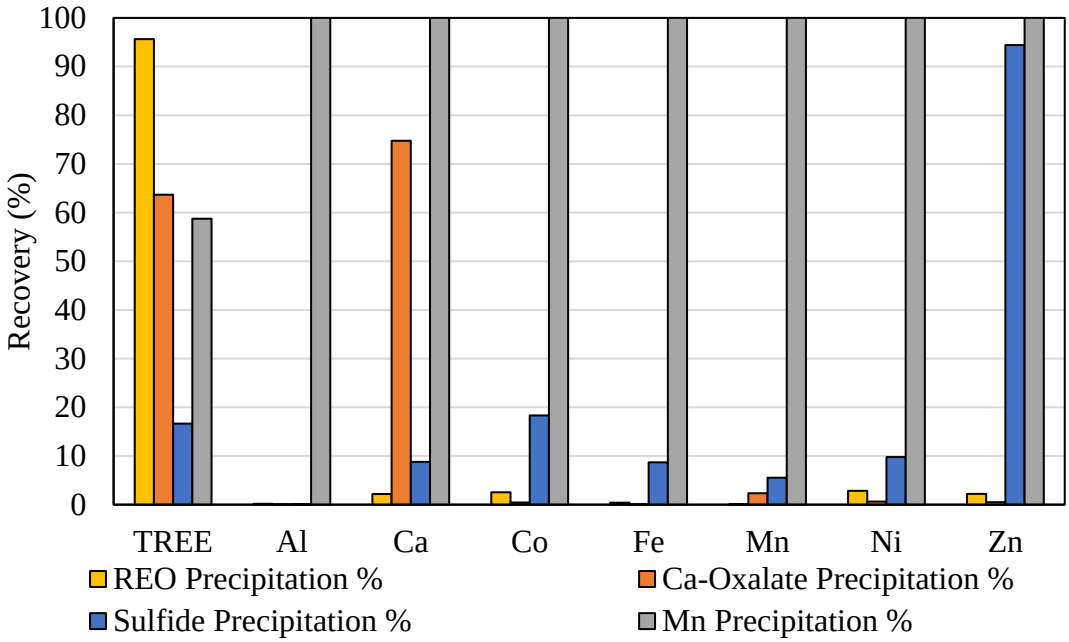



Figure 3.12.38 Precipitation efficiency of different elements in the final product precipitation stages during the pilot scale testing.

The overall recovery data demonstrated the poor leaching characteristics of the material with respect to the REEs (Figure 3.12.39). As explained previously, association of REEs with difficult-to-leach minerals such as monazite, xenotime and zircon coupled with the 1mm particle size used in the leaching was the primary reason for poor leaching characteristics. Even though most of the Al, Ca and Fe content was rejected to leaching solid residue cake, the PLS contained elevated content of these elements due to their significantly higher initial concentrations. The Al and Fe precipitation stage removed roughly 4 and 5 percent of the total Fe and Al in the system, respectively and only 0.1% Al and 0.6% Fe precipitated with the RE-CM cake. Similarly, approximately 10%, 8%, 2%, 5%, and 3% of the total REEs, Zn, Ni, Mn, and Co were lost to the Al and Fe cake, respectively (Figure 3.12.39). The RE-CM cake only recovered 15% of the total REEs whereas Co, Ni, Mn, and Zn recovery was 10%, 48%, 14%, and 18%, respectively (Figure 3.12.39). The oxalic acid cake recovered only 13 percent of the total REEs while the recovery of other elements was less 1 percent, signifying high selectivity of the precipitating agent. The oxide product contained 93 percent REE-Oxide with calcium as the primary contaminant in the product (Figure 3.12.40). It should be noted here that even though iron was identified to be in the ferrous form in the solid leaching stage, significant iron contamination of the oxalate product did not occur. This corroborates the claim that ferrous ions can be converted to ferric ions by increasing the residence time of the RE-CM cake leaching. The calcium precipitation stage recovered most of the leached calcium content and the distribution data showed that the subsequent cake contained more than 90 percent calcium oxalate (Figure 3.12.40).

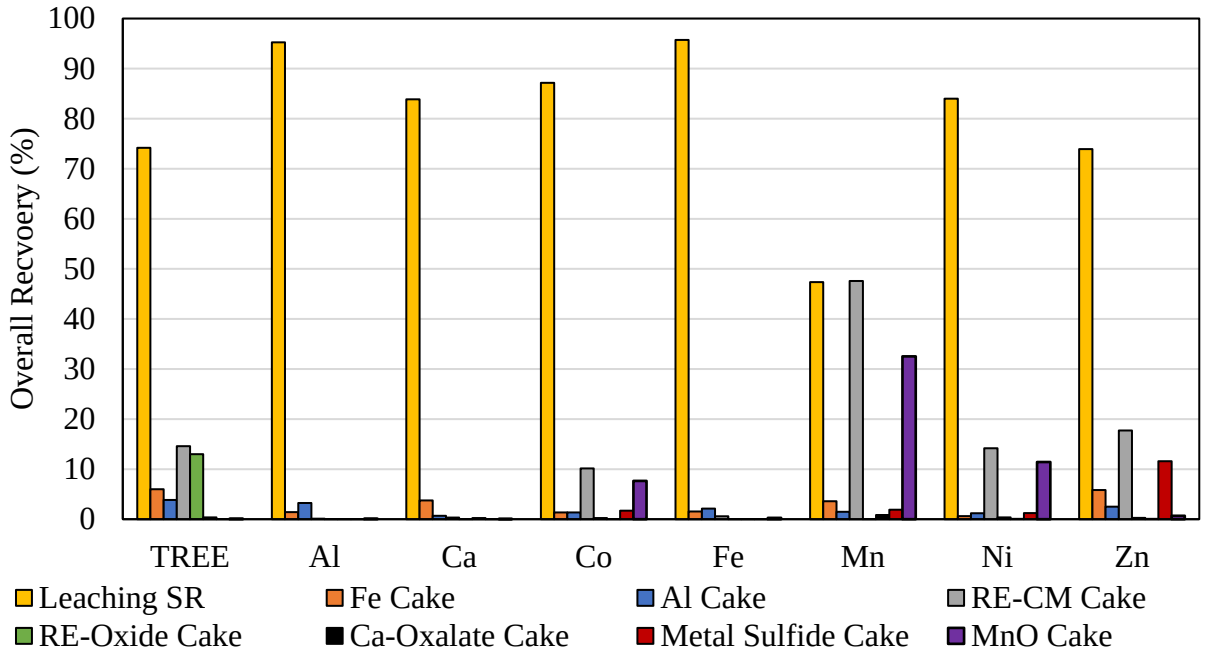



Figure 3.12.39 Overall recovery of various elements in the cakes generated during the pilot experiment.

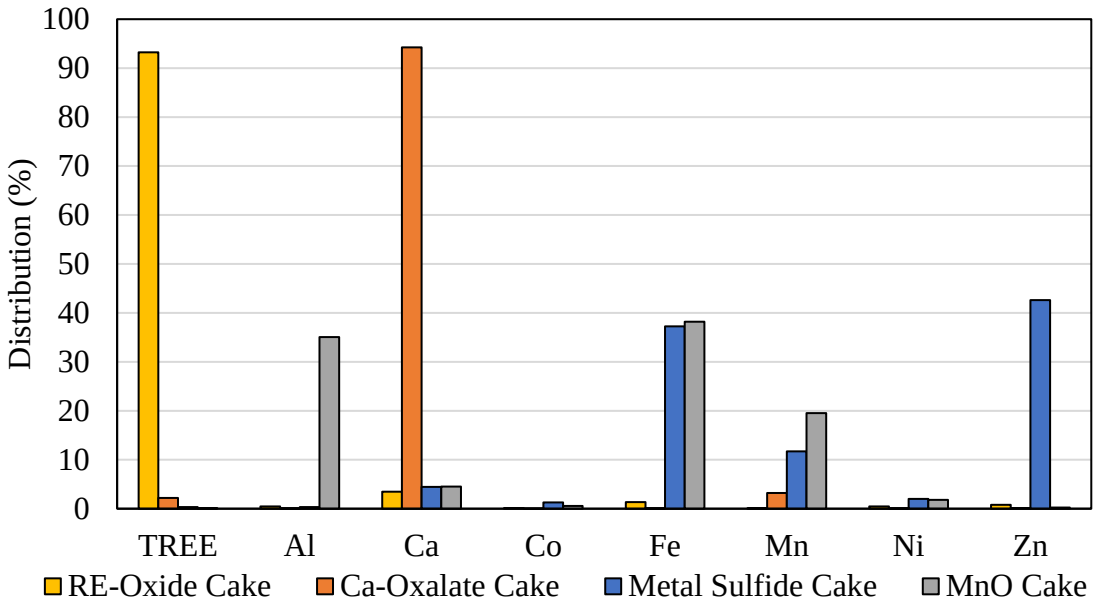



Figure 3.12.40 Distribution of different elements in the final product cakes generated in the pilot experiment.

As recovery of Co and Ni in the sulfide precipitation was low, only 2 percent of the total Co and 1% Ni was recovered in the metal sulfide cake whereas the overall zinc recovery was 12 percent. Since most of the Co and Ni content bypassed to Mn circuit, the remaining content precipitated with Mn product. Because Co and Ni precipitation in the Mn would be considered a contamination, therefore, it is crucial to identify the optimal chemical dosages with utmost prudence. As for the product grade, it was found that the metal sulfide cake contained 38 percent iron, 5 percent calcium and 43 percent zinc. The Co and Ni content was only 1 and 2 percent, respectively. Finally, the manganese product recovered 33% of the overall manganese. However, as the remaining Al, Fe, Ca, and other critical metals also precipitated during the

stage, the product grade was impacted. It was determined that Fe, Al, and Ca accounted for roughly 77 percent of the total mass in the manganese cake. The Mn content was approximately 20% whereas Co and Ni content was 0.6% and 2%, respectively. Overall, this circuit arrangement showed discouraging recoveries of REEs and CMs primarily due to the significantly lower REE recoveries in the leaching stage. However, this flowsheet can still be employed for REE and CM extraction from the sources demonstrating favorable leaching characteristics.

**Baker Seam Experiment Run-2 (WK-2):** The second pilot scale experiment, denoted WK-2, was also performed using 0.5M sulfuric acid but at different solids and liquid feeding rates to provide 5 hrs residence time for leaching. The process flow and flow rates in the leaching and iron precipitation stage in this experiment are shown in Figure 3.12.41. The solids and acid were added at 120 lb/hr and 1.25 gpm to Tank-2, overflowing into Tank-3. The tank-2 underflow was pumped at 50 gpm and added to tank-3. Overflow from tank-3 was filtered whereas the underflow was recycled into tank-1, where the 0.5M sulfuric acid solution was added at 1.25 gpm. The underflow from tank-1 was filtered and the overflow was fed to tank-2. The raffinate from filtration was transferred to tank-2 at 1.25 gpm for iron precipitation which was subsequently conducted in the same manner as the previous experiments only at a different flow rate.

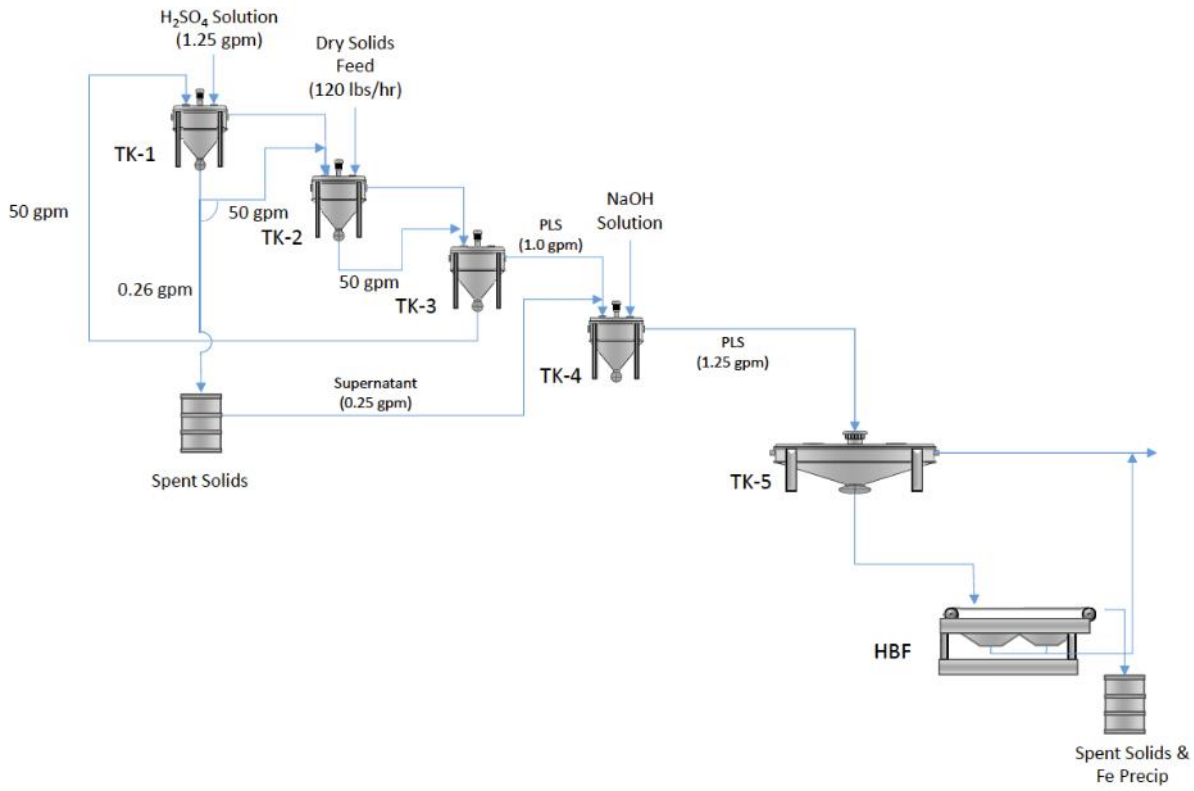



Figure 3.12.41 Leaching and iron precipitation process schematic and flowrates for the long leaching pilot scale test.

The REE leaching recovery in this experiment was 32 percent, 9 absolute percentage points higher than the previous experiment (Figure 3.12.42). Recovery of Co, Ni, Mn and Zn was 14%, 64%, 21%, and 30%, respectively, which was also higher than the previous experiment. Unfortunately, the recovery of other contaminants such as Fe, Al and Ca also increased significantly. The improvement in leaching characteristics of the material was likely a result of higher leaching time, which increased the decomposition of REE and CM bearing minerals as

well as clays and pyrite. The iron precipitation step removed 42 percent Fe and 40 percent Al as well as 23% REEs, 15% Co, 10% Mn, 22% Ni, and 22% Zn (Figure 3.12.42). A relatively higher loss of Al, Fe and CMs in this stage was likely a result of the elevated solution pH. The designed solution acidity level for this experiment was 3.1-3.3, however, the average pH during the experiment was 3.6. Hence, higher elemental losses than the previous experiment were observed. The aluminum precipitation removed 90% Al and only 46% Fe along with minute losses of REEs and other critical metals. Lower than anticipated iron recovery in this stage was consistent with the previous experiment due to the presence of iron as ferrous ions in the solution. However, the iron recovery in the earlier test during the Al precipitation stage was significantly higher than this experiment. One contributing factor may be the higher pH level in the iron removal step that removed the iron content that otherwise would have precipitated in the Al precipitation stage. It is interesting to note the relatively higher loss of REEs and other critical metals during the iron precipitation stage relative to the aluminum precipitation (Figure 3.12.42). The raffinate from Al removal step was treated for RE and CM precipitation at pH 9.0. Any iron and aluminum bypassing the initial precipitation stages were also recovered in the RE-CM cake.

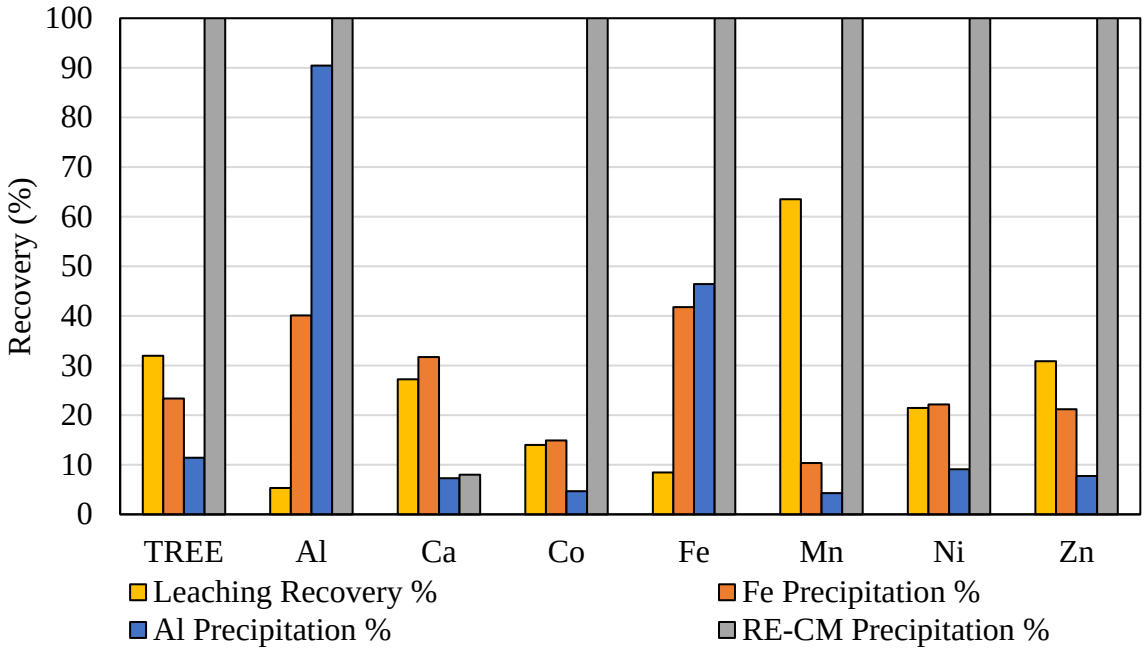



Figure 3.12.42 Recovery of various elements in different precipitation stages of the pilot scale test.

Due to the lower removal efficiency of iron and aluminum in this test, the concentration of both contaminants in the RE-CM cake was significantly higher compared to the previous test (Table 3.12.13). Therefore, PLS generated through leaching at pH 2.5 also contained elevated iron and aluminum concentrations (Table 3.12.13). Despite the earlier process inefficiencies, the PLS had roughly 3 times higher REE and CM concentrations, whereas the contaminant content decreased 3.5-4.5 folds. Similar to previous experiments, the leachate samples were collected for bench-scale experimentation to identify the optimum chemical dosage.

Table 3.12.13 Elemental content (ppm) in the RE-CM cake and the PLS generated from the leaching.

| | **TREE** | **Al** | **Ca** | **Co** | **Fe** | **Mn** | **Ni** | **Zn** |
|---|---|---|---|---|---|---|---|---|
| **RE-CM Cake** | 5455.9 | 25274.3 | 14873.0 | 195.6 | 92210.1 | 11474.3 | 527.8 | 1964.2 |

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| **RE-CM PLS** | 41.0 | 237.7 | 150.1 | 1.5 | 155.5 | 89.7 | 3.8 | 15.5 |

Rare earth element recovery in the oxalic acid stage was 94 percent whereas Ca, Co and Ni recovery was 2%, 1%, and 1%, respectively (Figure 3.12.43). Interestingly, the calcium precipitation stage only recovered 20% calcium, considerably lower than the previous test. This anomaly was likely due to insufficient residence time in the mixing tank. The REE recovery was 63 percent, which was an anticipated outcome. The mechanism for precipitation has been explained in earlier experiments. As most calcium was still present in the solution, it precipitated in the sulfide precipitation stage, contaminating the product.

Interestingly, the CM precipitation in this experiment was also substantially impacted. The Co and Ni recovery was merely 26% and 25%, respectively (Figure 3.12.43). Similar to the previous experiment, Zn recovery was significantly higher than cobalt and nickel. It is likely that lower concentrations of cobalt and nickel in the Baker seam PLS compared to Shakespeare material PLS shifted the optimum precipitation pH of these elements. Finally, the filtrate from the sulfide precipitation stage was processed for manganese precipitation and over 90 percent recoveries were obtained for the REEs, CMs and contaminants.

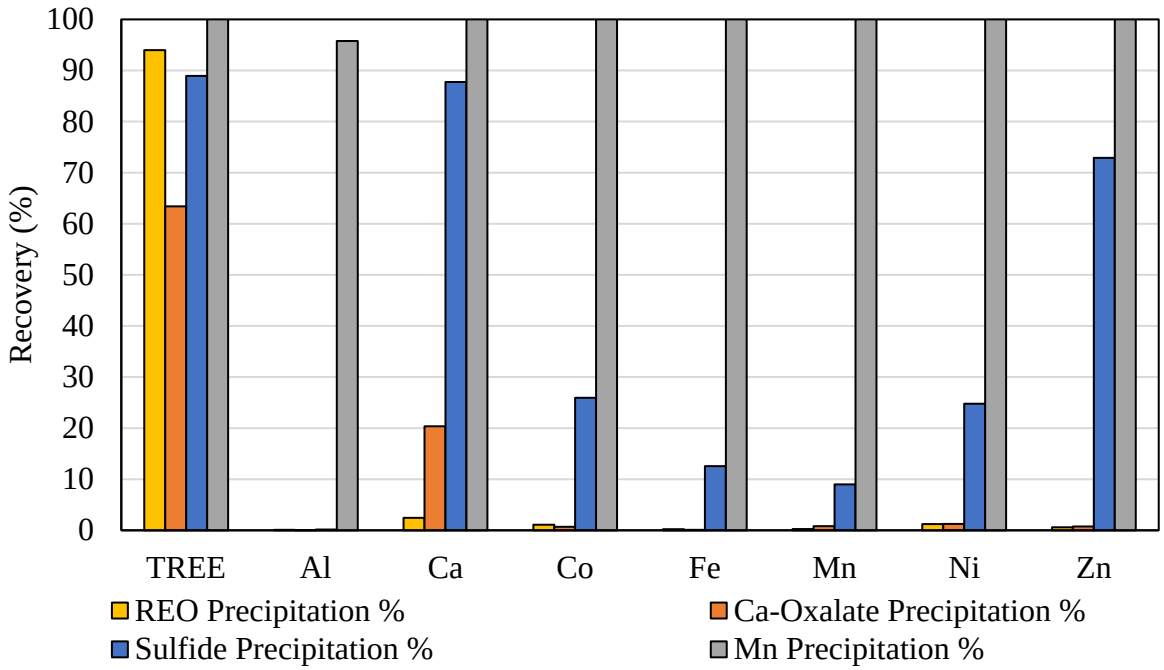



Figure 3.12.43 Precipitation efficiency of various elements in the different final product precipitation stages during the pilot scale experiment.

Similar to the previous experiment, most of the REE, CM, and contaminant content was lost to the leaching solid residue due to the poor leaching characteristics of the material (Figure 3.12.44). The Al and Fe precipitation stages collectively removed 5-6% of the total Al and Fe as well as 10% REEs, 3% Co, 9% Mn, 6% Ni and 8% Zn. Due to the process inefficiencies in earlier stages, the RE-CM cake recovered only 21% of the total REEs and 11% Co, 15% Ni and 22% Zn (Figure 3.12.44). Manganese recovery was the highest (~55%) mainly due to its better leaching efficiency. The RE-Oxide cake only recovered 16% of the total REEs which was still 3 absolute percentage points higher than the previous run. The rare earth oxide product was high purity with approximately 90 percent REE and 9 percent calcium content (Figure 3.12.45). Lower iron contamination should also be noted in this test. Additionally, it was found that the leaching time did not cause a significant change in the distribution of light and heavy REEs. In the WK-1 test, LREE and HREE distribution was 94% and 6% but 95% and 5% in

the WK-2 (this experiment), respectively. Hence, disproving the hypothesis that long leaching times would remove light REEs from PLS through isomorphic substitution with gypsum. Furthermore, the calcium concentration in the leachate generated from the short and long leaching time was 681 ppm and 690 ppm, respectively. Since gypsum formation would cause a drop in calcium content, similar concentrations suggested that gypsum formation behavior did not change in both experiments. Therefore, no difference in the RE-oxide cake composition was observed.

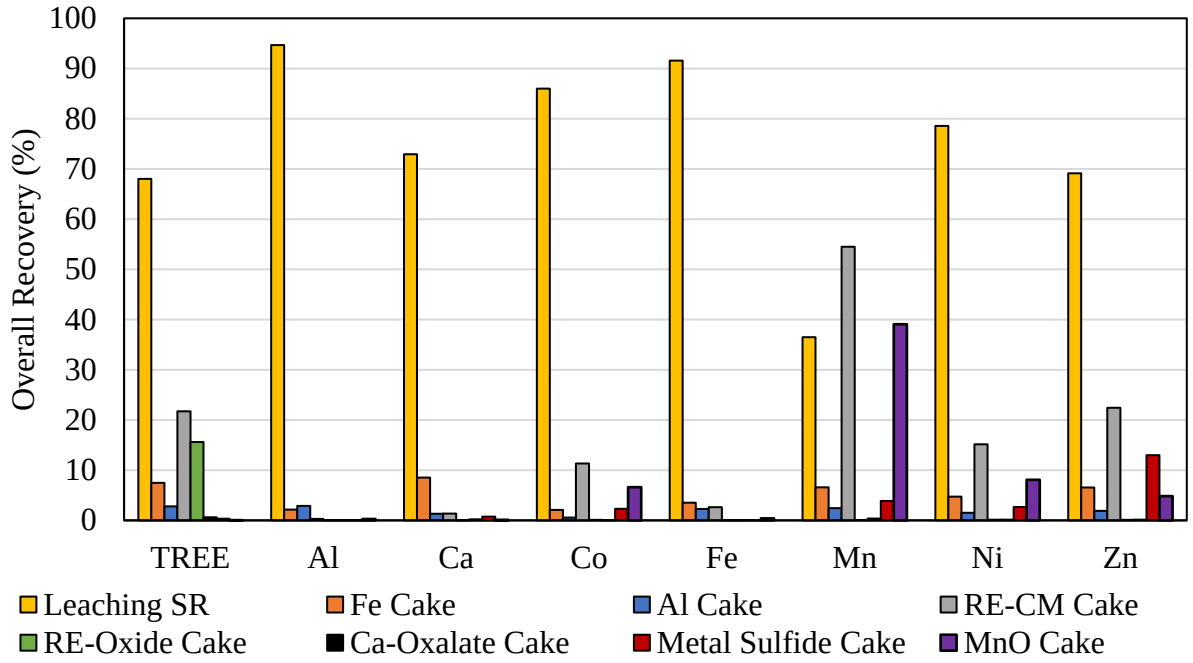



Figure 3.12.44 Overall recovery of different elements in the cakes generated in pilot scale experiment.

The calcium oxalate cake comprised of 92% calcium and 5% REE content. Due to the lower precipitation efficiency of calcium in its precipitation stage, the metal sulfide cake was contaminated with Ca and Fe, accounting for roughly 85 percent of the total mass (Figure 3.12.45). Unfortunately, zinc content was only 8 percent and the remaining mass was made up of manganese, which was a contaminant in this stage. The overall recovery of Co, Ni and Zn was merely 2%, 4%, and 13% in the metal sulfide cake whereas 7%, 8%, and 5% in the manganese cake, respectively (Figure 3.12.44). As some of the manganese was lost to the sulfide cake, overall Mn recovery was 39% in its cake. Similar to the previous test, Al and Fe content accounted for 78 percent of the total mass with only 17% Mn content (Figure 3.12.45).

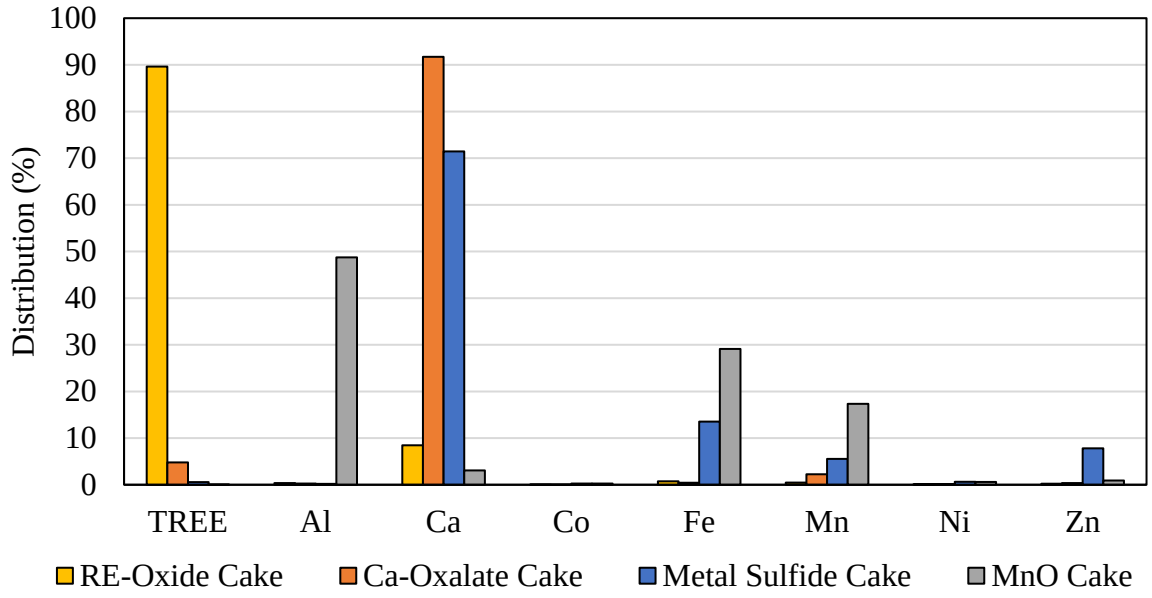



Figure 3.12.45 Elemental distribution in the final cakes obtained from pilot scale experiment.

**Baker Seam Experiment Run-3 (WK-3):**

The WK-3 test was conducted using bio-acid as a lixiviant generated through the bacterial decomposition of pyrite in the presence of nutrients. The mechanism for the acid formation has been discussed in the earlier sections. Similar to the previous experiment, this was a long leaching time test with the same process flow as shown in Figure 3.12.41. Molar strength of the acid was determined to be equivalent to 0.09M sulfuric acid. Elevated iron concentration was observed in the bio-acid test presented in 0. Surprisingly, the bio-acid used in this test contained more than double the iron content observed previously, i.e., 14621 ppm (Table 3.12.14). It is interesting to note that the calcium content dropped in this batch, likely due to gypsum formation. As bacterial decomposition of pyrite generated sulfate ions, elevated iron content was symbolic of higher pyrite decomposition. As such, in the presence of higher sulfate ions, gypsum formation likely increased considerably, resulting in calcium loss. The concentrations of REEs and CMs were also higher than the previous bio-acid. It was concluded that high-strength bio-acid can be produced through feeding of more pyrite and nutrients in the bio-reactor. However, as acid generation is directly associated with pyrite decomposition, higher acid strength would result in elevated iron content in the leachate, potentially creating techno-economic challenges in the downstream processes such as higher REE losses as well as higher chemical cost. As such, a detailed cost-benefit analysis needs to be conducted to identify the optimal operating points for bio-reactor operations.

Table 3.12.14 Elemental content (ppm) in the bio-acid used for the leaching in the WK-3 pilot scale experiment.

| | TREE | Al | Ca | Co | Fe | Mn | Ni | Zn |
|---|---|---|---|---|---|---|---|---|
| **Bio-Feed** | 2.2 | 36.7 | 466.3 | 2.0 | 14621.4 | 53.0 | 7.3 | 76.5 |

The REE leaching recovery in this experiment was 36%, 4 absolute percentage points higher than the previous test. Similarly, Co, Ni, Mn, and Zn recovery was 40%, 74%, 56%, and 65%, respectively. For reference, recovery of these elements in the previous experiment was 14, 64%, 21%, and 31%, respectively. Aside from the REEs and CMs, the recovery of iron and aluminum

also increased with no impact on calcium recovery. Better leaching recovery in this experiment was likely due to the REE content present in the bio-acid. Interestingly, iron content in the PLS decreased significantly compared to its initial concentration in the bio-acid (Table 3.12.15, Table 3.12.14). As the average pH during the leaching operation was 1.8, iron precipitation was likely due to the jarosite formation during the leaching. This hypothesis was supported by the drop in K content from 63 ppm to 29 ppm in the bio-acid and PLS, respectively.

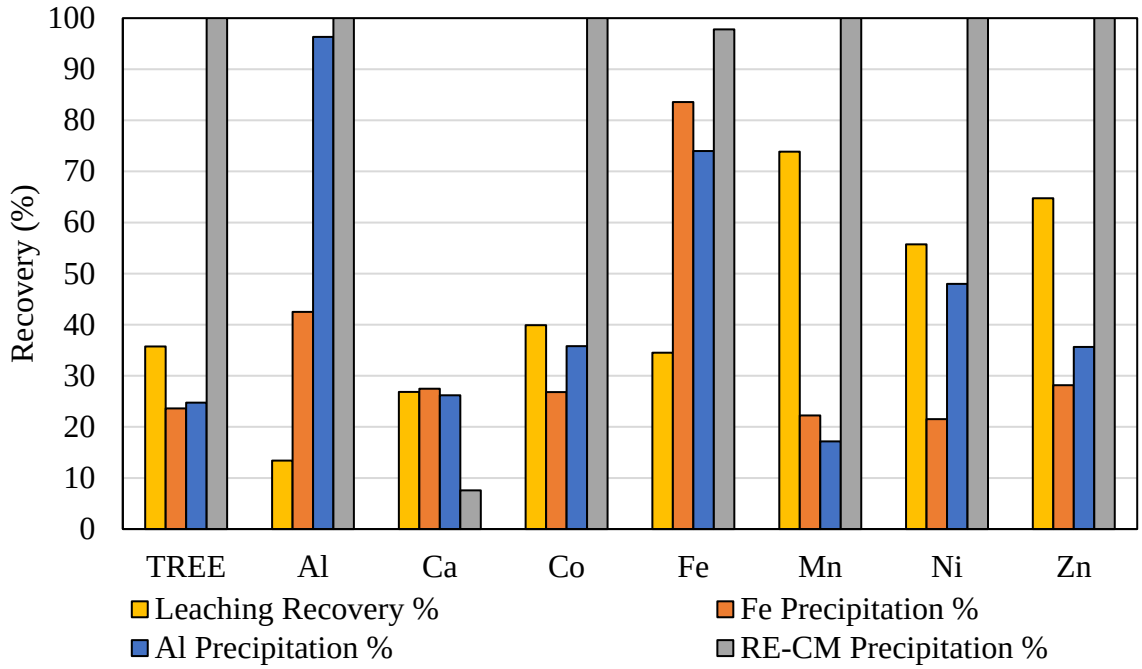



Figure 3.12.46 Elemental recoveries for various precipitation stages in the pilot scale experiment.

Table 3.12.15 Elemental content (ppm) in the leachate solution generated from the bio-acid leaching of Baker seam coarse refuse material.

| | TREE | Al | Ca | Co | Fe | Mn | Ni | Zn |
|---|---|---|---|---|---|---|---|---|
| **Leaching-PLS** | 19.8 | 2477.4 | 863.7 | 2.3 | 8053.6 | 62.8 | 9.0 | 66.8 |

The iron removal step recovered 84% Fe and 43% Al along with 24% TREEs, 27% Co, 22% Mn, 22% Ni, and 28% Zn (Figure 3.12.46). Interestingly, the pH of the iron mixing tank was, on average, 2.6 throughout the test. Considering the low pH range, recovery of all the elements was significantly higher than anticipated based on the previous results. As iron precipitation from PLS starts around pH 1.8 as jarosite, the iron loss was likely as jarosite and iron hydroxide. REEs and CMs likely co-precipitated with iron due to its elevated concentrations. The aluminum precipitation stage recovered 96% Al and 74% Fe as well as 25% TREEs, 36% Co, 17% Mn, 48% Ni, and 36% Zn. Due to high REE and CM elemental losses in the iron and aluminum stages would result in significantly lower recovery in the RE-CM cake. Furthermore, as iron content was significantly higher than the previous tests, removal inefficiencies in the iron and aluminum precipitation steps would translate into even greater concentrations in the RE-CM cake (Table 3.12.16). The concentrations of different elements in the leachate solution produced by leaching at pH 2.5 is shown in Table 3.12.16. The previous experiments reported 3-4 times upgradation in RE-CM PLS content. However, REE and CM concentration in this test increased by only 1-1.7 factor. However, the contaminant content decreased 15-26 folds.

Table 3.12.16 Elemental content (ppm) in the RE-CM cake (dry, whole mass basis) and the PLS generated from its leaching during the pilot scale test.

| | TREE | Al | Ca | Co | Fe | Mn | Ni | Zn |
|---|---|---|---|---|---|---|---|---|
| **RE-CM Cake** | 2125.6 | 9791.0 | 3966.4 | 203.2 | 62627.8 | 7542.1 | 685.5 | 5754.8 |
| **RE-CM PLS** | 30.1 | 154.1 | 58.5 | 2.6 | 94.9 | 108.1 | 7.9 | 73.0 |

The oxalic acid precipitation recovered over 85% REEs whereas recovery of Ca and Fe was 12% and 4%, respectively (Figure 3.12.47). Surprisingly, the recovery of critical metals was significantly higher than the previous tests. It was found that the Co, Ni, and Zn recovery was 16%, 14% and 19%, respectively. It is an interesting finding because the CMs were insolubilized previously using sodium sulfide, notably high recoveries in the oxalic acid stage indicated a possible interaction between oxalate anions and CMs under some special conditions. This statement is based on the fact that oxalic acid has been used for the precipitation of Co, Ni and Mn, albeit in much higher concentrations than the current solution content (Gerold et al., 2020). As zinc concentration was almost twice the REE content, even a small percentage would result in significant contamination. The oxalic acid raffinate was treated with sodium hydroxide at pH 3.0 for calcium precipitation. It was determined that the calcium and REE recovery was approximately 80 percent whereas the recovery of other elements was minute (Figure 3.12.47). Surprisingly, even with the high oxalate content at elevated pH levels, CM recovery did not increase. A systematic study would be required to understand the precipitation mechanism of CM extracted using bio-acid. The sulfide precipitation stage reported 76%, 60%, and 99% recovery of Co, Ni and Zn, respectively. Interestingly, the REE and Ca recovery dropped to only 15% in this stage (Figure 3.12.47). The iron recovery was 78%, which resulted in significant product contamination. Another interesting outcome in this step was 38% Mn recovery, despite its relatively lower recoveries in the previous stages. Subsequently, the REEs, CMs, and other contaminants were recovered in the manganese precipitation stage.

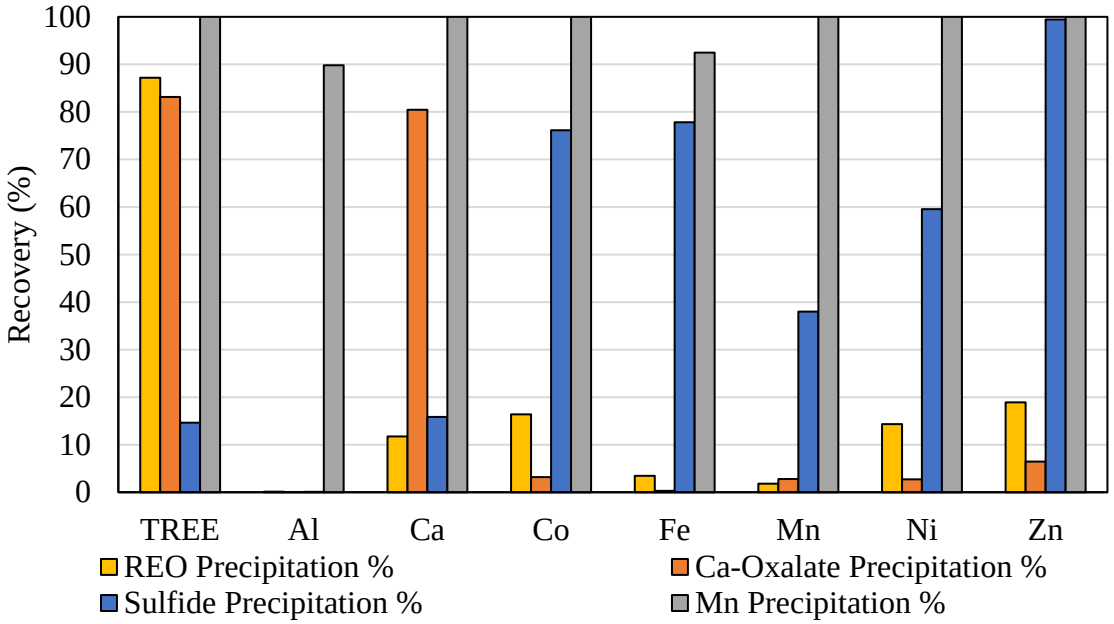



Figure 3.12.47 Recovery of various elements of interest in the final product precipitation stages during the pilot scale experiment.

It was determined that iron and aluminum precipitation stages removed 13% and 33% of the total Al and Fe content. Only 0.3% and 1.5% of the total Al, and Fe precipitated with the RE-

CM cake. However, considering the iron content present in the system, even a small percentage could cause significant contamination in the downstream processes. Additionally, the Fe and Al steps resulted in significant REE and CM losses (Figure 3.12.48). The REE, Co, Ni, Mn, and Zn lost in these stages represented 15%, 21%, 26%, 33% and 35% of the total content in the system. The RE-CM cake only recovered 21% REEs, 19% Co, 28% Mn, 23% Ni and 30% Zn.

Lower redissolution efficiency of the RE-CM cake coupled with drop in precipitation efficiency resulted in the recovery of only 15% of the total REEs (Figure 3.12.48). Unfortunately, the RE-Oxide cake produced in this experiment was low purity due to the precipitation of Zn. It was found that RE-Oxide cake was comprised of only 49% REE mass. The Ca and Zn content represented roughly 38% of the total content. Additionally, iron and manganese contamination was 6% and 3%, respectively (Figure 3.12.49). Long leaching time in this experiment also did not impact the REE distribution significantly. The LREE and HREE content in the RE-Oxide cake was 93% and 7%, respectively. The calcium oxalate cake contained 80% Ca, 6% REEs, 6% Mn and 7% Zn. Lower Co and Ni precipitation efficiency resulted in the significant loss to Mn cake. The metal sulfide cake recovered 9%, 8, and 16% of the total Co, Ni and Zn, respectively. However, since 15% of the total Mn precipitation along with iron, the final product was lower purity (Figure 3.12.49). The sulfide cake contained only 35% mass of CM, most of which was due to zinc (32%). Iron and manganese accounted for 64% of the total mass in the sulfide cake. The Mn precipitation step recovered the remaining elemental content. Like the previous experiments, the Mn cake was contaminated with Al and Fe, making up 67% of the total mass. The Mn content was only 27% with 4% calcium.

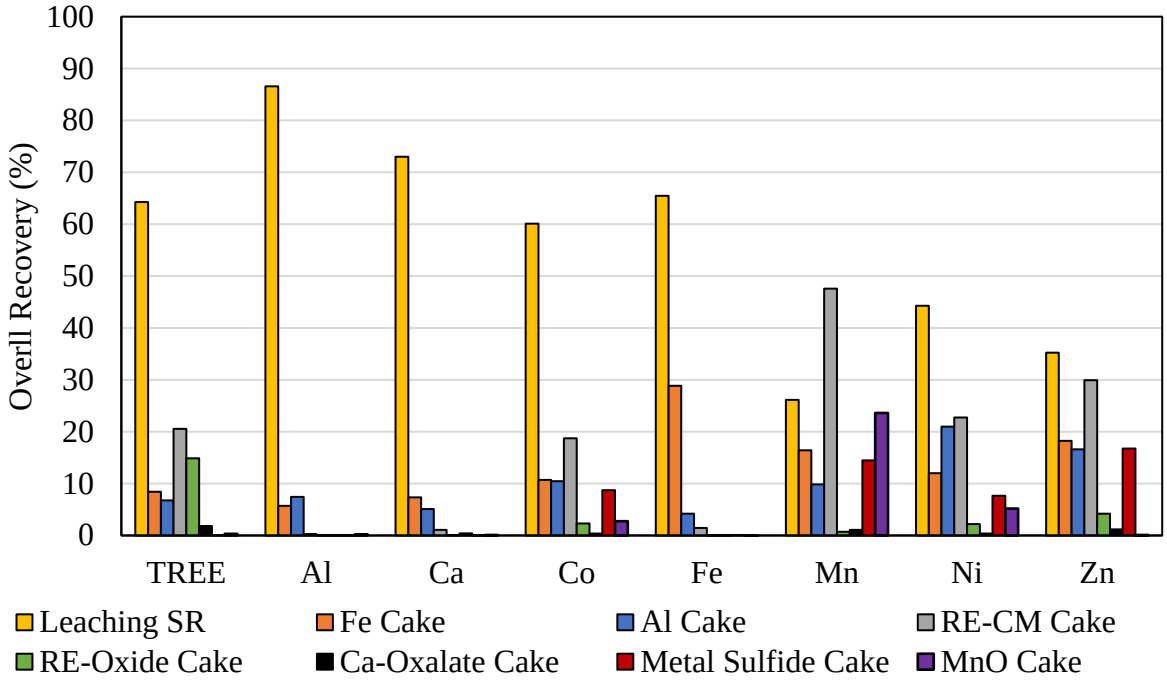



Figure 3.12.48 Overall recovery of various elements in the cakes generated during the pilot scale test.

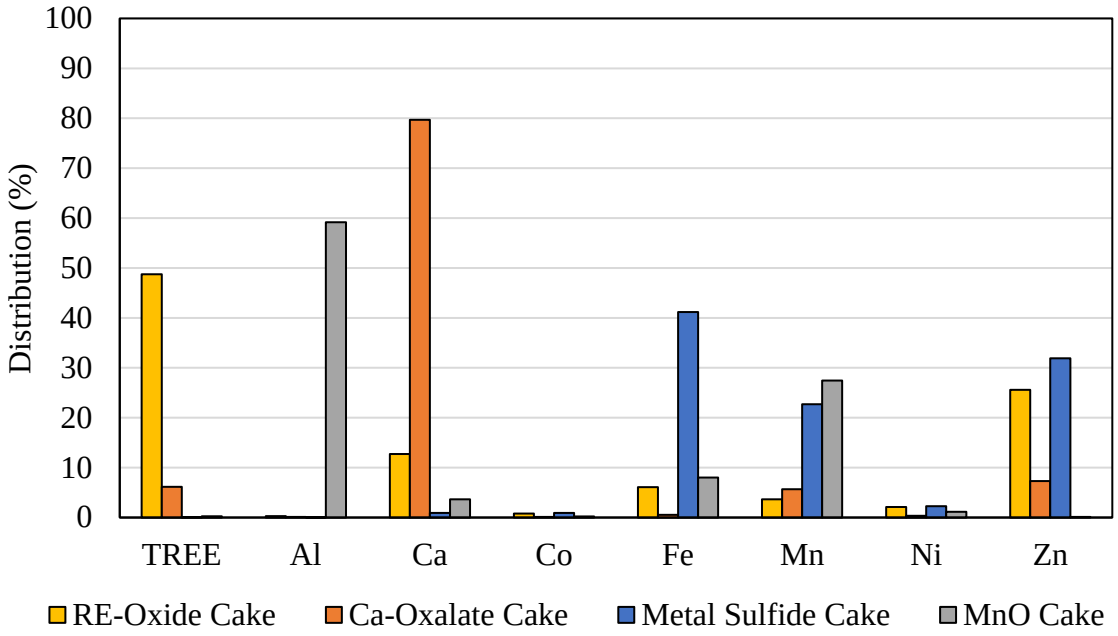



Figure 3.12.49 Elemental distribution in the final products generated in the pilot scale experiment.

**Baker Seam Experiment Run-4 (WK-4):** The final pilot scale experiment on the Baker seam coarse refuse material was also performed using bio-acid but with a relatively higher molar strength of acid. All other leaching conditions were the same as previous tests. The bio acid used in this test was equivalent to 0.65 M sulfuric acid, considerably higher than the earlier bio-acid batches. Similar to the previous bio-acids, the iron content in this batch was also anticipated to be high. Surprisingly, the new batch contained 24196 ppm of iron, approximately 10,000 ppm higher than the previous batch (Table 3.12.17). The concentrations of other elements such as Co, Ni, Zn and Mn were also more than the previous bio-acid solution. As discussed previously, elevated pyrite and nutrient concentrations increased the acid molarity of the solution through higher pyrite decomposition, which likely resulted in elevated solubilization of different minerals. Hence, elevated concentrations of other elements were acquired in the solution.

Table 3.12.17 Elemental content (ppm) in the bio-acid used in the pilot-scale experiment.

| | TREE | Al | Ca | Co | Fe | Mn | Ni | Zn |
|---|---|---|---|---|---|---|---|---|
| **Bio-Feed** | 2.1 | 36.1 | 731.9 | 2.4 | 24196.8 | 73.4 | 8.1 | 85.9 |

Despite the higher molarity of the bio-acid in this test, the REE recovery was only 39 percent, approximately 3 absolute percentage points higher than the earlier experiment. Recovery of Co, Ni, Mn, and Zn was 40%, 53%, 65%, and 65%, respectively, similar to the previous test. The iron and aluminum recoveries were also roughly the same. As identified earlier, Fe insolubilization in the leaching decreased the iron content from 24196 ppm to only 15606 ppm in the PLS (Table 3.12.18). The iron precipitation stage removal efficiency for Al and Fe was 61% and 93%, considerably higher than the earlier test. Higher iron precipitation efficiency was likely a result of the elevated iron content, which pushed the precipitation pH towards relatively more acidic pH levels. Unfortunately, the REE recovery to iron removal step was 46%, whereas the precipitation efficiency of CM in this stage was less than 20 percent. Considering the abnormally high recovery of calcium in the iron precipitation step, it was likely that REEs precipitated as sulfates through isomorphic substitution. Hence, despite the relatively higher recovery in the leaching stage, approximately half of the leached REEs were lost to the iron cake.

Table 3.12.18 Elemental content (ppm) in the PLS generated using bio-acid leaching in the pilot scale experiment.

| | TREE | Al | Ca | Co | Fe | Mn | Ni | Zn |
|---|---|---|---|---|---|---|---|---|
| **Leaching-PLS** | 23.7 | 2692.8 | 1519.9 | 2.7 | 15606.2 | 69.8 | 9.1 | 74.2 |

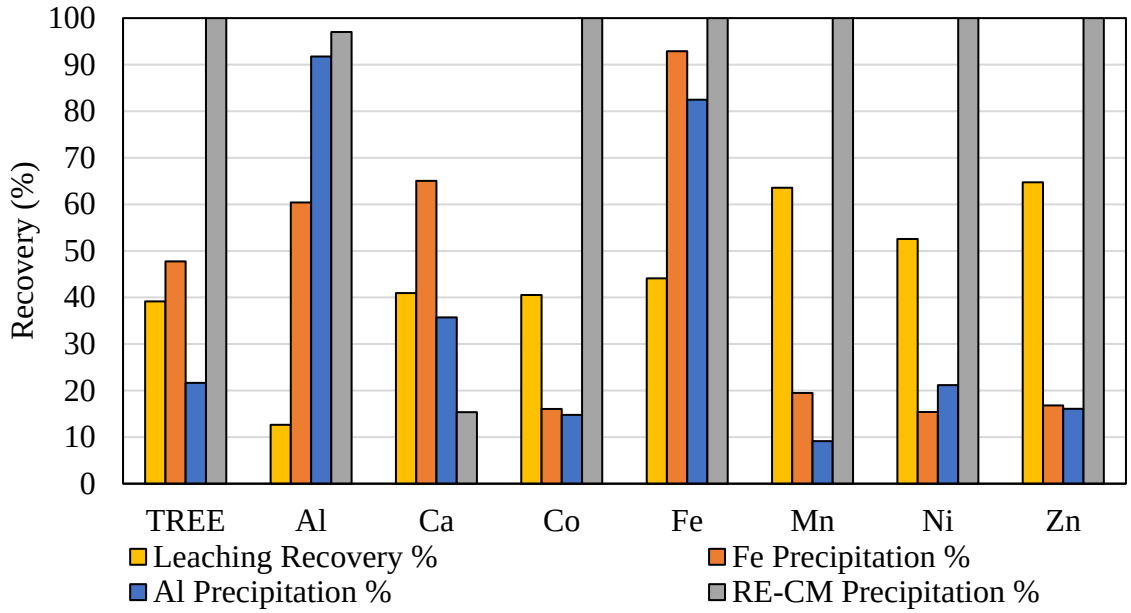



Figure 3.12.50 Recovery of various elements in the pilot scale testing of Baker seam coarse refuse material using bio-acid.

The Al removal step recovered 92% Al and 83% Fe along with 35% Ca in the solution. The REE and Ni precipitation efficiency in this stage was approximately 21 percent whereas Co and Zn recovery was 15 and 16 percent, respectively (Figure 3.12.50). Interestingly, Mn loss to the Al cake was relatively lower than the other elements. Subsequently, the aluminum precipitation raffinate was treated for RE-CM precipitation at pH 9.0. The results showed that virtually all the elements of interest except Al demonstrated 100% recovery in this step. Concentrations of different elements in the RE-CM cake are shown in Table 3.12.19. It is evident that the REE content is significantly lower than the RE-CM cakes generated in the previous experiments, stemming from the losses in earlier stages. Furthermore, while the PLS generated from the leaching of RE-CM cake contained significantly lower contaminant content, the REE content did not improve. Contrarily, Co, Ni, Mn and Zn content improved 1.5 folds relative to the concentrations in the PLS generated from the leaching stage (Table 3.12.19).

Table 3.12.19 Elemental content (ppm) in the RE-CM cake (dry, whole mass basis) and the PLS produced from its leaching in the pilot scale experiment.

| | **TREE** | **Al** | **Ca** | **Co** | **Fe** | **Mn** | **Ni** | **Zn** |
|---|---|---|---|---|---|---|---|---|
| **RE-CM Cake** | 1480.3 | 13030.3 | 6233.0 | 292.1 | 29709.2 | 7384.7 | 929.6 | 7909.4 |
| **RE-CM RD R** | 19.1 | 206.3 | 102.8 | 4.3 | 265.1 | 109.4 | 13.4 | 117.2 |

REE removal efficiency in the oxalic acid precipitation step was 82% whereas Co, Ni, Ca, and Zn recovery was 3%, 3%, 2%, and 6%, respectively. As mentioned previously, the CM recovery in oxalic acid precipitation stage was an anomaly particular to the processing of bio-acid treated material. Increasing the solution pH to 3.0 for calcium precipitation resulted in approximately 90 percent recovery of REEs and calcium. Interestingly, contrary to the earlier test, the recovery of CM also improved considerably during the calcium precipitation step, implying complex solution chemistry for the material treated with the bio-acid. The raffinate from the calcium precipitation step was further treated for CM recovery. It was determined that the precipitation efficacy of Co, Ni, and Zn in the sulfide precipitation step was 90%, 82%, and 99%, respectively. Similarly, the Fe, Ca, and REE recovery was 89%, 42%, and 30%, respectively.

As observed in the previous experiment, significant Mn content precipitated in the metal sulfide cake, resulting in product contamination. Finally, the remaining REE and CM content was recovered in the final precipitation step.

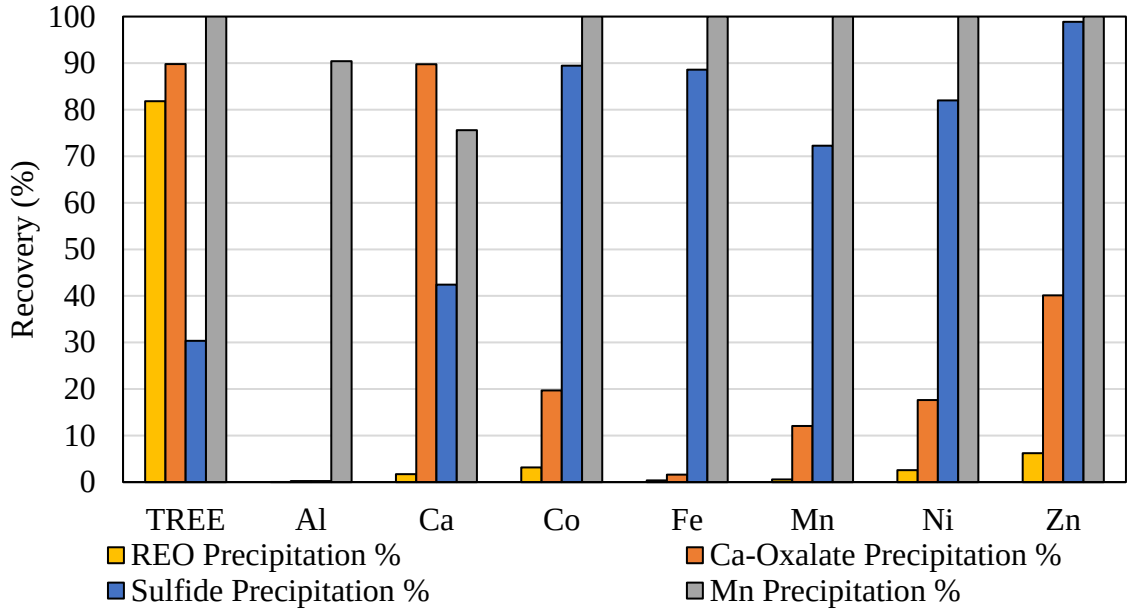



Figure 3.12.51 Elemental recovery in the final precipitation stages during the pilot scale experiment.

The overall recovery shown in Figure 3.12.52 indicated relatively lower REE and CM loss to the leaching solid residue stage due to higher leaching recoveries. However, approximately 23% of the total REEs were lost to the Fe and Al cakes, decreasing the overall recovery in the RE-CM cakes. The contaminant precipitation stages were highly efficient in the iron removal from the bio-acid. Overall, the Al and Fe precipitation steps removed 44% and 12% of the total Fe and Al content from the solution (Figure 3.12.52). The RE-CM step recovered only 0.4-0.5% of the total Al and Fe content in the system. Due to the CM content losses in the earlier steps, the overall recovery of Co, Ni, Mn, and Zn recover was approximately 29%, 47%, 35%, and 45%, respectively (Figure 3.12.52).

Despite the high precipitation recovery in the oxalic acid precipitation stage, only 10% of the total REEs were recovered in the oxide cake. Furthermore, as Ca and other CMs precipitated in this stage, the purity of the product was negatively impacted. The results indicated that the REE content in the oxide cake only accounted for 58% of the total mass with Zn, Fe and Ca as the main contaminants (Figure 3.12.53). The oxide product also contained a relatively lower Co, Ni and Mn content. The calcium precipitation stage removed most of the Ca content in the solution. However, as considerable CM content precipitation occurred in this stage, the final product only contained 57% Ca content. The CM content accounted for approximately 38% of the total mass whereas the remaining content was comprised of iron and REEs. Due to the change in Zn precipitation behavior, roughly 17% of the total Zn content was lost in the oxalic and calcium precipitation stage.

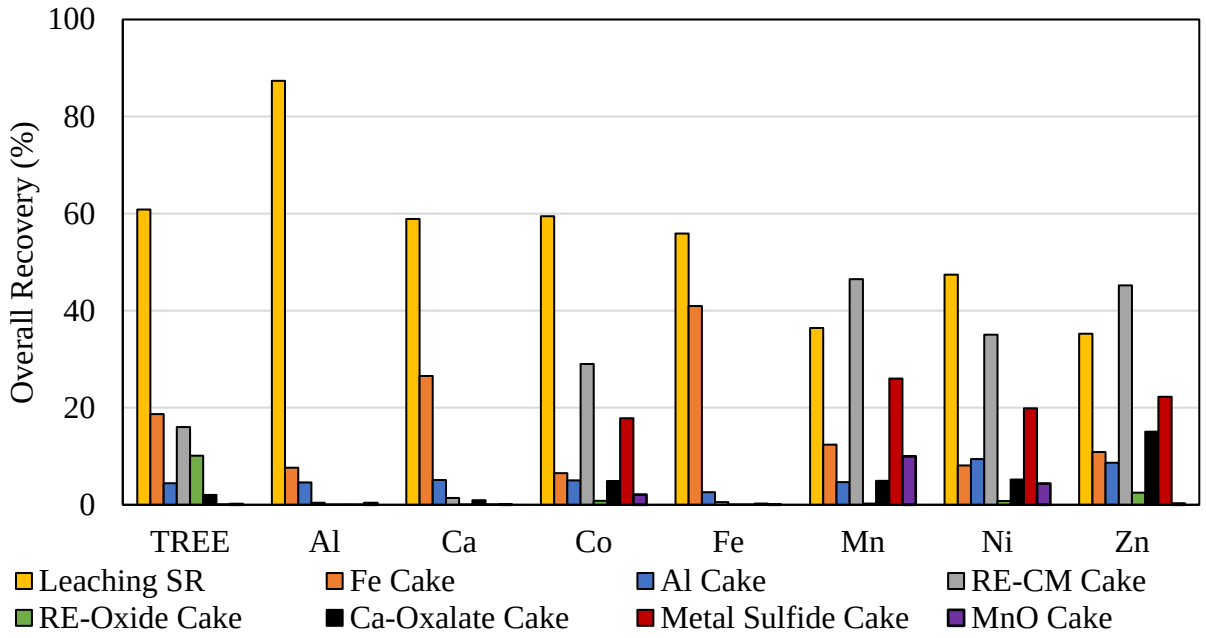



Figure 3.12.52 Overall recovery of various elements of interest in the cakes produced during the pilot scale test.

The sulfide precipitation stage recovered 18%, 20%, and 22% of the total Co, Ni and Zn content (Figure 3.12.52). Even though only 0.3 percent of the total Fe content was recovered in this stage, it accounted for 61% of the total content in the cake. Similarly, due to high precipitation efficiency of Mn in the CM removal step, the Mn content represented approximately 18% of the total mass (Figure 3.12.53). Despite the high precipitation efficiency of CMs, the sulfide cake only contained 20% CM content. Finally, the MnO cake merely recovered 10% of the overall Mn content. Since Al and the remaining Fe concentrations from the sulfide precipitation stage were recovered in this stage, combined Al and Fe content accounted for 86% of the total mass with only 11% Mn content.

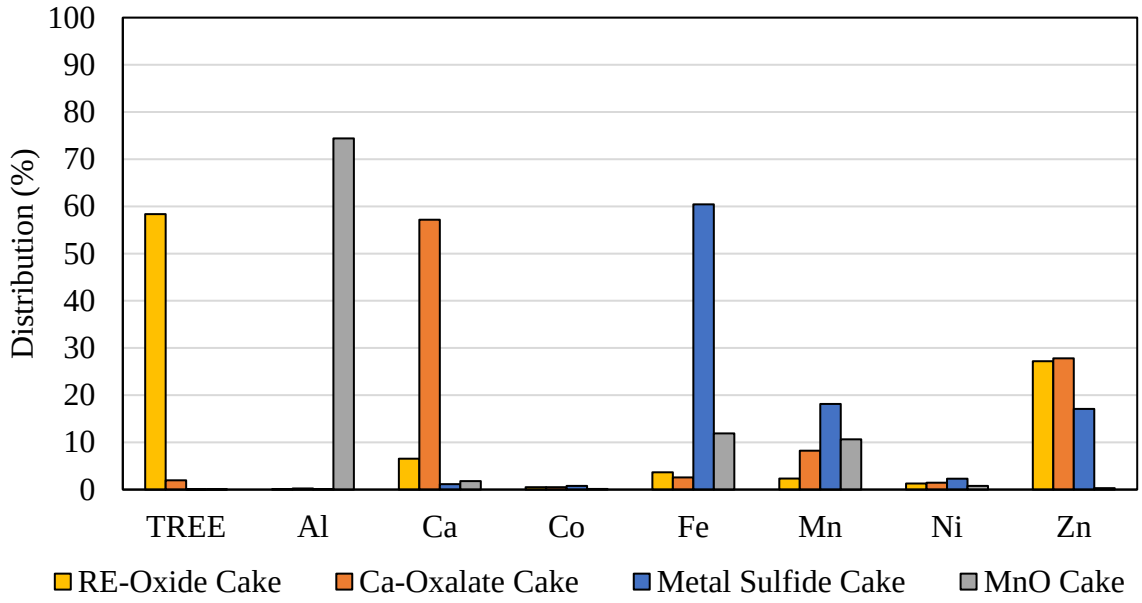



Figure 3.12.53 Distribution of various elements of interests in the final products generated during the pilot scale experiment.

### *3.12.4 Summary*

- **Circuit arrangement 1** provided promising means to enhance the recovery of rare earth elements and other critical metals while effectively reducing the cost of the processes. However, due to the high calcium content of the material, the purity of the final products was considerably impacted. As such, while this circuit arrangement can be employed to recover REEs and critical metals from a source rich in REEs and low calcium content, it is not favorable for the Shakespeare material.
- **Circuit arrangement 2** tested the efficiency of the separate precipitation stage in an attempt to increase both purity and recovery of the final products. The results indicated that the separate precipitation stages were advantageous to the purity of the products as the final products with more 90 percent purities were obtained. However, a substantial decrease in the CM recovery was observed due to the loss of critical metals to the RE cake.
- **Circuit arrangement 3** investigated the modified circuit arrangement to maximize the REE and CM recovery. It was determined that granted strict process controls, high purity of REEs and CMs can be achieved. However, mixing the RE and CM cake solution would be required to achieve a high recovery of REEs. The costs associated with separate precipitation and leaching steps may result in relatively more unfavorable economics.
- **Circuit arrangement 4** examined the potential of combined precipitation of REEs and CMs at pH 9.0. It was determined that high product recovery and purity could be obtained while decreasing the CAPEX and OPEX of the operation. The economic benefits were realized by eliminating an extra leaching setup previously required for the separate treatment and processing of RE and CM cakes. Similarly, elevated recovery was obtained as all the REE and CMs were recovered in one cake.
- **Circuit arrangement 5** added a calcium precipitation stage in the previous process flow in an attempt to improve the purity of the metal sulfide cake. It was found that the overall product grade increased. However, since the PLS generated from the Baker seam material was rich in iron concentrations, its precipitation in the sulfide cake did not produce the anticipated improvement in the grade. Additionally, as the iron was primarily as ferrous ions, most of the iron content bypassed the contaminant precipitation stages and precipitated with RE-CM cake. As such, in order to maximize the purity of the sulfide cake, iron oxidation to ferric form would be required when treating the PLS generated from the Baker seam material.

### *3.12.5 Product Grade Summary*

The rare earth oxide content of the products resulting from the pilot scale testing of the various feedstocks is shown in Table 3.12.20. It is evident that high RE-Oxide products were consistently generated in this project. The highest purity product with 88% RE-Oxide content was generated in the WK-2 test whereas the least pure product with only 36% RE-Oxide content was produced in the SHXP-C2 experiment. Discussion on the factors resulting in the reduction of oxide product purity has been provided previously. It is interesting to note that heavy REE content in the products generated from the PLS treatment is substantially higher than the Shakespeare and Baker seam coarse refuse material products. This anomaly was due to the change in REE distribution as a function of changing the feedstock. The heap leach PLS, similar to AMD, contained elevated HREE content compared to LREEs due to the loss of light REEs through isomorphic substitution with gypsum. Conversely, the leachate solution produced from the leaching of Baker seam and Shakespeare material was primarily comprised of LREEs, resulting in the oxide product containing elevated LREE content.

Table 3.12.20 Grades of the highest RE Oxide products generated in this project.

| Test Type-Number | % REO | %Heavy REO |
|---|---|---|
| Heap Leach PLS / Acid Mine Drainage | | |
| PLS-3 | 87.61 | 55.66 |
| PLS-4 | 87.15 | 41.63 |
| Coal Refuse-Western Kentucky No.13 | | |
| WK-1 | 82.23 | 10.39 |
| WK-2 | 88.19 | 5.37 |
| WK-3 | 39.59 | 3.01 |
| WK-4 | 42.44 | 3.18 |
| Secondary Source-Shakespeare | | |
| SHXP-1 | 61.75 | 7.49 |
| SHXP-2 | 36.20 | 10.03 |
| SHXP-3 | 85.28 | 29.54 |
| SHXP-4 | 70.03 | 21.67 |
| SHXP-5 | 68.39 | 18.7 |

*3.12.6 Sample Characterization*

**ICP Results Comparison:** All the samples generated in the experiment were analyzed using Inductively coupled plasma-optical emission spectrometry (ICP-OES) analysis at the Kentucky Geological Survey (KGS) and Mining Engineering Department of the University of Kentucky (MIN). Samples for the analysis were prepared using a representative solid sample from each final product and digested using high-strength acids. The resulting liquid samples were subsequently used for the ICP analysis. The ICP-OES analysis conducted at KGS only reported rare earth elements whereas the instrument at Mining Engineering Department provided analysis on REEs as well as 34 other elements. The data from Mining Engineering Department ICP-OES was used for all the data analysis, whereas KGS data was used to gauge the repeatability and accuracy of the results. At the end of the project, RE-Oxide samples from four distinct tests were sent for analysis at an external laboratory. The presented in table below provide a direct comparison between the RE-Oxide content obtained from three different labs.

The results indicated a fairly good agreement between the SGS and MIN data. However, there was a significant difference in the estimated RE concentrations between the SGS and KGS labs. An in-depth data analysis revealed that the disparity between these two labs was primarily because of cerium and samarium contents, which were overestimated at elevated concentrations. This finding was also supported by the relatively smaller difference in the PLS3 test REO content. Since a typical AMD is primarily heavy REE dominant owing to the light REE co-precipitation with gypsum, the estimated cerium and samarium concentrations were fairly close between the labs. As such, the SGS data was determined to be more accurate compared to the KGS. Overall, a decent agreement between the MIN and SGS ICP data suggested high accuracy of the unit available at the Mining Engineering department of the University of Kentucky.

Table 3.12.21 RE analysis comparison of concentrated rare earth oxide product from 3 labs (SGS: Commercial lab, KGS: Kentucky Geological Survey, MIN: Mining Engineering)

| Cake | TREO | LREO | HREO |
|---|---|---|---|
| **PLS-3 REO** | | | |
| **SGS** | 89.52 | 32.07 | 57.44 |
| **KGS** | 87.02 | 33.46 | 53.57 |
| **MIN** | 87.61 | 31.95 | 55.66 |
| **SHXP-C3 REO** | | | |
| **SGS** | 87.43 | 56.82 | 30.61 |
| **KGS** | 96.37 | 65.58 | 30.79 |
| **MIN** | 85.28 | 55.75 | 29.54 |
| **SHXP-C4 REO** | | | |
| **SGS** | 71.76 | 49.68 | 22.08 |
| **KGS** | 80.52 | 57.41 | 23.11 |
| **MIN** | 70.03 | 48.36 | 21.67 |
| **WK-4 REO** | | | |
| **SGS** | 45.13 | 41.24 | 3.89 |
| **KGS** | 49.75 | 46.36 | 3.39 |
| **MIN** | 42.44 | 39.25 | 3.18 |

**XRF Analysis:** The ICP data presented previously identified the distribution of the oxide and metal sulfide products generated in the pilot scale experiments. However, the data was normalized to the elements shown in the plots. Therefore, while the data provides a decent approximation of the elemental content, it does not provide any information regarding the phases in the cakes and other contaminants in the final products. As such, XRF analysis was performed on select RE-Oxide and CM sulfide samples to identify the concentration of contaminants in select product samples. As the silicon content was not included in the distribution due to analytical challenges of the silica identification using ICP-OES, XRF analysis would also assist in its quantification. The results from XRF analysis on the RE-Oxide and metal sulfide samples are shown in Table 3.12.22.

It was found that all the RE-Oxide samples contained significant calcium oxide content. As discussed previously, calcium tends to precipitate as calcium oxalate during the oxalic acid precipitation of the PLS. The aluminum oxide content was around 1 percent, likely due to aluminum precipitation as hydroxide or oxalate complexes. Similarly, the iron precipitation was likely due to its precipitation as ferrous oxalate. The silica content in the samples was approximately 1 percent, which is significant considering the difficulties silica presents in the further purification of the RE-oxides. Additionally, trace concentrations of potassium, titanium, and magnesium oxide were also identified in the analysis. Interestingly, iron, calcium and manganese oxide content was relatively high in the WK-4 test, which was conducted using the bio-acid. The RE-Oxide generated from the treatment of Shakespeare material using separate precipitation of REEs and CMs had elevated purity of the precipitates compared to PLS and WK products which involved combined precipitation of REEs and CMs.

Table 3.12.22 XRF analysis of the major and minor phases (%) in the different RE-Oxide cakes generated in the pilot scale testing (ND=Not Detected).

| | $SiO_2$ | $Al_2O_3$ | $Fe_2O_3$ | MgO | CaO | $K_2O$ | $TiO_2$ | MnO | $Cr_2O_3$ | $V_2O_5$ |
|---|---|---|---|---|---|---|---|---|---|---|
| **PLS-4** | 1.74 | 0.69 | 3.23 | 0.22 | 6.80 | 0.08 | 0.05 | 0.29 | 0.05 | 0.02 |
| **SHXP-C3** | 1.15 | 0.34 | 0.44 | 0.05 | 6.00 | 0.07 | 0.03 | 0.04 | ND | 0.01 |
| **WK-4** | 1.71 | 0.69 | 4.92 | 0.69 | 10.30 | 0.16 | ND | 2.49 | ND | ND |

The XRF analysis on select metal sulfide products revealed that the Baker seam coarse refuse material processed with the bio-acid contained significant iron and manganese content (Table 3.12.23). While elevated iron content was also observed in the Baker seam coarse refuse samples treated with sulfuric acid, high manganese precipitation in the metal sulfide cake was an interesting outcome from the test using bio-acid. Both samples contained over >30 percent iron and >12 percent manganese oxide content. Interestingly, elevated concentrations of magnesium were also identified in all the samples. Similar to the RE-Oxide product obtained through separate precipitation stages, the metal sulfide product also contained significantly lower concentrations of contaminants relative to the products generated from the processing of combined RE-CM cakes. This finding suggested that separate precipitation stages could effectively produce high purity products. However, as significant recovery losses were observed in the pilot-scale experiments, this limits the applicability of this process flow in the current system. As such, if the overlapping precipitation behavior of REEs and CMs could be avoided (by using a different pH modifier or a combination of pH modifiers), separate precipitation stages could likely generate a high purity product with elevated recovery.

Table 3.12.23 XRF analysis of the major and minor phases (%) in the different metal sulfide products generated during the pilot scale testing (ND=Not Detected).

| | $SiO_2$ | $Al_2O_3$ | $Fe_2O_3$ | MgO | CaO | $K_2O$ | $TiO_2$ | MnO | $Cr_2O_3$ | $V_2O_5$ |
|---|---|---|---|---|---|---|---|---|---|---|
| **PLS-3** | 3.6 | 1.4 | 1.8 | 3.0 | 0.3 | 0.2 | ND | 1.9 | ND | ND |
| **WK-3** | 0.6 | 0.2 | 30.1 | 11.9 | 1.2 | ND | ND | 14.6 | ND | ND |
| **WK-4** | 0.3 | 0.1 | 48.1 | 6.5 | 0.9 | ND | ND | 12.1 | ND | ND |

Similar to the final product samples, the XRF analysis was also conducted on a set of samples generated from the pilot scale testing. Samples produced from WK Run-4 were selected due to the elevated iron concentration of bio-acid used in this experiment. The Baker seam coarse refuse sample contained elevated concentrations of silica, alumina, iron and calcium oxide. High silica and aluminum content was due to the presence of clays such as kaolinite and illite whereas elevated iron oxide content was because of the pyrite in the material. It is evident from the results that most of the silica, alumina and iron oxide was rejected in the solid residue which complimented the experimental finding previously. Similarly, the iron and aluminum precipitation stage rejected most of the contaminants in the PLS. Interestingly, majority of the silica content was rejected to the left over RE-CM cake from leaching, suggesting high selectivity of the leaching process. Subsequently, the REE-Oxide cake had elevated concentrations of calcium, iron, manganese and silicon oxides whereas magnesium and aluminum oxides were in trace concentrations. Finally, the NaS cake was contaminated with iron, manganese and magnesium oxide.

Table 3.12.24 XRF analysis of the major and minor phases (%) in the cakes generated during the pilot scale testing of Baker seam coarse refuse in WK Run 4 (ND=Not Detected).

| | $SiO_2$ | $Al_2O_3$ | $Fe_2O_3$ | MgO | CaO | $K_2O$ | $TiO_2$ | MnO | $Cr_2O_3$ | $V_2O_5$ |
|---|---|---|---|---|---|---|---|---|---|---|
| **Feed** | 57.9 | 22.2 | 6.5 | 1.3 | 2.4 | 3.1 | 0.9 | 0.1 | 0.0 | 0.0 |
| **Leaching SR** | 43.2 | 15.5 | 25.4 | 0.8 | 1.3 | 2.5 | 0.7 | 0.1 | ND | 0.0 |
| **Fe Cake** | 47.7 | 11.5 | 19.1 | 0.5 | 1.7 | 2.8 | 0.4 | 0.1 | ND | 0.0 |
| **Al Cake** | 29.7 | 21.6 | 12.8 | 0.3 | 1.3 | 1.5 | 0.0 | 0.1 | ND | ND |
| **RECM Cake** | 14.0 | 10.3 | 16.7 | 6.0 | 1.3 | 0.6 | ND | 2.8 | ND | ND |
| **Redis Cake** | 71.2 | 11.5 | 5.0 | 0.3 | 0.6 | 3.9 | 0.1 | 0.1 | ND | ND |
| **REO Cake** | 1.7 | 0.7 | 4.9 | 0.7 | 10.3 | 0.2 | ND | 2.5 | ND | ND |
| **CaP Cake** | 1.1 | 0.4 | 2.7 | 1.9 | 42.2 | 0.1 | ND | 8.0 | ND | ND |
| **NaS Cake** | 0.3 | 0.1 | 48.1 | 6.5 | 0.9 | ND | ND | 12.1 | ND | ND |

**Environmentally Sensitive Element Analysis:** Several studies have found trace concentrations of environmentally sensitive elements (ESA) such as mercury, arsenic, thorium, and uranium, associated with different minerals in the inorganic matter of coal. Since the primary feedstock used in this project was coal coarse refuse from a preparation plant consisting primarily of inorganic matter, an elevated concentration of ESAs was possible, which could raise environmental concerns if their final form is easily leachable and/or operational difficulties in further purification of the REEs. As such, a complete set of solid samples from two distinct pilot-scale tests were sent to a commercial lab and the University of Kentucky Center for Applied Energy Research for As, Th, and U analysis. Same set of samples were also sent to an external lab (SGS) for Hg analysis.

The results suggested that uranium was concentrated in the RECM cake relative to the leaching solid residue (LSR) and aluminum cake. Similarly, trace concentrations of As and Th also precipitated with the RE-CM cake, contaminating the solids. Elevated concentrations of As and Th were observed in both oxide cakes, which suggested that the oxalic acid precipitated these elements. Elevated uranium content in RE-CM cake and Mn cake suggested that uranium formed hydroxide complexes in the basic pH ranges. The Hg analysis for WK4 and PLS3 test run solid samples was inconclusive because mercury content was below the detection limit of the instrument.

Table 3.12.25 Elemental analysis of environmentally sensitive elements on the various products generated from the pilot-scale testing of primary and secondary feedstocks.

| Cake | As | Th | U | Hg |
|---|---|---|---|---|
| **WK-4** | | | | |
| **LSR-Cake** | 82 | 17 | 6 | <0.3 |
| **Al-Cake** | 11 | 49 | 4 | <0.3 |
| **RECM-Cake** | 11 | 16 | 117 | <0.3 |
| **RECM-Redis-cake** | 3 | 28 | 8 | <0.3 |
| **RE Oxide-cake** | 738 | 197 | 12 | <0.3 |
| **CaP-Cake** | 11 | 22 | 2 | <0.3 |
| **NaS-Cake** | 7 | 0 | 6 | <0.3 |
| **Mn-Cake** | 1 | 4 | 184 | <0.3 |
| **PLS-4** | | | | |
| **Fe-cake** | 218 | 5 | 1 | - |
| **Al-cake** | 5 | 12 | 2 | - |
| **RECM-cake** | 4 | 2 | 31 | - |
| **RECM-Redis-cake** | 15 | 17 | 13 | - |
| **RE Oxide-cake** | 617 | 105 | 5 | - |
| **NaS-cake** | 3 | 1 | 6 | - |
| **Mn-cake** | 2 | 1 | 73 | - |

**SEM-EDX Analysis:** The XRF data presented previously was limited to the elements available for the analysis. For instance, zinc content was not identified in the XRF analysis due to technical challenges. As such, a systematic study using scanning electron microscopy with energy dispersive x-ray was conducted on the rare earth oxalate (REO) cake produced from different pilot scale tests. The objective was to identify the surface morphology of the samples as well as the primary and trace contaminants in the REO products. The individual weight percent of each element as well as the electron image with elemental mapping is shown in Figure 3.12.54-Figure 3.12.64 and Table 3.12.26-3.12.31.

Similar to the previous results, elevated calcium content was identified in all the samples which was due to its precipitation as calcium oxalate. Interestingly, it was determined that all the RE-Oxide samples contained fairly high concentrations of sodium. The main source of Na in the solution was sodium hydroxide base used to adjust the solution pH. Since the RE-CM precipitation required raising the solution pH from 1 to as high as 6.0 and/or 9.0, the generated cake contained elevated sodium content primarily due to the entrainment and/or precipitation as a complex. As the cake was redissolved in the solution which, at occasions, also required addition of NaOH in the solution to achieve the desired pH level, introducing more sodium content. As such, it is possible that sodium precipitated as an oxalate complex. Beside Na and Ca, silica content in the samples ranged from 0.3% to as high as 5%. Similarly, trace concentrations of other contaminants such as Mg, Ni, K, Cl, Fe, and S were also identified.

Table 3.12.26 Weight percentages of individual elements, as calculated via SEM-EDX mapping, for two sites of the PLS 3 REO sample

| Element | Site 1 | | Site 2 | |
|---|---|---|---|---|
| | Wt% | Wt% Sigma | Wt% | Wt% Sigma |
| **O** | 33.79 | 0.10 | 30.02 | 0.09 |
| **Y** | 28.26 | 0.09 | 29.75 | 0.09 |
| **Gd** | 9.64 | 0.08 | 10.18 | 0.07 |
| **Dy** | 6.80 | 0.08 | 7.17 | 0.07 |
| **Nd** | 6.14 | 0.06 | 6.60 | 0.06 |
| **Sm** | 3.63 | 0.07 | 3.90 | 0.06 |
| **Ce** | 2.94 | 0.05 | 3.16 | 0.05 |
| **Er** | 2.80 | 0.08 | 3.01 | 0.07 |
| **Ca** | 2.72 | 0.02 | 2.86 | 0.02 |
| **Yb** | 1.64 | 0.09 | 1.72 | 0.08 |
| **Na** | 0.64 | 0.03 | 0.64 | 0.02 |
| **S** | 0.43 | 0.02 | 0.35 | 0.01 |
| **Si** | 0.31 | 0.02 | 0.35 | 0.02 |
| **Cl** | 0.20 | 0.01 | 0.24 | 0.01 |
| **K** | 0.06 | 0.01 | 0.05 | 0.01 |

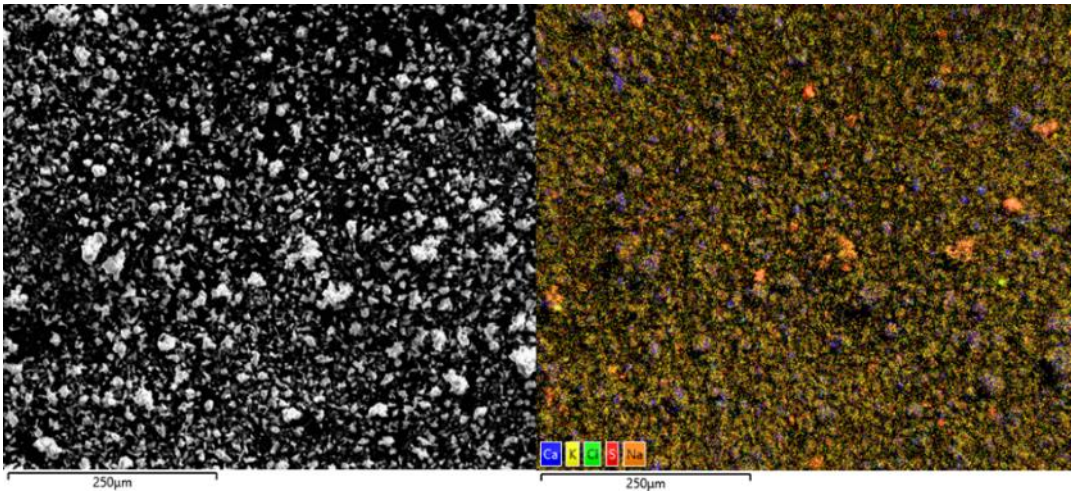



Figure 3.12.54 Electron image of site 1 from PLS 3 REO sample (left) and elemental mapping of contaminant elements (right).

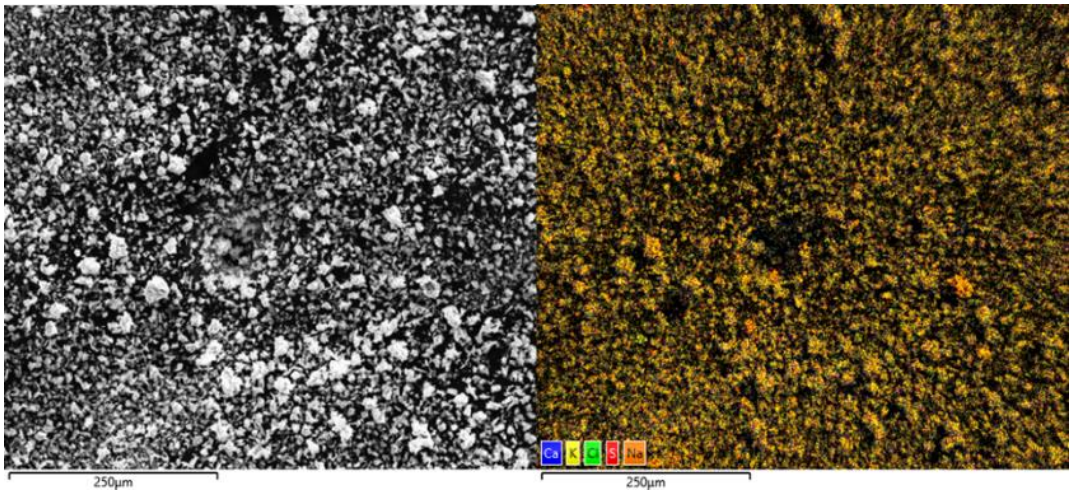



Figure 3.12.55 Electron image of site 2 from PLS 3 REO sample (left) and elemental mapping of contaminant elements (right).

Table 3.12.27 Weight percentages of individual elements, as calculated via SEM-EDX mapping, for two sites of the Shakespeare C1 REO sample.

| Element | Site 1 | | Site 2 | |
|---|---|---|---|---|
| | Wt% | Wt% Sigma | Wt% | Wt% Sigma |
| **O** | 25.71 | 0.06 | 26.47 | 0.07 |
| **Ce** | 19.15 | 0.07 | 18.69 | 0.07 |
| **La** | 16.28 | 0.06 | 15.90 | 0.06 |
| **Nd** | 7.30 | 0.06 | 7.17 | 0.06 |
| **Na** | 6.05 | 0.06 | 6.30 | 0.06 |
| **Y** | 4.81 | 0.05 | 4.71 | 0.05 |
| **Si** | 4.46 | 0.02 | 4.52 | 0.02 |
| **Zn** | 2.13 | 0.03 | 2.18 | 0.03 |
| **Pr** | 1.95 | 0.06 | 1.98 | 0.06 |
| **Ni** | 1.88 | 0.03 | 1.83 | 0.03 |
| **Mn** | 1.70 | 0.03 | 1.67 | 0.03 |
| **Gd** | 1.48 | 0.07 | 1.38 | 0.07 |
| **Co** | 1.39 | 0.02 | 1.37 | 0.03 |
| **Ca** | 1.22 | 0.01 | 1.21 | 0.01 |
| **Al** | 1.02 | 0.02 | 0.99 | 0.02 |
| **Sm** | 0.99 | 0.06 | 1.01 | 0.06 |
| **Mg** | 0.77 | 0.02 | 0.79 | 0.02 |
| **Cu** | 0.62 | 0.03 | 0.63 | 0.03 |
| **Dy** | 0.56 | 0.05 | 0.68 | 0.05 |
| **K** | 0.22 | 0.01 | 0.22 | 0.01 |
| **S** | 0.17 | 0.01 | 0.18 | 0.01 |
| **Cl** | 0.11 | 0.01 | 0.11 | 0.01 |

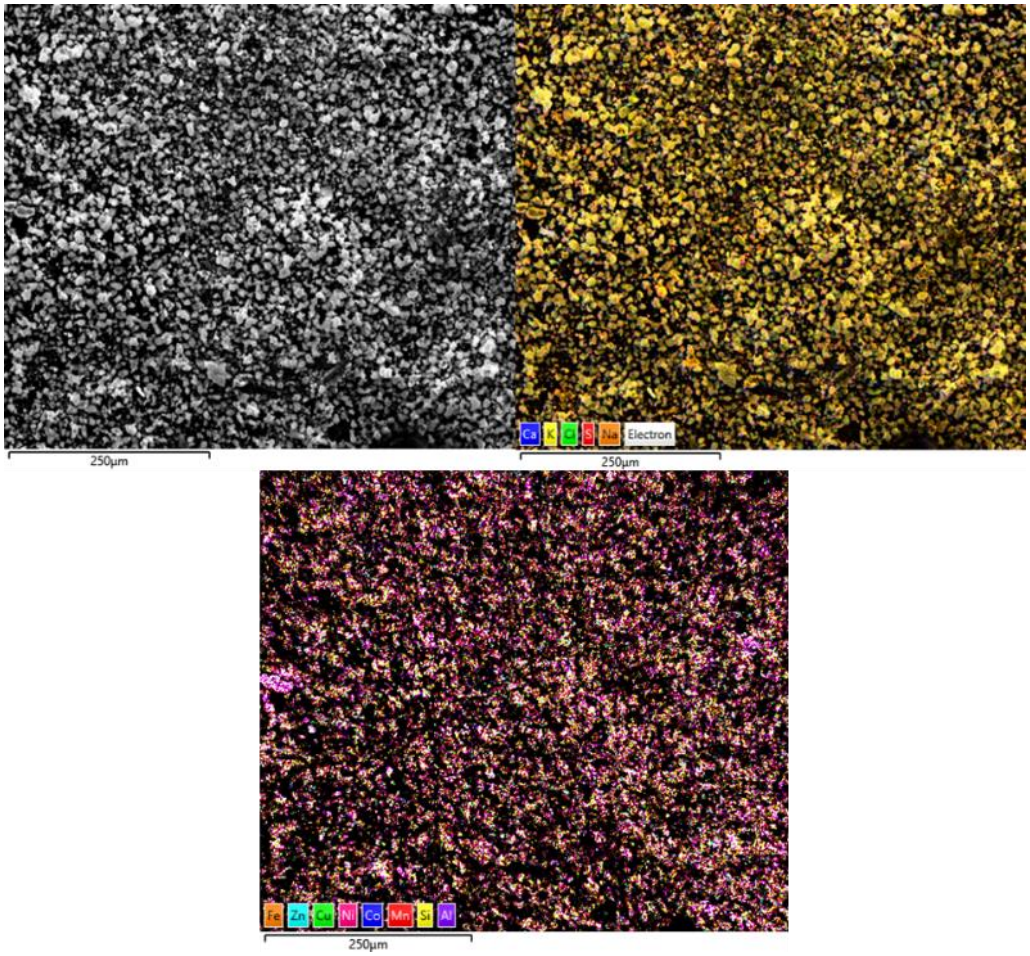



Figure 3.12.56 Electron image of site 1 from Shakespeare C1 REO sample (top left), elemental mapping of contaminant elements (top right), and elemental mapping site of critical metal/metal contaminants (bottom center).

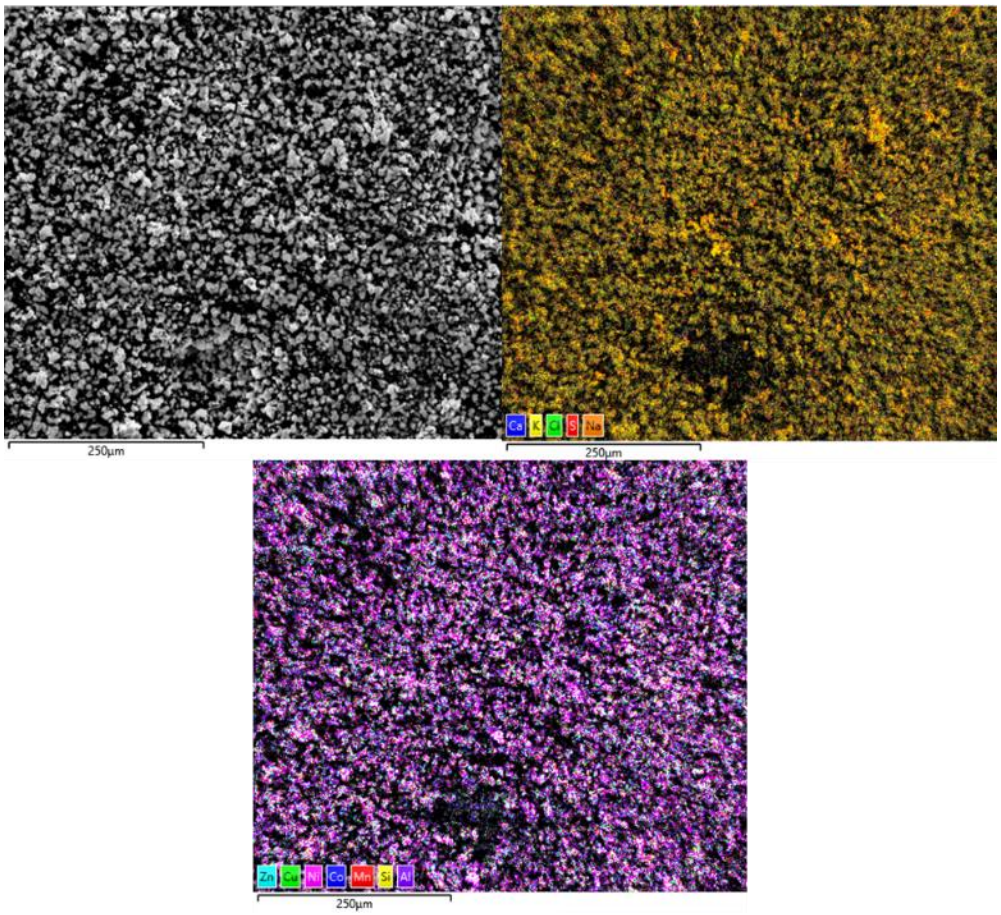



Figure 3.12.57 Electron image of site 2 from Shakespeare C1 REO sample (top left), elemental mapping of contaminant elements (top right), and elemental mapping site of critical metal/metal contaminants (bottom center).

Table 3.12.28 Weight percentages of individual elements, as calculated via SEM-EDX mapping, for two sites of the Shakespeare C2 REO sample.

| Element | Site 1 | | Site 2 | |
|---|---|---|---|---|
| | Wt% | Wt% Sigma | Wt% | Wt% Sigma |
| **O** | 62.54 | 0.10 | 61.77 | 0.10 |
| **Ca** | 26.48 | 0.06 | 27.06 | 0.06 |
| **Na** | 2.52 | 0.05 | 2.25 | 0.05 |
| **Ce** | 1.59 | 0.04 | 1.67 | 0.04 |
| **Y** | 1.07 | 0.04 | 1.12 | 0.04 |
| **S** | 0.88 | 0.01 | 0.71 | 0.01 |
| **Nd** | 0.88 | 0.04 | 1.00 | 0.04 |
| **La** | 0.88 | 0.04 | 0.87 | 0.04 |
| **Si** | 0.82 | 0.01 | 0.93 | 0.01 |
| **Ni** | 0.65 | 0.03 | 0.82 | 0.03 |
| **Mn** | 0.38 | 0.02 | 0.49 | 0.02 |
| **Al** | 0.28 | 0.01 | 0.29 | 0.01 |
| **Zn** | 0.23 | 0.03 | 0.24 | 0.03 |
| **Yb** | 0.21 | 0.07 | 0.09 | 0.07 |
| **Co** | 0.19 | 0.02 | 0.22 | 0.02 |
| **Dy** | 0.18 | 0.05 | 0.16 | 0.05 |
| **Mg** | 0.13 | 0.02 | 0.18 | 0.02 |
| **Fe** | 0.11 | 0.02 | 0.13 | 0.02 |

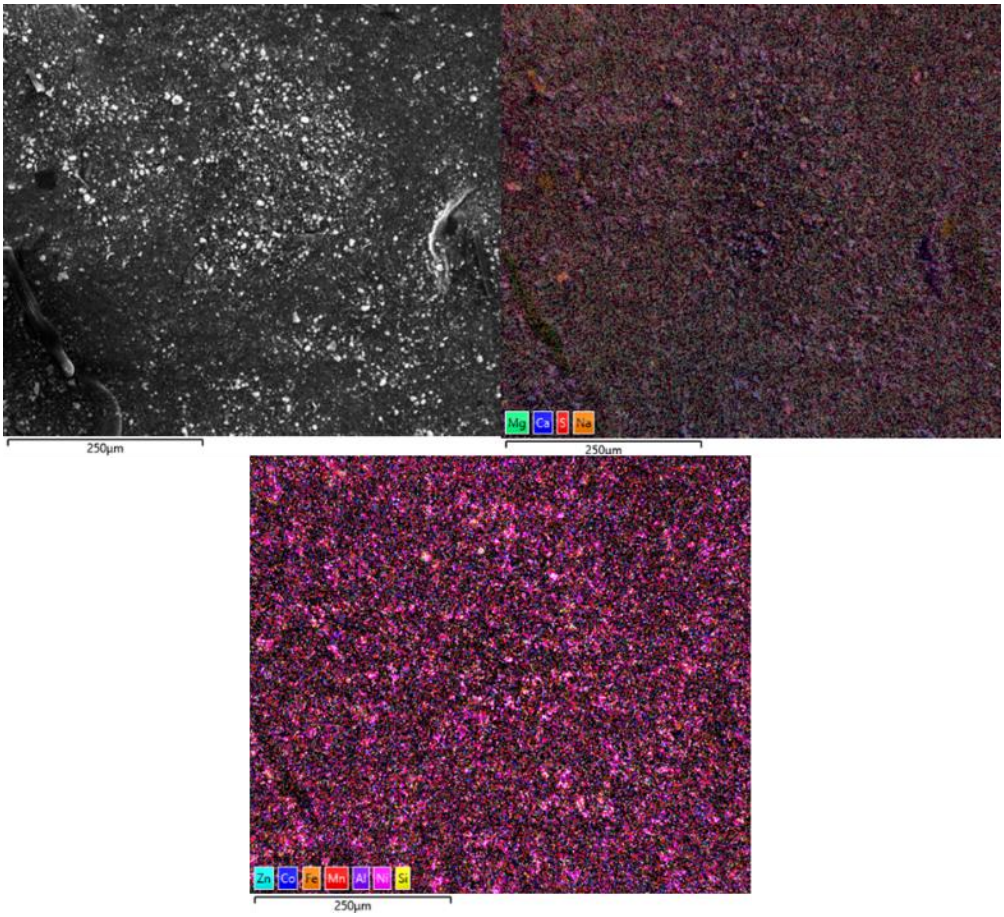



Figure 3.12.58 Electron image of site 1 from Shakespeare C2 REO sample (top left), elemental mapping of contaminant elements (top right), and elemental mapping site of critical metal/metal contaminants (bottom center).

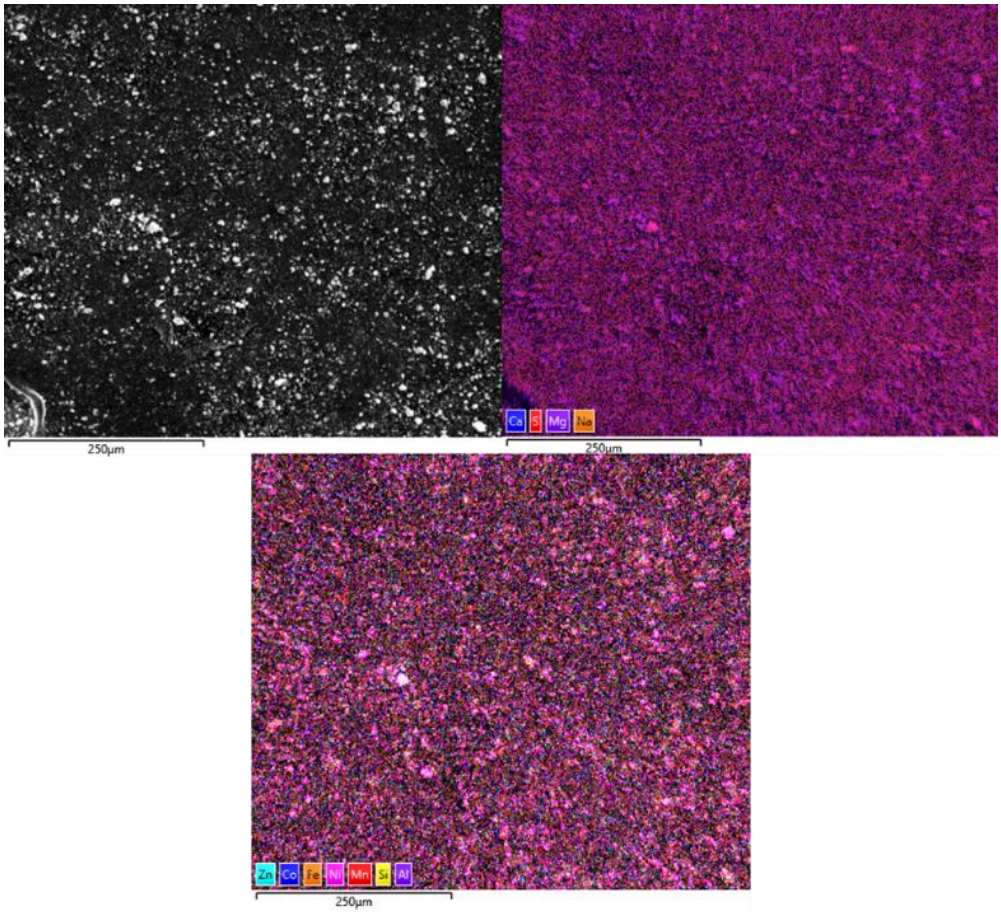



Figure 3.12.59 Electron image of site 2 from Shakespeare C2 REO sample (top left), elemental mapping of contaminant elements (top right), and elemental mapping site of critical metal/metal contaminants (bottom center).

Table 3.12.29 Weight percentages of individual elements, as calculated via SEM-EDX mapping, for two sites of the Shakespeare C3 REO sample.

| Element | Site 1 | | Site 2 | |
|---|---|---|---|---|
| | Wt% | Wt% Sigma | Wt% | Wt% Sigma |
| **O** | 39.12 | 0.13 | 24.39 | 0.07 |
| **Ce** | 15.46 | 0.09 | 19.76 | 0.07 |
| **Y** | 12.32 | 0.09 | 15.39 | 0.07 |
| **Nd** | 10.69 | 0.09 | 13.37 | 0.07 |
| **La** | 4.62 | 0.08 | 5.87 | 0.05 |
| **Gd** | 3.39 | 0.09 | 4.28 | 0.07 |
| **Dy** | 2.82 | 0.09 | 3.56 | 0.06 |
| **Sm** | 2.68 | 0.09 | 3.39 | 0.07 |
| **Pr** | 2.35 | 0.09 | 2.92 | 0.06 |
| **Ca** | 2.05 | 0.02 | 1.96 | 0.01 |
| **Er** | 1.40 | 0.09 | 1.84 | 0.06 |
| **Yb** | 0.92 | 0.10 | 1.33 | 0.07 |
| **S** | 0.50 | 0.02 | 0.41 | 0.01 |
| **Si** | 0.39 | 0.02 | 0.37 | 0.01 |
| **Na** | 0.33 | 0.05 | 0.27 | 0.03 |
| **Ni** | 0.31 | 0.04 | 0.32 | 0.03 |
| **Sc** | 0.20 | 0.02 | 0.21 | 0.01 |
| **Cl** | 0.20 | 0.02 | 0.20 | 0.01 |
| **Al** | 0.19 | 0.03 | 0.11 | 0.02 |
| **K** | 0.07 | 0.02 | 0.04 | 0.01 |

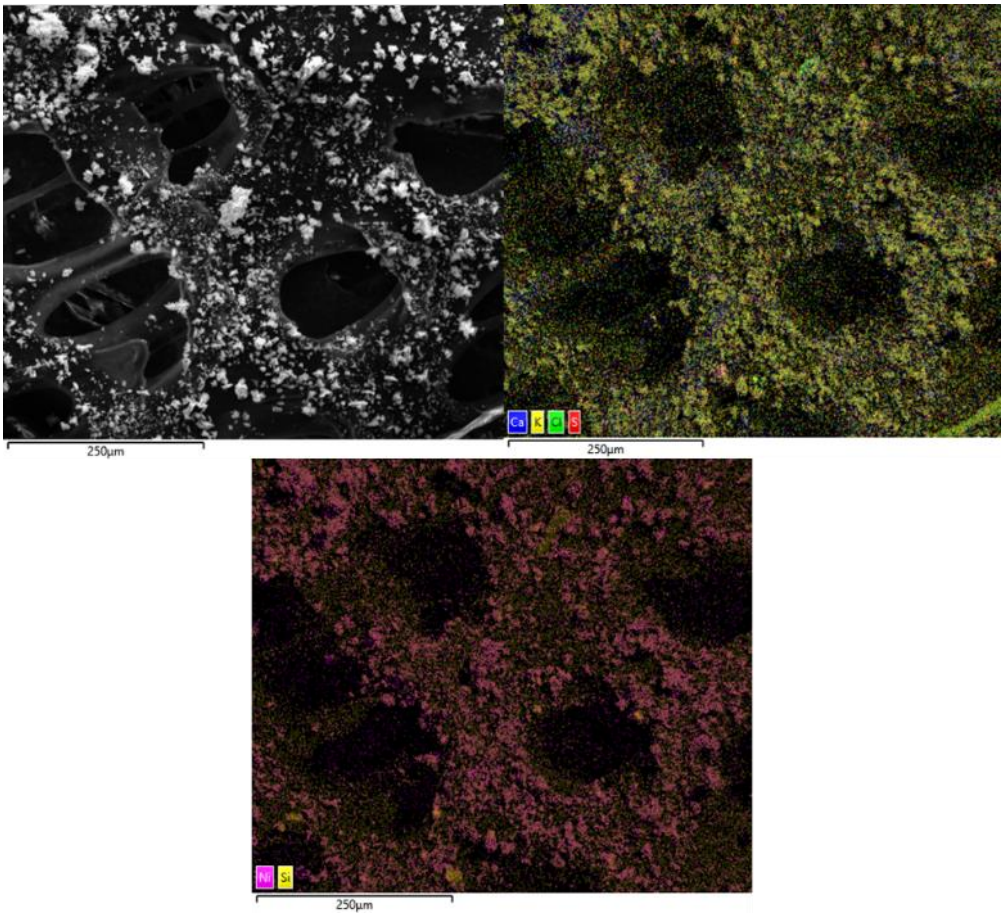



Figure 3.12.60 Electron image of site 1 from Shakespeare C3 REO sample (top left), elemental mapping of contaminant elements (top right), and elemental mapping site of critical metal/metal contaminants (bottom center).

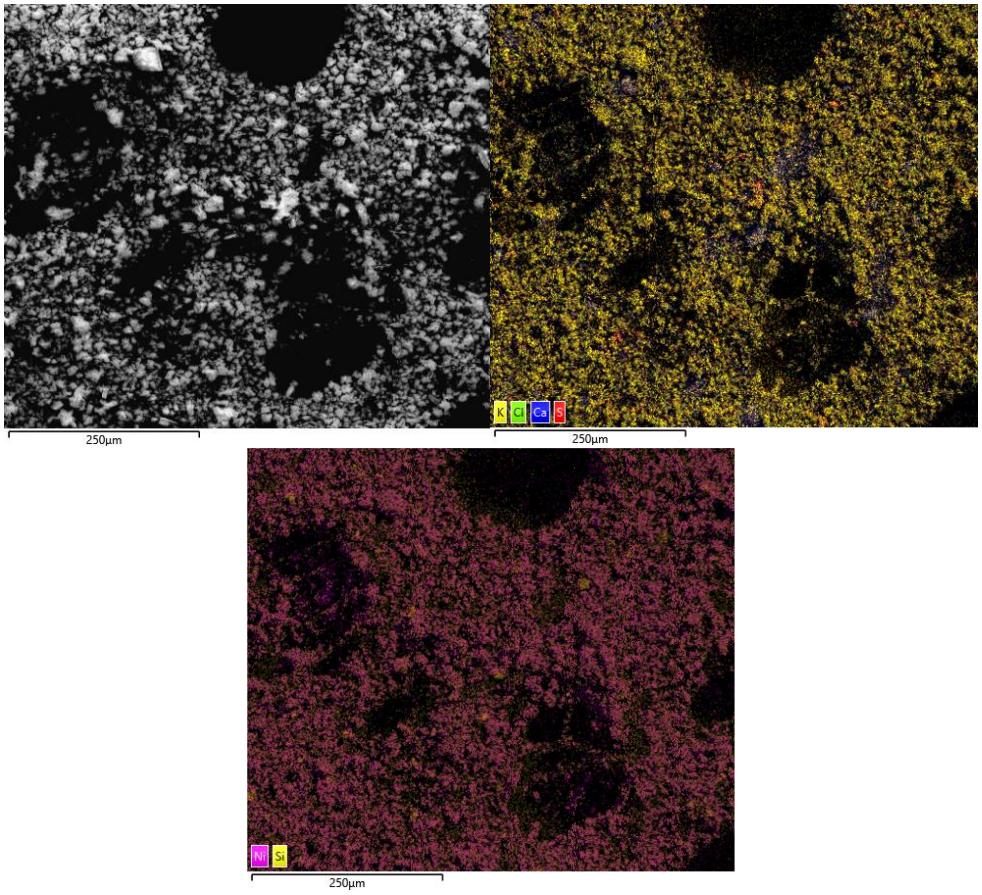



Figure 3.12.61 Electron image of site 2 from Shakespeare C3 REO sample (top left), elemental mapping of contaminant elements (top right), and elemental mapping site of critical metal/metal contaminants (bottom center).

Table 3.12.30 Weight percentages of individual elements, as calculated via SEM-EDX mapping, for two sites of the Shakespeare C4 REO sample

| Element | Site 1 | | Site 2 | |
|---|---|---|---|---|
| | Wt% | Wt% Sigma | Wt% | Wt% Sigma |
| **O** | 27.01 | 0.07 | 28.23 | 0.08 |
| **Ce** | 18.40 | 0.07 | 17.92 | 0.07 |
| **Y** | 12.72 | 0.06 | 12.27 | 0.07 |
| **La** | 9.91 | 0.06 | 9.63 | 0.06 |
| **Nd** | 8.17 | 0.07 | 8.07 | 0.07 |
| **Na** | 5.03 | 0.05 | 5.02 | 0.05 |
| **Mg** | 3.00 | 0.03 | 2.89 | 0.03 |
| **Pr** | 2.23 | 0.07 | 2.19 | 0.07 |
| **Si** | 2.23 | 0.02 | 2.57 | 0.02 |
| **Dy** | 2.09 | 0.06 | 1.99 | 0.06 |
| **Gd** | 1.65 | 0.07 | 1.77 | 0.07 |
| **Sm** | 1.64 | 0.06 | 1.67 | 0.07 |
| **Ni** | 1.47 | 0.03 | 1.41 | 0.03 |
| **Co** | 1.30 | 0.03 | 1.20 | 0.03 |
| **Ca** | 1.27 | 0.01 | 1.20 | 0.01 |
| **Zn** | 1.01 | 0.04 | 0.93 | 0.04 |
| **Al** | 0.58 | 0.02 | 0.67 | 0.02 |
| **S** | 0.23 | 0.01 | 0.27 | 0.01 |
| **K** | 0.09 | 0.01 | 0.12 | 0.01 |

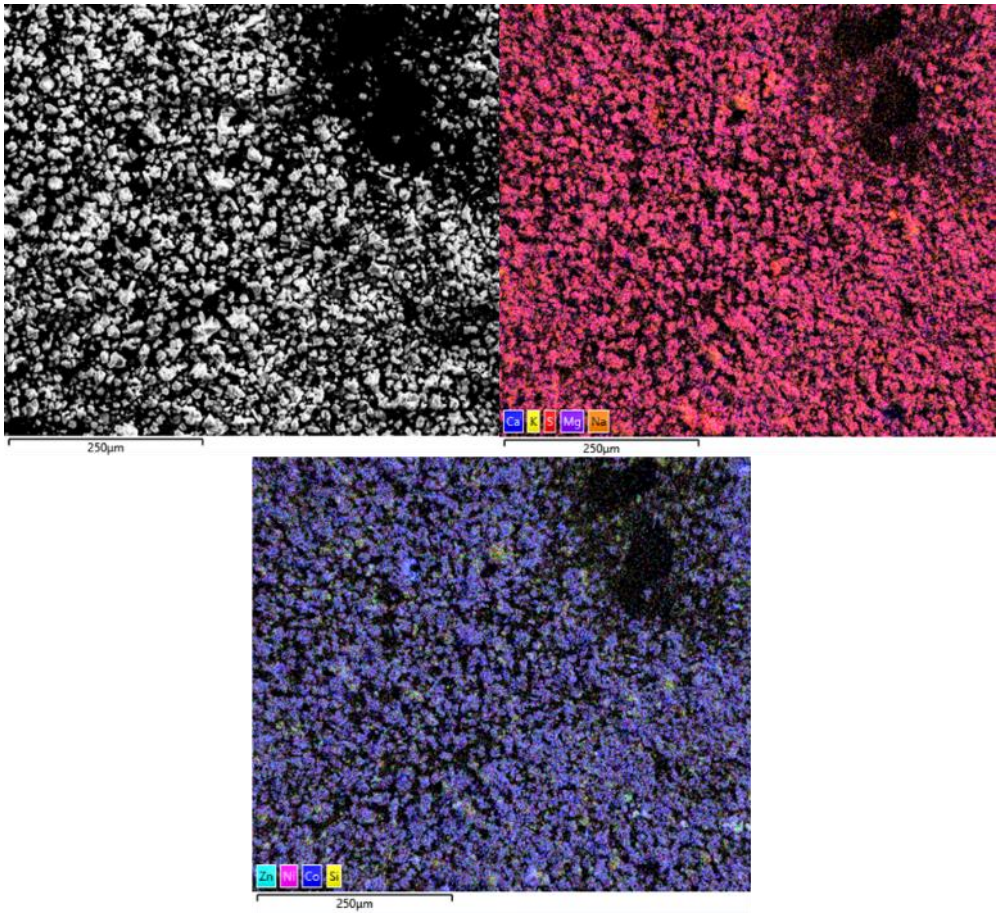



Figure 3.12.62 Electron image of site 1 from Shakespeare C4 REO sample (top left), elemental mapping of contaminant elements (top right), and elemental mapping site of critical metal/metal contaminants (bottom center).

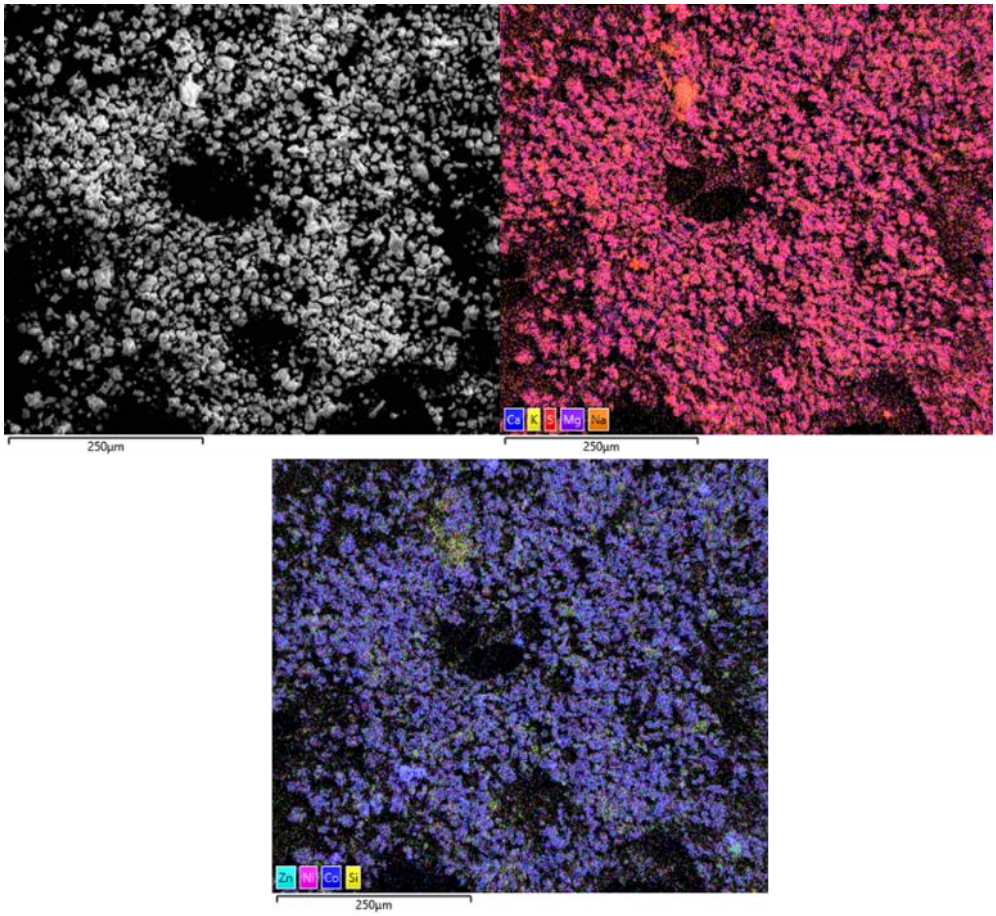



Figure 3.12.63 Electron image of site 2 from Shakespeare C4 REO sample (top left), elemental mapping of contaminant elements (top right), and elemental mapping site of critical metal/metal contaminants (bottom center).

Table 3.12.31 Weight percentages of individual elements, as calculated via SEM-EDX mapping, for one site of the Shakespeare C5 REO sample.

| Element | Site 1 | |
|---|---|---|
| | Wt% | Wt% Sigma |
| **O** | 38.88 | 0.11 |
| **Ce** | 12.79 | 0.07 |
| **Ca** | 10.47 | 0.04 |
| **Nd** | 7.80 | 0.07 |
| **Y** | 7.79 | 0.06 |
| **La** | 5.73 | 0.06 |
| **Fe** | 2.94 | 0.04 |
| **Gd** | 1.93 | 0.07 |
| **Sm** | 1.89 | 0.07 |
| **Si** | 1.86 | 0.02 |
| **Dy** | 1.85 | 0.08 |
| **Pr** | 1.80 | 0.07 |
| **Zn** | 1.13 | 0.04 |
| **Na** | 0.70 | 0.05 |
| **Yb** | 0.67 | 0.08 |
| **Ni** | 0.51 | 0.03 |
| **Al** | 0.51 | 0.02 |
| **S** | 0.49 | 0.01 |
| **Cl** | 0.15 | 0.01 |
| **K** | 0.12 | 0.01 |

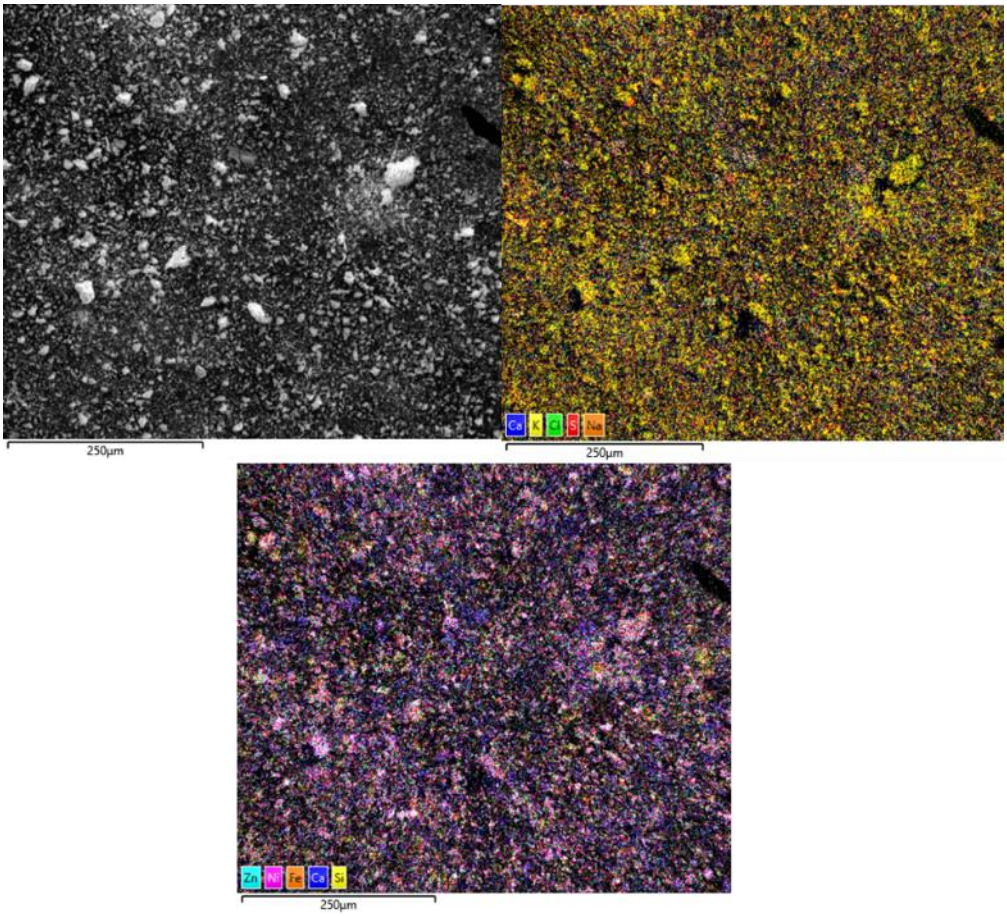



Figure 3.12.64 Electron image of site 1 from Shakespeare C5 REO sample (top left), elemental mapping of contaminant elements (top right), and elemental mapping site of critical metal/metal contaminants (bottom center).

### *3.12.7 Bio Acid Reactor Operations*

Acidithiobacillus Ferrooxidans are a bacterium that oxidizes iron to create sulfuric acid. If the temperature is lower than 30 degrees Celsius, the bacteria's metabolism begins to slow. If the temperature is higher than 37 degrees Celsius, the bacteria overheat and begin to die. As such, the bacteria are most effective around 34.5 degrees Celsius, or 94 degrees Fahrenheit. To generate sulfuric acid, *A. Ferrooxidans* require a source of iron, sulfur, and oxygen. Iron and sulfur are provided through the addition of a pyrite slurry and the oxygen is provided by aerating the solution.

Three 3000-gallon bioreactors were utilized for the production and storage of the generated acid on site. The concentrated pyrite material was fed to Bioreactor No. 1 and, as a result, the majority of the oxidation reaction took place in this reactor. Bioreactor No. 2 received liquid overflow from the top of Bioreactor No. 1 and provided additional residence time to oxidize residual solids from the Bioreactor No. 1. Bioreactor No.3 was the overflow tank, so liquid from the top of Bioreactor No. 2 overflowed into Bioreactor No. 3, and acid was extracted from Bioreactor No. 3. Bioreactor No 3 further served as a storage volume in case bacteria in reactors No. 1 or No. 2 are lost. Bioreactors No.1 and No.2 both contain 40 hp Denver flotation aeration units. These units received air from a blower an dispersed the air in the form of bubbles into the bioreactors. Dissolved air in the solution is required for bacteria population growth. These tanks also had a ventilation system to exhaust fumes and 8-tons of colling chillers each to overcome the heat generated by mechanical and chemical energy.

**Indicators of a healthy bacterial conversation:**

A healthy population of *A. Ferrooxidans* has an Eh, or Reduction/oxidation potential between 600 and 700 mV. The bioreactors were typical operated with an Eh value of 630 mV. The population of *A. Ferrooxidans* was approximated by the time it took for a known quantity of ferrous sulfate to be digested by the population. The addition of ferrous sulfate into a bioreactor caused the Eh to sharply drop. Once the population digested all of the ferrous sulfate, the Eh returned to its original value.

Eh of the solution as a function of time is shown in Figure 3.12.65. During this time frame, 12.5 kg of ferrous sulfate was added. This test was performed on July 9$^{th}$, 2021 and it only took 25 minutes for the population to fully digest all of the ferrous sulfate. The test should take around 5 minutes to return to normal Eh values if bacteria was at full population/concentration,. During the growth from 200 gallons to 3000 gallons, the bacteria population had to be heavily diluted to fill the bioreactors to the minimum level. The results of this oxidation test show that the bacteria needed additional time to grow in the 3000-gallons reactors.

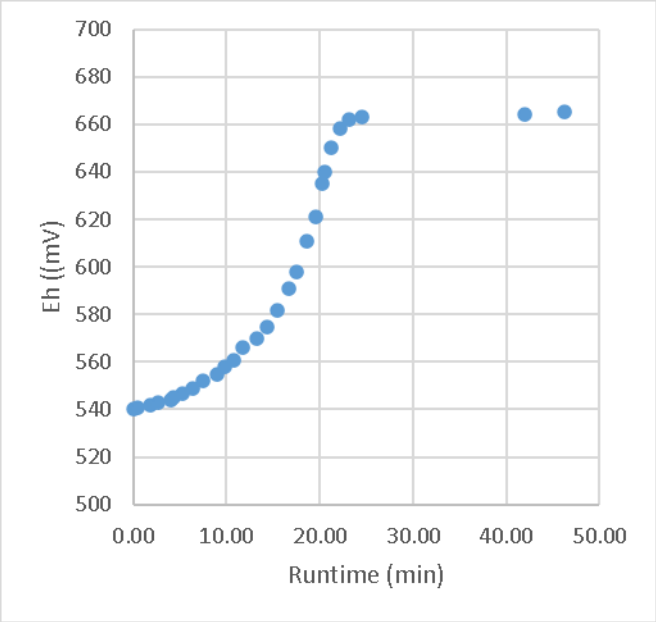



Figure 3.12.65 Impact of time on Eh of the solution.

**Heating Issues:** The initial bacteria population was grown in a 250-gallon reactor equipped with a slam-jet bubble generator to disperse the air. While the bacteria population was growing in a 250-gallon reactor, heat had to be added to maintain the ideal temperature of 34 degrees Celsius. In prior experiences with Bioreactors No. 1 and No. 2 units, the slurry had a much shorter residence time, so heat was able to be removed as fluid was being extracted. In the current situation where the population is growing and no fluid is being extracted, any heat input by the floatation unit remains inside of the system.

The initial charging of the *A. Ferrooxidans* into Bioreactor No. 1 involving pumping 1750 gallons of pyrite feed into the reactor. To keep the 8% solids suspended, the agitator was left on overnight. By the next morning, the pyrite feed in the tank was up to 50 degrees Celsius. To confirm, Bio-reactor #2 was filled with water and the agitator was left on. The same temperature was reached by the water. This confirmed mechanical agitation of the aerator imparted significant quantities of energy. This was unanticipated as the heat load due to mechanical energy was calculated. This finding was significant as the cooling capacity was not sufficient to maintain optimum temperatures.

In response, temperature versus time data was collected as the tanks were heated or cooled by various methods. The findings are presented in Table 3.12.32. For the purposes of these calculations, the heat capacity and density of the pyrite slurry was approximated by that of water. In reality, the density of the pyrite or agitated slurry is higher than that of water, meaning that the Q values for the tests are smaller than they should be. For the first test shown in Table 3.12.32, the Denver aeration unit was switched off and water was left to cool through the tank walls. Since this cooling effect will be present during all tests, it was subtracted to determine the heat load. At 94 degrees F, about 0.28 degrees F were lost per hour just through the tank wall. This comes out to about 4000 Btu/hr of cooling. When applied to the full tank, there should be a higher amount of heat lost through the tank walls due to more surface area being covered.

Table 3.12.32 Temperature response data for bio-reactors cooling with no agitation.

| 1750 gallons of water cooling through tank walls | | | |
|---|---|---|---|
| Variable | Value | Units | Assumptions/Notes |
| Slope | -33.852 | | Data needs to get down to 90 C before starting the next test. Ambient Temp for all of this data was around 80 F |
| Intercept | 256.0271 | | |
| Tank Volume | 1750 | gal | |
| Fluid Density | 62.11 | lbs/ft3 | |
| Fluid Heat Capacity | 0.998 | Btu/lbm-F | |
| Goal Temp | 94 | F | $94^o$ F is the ideal operating temperature. |
| Slope at Goal Temp | -0.28242 | F/hr | This represents heat loss through tank walls, the basis for the rest of the tests |
| $Q_{tot}$ | -4095.71 | Btu/hr | |

The next experiment provided the results in Table 3.12.33. The test was involved allowing the water to heat while due to the mechanical energy of the aerator. When the temperature rose past 94 degrees F, it was rising at a rate of 1.7 degrees F per hour. Accounting for the heat lost through the tank walls, the agitator alone was raising the water temperature by 2 degrees F per hour, or almost 30,000 Btu/hr. Forty (40) horsepower converts to about 100,000 Btu/hr, so 29% of the energy input by the motor was expended by the heating of the mixture.

Table 3.12.33. Temperature response data for bio-reactors due to mechanical energy.

| Heating of 1750 gallons of water due to Agitator | | | |
|---|---|---|---|
| Variable | Value | Units | Assumptions/Notes |
| Slope | 242.9334 | | |
| Intercept | -1105.43 | | |
| Tank Volume | 1750 | gal | |
| Fluid Density | 62.11 | lbs/ft3 | |
| Fluid Heat Capacity | 0.998 | Btu/lbm-F | |
| Goal Temp | 94 | F | |
| Slope at Goal Temp | 1.742837 | F/hr | Heat loss through walls + Agitator heat |
| Qtot | 25274.66 | Btu/hr | |
| Agitator dT/dt | 2.025261 | F/hr | Heat generated just by agitator |
| Q_Agitator | 29370.38 | Btu/hr | |

The initial setup had 3-ton chiller units installed on each bioreactor to control the temperature of the slurry. For the next test scenario (Table 3.12.34), the single chiller was connected to a pair of cooling coils submerged in the tank. The coils provided 5-$ft^2$ of contact area. With this setup, the chiller was unable to counteract the heat generated by the agitator and the pyrite oxidation reaction. A 3-ton chiller is capable of providing 36000 Btu/hr of cooling but the unit was only able to provide 1.5 tons of chilling to our system.

Table 3.12.34. Temperature response to a single 3-ton chiller to the heat load.

| Heat removed from 1800 gallons of agitated slurry by 1 Chiller | | | |
|---|---|---|---|
| Variable | Value | Units | Assumptions/Notes |
| Slope | 34.22299 | | This agitated slurry contains pyrite, which adjusts some of the fluid properties. |
| Intercept | -54.127 | | |
| Tank Volume | 1800 | gal | |
| Fluid Density | 62.11 | lbs/ft3 | |
| Fluid Heat Capacity | 0.998 | Btu/lbm-F | |
| Goal Temp | 94 | F | |
| Slope at Goal Temp | 0.451404 | F/hr | Heat loss through walls + Agitator heat + heat loss from chiller |
| Qtot | 6733.304 | Btu/hr | |
| Chiller dT/dt | -1.24302 | F/hr | Heat removed by one chiller operating on both tubes |
| Q_Chiller | -18541.4 | Btu/hr | |

The next test involved installing both three-ton chillers on one bioreactor. For this setup, each cooling coil in the tank had its own chiller. The test revealed that the chilling units were able to overcome the energy input by the agitator and allow for the temperature to fall in the tank. The six tons of chilling capacity were only removing 30000 Btu/hr from the system, or only 2.5 tons of chilling, so they were even less efficient than the single chiller setup.

| Heat removed from 1800 gallons of agitated slurry by 2 Chillers | | | |
|---|---|---|---|
| Variable | Value | Units | Assumptions/Notes |
| Slope | -54.2418 | | |
| Intercept | 364.1524 | | |
| Tank Volume | 1800 | gal | |
| Fluid Density | 62.11 | lbs/ft3 | Assume density of water |
| Fluid Heat Capacity | 0.998 | Btu/lbm-F | |
| Goal Temp | 94 | F | |
| Slope at Goal Temp | -0.37267 | F/hr | Heat loss through walls + Agitator heat + heat loss from two chillers |
| Qtot | -5558.87 | Btu/hr | |
| 2Chiller dT/dt | -2.06709 | F/hr | Heat removed by 2 chillers operating on one tube each. |
| Q_2Chiller | -30833.5 | Btu/hr | |

*Solution* - To address the deficiency in chilling capacity, two 5-ton chillers were installed. Each bioreactor had a 3-ton chiller connected to one cooling coil, and a 5-ton unit on the other cooling coil. The total 8 tons of chilling capacity per bio-reactor was capable of overcoming the heat added by the agitator, as well as the chemical heat generated by pyrite oxidation. To prevent an overheating event from happening again, these units were connected to a PLC. The control system was able to maintain the bioreactor temperature at a steady 94 F (34.5 C). This stabilization of temperature created a more consistent environment for the growth of the *A. Ferrooxidans* populations and even contribute to a faster growth rate.

**Failure of Bioreactor No. 1:** Loading of the initial charge into Bioreactor No. 1 (see Figure 3.12.66 for pictures of installed bio-reactors) from the 250-gallon incubation tank occurred on or about the 11$^{th}$ of June 2021. Operation of the bio-reactor proceeded without incident until Friday the 25$^{th}$ of June 2021. During inspection, the operator observed that agitation was not being provided by the Denver aerator mechanism even though the shaft was spinning and supplied with power. At this point, problem solving activities commenced which included turning off the agitator and observing. It was found that there was little to no resistance on the

mechanism. To further determine root cause, the team then proceeded to remove the bio-acid solution from reactor No. 2 to allow for visual inspection of the mechanism and particularly the rotor.




Figure 3.12.66. Picture of Bioreactors No. 1 and No. 2 with associated 40-hp Denver flotation aeration mechanisms as installed in the pilot plant.

The investigation revealed that the rotor had become disconnected from the shaft along with the diffuser and was laying on the bottom of the tank. Figure 3.12.67 shows the state of the mechanism when the solution was removed from the tank. It was observed that both the rotor and the diffuser were laying on the bottom of the tank.



Figure 3.12.67. Picture of the state of the Denver flotation mechanism after initial pump down of bio-reactor #1 showing the rotor laying on the bottom of the tank. The vanes of the rotor are visible through the inlet along with the diffuser laying on top of the rotor.

For reference, manufacture supplied drawings are shown in Figure 3.12.68 and Figure 3.12.69. Figure 3.12.68 shows the mounting points for the assembly and the placement of the bearings. The drive shaft is shown by the hidden lines running from the rotor to the bearings. The stator is attached to the inlet tube which is shown mounted to the air tube which is attached to the mounting plate containing the bearings. A detail view of the stator/shaft attachment is shown more fully in Figure 3.12.69. A 1-¼ bolt secures the stator to the shaft with three threaded dowel pins to prevent the stator from spinning on the shaft. The assembly is coated throughout with natural rubber which has been vulcanized in place.

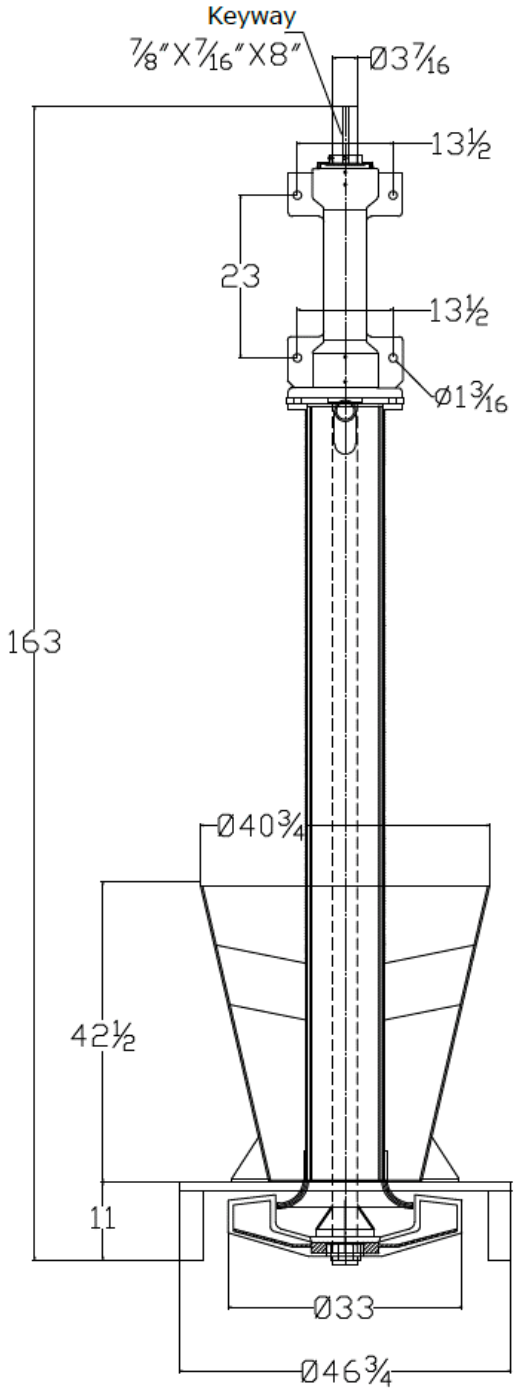



Figure 3.12.68. Diagram with dimensions showing the rotor, stator, diffuser, inlet, air tube and drive shaft.

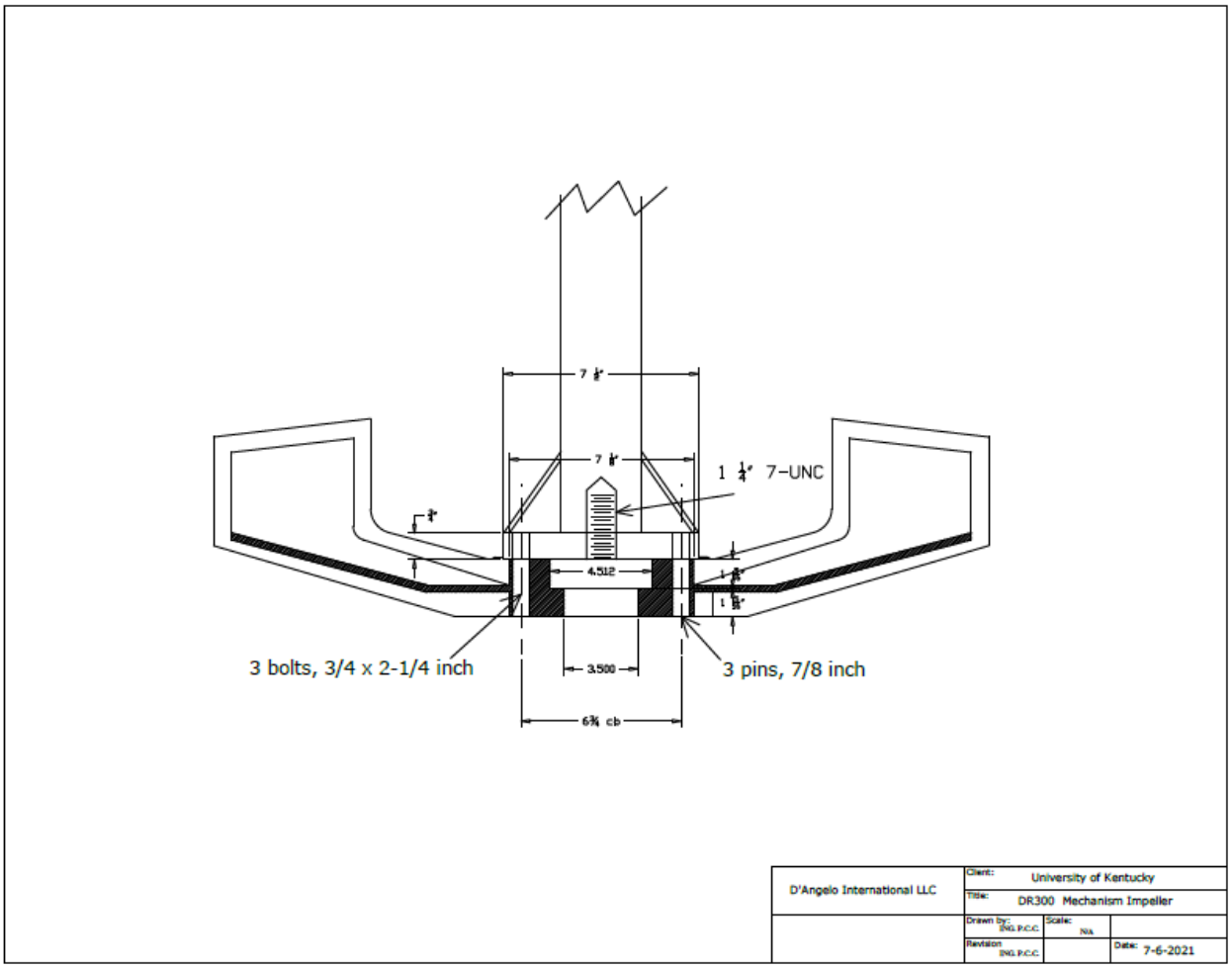



Figure 3.12.69. Detailed diagram showing the connection between the drive shaft and the rotor with a center bolt and retaining pins to prevent rotation of the rotor on the shaft.

The rotor and stator were removed from the bottom of Bioreactor No. 2 and cleaned. The team discovered that over the course of two weeks the bacteria had corroded up to 1.5 inches of steel in some places. The failure mode was corrosion of the steel which removed the locating dowels allowing the rotor to spin on the shaft and unscrew the center bolt, thereby releasing the rotor. Figure 3.12.70 shows the state of the rotor immediately upon removal from the tank. Note the three holes for the locating pin and the erosion of the flat mounting point. Figure 3.12.71-Figure 3.12.73 are provided to show and document the extent of the damage. Figure 3.12.73 shows the condition of the center bolt which remained intact. The impeller vanes on the inside of the stator also exhibited an interesting wear pattern as shown in Figure 3.12.74. This wear is characterized by an uneven erosion of the rubber toward the top of the rotor vanes.



Figure 3.12.70. Rotor showing eroded mounting point to the drive shaft.



Figure 3.12.71. Close up view of the rotor shaft mounting point showing erosion damage and the remnants of the 3 locating dowels.



Figure 3.12.72. Detail view of erosion profile on rotor and remnants of sealing compound where the rotor sealed to the shaft.



Figure 3.12.73. Picture of center bolt showing little to no damage.



Figure 3.12.74. Rubber wear on the rotor vanes on the center facing surfaces.

As mentioned, the diffuser was also found detached from the air inlet tube. The diffuser is the bell-shaped member which attaches to the air inlet shaft and provides guidance to the slurry and air as upon entry into the rotor eye as shown in Figure 3.12.68. It attaches to the air inlet with an overlapping joint and was attached with 4 stainless steel bolts. The erosion on both the topside and the bottom side of the diffuser was extensive. Figure 3.12.75-Figure 3.12.77 document the extent of the corrosion in the order of 2-3 inches depending on location.

The drive shaft and air tube are shown in Figure 3.12.78-Figure 3.12.81. The drive shaft flange showed significant erosion corresponding to that seen on the rotor mating surface. The exterior of the flange was destroyed outboard of the welded joint. The surface of the shaft interior to this welded joint was mostly intact, which was convenient for affecting repairs on the unit. The termination of the air tube was found to be entirely missing from bacterial enhanced corrosion. The figures show that the entirety of the overlapping joint between the diffuser and the air inlet tube was missing. This allowed the diffuser to separate and was believed to have contacted the interior faces of the rotor. This is believed to have caused the wear pattern exhibited on the interior surfaces of the rotor vanes. The reminder of the inspection is shown in Figure 3.12.82-

Figure 3.12.84 detailing the top of the stator and the condition of the stator vanes. No additional issues were documented during the inspection.



Figure 3.12.75. Bottom edge of the diffuser which is closest to the impeller showing the erosion of the steel from the edge in.



Figure 3.12.76. Detail view of the steel surface on the diffuser showing about 2 inches of erosion.



Figure 3.12.77. Cut way view of the diffuser from bio-reactor #1 show the extent of erosion. The red and black surfaces where the rubber has been cut away is the remain steel.



Figure 3.12.78. View of the drive shaft mating surface where the rotor attached showing the center threads and flange. Also shown is the bare and eroded air tube. The view is from the bottom of the tank looking upward.



Figure 3.12.79. Alternate view showing the drive shaft flange, residual joint sealing compound and the termination end of the air tube. Note the difference in the length of the rubber on the air tube. This corresponds to mating surface for the diffuser.



Figure 3.12.80. Alternate view showing the position of the drive shaft and the condition of the rubber on the stators.



Figure 3.12.81. Picture showing the erosion on the air tube termination where the diffuser connects.



Figure 3.12.82. View of the top of the stator and inlet during inspection.



Figure 3.12.83. Condition of the rubber lining on the stator during inspection.



Figure 3.12.84. View of the stator and inlet throat showing the rubber condition.

**Startup and failure of Bioreactor No. 2:** Due to the failure of Bioreactor No. 1, it was discovered that the coating on the joints and quality of construction was not suitable for the application. While the rubber itself seemed to stand up to the conditions, the quality of construction and method of sealing joins was not sufficient to prevent catastrophic failure. Due to the need to rebuild, the decision was made to continue operations in Bioreactor No. 2 to not lose time and to provide a time buffer whilst Bio tank #3 was converted into a bioreactor for contingency. After 2 weeks and 4 days, Bioreactor No. 2 developed a significant gyration. At this juncture, Bioreactor No. 2 was shut down and pumped out to Bioreactor No. 3. This involved a significant release to the secondary containment when a fitting fatigued and broke. All but 300 gallons was recovered. Inspection revealed that the air inlet tube had failed in a similar manner to that of Bioreactor No. 1 (see Figure 3.12.85). This released the air diffuser, which caused the imbalance observed in the bioreactor.

Inspection of the joint between the rotor and the drive shaft show a seal that appeared to be intact (see Figure 3.12.86) as compared to Bioreactor No. 1. It appears that the coating on the underside of the rotor was not of sufficient thickness to prevent the ingress of bacteria as shown in Figure 3.12.87. The corrosion at two locations on the underside of the rotor were about 3 inches in maximum depth. Corrosion of this nature was not observed in Bioreactor No. 1.



Figure 3.12.85. Damage to the air inlet tube on bio-reactor #2 showing where the air diffuser was connected.



Figure 3.12.86. View of the rotor/drive seal which appears to be still intact.



Figure 3.12.87. Damage on the underside of the rotor on what appears to be a thins spot on the rubber. This shows that the bacteria have penetrated the coating and have corroded out a significant portion of the rotor vane.



Figure 3.12.88. Wear on the exterior of the stator.

*3.12.8 Bio-Reactor Disaster Recovery Plan*

As described previously, failures in Bioreactors No. 1 and No. 2 were experienced. Beginning with Bioreactor No. 1 with two weeks of run time, a decision was made to commence acid production in bio-reactor #2 for three specific reasons. The first was to not lose two weeks of continuing to grow the bacteria and, secondly, based on learnings from Bioreactor No.1, the second reactor would require removal and rebuilt. The third reason was to allow time to convert Bio-tank No. 3, which only had a propeller agitator, to a bioreactor by the addition of gas sparging lines. It was anticipated that the use of the stainless steel agitator and gas addition will be a stop gap measure until the Denver aeration units could be rebuilt.

To rebuild the units, a local contractor was solicited to remove a panel of the roof and remove the agitation mechanisms from reactors 1 and 2. The rebuild of the rotor and drive shaft was simple and a shop was contracted locally to rebuild the corroded steel. Due to the welding that was needed, the rubber lining was cut back. To replace the rubber, an epoxy coating from Belzona was sourced which could be used in high abrasion situations. This epoxy was utilized to replace the rubber and seal joints which experienced bacterial attack. Further, alternate diffusers were constructed out of stainless steel to complete the repair. The repair and refit took approximately 1.5 months before reinstalling.

*1.1.23 Continuous Acid Production*

The rebuild was successfully achieved and the motor and shaft were installed back into the Bioreactors No. 1 and No. 2. The bacteria population that was incubated in Bioreactor No. 3 during the repair of Bioreactors No. 1 and No. 2 was moved back to Bioreactor No. 1 to

continue growth on August 20th, 2021. Ferrous oxidation test was carried out on Bioreactor No. 1 to keep track of the bacteria activities. For each of these tests, a known amount of ferrous sulfate (between 2 to 8 kg) was added into the bioreactor. This dropped the Eh to a minimum, and the return to the original value was tracked. The time taken for the Eh to return to its original values can be used to estimate the ferrous oxidation rate (Figure 3.12.89). The bacteria population was low when the bugs were initially transferred into Bioreactor No. 1, which was indicated by low initial ferrous ion consumption rate. As the population increased, the bioreactor was able to consume the known amount of ferrous sulfate at a faster rate. The high point on September 3rd may be because two ferrous sulfate tests were done in the same day. The datalogger went down during the test and the Eh values were not being recorded. An hour or so later, another test was carried out, which caused the result to be significantly higher than the previous data.

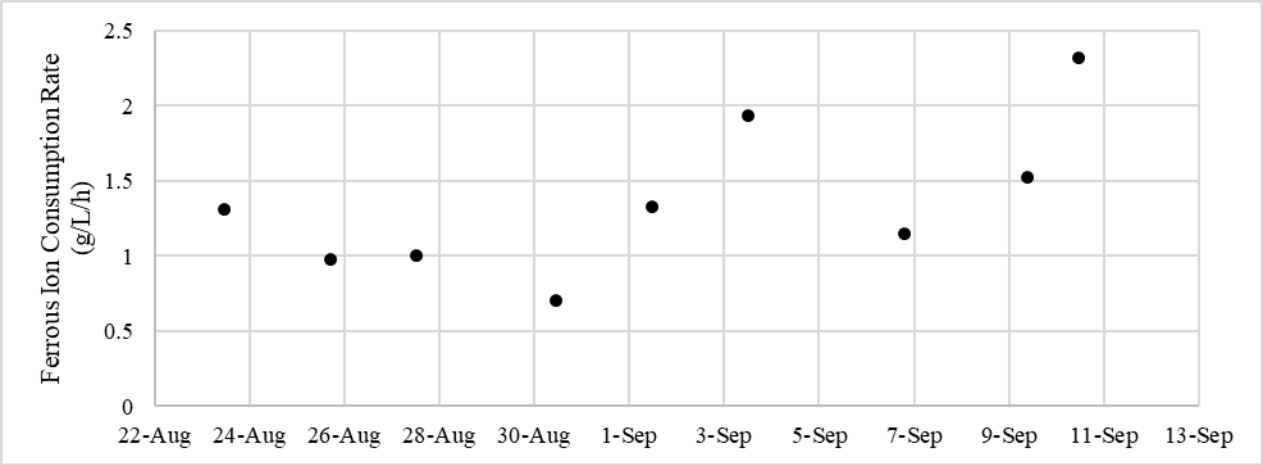



Figure 3.12.89. Ferrous ion consumption rate in Bio #1 which indicate the bacteria

The Eh, pH and temperature of the Bioreactor No. 1 were monitored and recorded after the bacteria were moved from Bioreactor No. 3 to Bioreactor No. 1 on August 20, 2021. The feeding schedule resumed to be Monday, Wednesday, and Friday each week. The Eh, pH and temperature data were recorded in the PLC system with a time stamp. Data collected every 3 hours are plotted in Figure 3.12.90. On August 27, 2021, there was a sharp drop in pH. This was because the probes were recalibrated that day. On August 30, 2021, the pattern on the Eh chart changed. When feeding MWF, the bugs had more time to recover and raise the Eh. With Eh values steadily over 700, the bacteria population was ready to be fed more, so the feeding/extraction rate went from 600 gal/week to 1000 gal/week. These shorter lines have a steeper slope than the long ones, meaning that the population was responding well to the change in pH.

On September 4 (2021), there was a steep rise in Eh as well as a sharp drop in temperature. This was because the belt between the 40 hp motor and the agitator shaft snapped. This caused the agitator to stop turning and bacteria and pyrite began to settle out. The blower was still on, so there was a small amount of agitation, but not what was needed to keep 14% solids in suspension. The belt was replaced on September 6th (2021) and this caused the temperature to start rising again, and the Eh value to return to normal levels.

With the current system of batch feeding, the solids concentration quickly rose during a 10-minute period while the feed was being added into the system. Solids were slowly being extracted with the bio acid we were extracting but solids were accumulating in Bioreactor 1. On September 10th, the agitator's amperage was monitored before and after feeding the reactor. It started at 17.5 amps and rose to 20.3 amps after the feeding. The motor was able to handle

50 amps, but this rise in current shows that the motor was working harder after a feeding. With the recent change to daily feedings and recycling of the bio leaching reactor thickener's underflow into Bioreactor No. 1, the resulting high torque required to move the slurry could explain the breakage of the belt. At this time, a multi belt pulley system was purchased and installed on both reactors.

Between September 15 to 21 (2021), data was not stored in the server due to OSI-soft testing and connections, and the pH probe wearing off. The probe was taking multiple days to calibrate, which was a sign of a bad probe. The probe was replaced with the one from Bioreactor No. 2 and the calibration issues persisted. On September 28th, the technician was able to calibrate the probe but it was showing a higher reading compared to the lab probe. The pH data from September 28th and later was about 0.2 higher than the lab probe.

On September 23rd (2021), the chiller for temperature control in Bioreactor 2 was tested. The agitator and chilling system started on September 23rd without adding any bacteria to the reactor as a test. This was a different chilling unit than Bioreactor No. 1, and thus needed to be tested. There were several spikes in Bioreactor No. 2's temp, and that happened when a high-pressure warning occurred and the chiller shut down. This was due to the refrigeration system containing too much refrigerant which was solved by bleeding off some refrigerant.

On September 28th, the overflow from Bioreactor No. 1 was transferred to Bioreactor No. 2. The aeration line for Bio 2 was turned on. The air line for Bioreactor 1 tees off of the line to Bioreactor No. 2. When the valve for Bioreactor No. 2 was opened, more of the air went into Bioreactor No. 2 instead of Bioreactor No. 1 due to the aqueous level differences. Bioreactor No. 2 created less resistance to air resulting in higher aeration than that of the Bioreactor No. 1. This essentially stopped Bioreactor No. 1 from getting the air it needed and caused a drop in Eh value similar to that realized when the blower was shut off. To remedy this, a three-inch PVC line was plumbed to replace the air split that had two valves on it. This allowed air to be equally distributed between Bioreactor No. 1 and Bioreactor No. 2.

Bioreactor No. 3 was converted to a feed holding tank. A 1.5-inch AODD line circulated fluid from the bottom to the top of the tank. Teeing off that line was a valve to a peristaltic. The peristaltic pump had two heads, one pushing feed into Bioreactor 1, and the other pulling acid out of Bioreactor 2. The bio acid then travelled to a thickener, where the overflow was run through an inline filter and pumped out to a bio-acid storage. The underflow contains about 16% pyrite. Given the fresh feed contained more than 50% pyrtite, the underflow was treated to produce a solids waste.

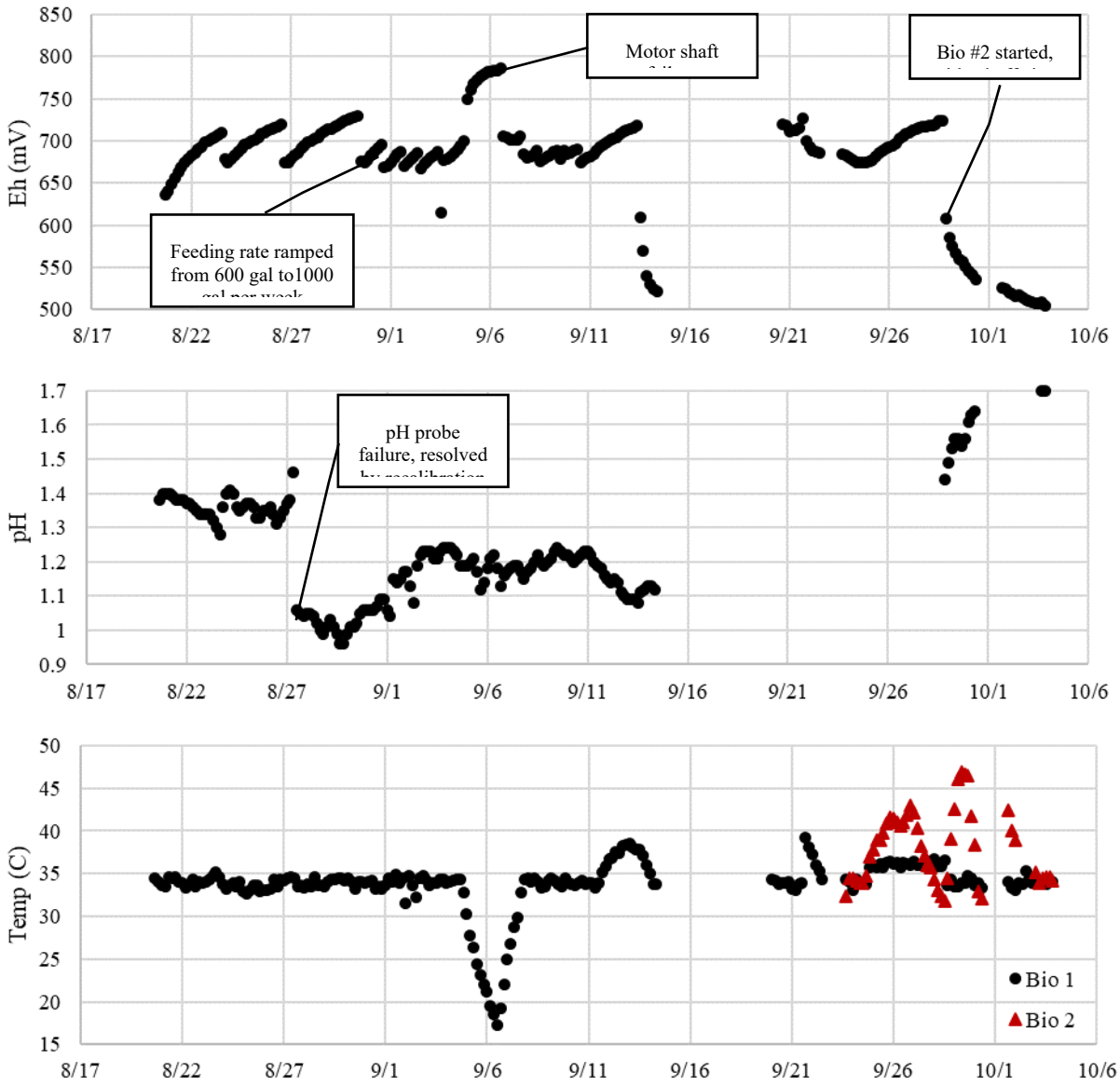



Figure 3.12.90. The Eh, pH, and temperature data recorded for Bioreactor No. 1.

In early October 2021, the bioreactor system switched to continuous feeding instead of batch feeding of pyrite feed. As shown in Figure 3.12.91, the ORP started off very low shortly after the transition to continuous instead of batch feeding. Bioreactor No. 3 in the reactor circuit held a surplus of feed which was low on both dipotassium phosphate and ammonium sulfate. Once this was realized, feed was shut off and the bacteria received additional nutrients each day until the ORP rose to meet expectation. Once the ORP reading reached a value above 600 mV, continuous feeding was resumed.

The practice of continuous feeding provided some learnings about the pyrite feeding system. As shown in Figure 3.12.91, the redox potential in bio reactor system showed several spikes in the month of November 2021 which was a result of inconsistent feed rate of pyrite. Pyrite has a relatively high settling velocity, and the inline velocity of the pyrite feed slurry was low in the half inch feeding and extraction lines, therefore causing some clogging issues in the lines. This provided a challenge for the objective to achieve a steady state for the bioreactor system. To resolve this issue, smaller tubing sizes were tested to prevent the pyrite particle from settling in the lines without causing particle bridging effects.

Due to the high settling rate of pyrite in the feed preparation tank (Bio 3), an air operated diaphragm pump was used to recirculate the material in the tank to maintain pyrite suspension. At the end of November 2021, the recirculation pump experienced a failure due to the abrasion effect on the pump interior which caused the plastic flanges to crack. On November 29 (2021), a peristaltic pump was installed to transport material from the feed tank to Bioreactor No. 1 to resume continuous feeding. A reduction in feed rate resulted in an increase of the residence time in the bioreactor, thereby promoting the production of bacteria and causing a significant improvement in the generation of acid. As shown in Figure 3.12.92, the pH in the bioreactor reached 0.8 in early December 2021. The pH went up when the feed rate was resumed to normal and residence times were reduced.

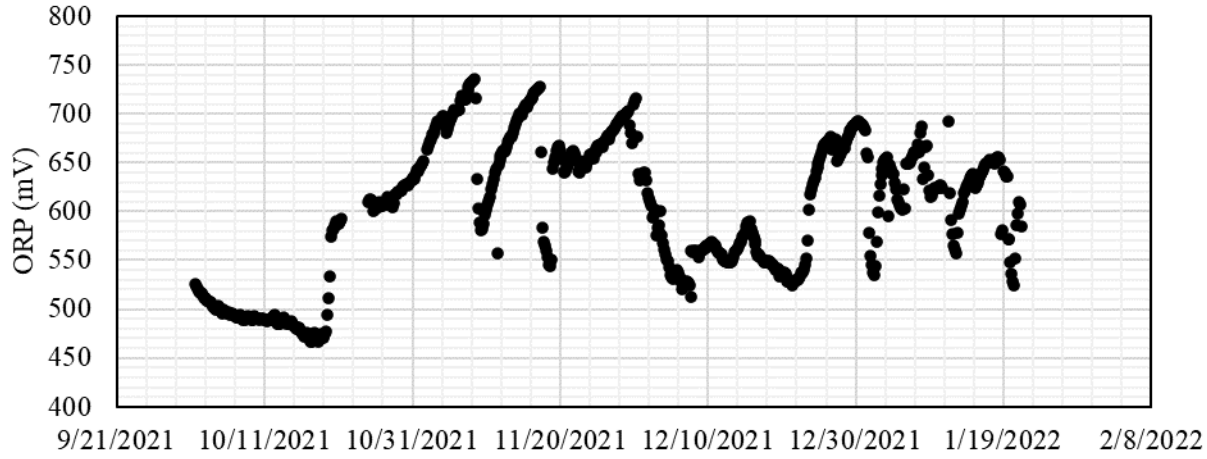



Figure 3.12.91. Redox potential of the 3000-gallon bioreactor over time.

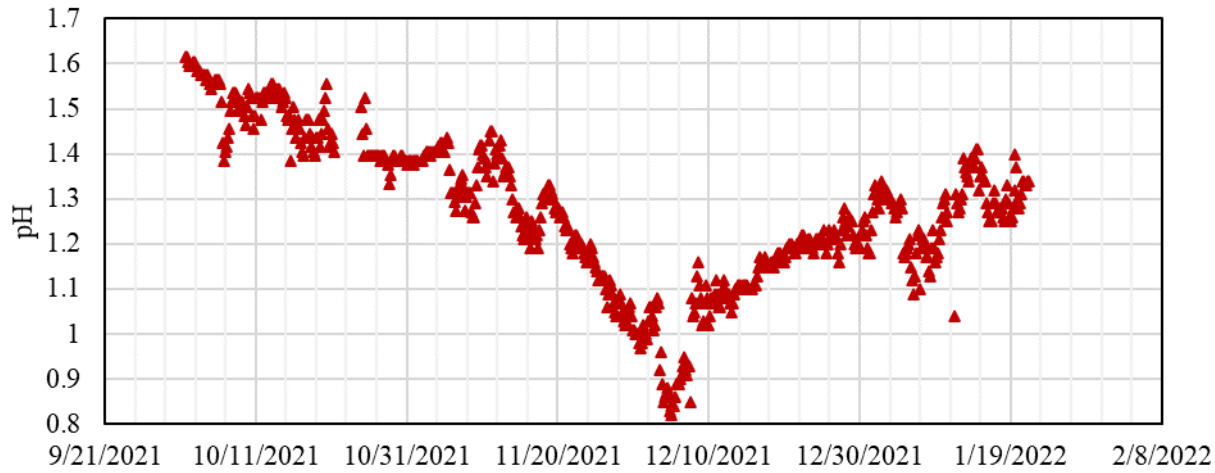



Figure 3.12.92. pH recording of the 3000-gallon bioreactor system over time.

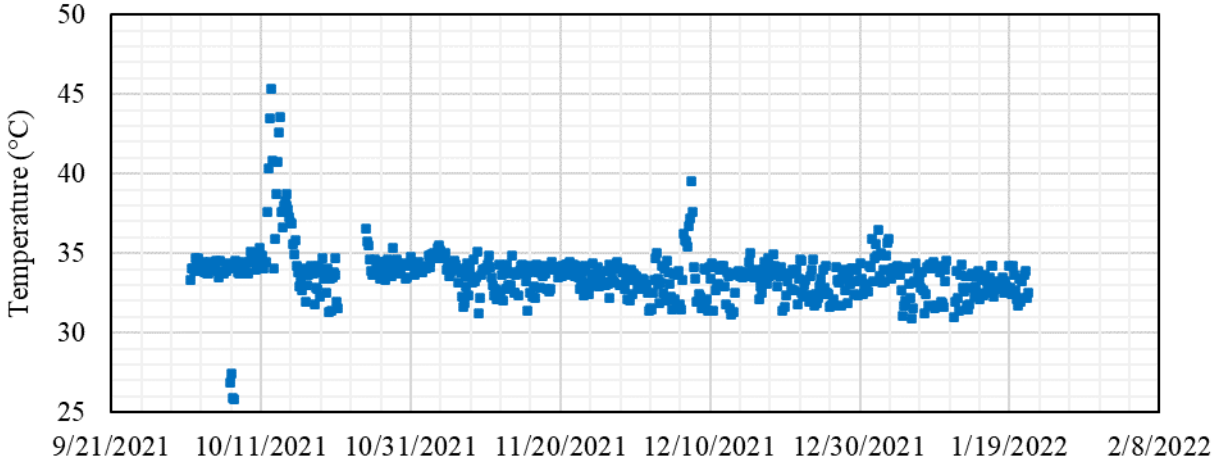



Figure 3.12.93. Temperature recording of the 3000-gallon bioreactor system over time.

In early January 2022, it was decided to test the effect of shorter residence times in Bioreactor No. 1. To reduce the residence time, Bioreactor No. 2 was temporarily bypassed by transferring the bio acid solution from Bioreactor No. 1 to the storage tank TK-26 directly using a peristaltic pump as shown in Figure 3.12.94. By doubling the volume of fluid in the process, the residence time could be reduced to half. By this action and slightly increasing the pyrite feed rate, residence time was changed from nearly 27 days down to 8 days. This will provide the acid generation rate as the demand for bio acid increases, the second reactor will be brought back online, and feed rates will increase.

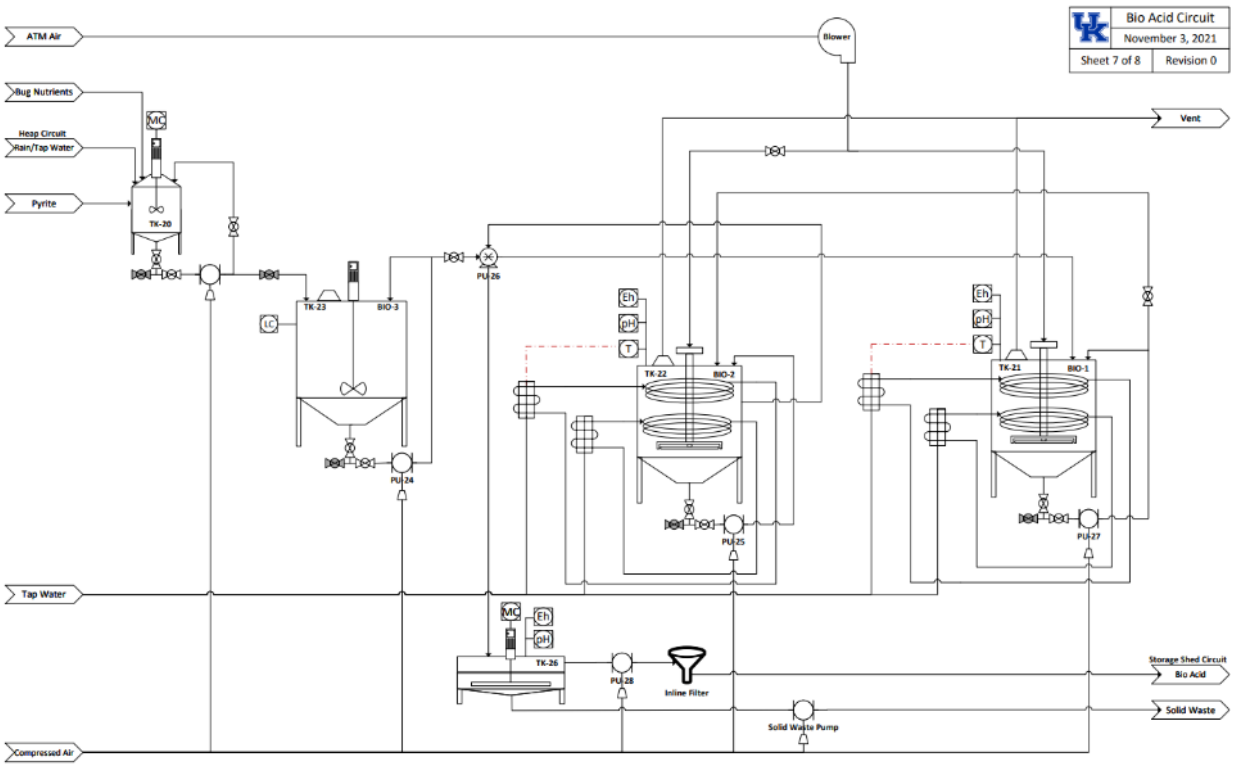



Figure 3.12.94 Modified bio-reactor circuit schematics to test the impact of shorter residence time.

### *3.12.9 Bio-Acid Reactor Operation (Short Residence Time)*

The Eh, pH, and temperature of Bioreactor No. 1 are shown in Figure 3.12.95. From December 2021 to February 2022, pH was a little high for the bioreactor, and the ferrous oxidation rate was very low. This led us to believe that the bacteria population inside of the reactor had mutated and stopped producing acid at a fast rate. Until February 1st (2022), there were no significant changes to the reactor feed rate. Afterward, the feed rate was adjusted to reduce the residence time in the reactor to 8 days, as those actions helped the University of Utah's smaller reactor when they were in a similar state. When the feed rate was changed, the Eh significantly dropped, and the pH started to quickly rise. Since the Eh was near 500, ferrous oxidation tests were not effective, as they do not cause a significant drop in Eh. Once we concluded that the population was mutated, the main reactor was shut down and efforts were made to restart the population in a 200-gallon reactor. To prepare for that, bio acid, pyrite, ammonium sulfate, and dipotassium phosphate were combined.

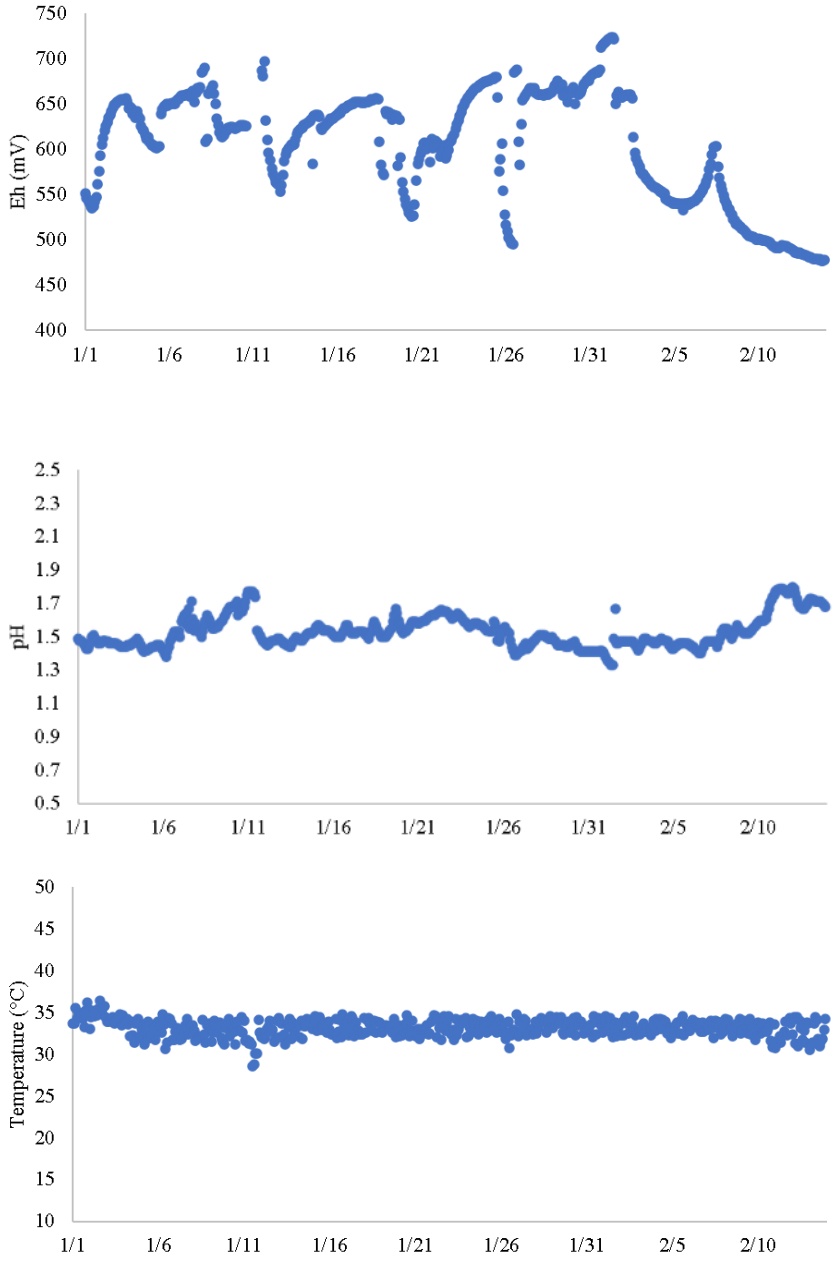


Figure 3.12.95. The Eh, pH, and Temperature of Bioreactor .

The Eh, pH, and temperature of the backup bioreactor is shown in Figure 3.12.96. The bioreactor took a while to restart and get going. On February 17th, sulfuric acid was added to the reactor to bring the pH to normal levels. On February 28th, the older sample of our bio acid thickener underflow was added into the bioreactor (Figure 3.12.97). These solids contained dormant A. Ferrooxidans, which can be reactivated when mixed into a warm aerated solution.

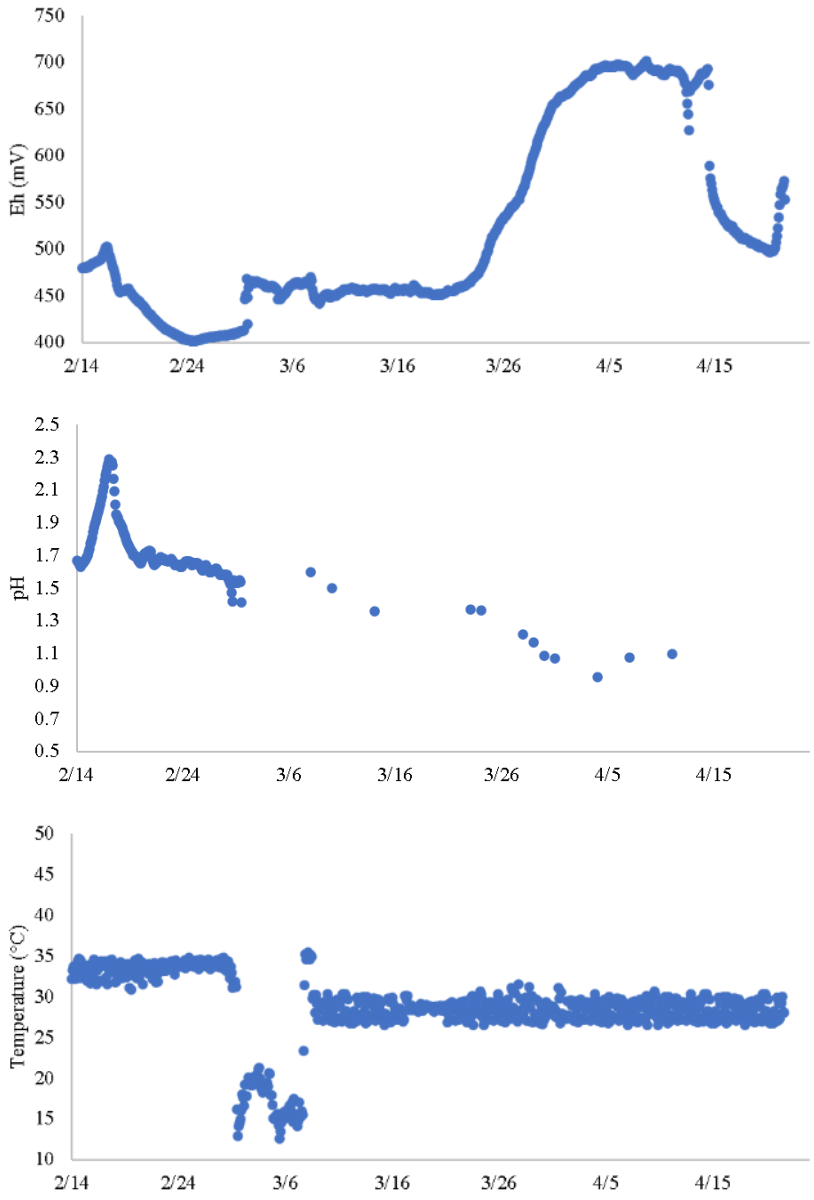



Figure 3.12.96. The Eh, pH, and Temperature of the backup bioreactor.

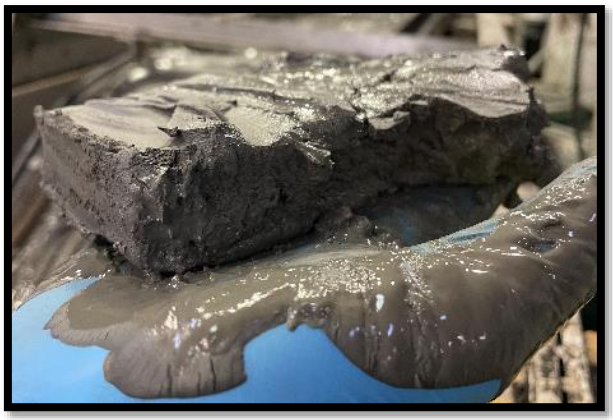


Figure 3.12.97. Pyrite and bacteria material from old bioreactor in the thickener underflow.

Around the time when bacteria were added into the reactor, a sharp drop occurred in the temperature of the solution. This was due to the spargers providing aeration to the system. As the compressed air of the spargers was released and expanded in the tank, it loses energy and cools down. Since the sparger creates a "cloud" of small air bubbles, there was a large surface area for heat to be pulled from the solution and into the rising air. This situation was the inverse of the larger bioreactor. The 40 hp motor on Bioreactor 1 produces bubbles internally through the Denver aeraotort, which also requires cooling coils inside of the reactor. This smaller bioreactor was retrofitted with an air bubble generator using air from an external compressed air unit and requires heating. Figure 3.12.98 shows the inside of the smaller bioreactor, including variable height electric heating coils. These heaters were installed in the reactor on March 4$^{th}$, which was when the smaller reactor's temperature began to rise.



Figure 3.12.98. The smaller bioreactor including variable height electric heating coils

Another issue with the spargers was the mass transfer rate going out the top of the reactor. These small bubbles were picking up droplets of solution and carrying them out of the tank. The reactor lost roughly 10 gallons a day, which was roughly 5% of its overall volume. The temperature of the reactor was reduced from 35 C to 28 C to combat the mass transfer, as seen in Figure 6 after March 9th.

Before March 24th, the 200-gallon reactor was not showing much activity. To resolve this issue, the recirculation of the reactor was increased to eliminate the solids buildup at the bottom of

the tank. Shortly after the action, a significant increase in the reactor's Eh was observed, which indicated that solids (bacteria) were most likely settled to the bottom of the tank. When Eh passed 600 mV, continuous feeding of the reactor was resumed. It started off at a 10-day residence time, which was equivalent to 15 gallons a day being fed into the tank. There was not a significant change in rise of Eh, so on April 4th, the feed rate was increased to a 5-day residence time. This may have been too fast for the reactor at this stage, so the Eh started to fall again. Figure 3.12.99 shows the ferrous ion consumption rate. This was calculated by adding a known mass of ferrous sulfate into the reactor, which caused the Eh to drop rapidly. After the fall, the rate that the Eh increases are proportional to the bacteria population in the reactor. A larger population of bacteria can consume the ferrous ions faster and will return the starting value faster. The first three points took place while the bioreactor system was functioning well. We started to see a decline in ferrous oxidation rate with the next 5 points. The last point was is nearly up to where the main bioreactor used to be.

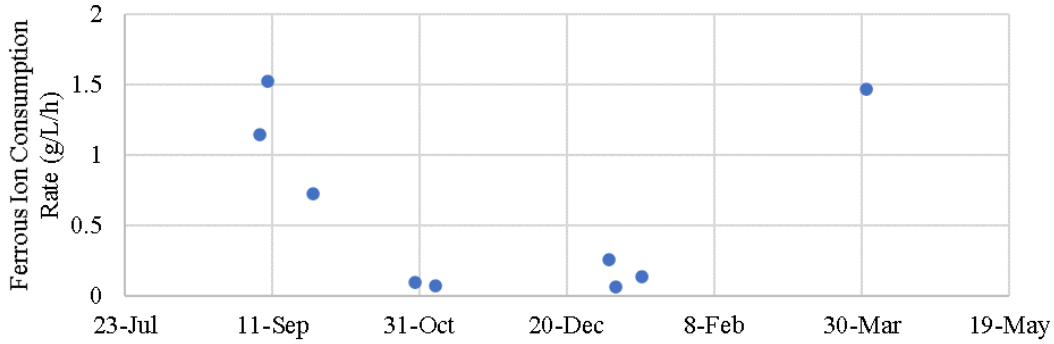



Figure 3.12.99. Ferrous ion consumption rate in the bioreactor.

### *1.1.24 Continued Bio Reactor Operations*

As discussed previously, key parameters indicated that the bacterial population in Bioreactor No. 1 had either mutated or an unknown bacterial strain became dominant over the preferred *A. ferrooxidans* species, which significantly lowered the rate of bio-acid acid production. Approximately 200 gal of a new bacterial population was cultivated in Baby Bio in February before being transferred into Bioreactor No. 2 in March. The pyrite feed slurry was added to bacterial population with a 5-day residence time from the beginning of April until late May 2022. Due to lack of promising trends from key indicators (i.e., pH, Eh, temperature), the somewhat inactive-active bacterial population in Bioreactor No.2 was killed off by increasing the temperature to above 49 °C (120 °F) for a sufficient time, leaving approximately 1700 gal of a pH 2.0 bio-acid/pyrite slurry solution.

A new bacterial population was cultivated in Baby Bio on May 23rd, 2022 until June 3rd, 2022. To start, a pyrite feed slurry comprised of 200-gal water, 1.2 kg ammonium sulfate, 0.6 kg dipotassium phosphate, and 60 kg of cleaned pyrite processed at the Pilot Plant was prepared in the 200-gallon bioreactor. The agitator in the 200-gallon bioreactor was operated with a tip speed within a 5-7 m $s^{-1}$ range; fast enough to keep the undissolved solids (i.e., pyrite) suspended, but not high enough to adversely impact the bacterial population. After the water-soluble species (i.e., ammonium sulfate, dipotassium phosphate) dissolved, this yielded an 8% w/v slurry, within the previously identified 6-10% w/v optimal range. This feed composition

was strictly followed on a volume adjusted basis for pyrite feed slurries prepared in the 200-gallon reactor and Bio Food during the period from May 23rd, 2022 to July 11th, 2022.

While the cleaned pyrite used in the feed slurries were processed identically, there were variations in the amount of water and/or calcite content across each barrel used. The variation in water content, as shown in Figure 3.12.100 can be attributed to inconsistent water exposure due to differences in seal qualities of the barrels and storage location (e.g., outside, in shed). Measures were put in place to use the driest pyrite available, but the impact and/or extent of this source of error was not quantified.



Figure 3.12.100 (left) Photo of a barrel of cleaned pyrite with high water content and (right) low water content

Higher calcite content in the cleaned pyrite negatively affected (i.e., increased) the pH of the feed slurry due to the alkalinity of calcite in solution; previous work identified this impact to be substantial enough to consume hydronium ions (i.e., $H_3O^+$) faster than the *A. ferrooxidans* bio-acid production can offset the impact in the system. Previous work identified that acceptable calcite content in the pyrite feed slurries resulted in a pH of 7.0-7.5 after stirring for 4 hr to ensure the contents were well-mixed. Any well-mixed pyrite feed slurries that exceeded this pH range were properly disposed of and a new barrel of cleaned pyrite was obtained to restart the process. Approximately 5 gal of the active bacteria culture, half being liquid and half being the settled solids, was obtained from the barrel shown in Figure 3.12.100 and added to the well-mixed pyrite feed slurry in the 200-gallon reactor, completing the preparation of the new bacterial culture.

Oxygen, a crucial component in facilitating aerobic metabolism in the *A. ferrooxidans* bacterial population, was introduced to the system at 23 ± 1 scf $min^{-1}$ through air spargers (SlamJet) installed in the 200-gallon reactor. While it was intended to measure dissolved oxygen (DO) to assess the effectiveness of the oxygen added to the system, probe malfunctions precluded any accurate measurements of this parameter for any plant-scale operations during the May 1st, 2022 to July 31st, 2022 period. Previous work identified that the system Eh can be used as a rudimentary assessment of air flow effectiveness in the absence of a DO probe, with readings of 420-500 mV (620-700 RmV) being ideal. The pH and Eh of the new bacterial population in the 200-gallon reactor during the cultivation period are shown in Figure 3.12.101. Previous work identified an optimal pH range of 1.4-2.0, but literature suggests an optimal range of 2.0-3.0 for *A. ferrooxidans* activity (Gao, Jiang, Mao, & Jiang, 2021). The new bacteria population was kept in an environment below the optimal *A. ferrooxidans* pH range, raising the concern that *A. ferrooxidans* may not have been the dominant species in this active bacterial population.

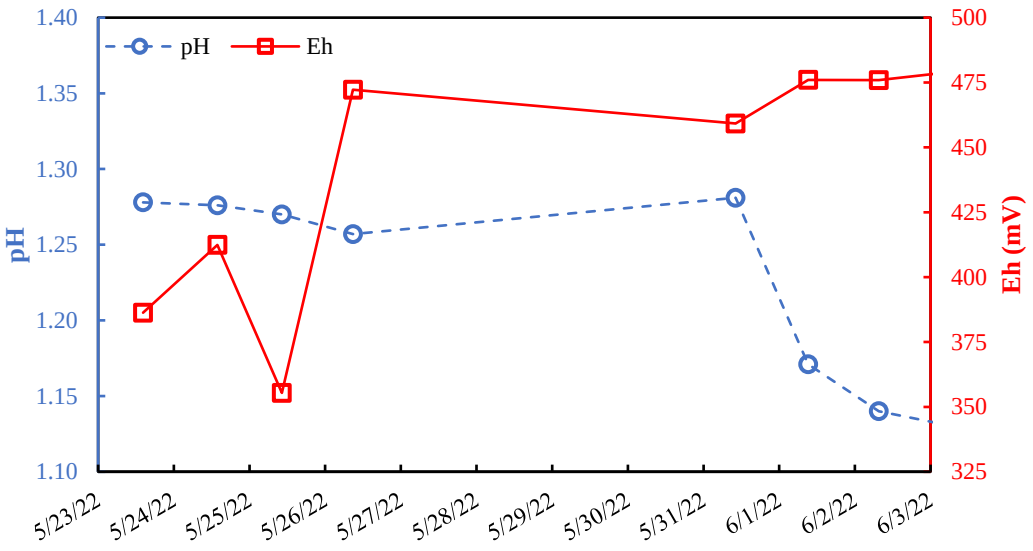



Figure 3.12.101 pH, Eh (vs. Ag/AgCl) of Baby Bio for the period from May 23rd, 2022 to June 3rd, 2022

Previous work identified an optimal temperature of 35 °C (95 °F) for bio-acid generation from *A. ferrooxidans*; below 32 °C (90 °F) slowed the metabolism of the bacterial population (i.e., dormancy), above 38 °C (100 °F) was sufficient to kill small portions of the population through protein denaturing, and above 49 °C (120 °F) was sufficient to kill large portions of, or entire populations. The temperature of the new bacterial population in the 200-gallon reactor during the cultivation period is shown in Figure 3.12.102. While the 200-gallon reactor was equipped with a heating element prior to the cultivation period, the element was removed prior to the cultivation of the new bacterial population. As a result, the bacterial population was never exposed to the optimal conditions for the *A. ferrooxidans* bacteria to activate from their dormant state.

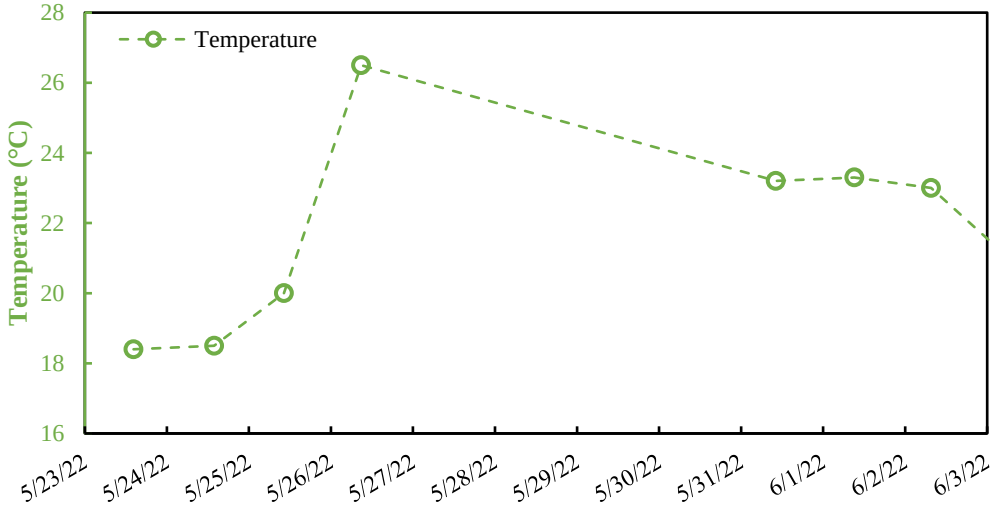



Figure 3.12.102 Temperature of Baby Bio for the period from May 23rd, 2022 to June 3rd, 2022

While the key parameters did not indicate the presence of an active bacterial population in the 200-gallon reactor, the contents were transferred into Bio-2 on June 4th, 2022 due to the project

timeline constraints. Once the bacterial population was transferred into Bioreactor No.2, the previously described optimal conditions were sought to be maintained, albeit at a scaled-up ratio given the 1:10 volume ratio between 200-gallon ractor and Bioreactor No. 2. Bioreactor No. 2 was fed pyrite feed slurries until June 24th, 2022, at which point the shutdown process for the bioreactor began. Feeding time was not dictated by a target residence time as had been previously done, but instead used the rate of change of $H^+$ (i.e., acid) production to dictate when feeding would occur as shown in Figure 3.12.103. The content of the pyrite feed slurry additions into Bio-2 was strictly maintained on a volume adjusted basis as shown in Figure 3.12.104.

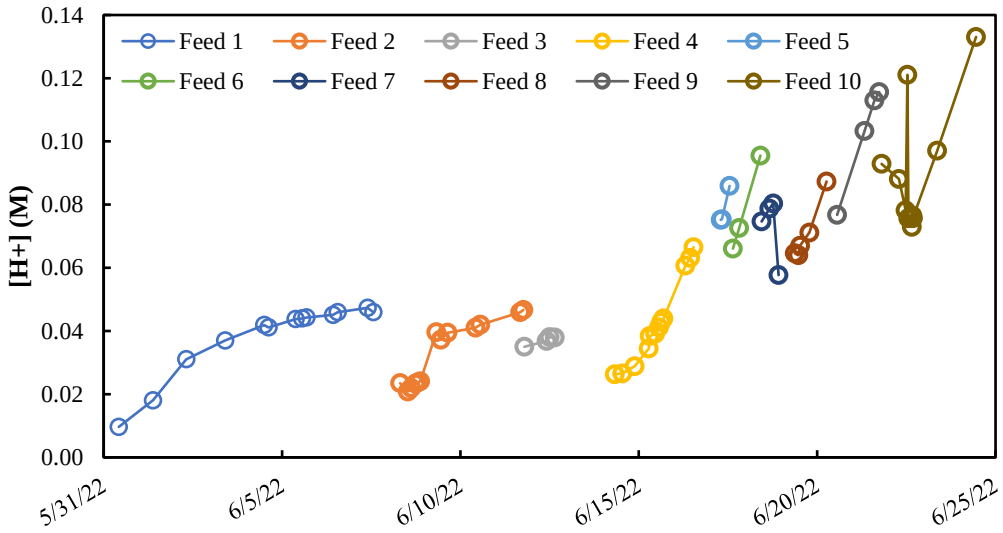



Figure 3.12.103 Acid production rate in Bio-2 for the period from May 31st, 2022 to June 25th, 2022

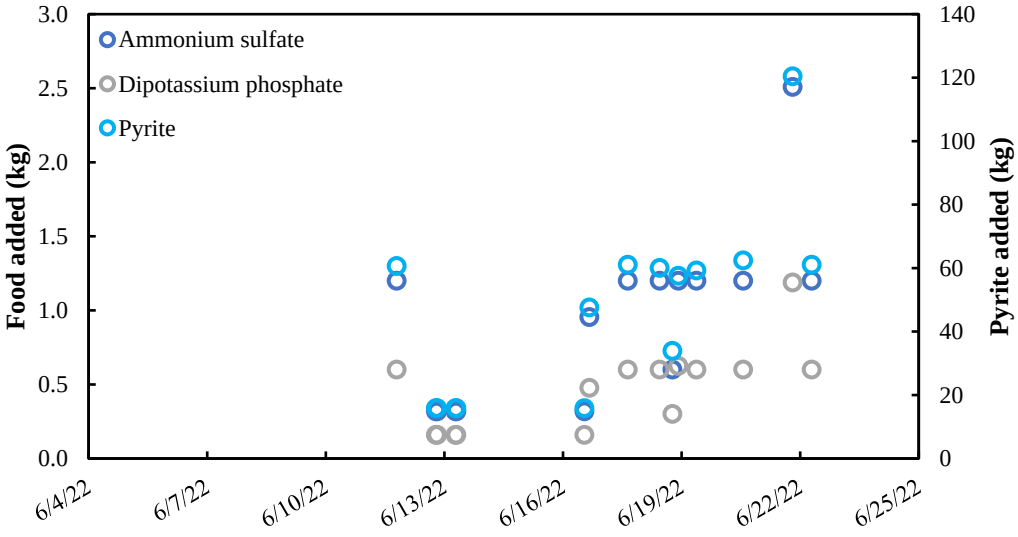



Figure 3.12.104 Content of pyrite feed slurry addition to Bio-2 for the period from June 4th, 2022 to June 25th, 2022

A nutrient mixture of 306.0 g ferrous sulfate, 30.6 g ammonium sulfate, and 5.1 g dipotassium phosphate per 2000 gal of bio-acid/pyrite slurry solution was regularly added to Bioreactor No. 2 to boost the active bacterial population activity; sulfuric acid was added to keep the pH below 1.8 as necessary as shown in Figure 3.12.105. Bio-2 was not equipped with a level sensor, making accurate determination of the bio-acid/pyrite slurry solution volume difficult. Thus, the nutrient mixture addition was not added on a strict volume adjusted basis, but rather on daily basis during plant operations. In addition to nutrient additions, large (i.e., 2-5 kg) amounts of ferrous sulfate were added into the system to determine the ferrous ion consumption rate, a quantitative measure of active bacteria population size, as shown in Figure 3.12.106.

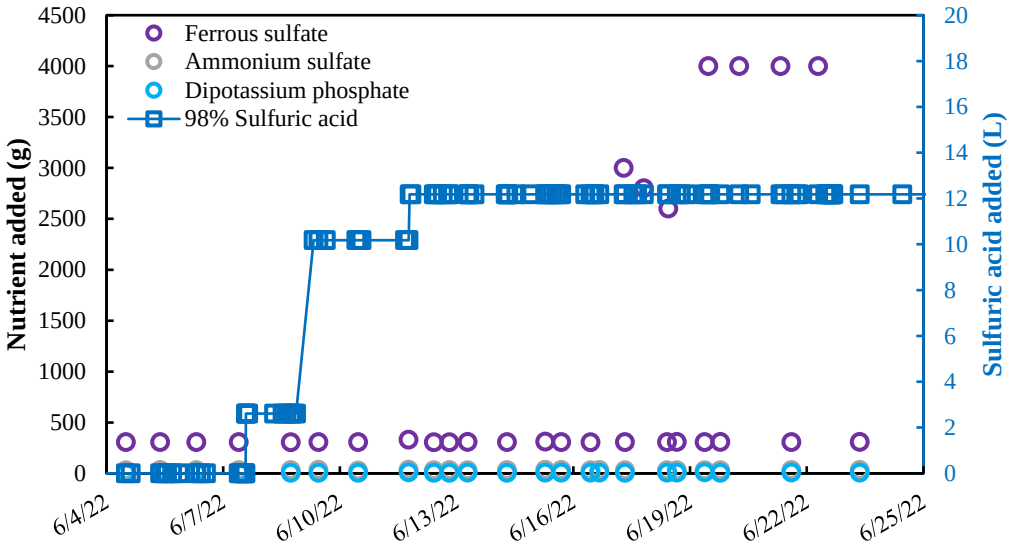



Figure 3.12.105 Nutrient, 98 wt% sulfuric acid addition to Bio-2 for the period from June 4th, 2022 to June 25th, 2022

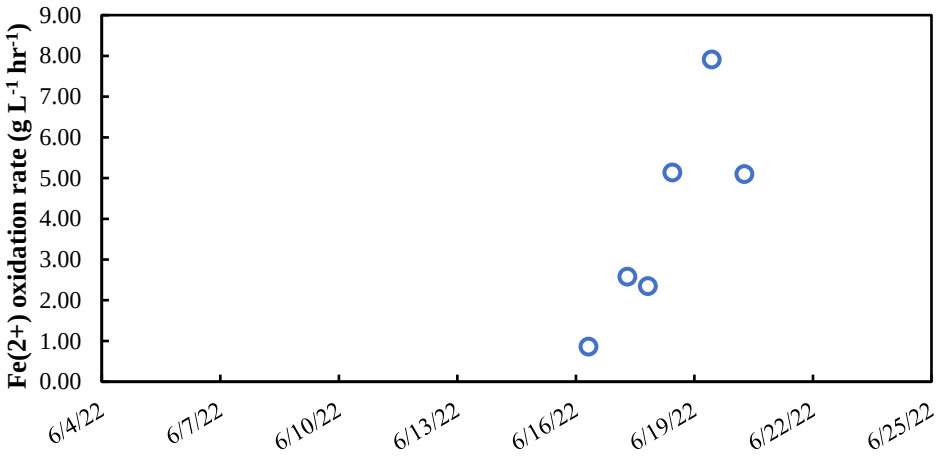



Figure 3.12.106 Ferrous oxidation rate of Bio-2 for the period from June 4th, 2022 to June 25th, 2022

The Eh of the bio-acid/pyrite slurry solution was continuously monitored using an Eh probe. A measured amount of ferrous sulfate was added into Bio-2 and the ferrous oxidation rate was calculated from the rate at which the Eh measurements recovered to 99.9% of the initial value. Previously (i.e., Q3 2021 to Q1 2022), the recovery criterion was arbitrarily varied across both operators and days from 97.0-99.9%. A sensitivity analysis conducted on the calculation

methodology employed determined that changing the recovery criterion from 97.0% to 99.9% results in ± 27.4% in the calculated value. Thus, a 99.9% recovery criterion was established as the only value to be used hence forth for consistency. To calculate the ferrous oxidation rate, the volume of the bio-acid/pyrite slurry solution is needed. As previously stated, accurate determination of the volume in Bio-2 is not possible due to the lack of level sensor. Best estimates can only be made within ± 500 gal using external markings on Bio-2; a sensitivity analysis conducted on the ferrous oxidation rate calculation methodology employed determined this tolerance results in an uncertainty of ± 62.1% in the calculated value. While volume estimates were made to the nearest 50 gal, it is worth noting that the accuracy of said estimates represents an unquantifiable source of uncertainty in the measurements.

The 0.86 g $L^{-1}$ $hr^{-1}$ rate measured on June 16th, 2022 was suspected to be artificially lowered due to insufficient mass of ferrous sulfate used in the test. Neglecting this measurement, an average ferrous oxidation rate of 4.61 ± 2.27 g $L^{-1}$ $hr^{-1}$ was measured during the peak operating period, including the highest rate ever observed in the pilot-plant bio-acid generation operations (i.e., 7.91 g $L^{-1}$ $hr^{-1}$). These ferrous oxidation rates represented a significant improvement from the maximum rate of 1.52 g $L^{-1}$ $hr^{-1}$ reported in the Q1 2022 report. The pH and Eh, temperature, and air flow in Bio-2 during the period from May 23rd, 2022 to July 11th, 2022 are shown in Figure 3.12.107, Figure 3.12.108, and Figure 3.12.109, respectively. Further analysis is currently being conducted to investigate the relationship(s) between these parameters on bio-acid generation.

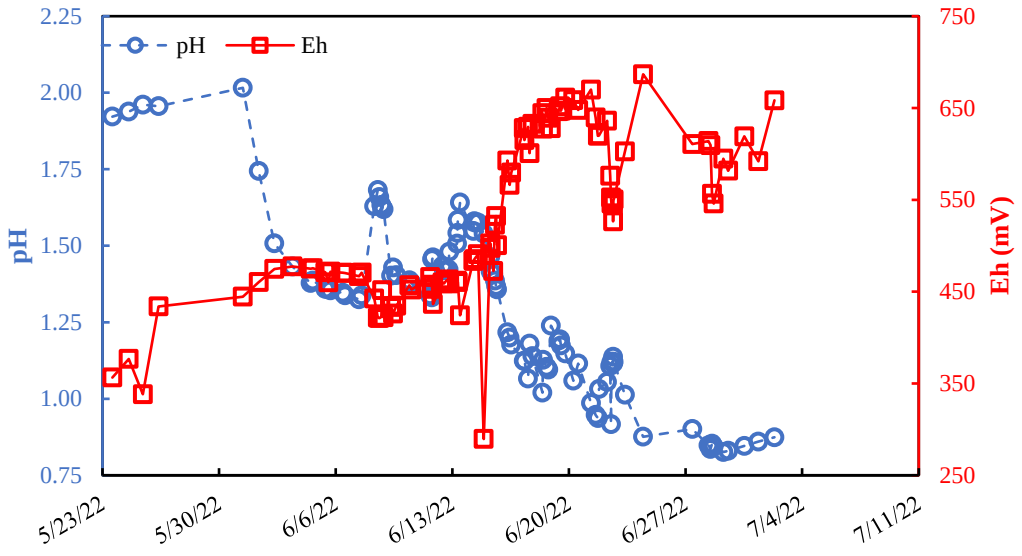



Figure 3.12.107 pH, Eh (vs. Ag/AgCl) of Bio-2 for the period from May 23rd, 2022 to July 11th, 2022.

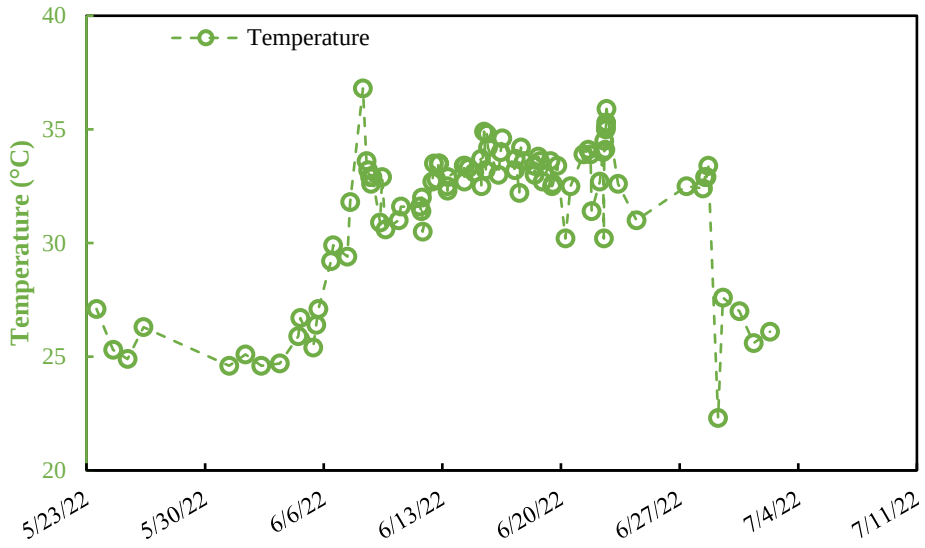



Figure 3.12.108 Temperature of Bio-2 for the period from May 23rd, 2022 to July 11th, 2022

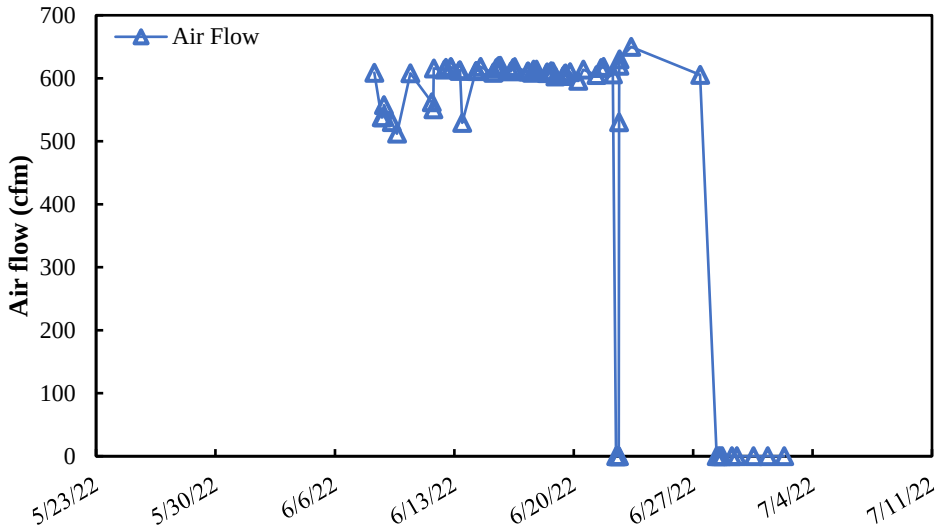



Figure 3.12.109 Air flow into Bio-2 for the period from May 23rd, 2022 to July 11th, 2022

Other closely related species that gain energy from excess iron and sulphur, such as *Acidithiobacillus thiooxidans* (*A. thiooxidans*) or *Leptospirillum ferrooxidans* (*L ferrooxidans*), respectively, may have been present or dominant in the active bacterial population. Two 50 mL samples of bio-acid/pyrite slurry solids were obtained from the Bio-2 underflow and sent to the University of Utah for further investigation of bacterial composition.

Bio-acid extraction began when the contents of Bioreactor No. 2 reached a sufficiently low pH (i.e., < 1.3) on June 13$^{th}$, 2022. Extraction was completed by using a peristaltic pump to bio-acid/pyrite slurry from the top of Bioreactor No. 2 through a ¼-in line into TK-28 until June 28$^{th}$, 2022. From June 28$^{th}$, 2022 to July 7$^{th}$, 2022, On June 28$^{th}$, 2022, a pump was installed to recirculate the bio-acid/pyrite slurry from the bottom to the top of Bio-2 using a 1-in line equipped with a tee that sent a separate 1-in line into TK-28. Extraction was then conducted by actuating a valve to send the recirculating bio-acid/pyrite slurry as needed until July 7$^{th}$, 2022. The pyrite solids settled in the bottom, leaving a clear, deep-red bio-acid at the top of TK-28

as shown in Figure 3.12.110; however, extracting the bio-acid/pyrite slurry too quickly resulted in a murky brown solution that proved difficult to clean until the solids were allowed enough time to settle as shown in Figure 3.12.110.



Figure 3.12.110 (left) Photo of TK-28 operating at an appropriate extraction rate and (middle) too quickly, and (right) settled pyrite solids removed from TK-28 via belt filter

The pH and Eh and temperature of TK-28 during the period from May 23rd, 2022 to July 11th, 2022 are shown in Figure 3.12.111and Figure 3.12.112, respectively. Further analysis is currently being conducted to investigate the relationship(s) between these parameters on bio-acid generation.

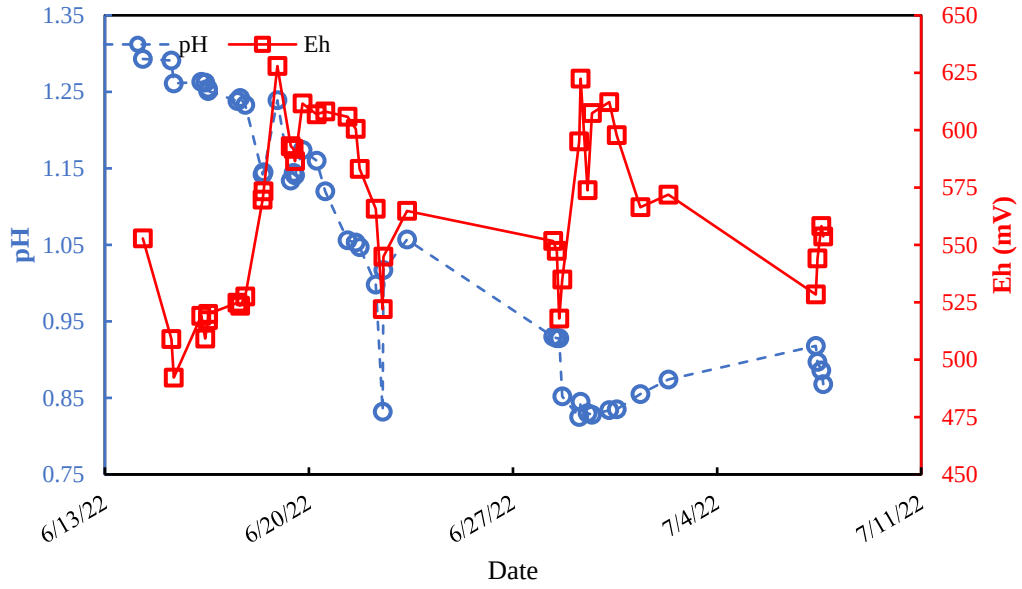



Figure 3.12.111 pH, Eh (vs. Ag/AgCl) of TK-28 for the period from June 13th, 2022 to July 11th, 2022

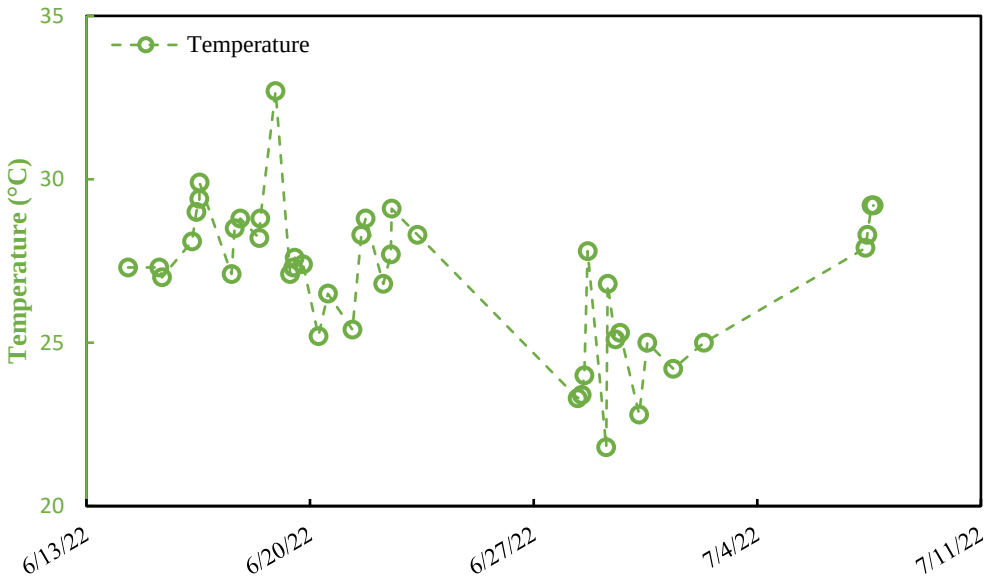



Figure 3.12.112 Temperature of TK-28 for the period from June 13th, 2022 to July 11th, 2022

The bio-acid overflowed into a 55-gal surge tank equipped with a level sensor to automatically pump this supernate through an in-line filter for additional filtration prior to being sent into Bioreactor No. 3 for storage. Approximately 1000 gal of bio-acid was processed from June 13th, 2022 through June 21st, 2022 before being used in the WK-3 pilot-plant test. Another 1600 gal of bio-acid was processed from June 28th, 2022 through July 7th, 2022 before being used in the WK-4 pilot-plant test. The pH and Eh and temperature of Bioreactor No. 3 during the period from June 13th, 2022 to July 11th, 2022 are shown in Figure 3.12.113 and Figure 3.12.114, respectively.

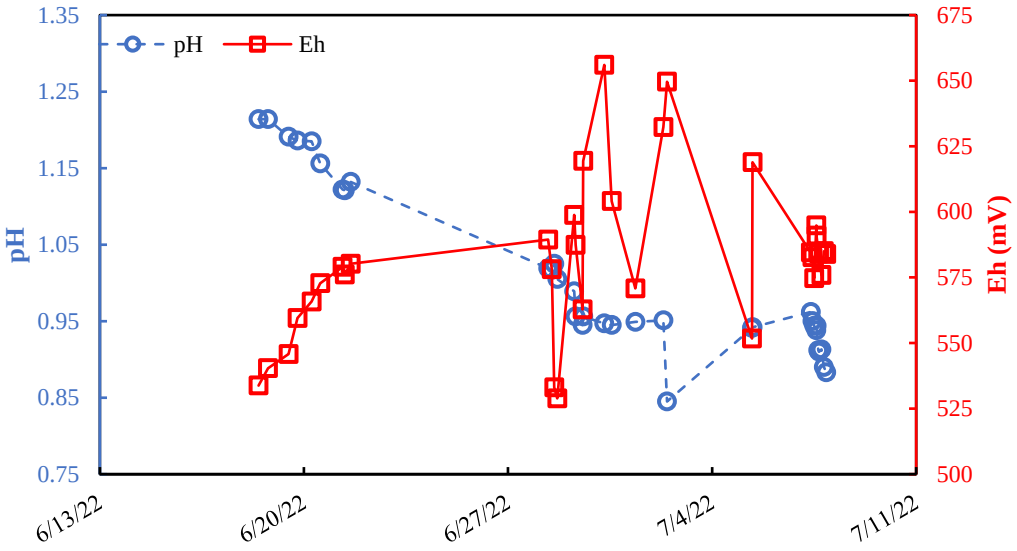



Figure 3.12.113 pH, Eh (vs. Ag/AgCl) of Bio-3 for the period from June 13th, 2022 to July 11th, 2022

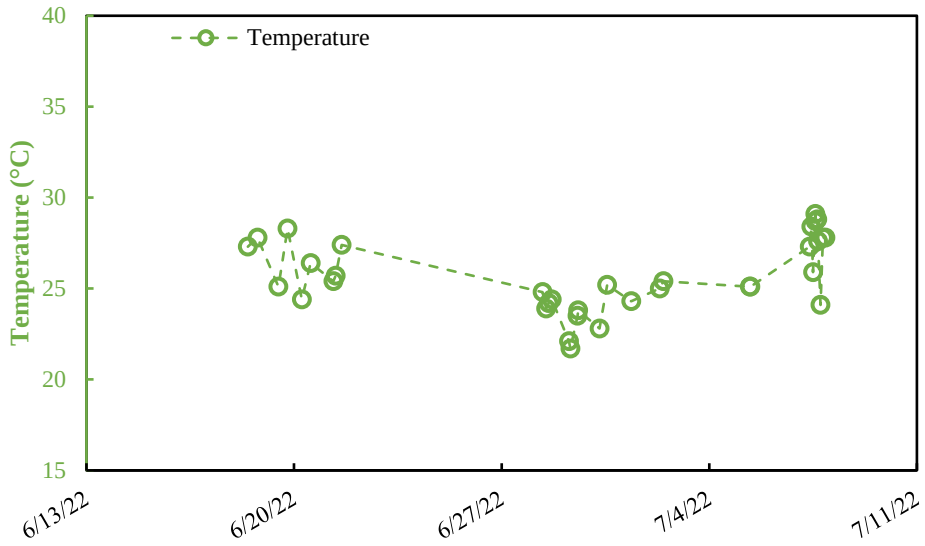



Figure 3.12.114 Temperature of Bioreactor No. 3 for the period from June 13th, 2022 to July 11th, 2022

After the target pH was reached, titration tests were conducted at multiple point during the bio-acid generation process to determine the acid concentration of the bio-acid solution. The test procedure is summarized below.

1. Obtained approximately 600 mL of the bio-acid/pyrite slurry or bio-acid from Bio-2 or Bio-3, respectively.
2. Filtered the liquid sample through a large medium pore, medium flow filter paper via vacuum filtration to remove solid particulate matter.
3. Used a graduated cylinder to measure 500 ± 1 mL of the filtered bio-acid solution and added to a 600 mL glass beaker equipped with a 1.5 in ribbed stir bar.
4. Placed the glass beaker onto a Cole-Palmer stir plate inside the fume hood and set the stir speed to the 3.25 marking.
5. Calibrated the pH probe using a five-point curve from 1.68 to 12.480 and the Eh (vs. Ag/AgCl) probe using a single-point standard before inserting into the stirred bio-acid solution. Recorded the initial pH, Eh, and temperature measurements.
6. Used a 100-1000 µL pipette to add 12.00 ± 0.001 M NaOH solution to the stirred bio-acid sample. Waited 2 min, recorded the aforementioned parameters, took a photo of the solution, and repeated until solids were visibly present in the stirred solution.

The acid equivalent value was determined using three methods for establishing the endpoint: the point at which the solution becomes cloudy, the point at which the solution pH reaches 2.2, and the point at which the solution begins to have visible precipitates. Examples of the visual appearance of the solutions at these endpoints are shown in Figure 3.12.115. The acid concentrations determined by each of these three methods are shown in Table 3.12.35.



Figure 3.12.115 Photos from left to right of the starting bio-acid and cloudy, Free method, and solid endpoints, respectively

Table 3.12.35 Bio-acid acid concentrations determined via 12 M NaOH titration, where molar equivalence denotes the free method.

| | | | **Endpoint Methodology** | | |
|---|---|---|---|---|---|
| **Test Date** | Source | Initial pH | Cloudy | Solids | Molar Equivalence |
| **yyyy/mm/dd** | | | M | M | M |
| **2022/06/01** | Bio-2 | 1.798 | 0.055 | 0.062 | 0.034 |
| **2022/06/10** | Bio-2 | 1.341 | 0.131 | 0.146 | 0.115 |
| **2022/06/15** | Bio-2 | 1.322 | 0.123 | 0.186 | 0.132 |
| **2022/06/16** | Bio-3 | 1.217 | 0.157 | 0.209 | 0.204 |
| **2022/06/21** | Bio-3 | 1.094 | 0.321 | 0.386 | 0.328 |
| **2022/06/29** | Bio-3 | 0.911 | 0.049 | 0.727 | 1.097 |
| **2022/07/25** | Bio-3 | 0.951 | 0.380 | 0.660 | 0.503 |

*3.12.10 Bioleaching Cost Analysis*

The TEA model developed under Task 3 was used to conduct single-factor sensitive analysis on the bioreactor operation. Results from the analysis show the equivalent acid cost as a function of key design variable, including retention time (Figure 3.12.116) and thermal management (expressed as equivalent power, Figure 3.12.117). Assumptions and fixed parameters used in these analyses include: 500 gpm solution production, 3 reactors in series, 0.7 mol/L output acid molarity, 3.5 kJ/kg-K solution heat capacity, 55% chiller efficiency, 0.1 kg/L pyrite consumption, 1.5 g/L ammonium sulfate consumption, 0.6 g/L $K_2HPO_4$ consumption, 0.1 lpm air/lpm solution, 20-year operating period, 10% discount rate. Equivalent acid cost is a combination of the operating cost and the capital cost amortized over the operating period. The data indicates that both factors are significant and can lead to nearly a 3x change in acid cost over the range of variables assessed. It should be noted, that in the higher range of values, the equivalent acid cost was less than that of bulk sulfuric acid, which is typically between $0.20 and $0.25.

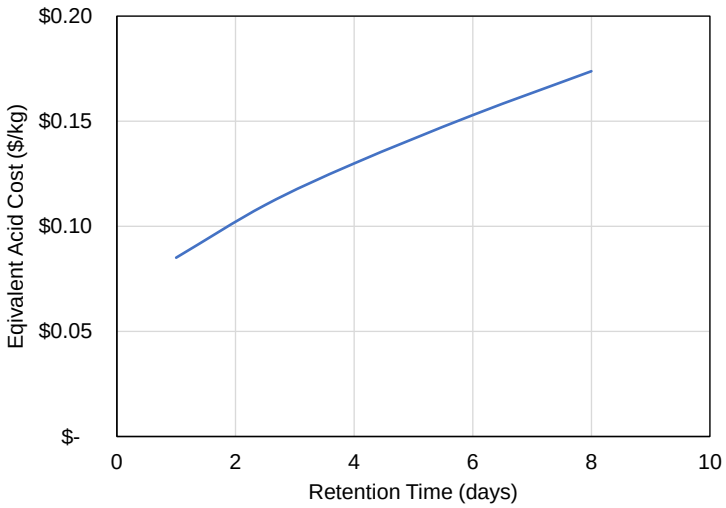



Figure 3.12.116. Bioreactor sensitive analysis showing equivalent acid cost (CAPEX+OPEX) as a function of total retention time.

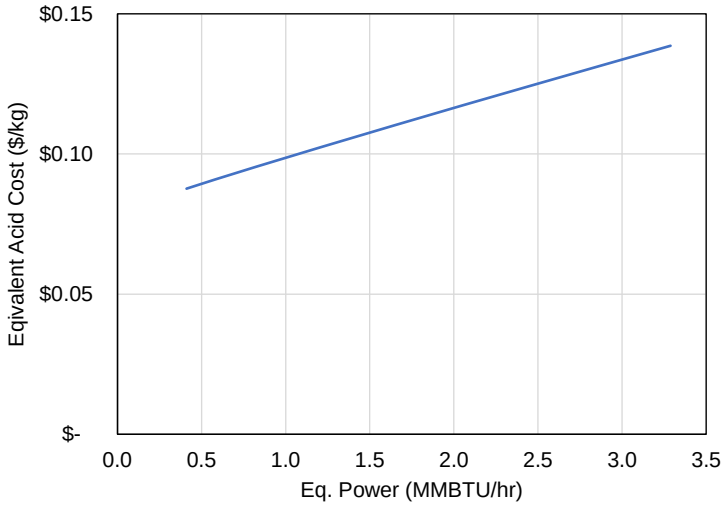



Figure 3.12.117. Bioreactor sensitive analysis showing equivalent acid cost (CAPEX+OPEX) as a function of equivalent cooling energy needed for thermal management.

*3.12.11 Scandium Recovery & Purification*

The effectiveness of solvent extraction (SX) for the selective removal of scandium was assessed using acidic mine drainage (AMD) solution obtained as runoff from the Dotiki West Kentucky-13 coal seam refuse pile (Alliance Coal Co.). $Fe^{3+}$ precipitation (Stage PT0) was carried out on 5000 gallons of this AMD solution at a pH of 3.2, removing an average of 81% of total iron content. A 50-gallon sample of the PT0 filtrate was set aside for use in the SX loading, stripping, and cleaning experiments.

**Bench-Scale Experimentation:**

**Loading Trials:** To remove scandium from the aqueous solution, an organic mixture of extractant and diluent was mixed with the aqueous solution, "loading" the organic layer prior to physical separation of the two immiscible phases in a settling apparatus. To determine optimal loading conditions, five lab-scale loading trials of various aqueous to organic (A:O) ratios were performed with a 300 mL sample of the 50 gallons of filtrate saved from PT0, and 500 mL of a 1:99 v:v mixture of Cyanex 272 and Orfom (serving as the extractant and diluent, respectively) made for use as the organic to be loaded. A:O ratios used were 1:10, 1:1, 5:1, 10:1, and 50:1. To ensure the pH of the aqueous phase remained relatively stable and did not decrease due to acid production during the loading process, the organic volume used for each of the five trials underwent saponification by adding an appropriate volume of 1M NaOH to the mixture (determined through calculations based on the total iron content in the volume of filtrate to be used) and was allowed to stir for 15 minutes prior to use. After the 15 minutes had elapsed, the corresponding volume of PT0 filtrate for each of the trials was added to the beaker to obtain the selected A:O ratio and then allowed to stir for an additional 15 minutes prior to introduction into the settling apparatus. Table 3.12.36 lists the approximate volumes used for each of these loading trials below.

Table 3.12.36. Approximate volumes and component ratios of aqueous loading feed, organic extractant/diluent mixture, and base addition used in solvent extraction loading tests.

| Test ID | A:O | Aqueous (mL) | Organic (mL) | 1M NaOH (mL) |
|---|---|---|---|---|
| DTK-AMD20-E40 | 1:10 | 10 | 100 | 0.08 |
| DTK-AMD20-E41 | 1:1 | 50 | 50 | 0.04 |
| DTK-AMD20-E42 | 5:1 | 50 | 10 | 0.02 |
| DTK-AMD20-E43 | 10:1 | 50 | 5 | 0.01 |
| DTK-AMD20-E44 | 50:1 | 50 | 1 | 0 |

Samples of the aqueous feed and of the raffinate obtained from the settling apparatus post-loading were analyzed using inductively coupled plasma – optical emission spectroscopic methods (ICP-OES). Elemental concentrations in the organic layer were determined by completing a mass balance around the concentrations, provided through ICP-OES, of the feed and raffinate samples. The Cyanex 272 extractant utilized in this experiment proved to be selective towards both scandium and iron, while loading negligible amounts of the other elements analyzed. With iron being the primary contaminant, and also a hindrance to the oxalic circuit efficiency in the final rare earth precipitation process, it was desirable to limit loading of iron, while still maintaining a high efficiency of scandium loading. Figure 3.12.118, below, plots the loading efficiency of both scandium and iron for each of the five aqueous to organic ratios trialed.

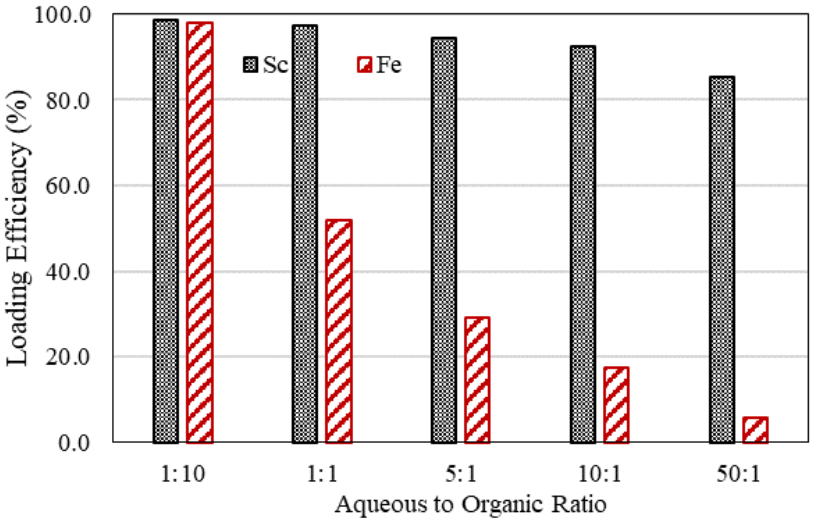



Figure 3.12.118. Loading test data using various aqueous:organic ratios. Scandium loading and iron loading are displayed.

While lower aqueous to organic ratios led to near 100% loading of scandium, the iron loading was also near 100%. At a 25:1 ratio, it was found to limit iron loading to 17.3%, while still loading 92.6% of the scandium. However, the additional decrease in iron loading at a 50:1 ratio, being 5.6% and a third of that loaded in the 25:1 trial, was significant enough to offset the 7.3% reduction in scandium loading. Therefore, the 50:1 aqueous to organic ratio was selected for further experimentation.

**Stripping Trials:** After loading the scandium into the organic phase, it is necessary to effectively "strip" the scandium back out into aqueous media for further processing. To accomplish this, lab-scale tests were performed using 8M, 6M, 4M, 2M, 1M, and 0.5M sulfuric acid, as well as a control test using deionized water, as the stripping agents. A 5:1 organic to aqueous ratio was initially used for the stripping trials. The organic used in the stripping tests was prepared via the methods described previously for the A:O ratio of 50:1, with the exception that the process was scaled to use 3.5 L of aqueous AMD filtrate and 70 mL of the Cyanex 272/Orfom mixture. After separation of the loaded organic and spent raffinate from the loading stage, 10 mL of loaded organic was utilized in each of the stripping trials with 2 mL of one of the seven sulfuric acid concentrations. Stirring continued after introduction of the stripping agent for 15 minutes before the mixture was poured into the settling apparatus for phase separation.

Assessment of the loading stage was performed using ICP-OES in the same fashion as described previously. From there, stripping was assessed by analyzing 10X dilutions of the used stripping acid, which were prepared with 1 mL of the stripping acid obtained from the settling apparatus and 9 mL of 5 vol% nitric acid. This was done because the high sulfuric acid concentrations utilized in the experimental plan were not compatible with the ICP-OES method that was employed at the time. Recovery calculations from the 5:1 organic to aqueous stripping trials are shown below in Figure 3.12.119.

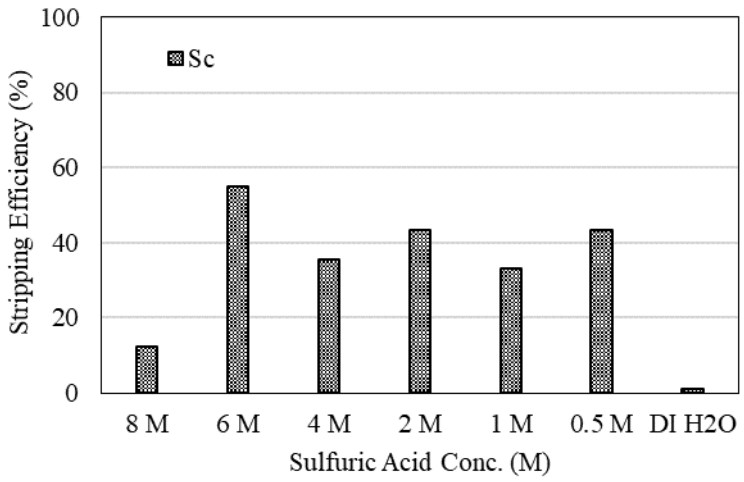



Figure 3.12.119. Scandium recovery as a function of sulfuric acid concentration in the stripping solution at an organic to aqueous ratio of 5:1.

As is shown, at best, scandium recovery from the loaded organic never exceeded 60%, and little difference between acid molarities is seen from 0.5 M to 6 M. As a result, the poor recovery of scandium in these trials led to further testing using a 1:1 organic to aqueous ratio. Again, the 50:1 aqueous to organic loading ratio was used, scaled to 3.5 L of aqueous material and 70 mL of organic material. The concentrations of sulfuric acid were also kept constant, with the only exception to the method described previously being that 10 mL of loaded organic were mixed with 10 mL of one of the seven sulfuric acid concentrations.

For the loading stage, it was found that 87% of the scandium was loaded into the organic phase, corroborating the results obtained during initial loading condition optimization tests. Too, only 4.0% of total iron content in the PT0 filtrate was loaded. The results of the stripping trials are seen below in Figure 3.12.120.

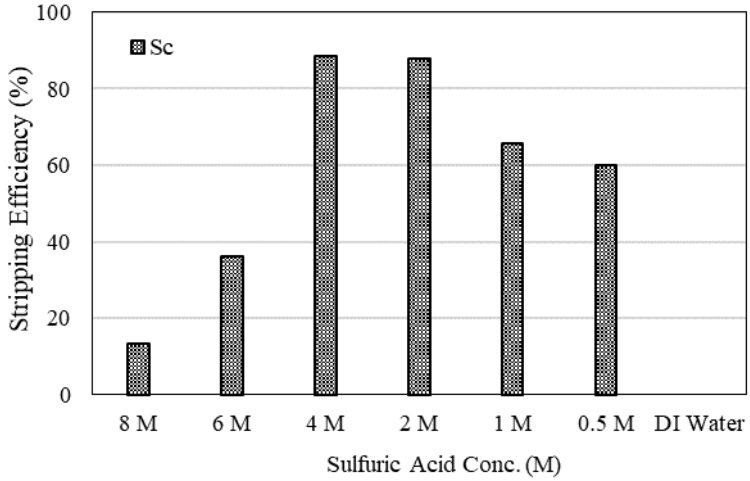



Figure 3.12.120. Scandium recovery as a function of sulfuric acid concentration in the stripping solution at an organic to aqueous ratio of 1:1.

The stripping efficiency is tied to the sulfuric acid molarity in a parabolic fashion, peaking between 2M and 4M, providing 89.1% and 89.8% recovery of scandium from the loaded organic, respectively. From this, a 2M sulfuric acid concentration was chosen as the stripping agent, since choosing 4M for a 0.7% increase in recovery would be cost-prohibitive. Unfortunately, iron numbers obtained from the stripping trials were inconclusive, as

contamination during testing led to higher iron concentrations in the used stripping acid than would be predicted through a mass balance on the system. Further testing is needed to establish what the expected iron concentrations in the stripping acid should be, theoretically.

### *3.12.12 Small Pilot-Scale Solvent Extraction Process*

From the loading and stripping test trials performed on the lab-scale, it was determined that two stages of loading, conducted using a 50:1 aqueous to organic ratio, and two stages of stripping, using a 1:1 ratio of organic to 2M sulfuric acid, would be sufficient to achieve near-complete scandium recovery. The small solvent extraction unit was plumbed for the testing of a continuous processing system using the remainder of the PT0 filtrate. The stages were connected in sequence, with the organic phase flowing left to right across all four. The aqueous phase was run countercurrent to the organic, and an approximate 1:1 internal recycling ratio of organic was added to each stage to aid in phase separation within the settling apparatuses. To ensure overall flowrates were in line with the conditions selected, three pumps were used to pump the fresh feed, the organic extractant mixture, and the 2M sulfuric acid at flowrates of 375 mL/min, 7.5 mL/min, and 7.5 mL/min, respectively. 7.5 mL/min was selected as the low-end flowrate as a result of being the minimum flowrate that could be achieved with the peristaltic pumps utilized in the set-up. A process schematic and picture of the set-up is seen below in Figure 3.12.121.

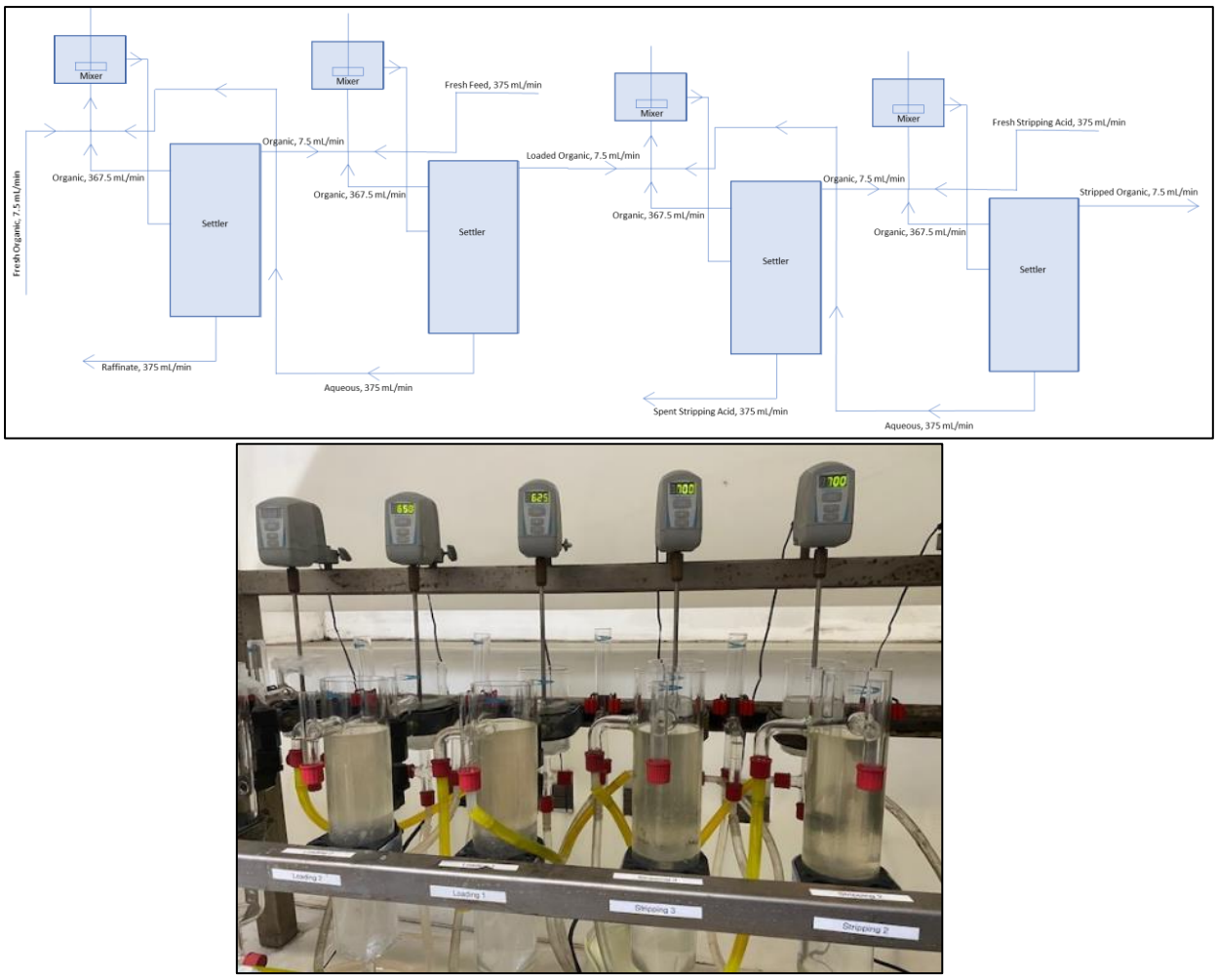



Figure 3.12.121. Process schematic (top) of small, pilot-scale solvent extraction unit used in initial scandium recovery experiments. Picture taken during processing (bottom) showing weir heights and mixer speeds after the completion of the adjustment period.

Mixer speed and weir height adjustments were made over the course of three days on the four-stage system using 9 gal. of recirculated PT0 filtrate feed, 3.5 L of organic, and 5 L of 2M sulfuric acid. Flowrates were measured to ensure mixer speeds were sufficiently high to pump the appropriate amount of organic recycle into each stage and move the aqueous phase through the system. After determining the working conditions of the system, an eight-hour initial processing run was completed using the remaining 41 gallons of feed and an additional 5 L of 2M sulfuric acid. The organic mixture utilized was saved from the adjustment period. Samples were taken of the feed, raffinate, and stripping acid before continuous processing commenced, at hour long intervals after commencement, and at the end of processing when the system had been drained and phases separated into appropriate containers. Table 3.12.37, below, displays this data as percent efficiencies, normalizing the concentrations provided through ICP-OES analysis.

Table 3.12.37. Scandium and iron recoveries during continuous processing in both the loading stages and stripping stages.

| Sample Pt. | Time (24 h) | Run Time (h) | Sc Loading (% Rec.) | Fe Loading (% Rec.) | Sc Stripping (% Rec.) | Fe Stripping (% Rec.) |
|---|---|---|---|---|---|---|
| 0 | 10:30 | 0:00 | - | - | - | - |
| 1 | 12:00 | 1:30 | 67.25 | -1.01 | 10.61 | -136.8 |
| 2 | 13:00 | 2:30 | 63.10 | -7.59 | 26.04 | -34.81 |
| 3 | 14:00 | 3:30 | 69.40 | 10.89 | 35.43 | 25.71 |
| 4 | 15:00 | 4:30 | 81.71 | 5.22 | 43.78 | 65.29 |
| 5 | 16:00 | 5:30 | 58.99 | 6.05 | 45.5 | 54.88 |
| Overall Efficiency | 16:20 | 5:50 | 74.88 | 12.21 | 60.48 | 78.97 |

Sampling of the feed, loading raffinate, and used stripping acid prior to the start of continuous processing was inconclusive, and the data is omitted from Table 3.12.37 as a result. Since adjustments were constantly being made on the system during the time in which those liquid volumes were produced, it is not representative of the process. Samples taken during continuous processing show much lower loading efficiencies than what would have been predicted from the lab-scale results discussed previously. This can largely be attributed to the loading stages only having a combined mixing time of approximately one minute instead of 15, due primarily to the limitations on the minimum organic flowrate that was able to be achieved, and the small volume, 200 mL of the mixers. From this, it is plausible equilibrium between the two phases was not reached before phase separation occurred.

It is imperative to note that samples of the feed and raffinate of the loading stages are not the same material, and material balances using the concentrations obtained from those samples (taken at corresponding time points) to assess loading efficiencies are approximations, at best, which is why negative percent recoveries are seen for some of the iron numbers. To account for this, representative samples of the raffinate and the used stripping acid were taken, and the feed concentrations averaged, after processing was complete. Weights and densities were also calculated in order to obtain accurate total volumes to normalize concentrations and complete the mass balance. As predicted from sampling during processing, loading efficiency of scandium was significantly reduced and iron loading increased compared to what was seen on the lab-scale, being only 74.88% for scandium and 12.21% for iron. This is again due to the limited retention time in the system. Stripping efficiency was also greatly reduced, being only 60.48% of that which was loaded and a 30% reduction from lab tests. From observations noted during the trial run, phase separation in the stripping stages was quite pronounced in the mixers, indicating phase mixing might not have been sufficient to ensure scandium stripping back into the aqueous ph.

**Cleaning Stage Solvent Extraction:** After processing of the saved AMD PT0 filtrate, the used stripping acid, a roughly 10 L volume, was sampled as part of further testing to establish if “cleaning” stages were necessitated. Much like the loading/stripping cycle described previously, the cleaning stages serve as a second round of scandium extraction in order to limit iron content in the final aqueous product that will be used in downstream processing, effectively “cleaning” the stripping acid of iron. Thus far, only the loading aspect of the cleaning cycle has been explored. Scandium and iron loading into the organic phase are largely pH dependent, so various amounts of base were trialed as part of the saponification process of

the organic, effectively determining the final pH of the loading raffinate. Table 3.12.38, below, lists the amounts of base used, and the final pH of the raffinate.

Table 3.12.38. pH and Eh measurements of the stripping acid raffinate, post-loading, as a function of base added to the organic during saponification.

| Sample ID | 50 wt% NaOH (mL) | pH | Eh |
|---|---|---|---|
| DTK-AMD20-CL-F | - | -0.289 | 742.2 |
| CL-R-1 | 0.002 | -0.252 | 729.4 |
| CL-R-2 | 0.010 | -0.277 | 742.5 |
| CL-R-3 | 0.020 | -0.285 | 746.7 |
| CL-R-4 | 0.050 | -0.270 | 752.6 |
| CL-R-5 | 0.100 | -0.243 | 741.5 |
| CL-R-6 | 0.500 | -0.155 | 739.9 |
| CL-R-7 | 1.000 | -0.072 | 748.9 |
| CL-R-8 | 1.500 | 0.053 | 738.7 |
| CL-R-9 | 2.000 | 0.209 | 746.4 |
| CL-R-10 | 2.500 | 0.411 | 736.7 |
| CL-R-11 | 3.000 | 0.545 | 663.8 |

Procedures followed for saponification and for loading testing were analogous to the procedures described previously, with the exceptions being that the aqueous and organic phases were in a ratio of 1:1, instead of 50:1, and volumes of organic and aqueous phases used for each trial were both 25 mL. Small changes in pH during the first five trials led to larger increases in the base aliquots into the organic phase during saponification during the last six trials. Upon reaching a pH of approximately 0.5, trials were stopped until ICP-OES analysis of the raffinate samples (CL-R-#) could be completed, in order to determine the appropriate next steps. Analysis of the stripping acid raffinate (post-saponification and post-loading) is seen below in Figure 3.12.122.

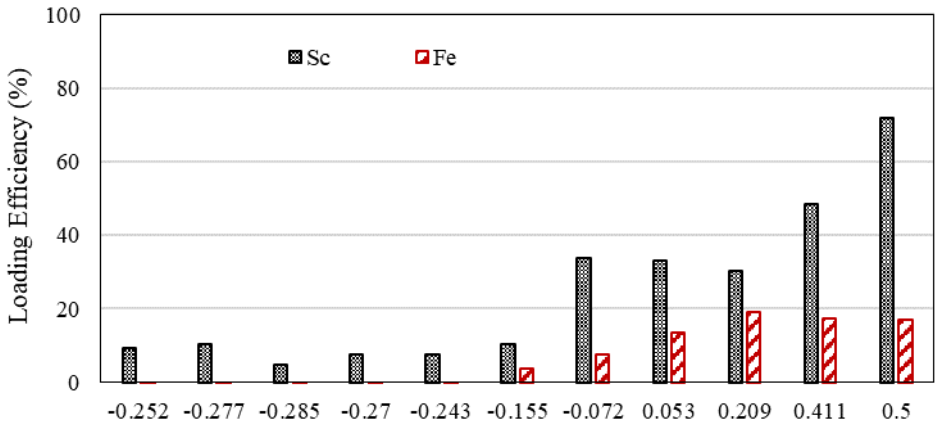



Figure 3.12.122. Recovery of scandium and iron during the second round of lab-scale loading as part of the stripping acid cleaning process.

As the pH went slightly above 0.5, scandium recovery from the stripping acid into the organic phase greatly improved, surpassing 70%. However, from a pH of roughly -0.2 and above, iron loading steadily increases, appearing to plateau between 15%-20%. From this, it is desirable to continue loading testing with larger aliquots of base to saponify the organic, as it may be possible to reach a pH of roughly 1 and recover nearly 100% of the scandium in the stripping

acid, while also only recovering less than 20% of the iron. Too, it will also be necessary to repeat the stripping trials, described previously, on the loaded cleaner-stage organic to determine if stripping conditions will be the same during the cleaning process. If this is proven to be the case, then the next round of continuous processing will at least contain one loading and one stripping stage for cleaning purposes.

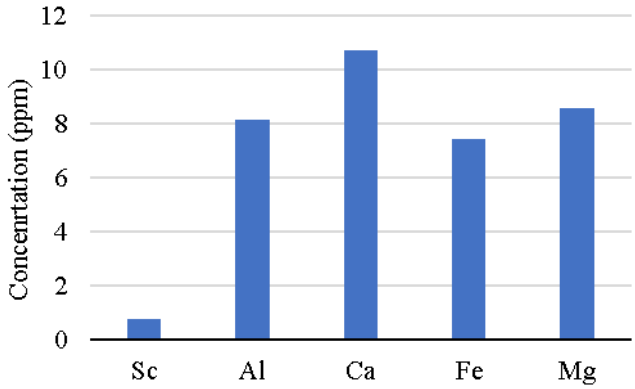



Figure 3.12.123. Concentration of Sc and major contaminant elements loaded in organic in the SX cleaner stage.

**Phosphate Precipitation for Scandium Recovery:** From a literature review on phosphate precipitation of leached bauxite residues, it was suggested that using sodium phosphate could shift the precipitation curves, in both sulfuric and nitric media, of iron, scandium, and aluminum such that scandium and iron would precipitate out of solution together, while aluminum would precipitate at a higher pH. Due to the similar precipitation behavior of aluminum and scandium, and low recovery rates of scandium resulting from this, it was desirable to test if similar precipitation behavior occurred in our pregnant leach solutions (PLS) as was described for the bauxite residues. As such, precipitation tests were conducted on 500 mL samples of Dotiki acid mine drainage (same material used in SX tests, DTKAMD20) and Fireclay PLS, (FC-C7) which were in sulfuric and nitric media, respectively. Precipitation of these two solutions was induced via 1M sodium phosphate aliquots, adjusted to achieve certain pH targets as sampling points. A comparison of sodium hydroxide precipitation curves and sodium phosphate precipitation curves are shown below in Figure 3.12.123 for the DTKAMD20 solution.

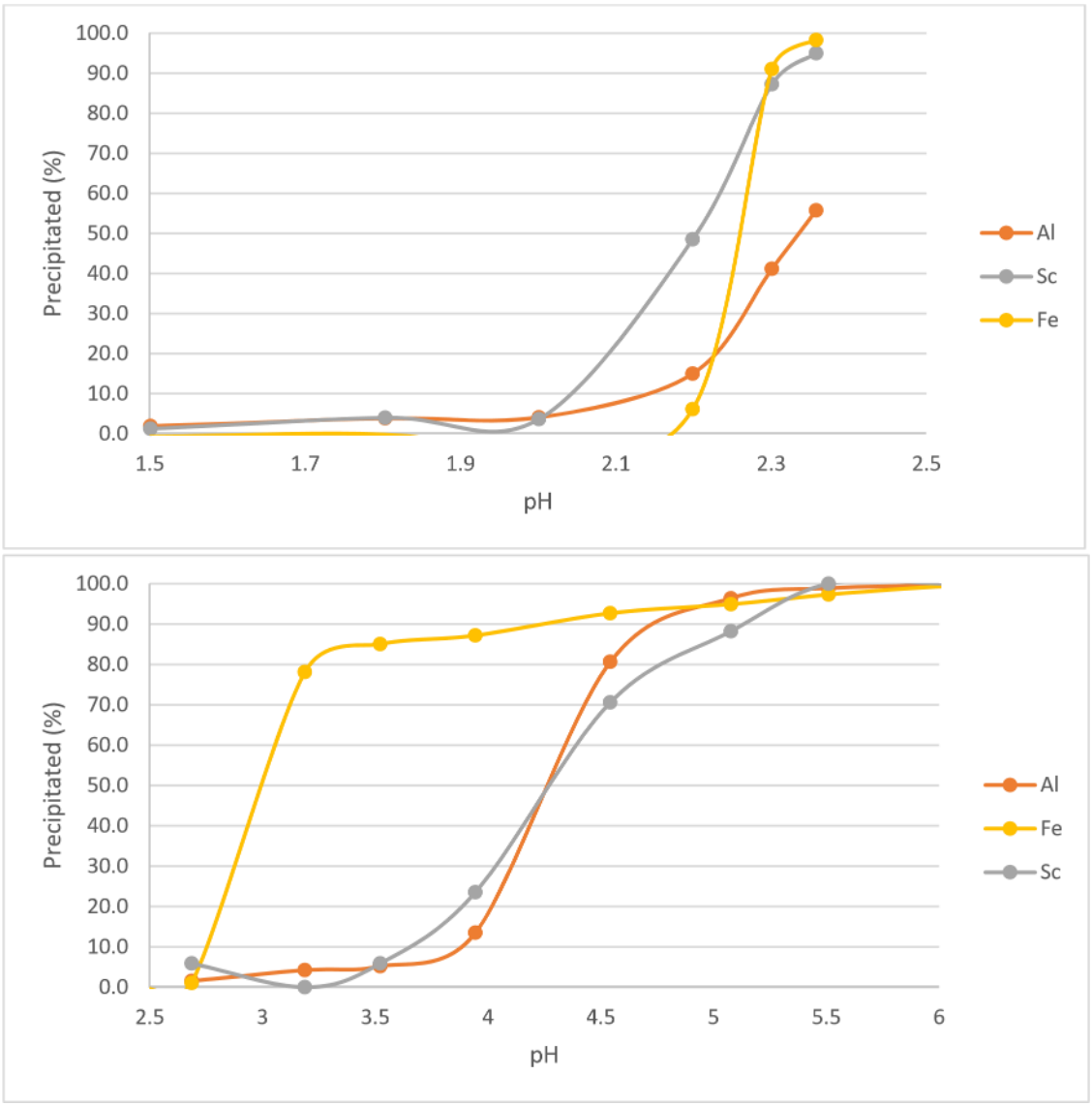



Figure 3.12.124 Sodium hydroxide (bottom) precipitation curves and sodium phosphate (top) precipitation curves on DTK-AMD20 solution.

Much like the behavior shown in the precipitation of the bauxite residues, the scandium and iron curves appeared to much more closely follow each other than the aluminum and scandium curves that are characteristic to our precipitation tests with sodium hydroxide. Since it is well established that iron precipitation can be initiated with sodium hydroxide at a pH of 3.2, a new stage in the process could be designed to precipitate the aluminum and scandium cake at a pH of ~4.5 with sodium hydroxide after the iron removal, then have the cake be redissolved in sulfuric acid at a pH of 2.0, and finally re-precipitate the scandium at a pH of 2.3 separate from the aluminum. This would effectively recover 90% of the scandium, while eliminating 60% of the aluminum. As seen in the lower graph, when aluminum and scandium curves are reversed during sodium hydroxide precipitation, to obtain a 60% cut of aluminum, scandium would follow in a 1:1 ratio, also losing nearly 60%. The issue here lies in the cost-effectiveness of adding the new stage to the process to recover 90% of the scandium, while also keeping 40% of the total aluminum in the cake for use in redissolution and downstream processing.

When working with nitric-leached material, a considerable difference occurs between the precipitation curves of aluminum, scandium, and iron when introducing phosphate into the system and those observed when using only sodium hydroxide. The main issue arising from

using nitric acid as the leaching acid, though it does provide a more efficient leach, is the inability to separate rare earths from iron and aluminum contaminants via normal precipitation procedures, as all elements precipitate at roughly the same pH as shown in Figure 3.12.125.

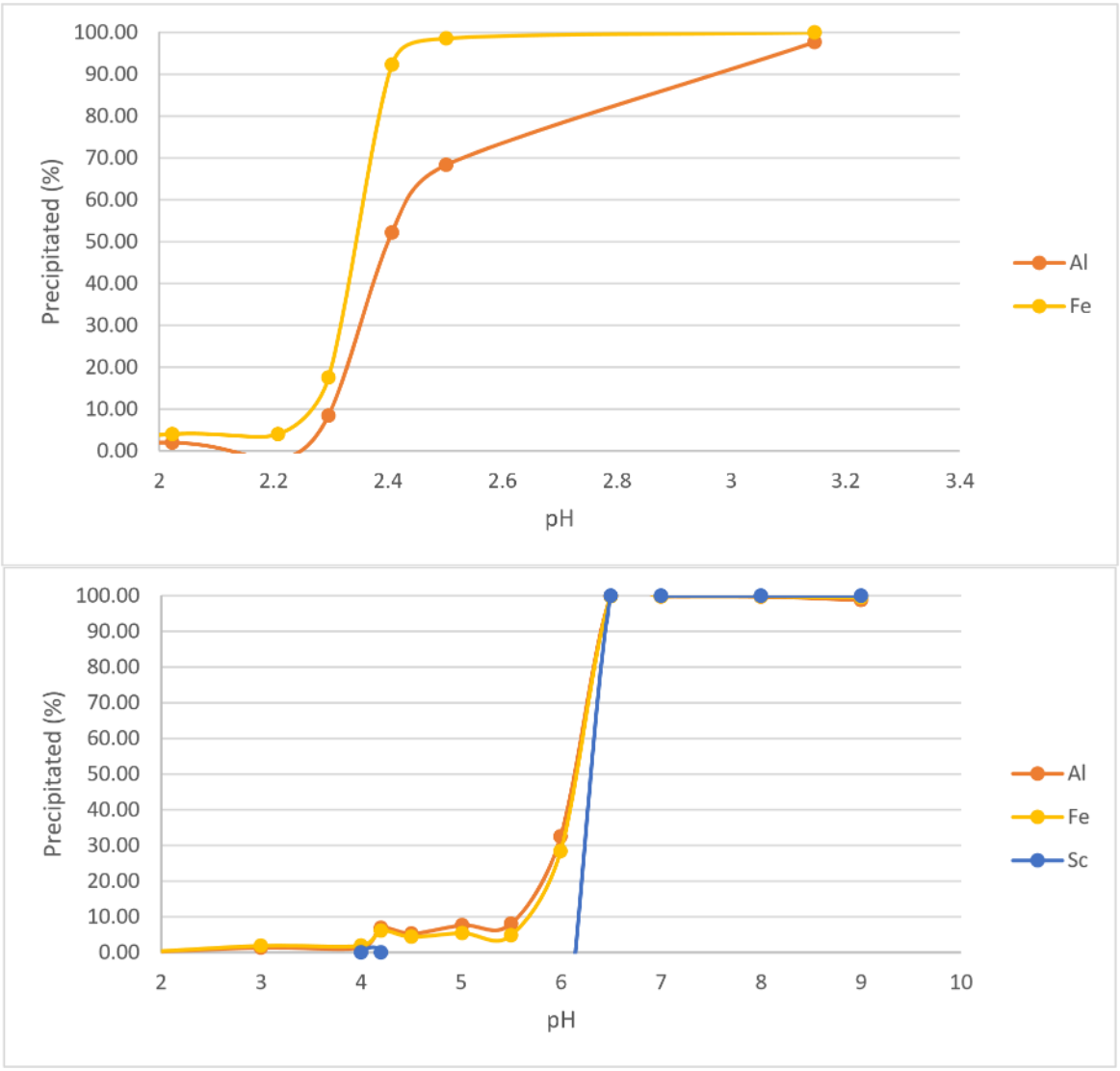



Figure 3.12.125 Sodium phosphate precipitation (top) and sodium hydroxide precipitation (bottom) curves of aluminum, scandium, and iron in nitric media.

While a clear delineation between iron and aluminum precipitation is seen when phosphate anions are introduced into the system, it is difficult to draw conclusions as to where the scandium and other rare earth precipitation curves will fall with regards to pH. Scandium concentrations in the nitric-leached FC-C7 material were not high enough to register during ICP-OES analysis. From the precipitation of other rare earths (not shown), it is possible the scandium precipitation would fall between the iron and aluminum curves seen in the top graph. Nonetheless, the limited pH range over which this separation occurs during phosphate precipitation is likely to be too narrow to control during continuous processing in the plant. Also, contamination with 70% or more aluminum when precipitating the rare earths post-iron precipitation is much higher than the maximum allowable for this method to remain cost-effective.

While nitric-leaching has been determined to not be compatible with the current process, further tests on various sulfuric-based leach solutions are still planned. Specifically,

determination of the minimum phosphate anion concentration needed in the solution to induce the scandium/aluminum precipitation curve separation, variations in precipitation curves both before and after redissolution of rare-earth hydroxide cakes with hydrochloric media, and cost analyses to determine if this could be a more effective approach than solvent extraction. From this initial testing, however, it appears solvent extraction will remain the more cost-effective solution.

### *3.12.13 Configuration of solvent extraction circuit for scandium extraction*

The existing solvent extraction unit was replumbed to accommodate new stages and additional treatment steps than had previously been incorporated into the system. The first new change to the system was to add a second loading stage to the unit to accommodate a higher feed flowrate of 2 gal/min. This new loading stage was plumbed to be parallel to the existing loading stage so that both are fed by separate but identical streams. Both stages flow directly into the same stripping stage. Therefore, the solvent extraction process now uses six total mixer-settlers with two going toward each step.

In addition to the second loading stage, a new step has been added to the solvent extraction unit directly before the feed enters the loading stages. The leach solution used as the feed for the solvent extraction contains iron (III) in high concentrations which will interfere with the extraction of the desired product, scandium. Steel wool can be used to reduce iron (III) to iron (II) and a reduction tank has been added to the system to prepare the feed stream to enter the unit. Steel wool will be packed in mesh bags which will be packed into the reduction tank to allow for feed interaction with the steel wool while allowing for easy retrieval and replacement.

Finally, the stripping solution lines were plumbed from the stripping solution holding tank into the system overall. A peristaltic pump was added after the aqueous overflow weir that returns to the stripping solution holding tank in order to control the flowrate entering the tank more strictly at 0.04 gpm. The 2 M stripping solution was prepared and pumped into the system to begin filling the system and preparing to begin the extraction process. In addition to loading the stripping solution into the solvent extraction unit, the organic solution was prepared and loaded into the system as well. The organic mixture for this process contains Cyanex-272 and Orfom as the extractant and diluent respectively. The 1% Cyanex solution was pumped into the loading and stripping stages of the system and required 130 gallons of organics in total.

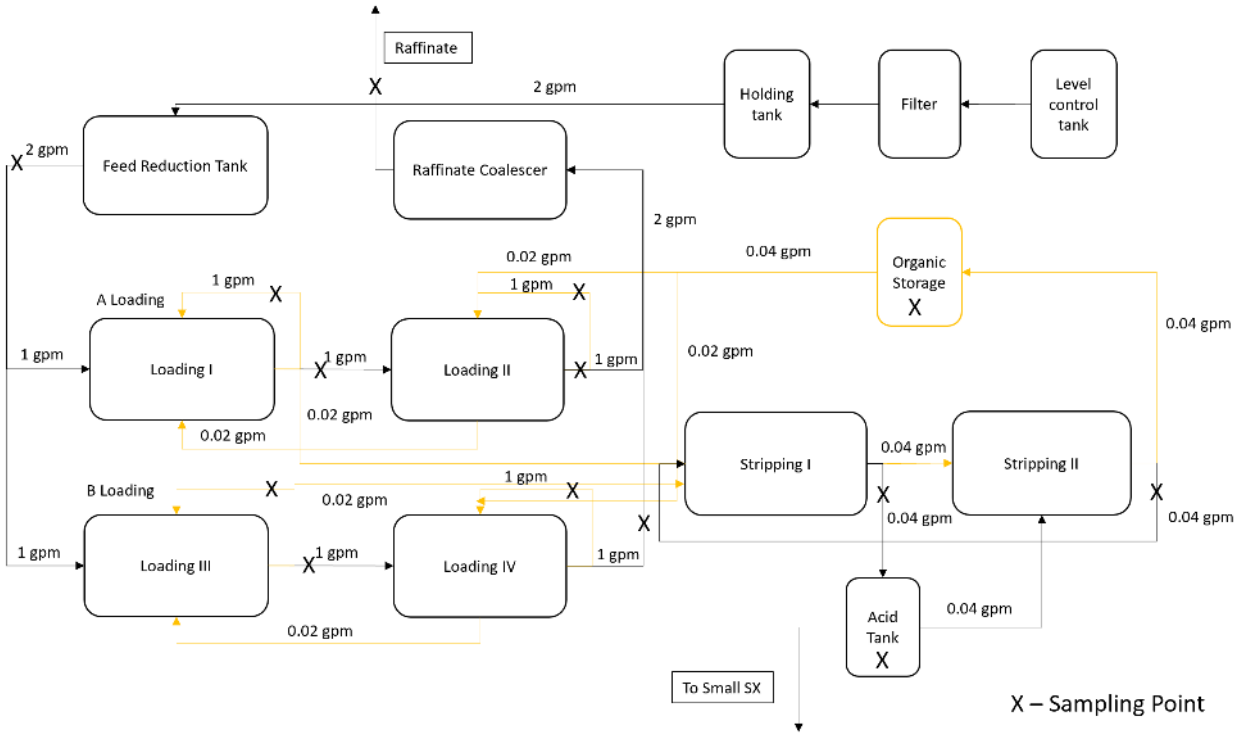



Figure 3.12.126. Solvent extraction Process Flow Diagram including locations to take data analysis samples.

**Pump Calibrations:** The system is controlled using several Masterflex L/S and I/P peristaltic pumps. Pump calibrations have already been performed on the I/P, but calibration was performed for the L/S pumps with the Masterflex L/S 77252-72 pump head and the Masteflex 6440-35 tubing. Results are reported below in Table 3.12.39.

Table 3.12.39. Data collected from pump calibration of Masterflex L/S pump with Masterflex 35 tubing.

| 35 Masterflex | | |
|---|---|---|
| | Setting | Flowrate |
| | 30 | 0.035663 |
| | 30 | 0.035135 |
| | 30 | 0.035135 |
| Average | 30 | 0.035311 |
| | 70 | 0.079516 |
| | 70 | 0.078723 |
| | 70 | 0.079252 |
| Average | 70 | 0.079164 |
| | 100 | 0.108311 |
| | 100 | 0.108311 |
| | 100 | 0.109103 |
| Average | 100 | 0.108575 |

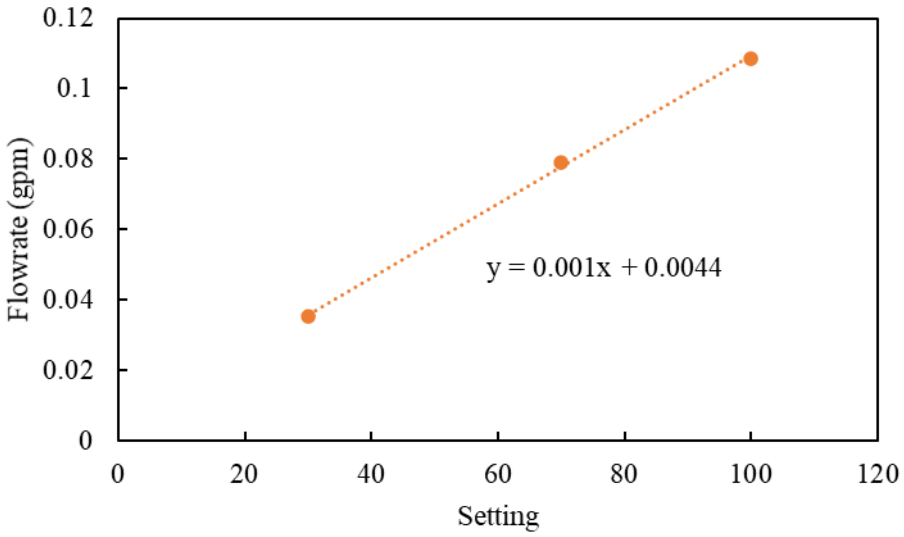



Figure 3.12.127. Correlation developed from calibration data points for Master flex L/S pump and 35 Master flex tubing.

**Ferric reduction using Steel Wool:** To remove scandium more effectively from the aqueous solution, the iron content of the feed solution can be reduced from $Fe^{3+}$ (ferric iron) to $Fe^{2+}$ (ferrous iron). Ferric iron will also be extracted by the organic extractant used in the process reducing the extraction of scandium over time significantly. Steel wool can be used to reduce ferric iron to ferrous iron. To determine the efficacy of steel wool reduction, lab tests were performed with grade 0 steel wool and a concentrated iron (III) sulfate solution. Materials used include Sigma Aldrich iron (III) sulfide hydrate and trace metal grade 98% sulfuric acid purchased from VWR International. Grade 0 steel wool was purchased from Lustersheen Products. The iron (III) sulfate solution was prepared to be representative of the aqueous feed of the solvent extraction. A 0.1 M sulfuric acid solution was used as the solvent for the representative feed solution and iron (III) sulfate hydrate was used to achieve a concentration of 7000 ppm iron. This stock solution had an initial pH of approximately 1 and Eh of 670 mV. Every experiment utilized a steel wool piece approximately 5 grams in weight.

First, two 50 mL centrifuge tubes are prepared by drilling holes in the bottom of the tube. One tube will have a hole of diameter 1/16-in, and the other is 1/8-in. The 5-gram piece of grade 0 steel wool is cut from a 5 lb. roll to be around 3.5 in in height and such that it will fit in the centrifuge tube and reach the 50 mL mark. The steel wool stuffed centrifuge tube is clamped above a 50 mL digestion tube for sample collection. A separatory funnel filled with the prepared stock solution is then suspended above the centrifuge tube to allow the stock solution to drip through the steel wool.

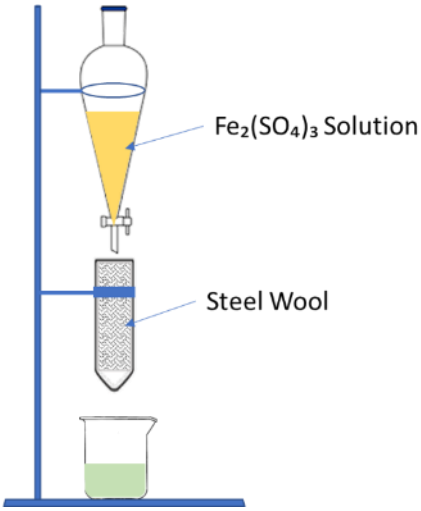



Figure 3.12.128. Illustration of steel wool testing arrangement.

To start a trial, begin a drip of stock solution at 90 - 100 drops per min and allow it to saturate the steel wool. Begin timing the trial once the first drop leaves the bottom of the selected centrifuge tube. Once 50 mL of solution has been collected in the digestion tube, stop timing, and stop the drops from the separatory funnel. This process is repeated until 10 collections of 50 mL have been completed. Eh and pH measurements were taken for all trials. Two variations on this experiment were performed for analysis. One complete experiment was run using the centrifuge tube with 1/16-in with an average feed flowrate. A second experiment was run using the centrifuge tube with the 1/8-in at a significantly faster feed flowrate (~150 drops/min) in order to reduce the residence time of the experiment.

**Results:**

The steel wool does successfully reduce the iron (III) sulfate solution which can be seen in the drop in Eh. The standard stock solution starts with an Eh of 670 mV and is reduced in both trials to 300-400 mV. When looking at the duration of both experiments as well as calculated residence times, the reduction does happen fairly quickly. Average residence time within the centrifuge tube based on 25 mL of used volume is 4.6 minutes.

Notable deterioration of the steel wool can be observed as iron reduction is completed. Steel wool was weighted before and after the complete experiment when completely dry. Significant weight loss was observed after use in reduction indicating iron dissolution. During the longer experiment using the 1/16-in hole centrifuge tube, 0.8 grams of steel wool was dissolved and after the shorter experiment with the 1/8-in hole centrifuge tube, 0.6 grams of steel wool was dissolved. During the 1/8-in experiment, the faster feed flowrate reduced the area and time of interaction between the iron (III) sulfide solution and the steel wool resulting in less steel wool dissolution and less feed reduction. ICP data also shows that iron is dissolving from the steel wool and reducing the iron which increases the overall iron concentration within the stock solution.

$$2Fe^{3+} + Fe \rightarrow 3Fe^{2+}$$

The samples collected increased in iron concentration from around 6000 ppm iron to 9000 ppm iron indicating that on average an additional 0.06 grams of iron was added to the samples from the steel wool. Steel wool is not pure iron and thus it is reasonable to assume that the additional mass lost from the steel wool is other materials breaking down under the wet and acidic conditions of the stock solution.

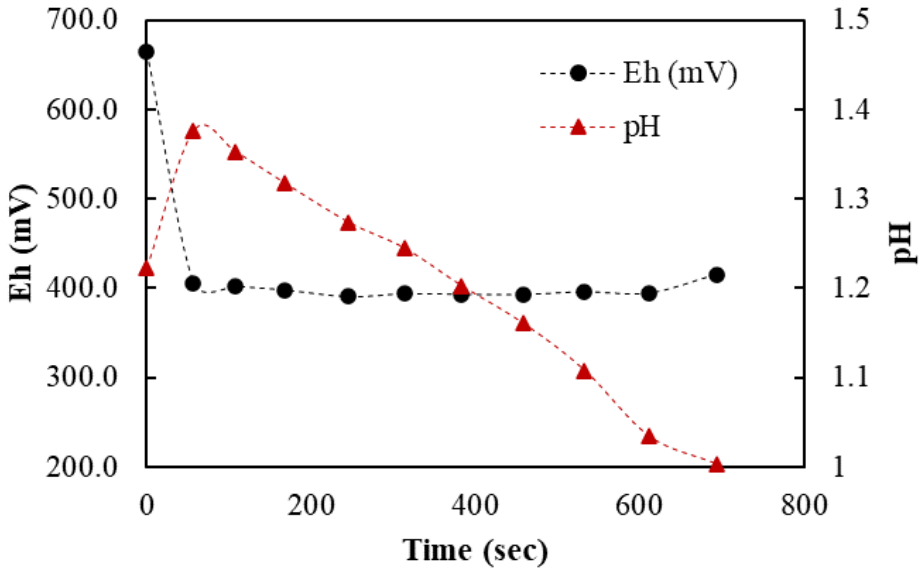



Figure 3.12.129. 1/8-in hole centrifuge tube steel wool test pH and Eh vs time.

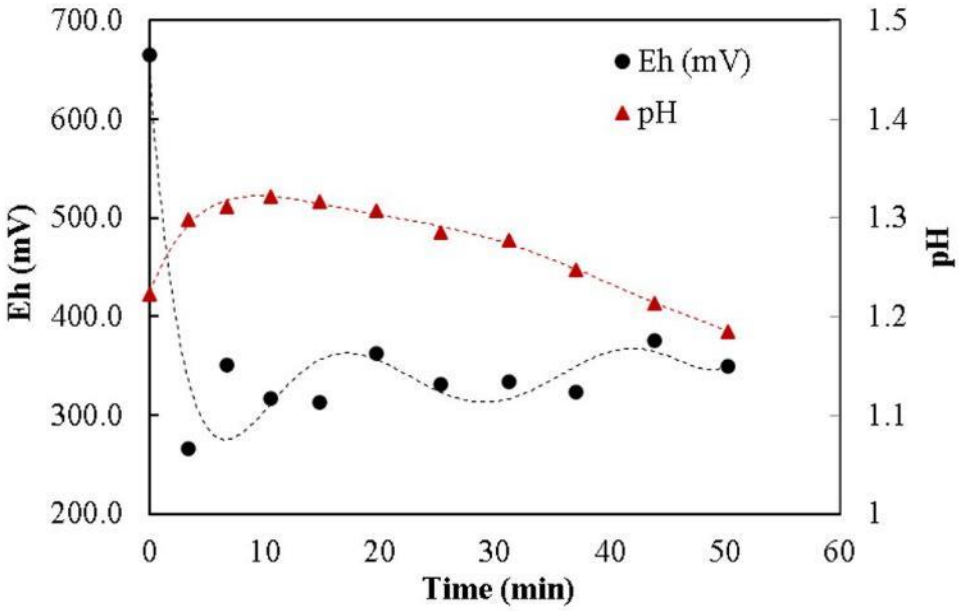



Figure 3.12.130. 1/16-in hole centrifuge tube steel wool test pH and Eh vs time.

Table 3.12.40. Data collection results of 50 mL collections from 1/8-in centrifuge tube test.

| Sample | Time (s) | Eh (mV) | pH |
|---|---|---|---|
| Feed | 0 | 664.7 | 1.223 |
| 1 | 56.78 | 406.1 | 1.377 |
| 2 | 51.71 | 402.1 | 1.353 |
| 3 | 59.72 | 397.8 | 1.318 |
| 4 | 77.85 | 390.4 | 1.274 |
| 5 | 67.85 | 393.7 | 1.245 |
| 6 | 69.17 | 393 | 1.203 |
| 7 | 74.9 | 392.8 | 1.161 |
| 8 | 75.09 | 395.9 | 1.108 |
| 9 | 78.7 | 394.5 | 1.034 |
| 10 | 81.46 | 415.4 | 1.003 |

Table 3.12.41. Data collection results of 50 mL collection from 1/16-in centrifuge tube test.

| Sample | Time (min) | Eh (mV) | pH |
|---|---|---|---|
| 0 | 0 | 664.7 | 1.223 |
| 1 | 03:19.0 | 266.2 | 1.298 |
| 2 | 03:20.8 | 350.6 | 1.311 |
| 3 | 03:53.0 | 316.9 | 1.322 |
| 4 | 04:13.0 | 313.4 | 1.316 |
| 5 | 04:55.7 | 363.2 | 1.307 |
| 6 | 05:33.9 | 331 | 1.285 |
| 7 | 05:55.9 | 334.4 | 1.278 |
| 8 | 05:51.0 | 324 | 1.247 |
| 9 | 06:35.7 | 376.1 | 1.214 |
| 10 | 06:24.1 | 349.6 | 1.185 |

### *1.1.25 Production of Scandium Concentrate*

Upon completion of pilot plant solvent extraction tests, laboratory tests were performed on the concentrated sulfuric stripping acid with concentrations shown in Table 3.12.42 to develop a purification pipeline. The stripping acid used in the pilot-plant test runs from

Laboratory tests were conducted to evaluate the effectiveness of using sodium hydroxide to adjust pH to selectively precipitate contaminant (i.e., gangue) elements. Tests were conducted by adding sodium hydroxide solutions (1.0 to 18.9 M) to 50 mL of organic solution (1%v/v Cyanex 272/orform). Sodium hydroxide precipitation did not elucidate a meaningful separation of scandium from any gangue elements as shown in Figure 3.12.131.

Steel wool was used to reduce the iron at a pH of 2. Loaded stripping acid was added to a 250 mL Pyrex separatory funnel, dispensed through approximately 5 g of steel wool packed in a 50 mL centrifuge tube with a 1/8 in discharge, a 1%v/v Cyanex 272/orform solution was added to the reduced SA in 1:1, 1:5, 1:10, and 1:50 organic to aqueous (O:A) ratios, shakeout tests were performed, and the aqueous samples were analyzed via ICP-OES upon completion of the solvent extraction process as shown in Figure 3.12.132. These tests were perturbed by the formation of white crystals which interfered with operation of separatory funnels. Precipitation

efficiency of scandium and other REEs was inversely proportional to organic to aqueous ratio, while gangue element precipitation efficiency was directly proportional.

A 1:1 organic to aqueous ratio was selected, which had 96.3% and 3.0% loading efficiencies for scandium and iron, respectively. The impact of stripping acid concentration on scandium and iron recovery was investigated as shown in Table 3.12.43. Recoveries of both scandium and iron increased with sulfuric acid concentration, and 3 M sulfuric acid was used for subsequent processing as it demonstrated the highest scandium recovery.

The small solvent extraction unit was cleaned and plumbed to process using 3 M sulfuric acid stripping solution as shown in Figure 3.12.133. The small solvent extraction unit set up for scandium recovery was operated for 72.6 hours from June 21$^{st}$, 2022 until July 20$^{th}$, 2022. pH, Eh was recorded at every 2 hour sample point during operation for the aqueous feed (Figure 3.12.134), aqueous raffinate (Figure 3.12.135), stripping acid output (Figure 3.12.136), and stripping acid input (Figure 3.12.137). Elemental analysis showed the final concentrated stripping acid had 22.3 ppm scandium and 181.9 ppm iron. A sodium hydroxide precipitation test was performed on 500 mL of the concentrated stripping acid as shown in Figure 3.12.138. The pH setpoint of 7 was selected because 98.0% of the scandium was precipitated.

Approximately 8.9 gal of loaded stripping acid was added to a 15-gal polypropylene tank equipped with a mixer (IKA RW 20 Digital) set to 500 rpm shown in Figure 3.12.139. Sodium hydroxide (50 wt%) and hydrochloric acid (18.1 M) were used to adjust to pH 7 setpoint. Sodium hydroxide addition (11.5 L, 50 wt%) increased the solution temperature from 25.4 °C to as high as 59.9 °C. When the solution cooled overnight, crystallization of sodium salts became a major processing challenge (Figure 3.12.139). A heating element set to 30 °C and 15 L deionized water were added to increase the solubility of the crystals formed. Analysis of the solution indicated only 82.7% of the scandium precipitated.

A filter press was used to filter the solid precipitates from the precipitation tank. Membranes were cleaned in dilute hydrochloric acid solution and dried before filter papers were added to aid filtration. The slurry was pumped into the filter press at 4.75 gpm and the filtrate was recycled until the solution became clear. Approximately 11.81 g of concentrate was generated with concentrations shown in table below.

Table 3.12.42. Concentrations of REEs, scandium, and major contaminants in stripping acid via ICP-OES.

| Element | Concentration (ppm) |
|---|---|
| **TREE** | 16.3 |
| **Sc** | 8.5 |
| **Al** | 787.8 |
| **Ca** | 129.6 |
| **Fe** | 3783 |
| **Mg** | 814.7 |
| **Na** | 343.4 |

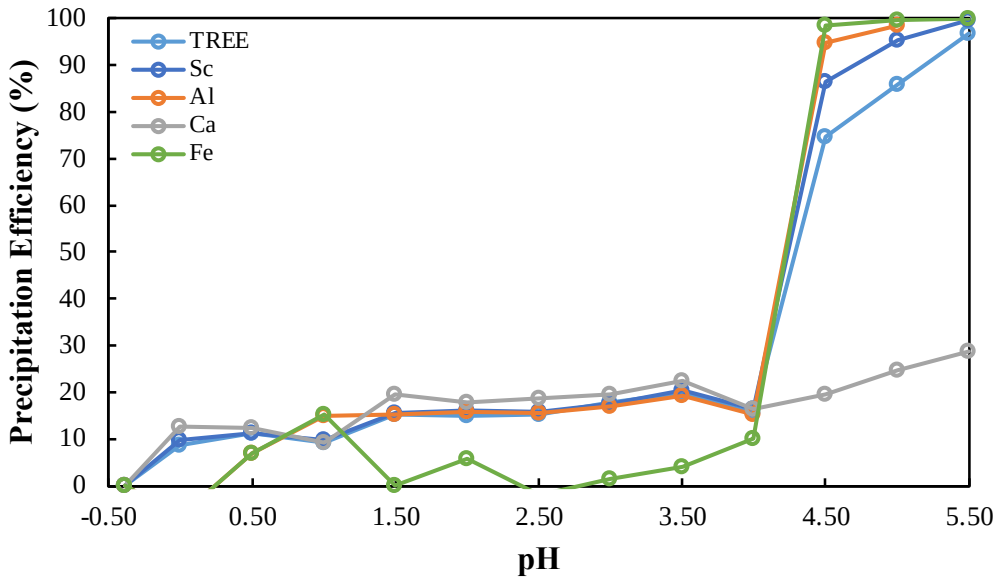



Figure 3.12.131. Sodium hydroxide precipitation of concentrated sulfuric stripping acid.

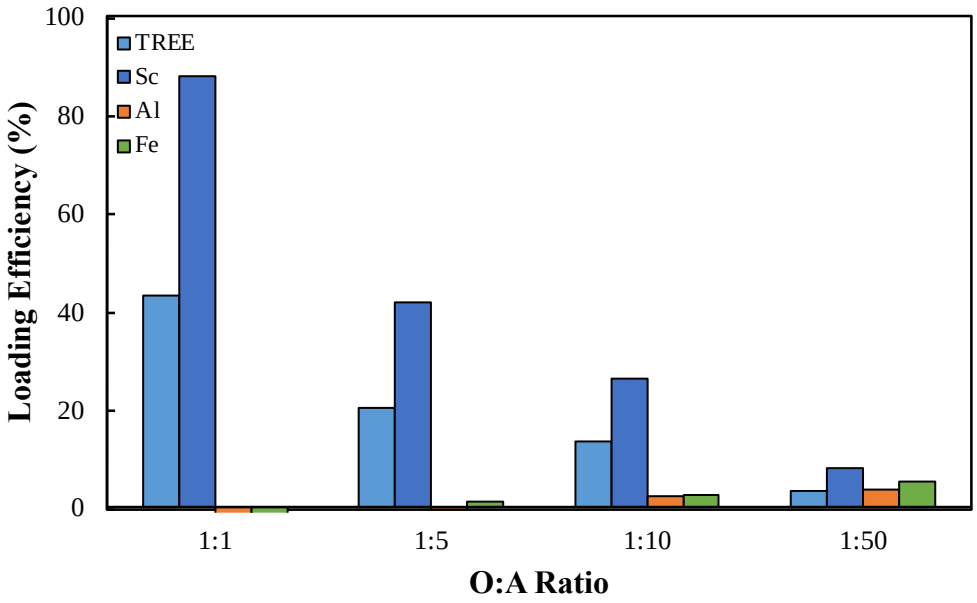



Figure 3.12.132. Loading efficiency for shakeout tests of multiple organic to aqueous ratios.

Table 3.12.43. Stripping acid concentration impact on scandium and iron recovery.

| | Recovery (%) | |
|---|---|---|
| **Acid Concentration (M)** | Scandium | Iron |
| **0.0** | 0.0 | 6.9 |
| **0.5** | 74.5 | 57.7 |
| **1.0** | 88.2 | 56.9 |
| **2.0** | 94.1 | 58.7 |
| **3.0** | 98.0 | 59.1 |



Figure 3.12.133. Picture taken of the small solvent extraction unit during operation.

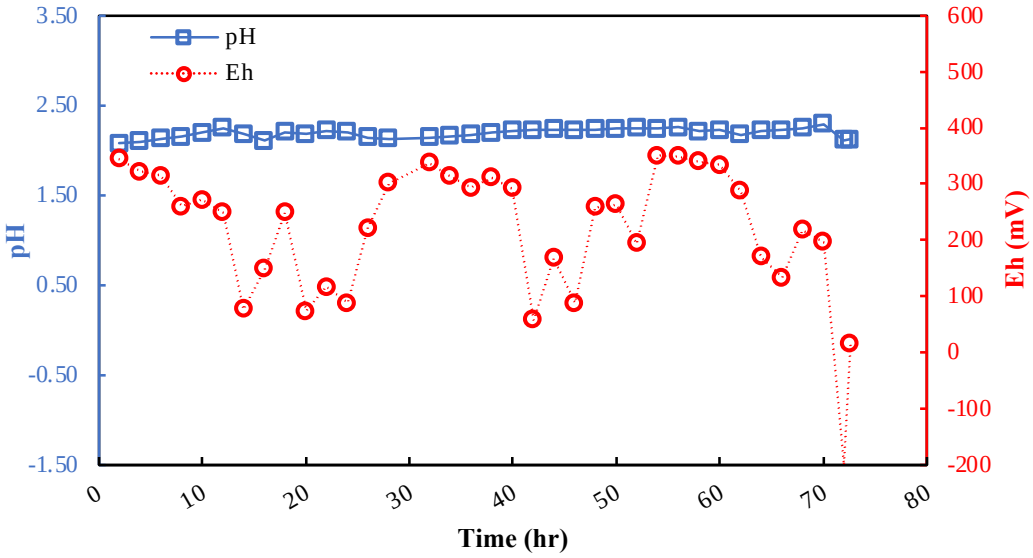



Figure 3.12.134. pH, Eh (vs. Ag/AgCl) of the aqueous feed during scandium cleaning process.

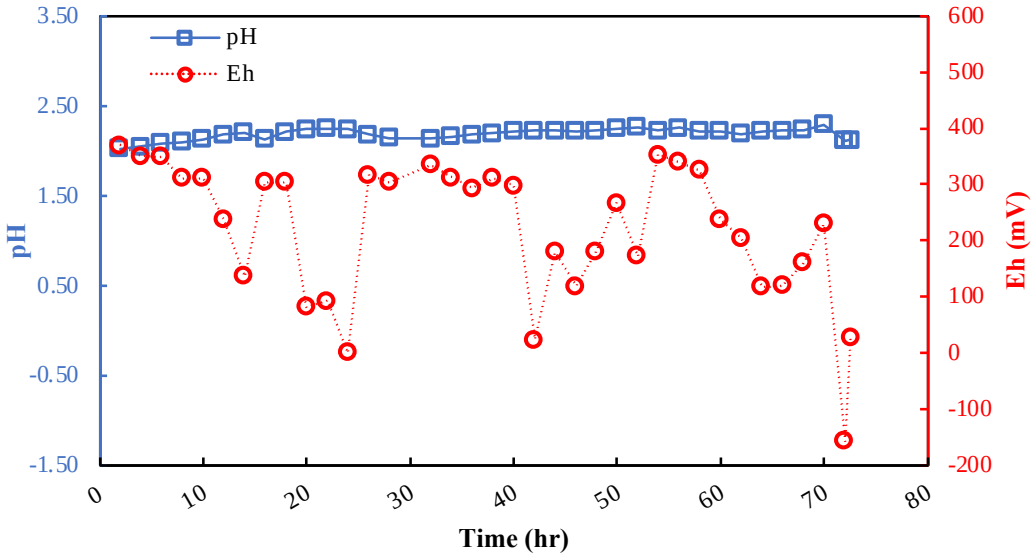



Figure 3.12.135. pH, Eh (vs. Ag/AgCl) of the aqueous raffinate during scandium cleaning process.

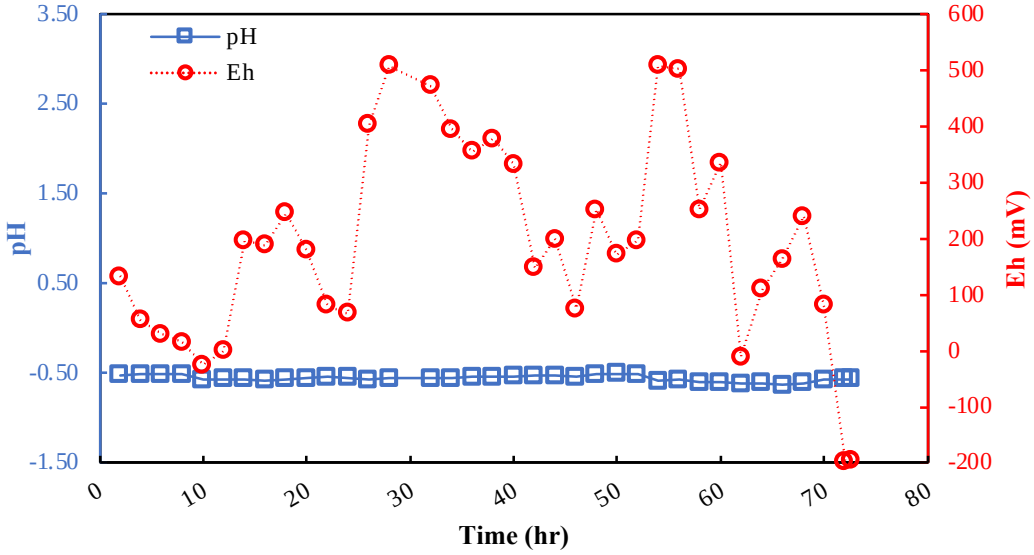



Figure 3.12.136. pH, Eh (vs. Ag/AgCl) of the stripping acid exiting the process during scandium cleaning process.

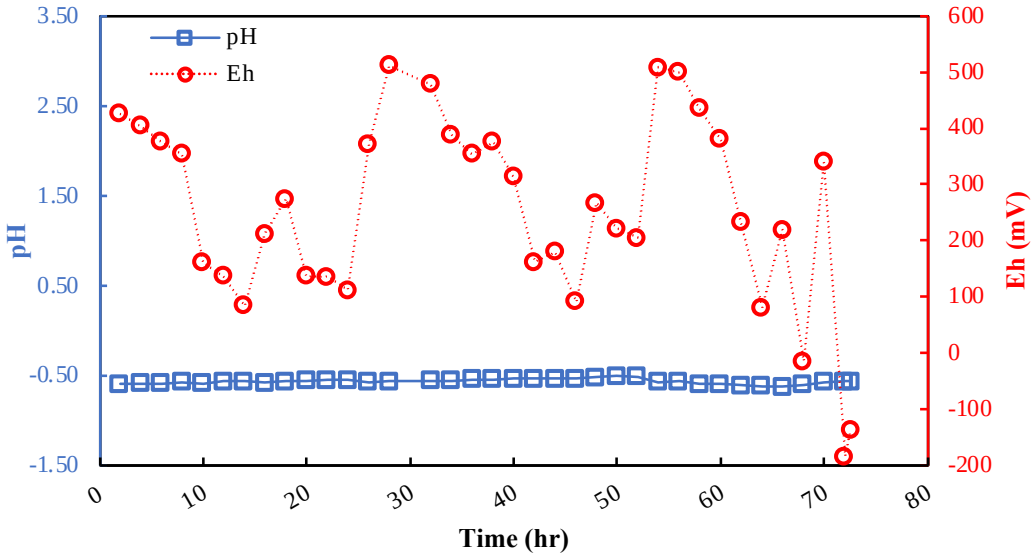



Figure 3.12.137. pH, Eh (vs. Ag/AgCl) of the stripping acid entering the process (i.e., bulk solution) during scandium cleaning process.

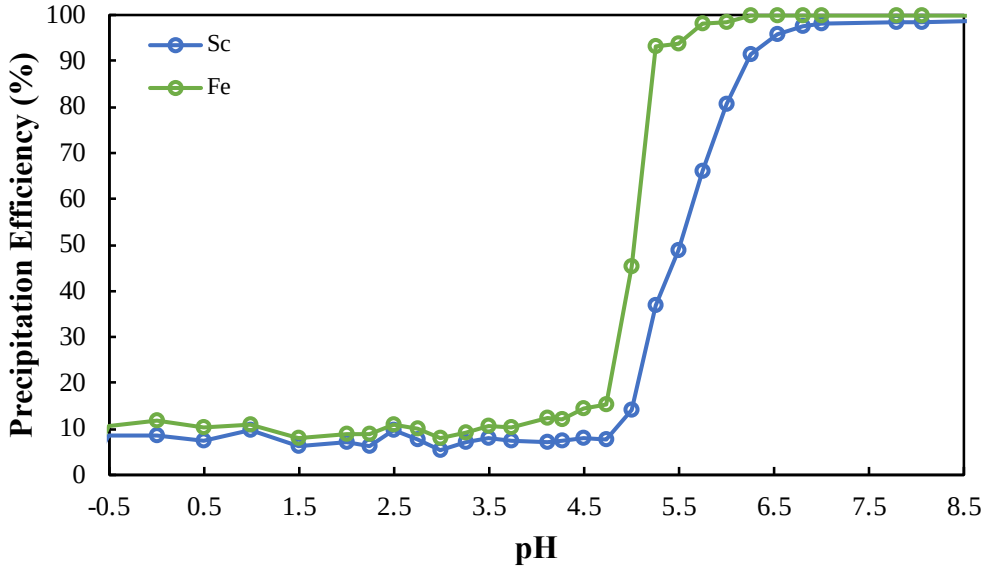



Figure 3.12.138. Sodium hydroxide precipitation of concentrated sulfuric stripping acid after steel wool reduction, dilution, and solvent extraction.

Table 3.12.44.Concentrated scandium product elemental concentrations and standard errors via ICP-OES.

| Element | Concentration (mg $kg^{-1}$) |
|---|---|
| **Sc** | 7743 ± 191 |
| **Y** | 2536 ± 89 |
| **La** | 2402 ± 67 |
| **Ce** | 530 ± 68 |
| **Nd** | 2772 ± 79 |
| **Sm** | 743 ± 22 |
| **Eu** | 154 ± 4 |
| **Gd** | 816 ± 25 |
| **Dy** | 565 ± 18 |
| **Ho** | 100 ± 3 |
| **Er** | 237 ± 6 |
| **Yb** | 161 ± 6 |
| **Lu** | 27 ± 1 |
| **Al** | 2124 ± 55 |
| **Ca** | 25894 ± 1576 |
| **Co** | 508 ± 9 |
| **Cr** | 2740 ± 181 |
| **Cu** | 621 ± 14 |
| **Fe** | 119749 ± 19082 |
| **Mn** | 1218 ± 30 |
| **Na** | 105087 ± 3002 |
| **Ni** | 1927 ± 37 |
| **Zn** | 12336 ± 216 |



Figure 3.12.139. (left) Picture taken of scaled scandium precipitation unit during operation and (right) after crystallization issues began.

### *1.1.26 Optimization of Solvent Extraction Circuit*

The TEA model developed under Task 3 was used to evaluate and optimize the flowsheet configurations and operational parameters for an REE solvent extraction system. The objective of this assessment was to determine the economic viability of directly treating the leach solution via solvent extraction for REE recovery. Earlier studies by the research team have shown that this approach is cost prohibitive, primarily due to the low REE concentrations in the leach solution, which are often on the order of 1 to 50 ppm. These prior data show that the cost vs. leachate concentration curve follows an exponential pattern and leachate concentrations of 800 ppm or more are required to achieve REE production costs lower than $50/kg. This prior analysis catalyzed the subsequent development of the precipitation and redissolution circuit that is currently being employed in the pilot plant.

Given new experimental data and recent results on the solvent extraction process, this techno-economic analysis was repeated to reassess the use of solvent extraction as the primary concentration unit in the circuit. A spreadsheet-based material balance and cost assessment model was developed using the methods and techniques described in prior reports. The model was tuned to calculate annual REE production, total solvent extraction capital cost, and total solvent extraction operating cost. Together, these outputs were combined to calculate total production cost (CAPEX + OPEX, $/kg REE) as the primary output metric. Given the objectives of this study, other cost factors, including that of upstream and downstream units were not evaluated or considered.

Capital cost factors included bare equipment costs for the tanks, mixers, and pumps as well as the initial reagent fills. The mixer settlers were sized based on retention time measurements, while pumps and other units were sized based on design flow rates. To account for ancillaries and installation, a total Lang factor of 3.72 was applied to the bare equipment cost. For the purpose of this analysis, the capital cost was annualized using an operating period of 20 years and a discount rate of 10%. The annualized capital cost was then divided by annual production to produce the capital cost portion of the production cost. Operating costs included itemized power and reagent requirements, including organic losses. All costs were calculated on an annual basis and normalized by the REE production. Labor costs and indirect/overhead costs were not included in this analysis.

Figure 3.12.140 shows the generic solvent extraction flowsheet utilized in this study. The configuration includes the option for multi-stage loading, multistage scrubbing, and multi-stage stripping. Internal recycles on the organic and various scrub/strip streams are configured to allow independent control of the advance O:A ratio and the mixer O:A ratio. For example, as shown in Figure 3.12.140, a significant portion of the loaded organic is internally recycled in the loading stage to maintain a mixer ratio of 1:1 but an advance ratio of 0.03:1. The 1:1 mixer ratio provides the proper volumes for adequate mixing and phase separation, while the low advance ratio produces a highly concentrated organic for the downstream scrub and strip units.

The various constant and variable design parameter used in the analysis are included in Table 3.12.45. To constrain the model and provide a fair basis for cost comparison, both the total REE production and the target strip solution concentration were set to fixed values, 1.5 t/day and 10 g/L, respectively. Given these parameters as well as well other inputted design factors (e.g. recovery), the PLS flow rate was calculated to reach the desired target production. Likewise, the O:A advance ratio in stripping was calculated to reach the desired strip solution concentration. These model constraints ensure that the system under consideration is performing the same output duty (i.e. production rate and strip solution composition) regardless of the variable model inputs, such as leachate REE content.

Table 3.12.45. Design Parameters used in Solvent Extraction Techno-Economic Model.

| Parameter | Value | Unit |
|---|---|---|
| REE content | variable | |
| Extraction O:A Advance Ratio | variable | |
| PLS Flow Rate | calcualted, based on target production | |
| Overal REE Recovery | calcualted, based on input extraction O:A ratio | |
| Stripping O:A Advance Ratio | calcualted, based on target strip concentration | |
| *Constants* | | |
| Target REE Production | 1.5 | t/day |
| Target Strip Concentration | 10 | g/L |
| Mixing Time | 2.50 | min |
| Mixing Intensity | 2.50 | HP/1000 gal |
| Settling Time | 15.00 | min |
| Number of Extraction Stages | 2.00 | -- |
| Number of Scrubbing Stages | 1.00 | -- |
| Number of Stripping Stages | 2.00 | -- |
| Number of Parallel Extraction Lines | 1.00 | -- |
| Number of Parallel Scrubbing Lines: | 1.00 | -- |
| Number of Parallel Stripping Lines: | 1.00 | -- |
| Extraction Mixer O:A | 1.00 | v/v |
| Scrubbing Mixer O:A | 1.00 | v/v |
| Stripping Mixer O:A | 1.00 | v/v |
| Scrubbing O:A Advance Ratio | 1.00 | v/v |
| Scrubbing Acid Molarity | 0.50 | mol/L |
| Scrubbing Acid Molar Mass | 36.46 | g/mol |
| Stripping Acid Stock Conc. | 37% | % |
| Stripping Acid Molarity | 6.00 | mol/L |
| Stripping Acid Molar Mass | 36.46 | g/mol |
| Stripping Acid Stock Conc. | 37% | % |
| Organic Entrainment (loss) | 60.00 | mg/L |
| Capital Period | 20.00 | years |
| Discount Rate | 10% | % |
| Lang Factor | 3.72 | -- |
| Cost Index Factor | 1.36 | -- |

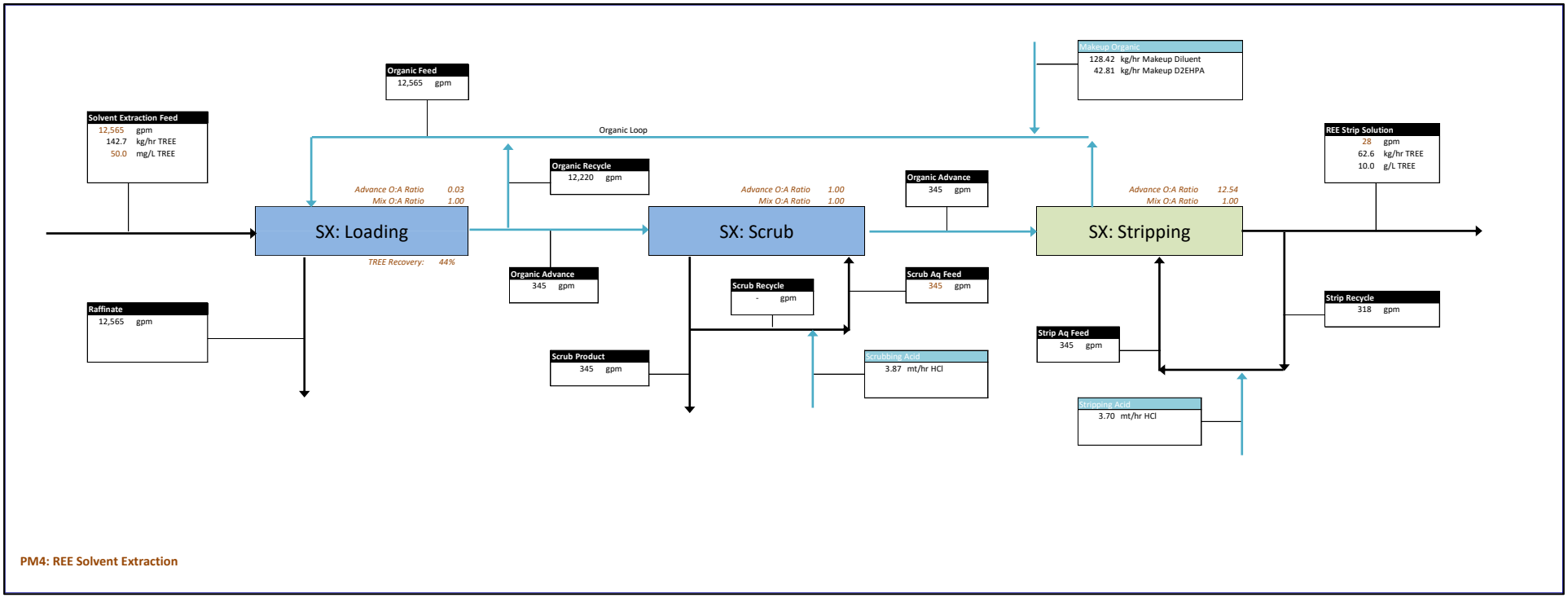



Figure 3.12.140. Generic solvent extraction flowsheet used in techno-economic analysis.

Initial model testing indicated that the O:A advance ratio in the extraction stage had a major impact on the system production cost. This O:A ratio directly controls the rate of internal organic recycle in the extraction stage, and control of this recycle creates a tradeoff between the REE concentration in the loaded organic and the REE recovery from the leachate. Lower O:A ratios (i.e. more organic recycle) leads to higher overall loading of REEs in the organic, which subsequently reduces the size and chemical consumption in the downstream units. However, the lower O:A ratio also inhibits REE extraction, thus reducing REE recovery from the leachate. Nevertheless, this tradeoff creates a techno-economic optimization problem where the ideal ratio will balance the cost reductions and recovery gains.

To model this tradeoff, a simple empirical recovery vs. O:A ratio model was constructed using data collected earlier in this project and published in a prior report. For this experiment, an organic solution comprised of 25% D2EHPA and 75% diluent was contacted with fresh PLS at O:A ratios ranging from 10:1 to 1:50. The data showed that REE recovery generally remained high (>90%) for ratios greater than 1:10; however, the recovery dropped precipitously at the lower ratios of 1:50. The empirical model produced from this data outputs extraction stage recovery when given the extraction O:A advance ratio. For the purpose of this analysis, all other recoveries (i.e. scrubbing and stripping) were assumed to be 100%.

**Extraction O:A Ratio Optimization:** Given the significant impacts of extraction O:A ratio on overall system cost that were observed during the initial model testing, a parametric sweep of O:A ratios was first conducted to identify the optimal value. Figure 3.12.141 shows the results from this parametric sweep with O:A ratios varying from 0.01:1 to 1:1 and REE concentrations in the PLS varying from 25 to 100 mg/L. This data clearly demonstrates the tradeoff described above, with high production costs at both low O:A ratios (due to low recovery) and at high O:A ratios (due to increased chemical consumption. Based on the assumptions utilized in this study, the data passes through an optimum at an O:A ratio of 0.03:1 or roughly 1:36.

While the data does show that increased REE concentration leads to lower production costs, it also interestingly shows that the optimal O:A ratio is not strictly dependent on the REE concentration, as all three curves in this figure show the same optimal point. Instead, the optimum is largely driven by the recovery vs. O:A ratio relationship, which was empirically determined for this study. Thus, one recommendation for future work is to conduct additional test work at relatively low O:A ratios to provide further resolution and validation of the model in those conditions.

Lastly, this data explains the high solvent extraction cost that were reported in earlier studies. Those studies consistently used an O:A ratio of 1:1, which this data confirms is the highest cost operational point. At this ratio, even moderate leachate concentrations of 50 mg/L produce untenable production costs of nearly $300 / kg. This result correlates well to the prior findings and further confirms the need to evaluate lower O:A ratios.

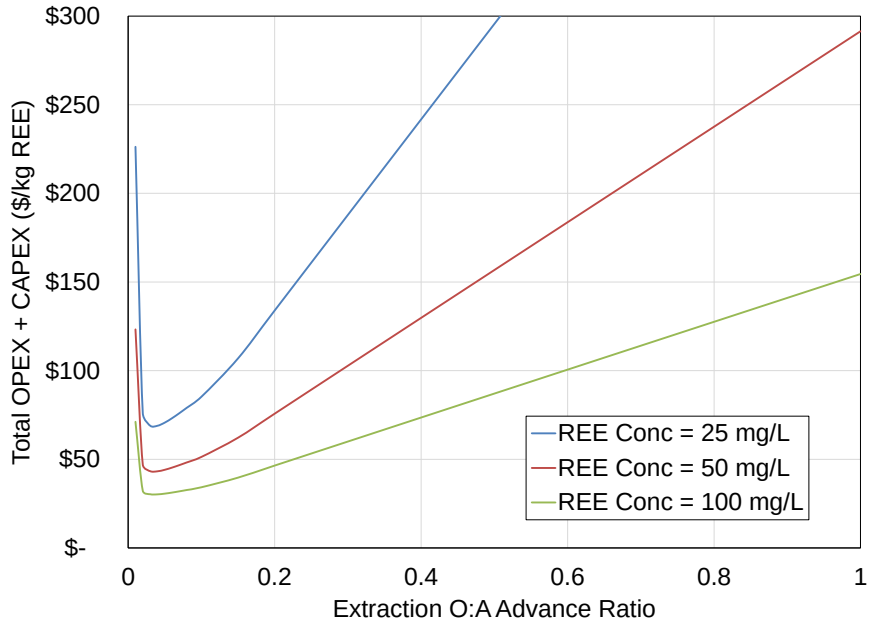



Figure 3.12.141. Techno-economic model results for multi-stage REE solvent extraction system treating raw leach solution. Data shows total solvent extraction production cost as a function of extraction O:A ratio for various REE concentrations in the leach solution. Model assumptions and parameters given in Table 3.12.45.

**Influence of Leachate Concentration:** To provide guidelines and benchmarks on the extent of leaching needed to promote an economically viable solvent extraction process, a second parametric sweep was conducted considering leachate concentration as the independent variable. The results from this analysis are shown in Figure 3.12.142 for various O:A ratios, including the optimal ratio of 0.03:1. As in the prior studies, this data confirms the exponential relationship between cost and leachate concentration and further supports the need to concentrate the leach solution as much as possible prior to solvent extraction. For the optimal case, the data shows that a cost target of $50/kg REE can be achieved with a leach solution concentration of 40 mg/L, while a cost target of $25/kg can be achieved with a solution concentration of 175 mg/L. Altogether, these results, and particularly the optimal result demonstrate a potential pathway for the integration of direct solvent extraction recovery in the extraction process. Additional data is needed to confirm the metallurgical performance of the process and the general conclusions of this analysis; however, this initial study does show promise and indicates that further study is warranted.

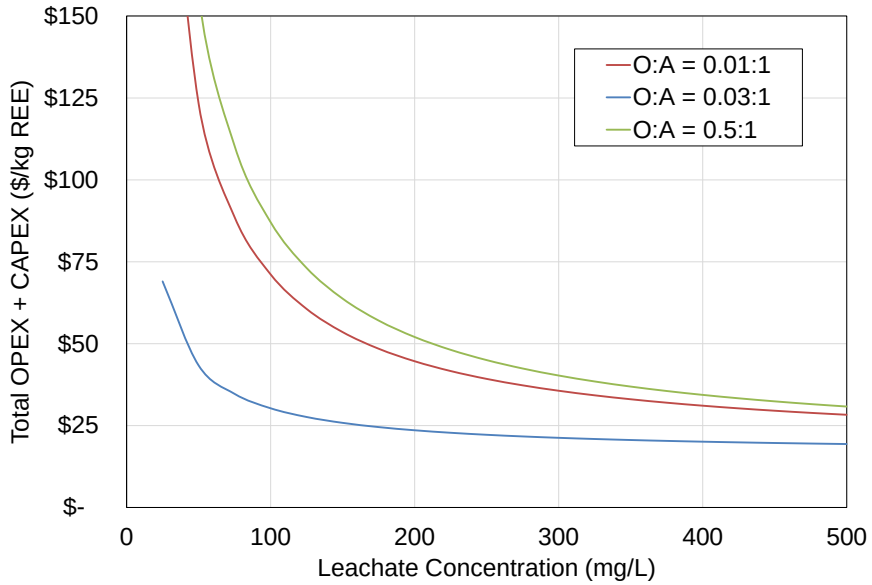



Figure 3.12.142. Techno-economic model results for multi-stage REE solvent extraction system treating raw leach solution. Data shows total solvent extraction production cost as a function of REE leachate concentration for various O:A ratios, including the optimal ratio of 0.03:1. Model assumptions and parameters given in Table 3.12.45.

# 4 PRODUCTS

## I. JOURNAL ARTICLES

Gupta, T., Nawab, A., & Honaker, R. (2023). Removal of Iron from Pyrite-Rich Coal Refuse by Calcination and Magnetic Separation for Hydrometallurgical Extraction of Rare Earth Elements. *Minerals*, *13*(3), 327. https://doi.org/10.3390/min13030327

Li Q., Ji B., Honaker R., Noble A., Zhang W., 2022. Partitioning behavior and mechanisms of rare earth elements during precipitation in acid mine drainage. *Colloids and Surfaces A*, 641, 128563. https://doi.org/10.1016/j.colsurfa.2022.128563

Li Q., Zhang W., 2022. Process development for recovering critical elements from acid mine drainage. *Resources, Conservation & Recycling*, 180, 106214. https://doi.org/10.1016/j.resconrec.2022.106214

Nawab, A., Yang, X., & Honaker, R. (2022). Parametric study and speciation analysis of rare earth precipitation using oxalic acid in a chloride solution system. *Minerals Engineering*, *176*, 107352. https://doi.org/10.1016/j.mineng.2021.107352

Nawab, A., Yang, X., & Honaker, R. (2022). An acid baking approach to enhance heavy rare earth recovery from bituminous coal-based sources. *Minerals Engineering*, *184*, 107610. https://doi.org/10.1016/j.mineng.2022.107610

Prasenjit Podder, Zongliang Zhang, Rick Q. Honaker, Michael L. Free, and Prashant K. Sarswat, "Evaluating and Enhancing Iron Removal via Filterable Iron Precipitates Formation during Coal-Waste Bioleaching", *Eng*. 2(4), 632-642, 2021.

Prashant K Sarswat, Prasenjit Podder, Zongliang Zhang, Michael L Free, "Autogenous acid production using a regulated bio-oxidation method for economical recovery of REEs and critical metals from coal-based resources" *Journal of Applied Surface Science Advances*, 11, 100283, 2022.

Michael L Free, Joel K Ilunga, Prasenjit Podder, Prashant K Sarswat, Assessing and Improving Biooxidation for Acid Generation and Rare Earth Element Extraction, *Processes,* 11(4), 1005.

Li, Q., Ji, B., Xiao, Z., & Zhang, W. (2022). Alkali Pretreatment Effects on Acid Leaching Recovery of Rare Earth Elements from Coal Waste of the Western Kentucky No. 13 and Fire Clay Seams. Miner. Miner. Mater, 1(7).

Gupta, T., Nawab, A., & Honaker, R. (2023). Effect of Calcination Atmosphere on the Leaching Recovery of Rare Earth Elements in Coal-Byproducts (Submitted).

Nawab, A., Yang, X., & Honaker, R. (2023). Parametric study of an acid baking process for rare earth element recovery from a bituminous-coal source (Submitted).

## II. PEER-REVIEWED CONFERENCE ARTICLES

None.

## III. NON-REVIEWED CONFERENCE ARTICLES

None.

## IV. ABSTRACT & CONFERENCE PRESENTATION

Li Q., Zhang W., "Process development for recovering critical elements from an acid mine drainage," 2022 SME Annual Conference & Expo., 02/27-03/02, 2022, Salt Lake City, Utah.

Nawab A. Yang X. Honaker R. "*Parametric study of an acid baking process for rare earth element recovery from a low-grade bituminous coal source*", SME Annual Conference and EXPO, Denver, CO, March 2023.

Nawab A. Yang X. Honaker R. "*Enhanced heavy rare earth recovery from bituminous coal-sources using acid baking*", SME Annual Conference and EXPO, Salt Lake City, UT, March 2022.

Nawab A. Yang X. Honaker R. "*Oxalic acid precipitation of rare earth elements at high contaminants-to-REE ratios*", SME Annual Conference and Expo. Salt Lake City, UT, March 2022.

Nawab A. Yang X. Honaker R. "*An acid baking approach to enhance heavy rare earth recovery from bituminous-coal sources*", Annual MPD Conference, Colorado Springs, CO, April 2022.

Michael L. Free, "Extraction and Recovery of Rare-earth Elements and Critical Materials from Coal Waste Using Low Cost Processing Methods", Keynote Presentation, Rare Metal Extraction and Processing, Presentation at TMS Annual Meeting, Anaheim, CA, USA, February 28, 2022.

Michael L. Free, Rare Earth and Critical Materials, UAMMI Cross Talk Presentation, Price, Utah, September 22, 2022.

Honaker, Rick, Josh Werner, Xinbo Yang, Aaron Noble, Wencai Zhang (2020). "Demonstration of Scaled-Production of Rare Earth Oxides and Critical Materials from U.S. Coal-Based Sources." Update on U.S. DOE's Office of Fossil Energy: Critical Minerals and Rare Earth Elements R&D. United States Energy Association. Online Conference. August 25, 2020.

Honaker, Rick, Josh Werner, Michael Free, Aaron Noble, Wencai Zhang (2020). "Demonstration of Scaled-Production of Rare Earth Oxides and Critical Materials from U.S. Coal-Based Sources." DOE-NETL's 2020 Integrated Project Review Meeting – Rare Earth Elements. Online Conference. September 15-16, 2020.

Honaker, Rick, Josh Werner, Xinbo Yang, Michael Free, Aaron Noble, and Wencai Zhang. (2021). "Demonstration of Scaled-Production of Rare Earth Oxides and Critical Materials from U.S. Coal-Based Sources." DOE-NETL's 2021 Project Review Meeting: Rare Earth Elements – Critical Minerals. Online Conference. May 25, 2021.

Honaker, Rick, Michael Free, Josh Werner, Aaron Noble, Wencai Zhang, and Xinbo Yang. (2022). "Scaled-Production of Rare Earth Oxides and Critical Materials from Coal-Based Sources." 2022 SME Annual Meeting and Exhibit. Salt Lake City, UT. February 27 – March 2, 2022.

Honaker, Rick, Xinbo Yang, Josh Werner, Michael Free, Aaron Noble, and Wencai Zhang. (2022). "A Pilot Plant Study of Rare Earth and Critical Material Extraction and Purification from Coal-Based Sources." The 46th International Technical Conference on Clean Energy. Clearwater, FL, August 2022

Noble, Aaron and Rick Honaker. (2022). “Demonstration of Scaled-Production of Rare Earth Oxides and Critical Materials from U.S. Coal-Based Sources (DE-FE00031827).” NETL Resource Sustainability Project Review Meeting. Pittsburgh, PA, October 25 – 27, 2022.

Li Q., Zhang W., “Process development for recovering critical elements from an acid mine drainage,” 2022 SME Annual Conference & Expo., 02/27-03/02, 2022, Salt Lake City, Utah. (Abstract and presentation)

## V. OTHER PUBLICATION PRODUCTS

Russell, Alexandra. (2021). Identification, Characterization, and Speciation of Rare Earth Elements in Coal Refuse. Master’s Thesis. Virginia Tech, Mining Engineering.

## VI. PATENTS & PATENT APPLICATIONS

Zhang W., Li Q., Honaker R., 2021. Innovative circuit for recovering critical elements from leachate of coal-based materials. US Patent, being processed (Application No: 63/266,414).

## 5 REFERENCES

Allegretta, I., Pinto, D., & Eramo, G. (2016). Effects of grain size on the reactivity of limestone temper in a kaolinitic clay. *Applied Clay Science*, *126*, 223–234. https://doi.org/10.1016/J.CLAY.2016.03.020

Arbuzov, S. I., Maslov, S. G., & Il'enok, S. S. (2015). Modes of occurrence of scandium in coals and peats (A review). *Solid Fuel Chemistry 2015 49:3*, *49*(3), 167–182. https://doi.org/10.3103/S0361521915030027

Ariffin, M. A. M., Bhutta, M. A. R., Hussin, M. W., Mohd Tahir, M., & Aziah, N. (2013). Sulfuric acid resistance of blended ash geopolymer concrete. *Construction and Building Materials*, *43*, 80–86. https://doi.org/10.1016/j.conbuildmat.2013.01.018

Balek, V., & Murat, M. (1996). The emanation thermal analysis of kaolinite clay minerals. In *Thermochimica Acta* (Vol. 282).

Banks, C., Carlson, O., Daane, A., Fassel, V., Fisher, R., Olson, E., Powell, J., & Spedding, F. (1959). *Studies on the preparation, properties and analysis of high purity yttrium oxide and yttrium metal at the Ames Laboratory*.

Belver, C., Bañares Muñoz, M. A., & Vicente, M. A. (2002). Chemical activation of a kaolinite under acid and alkaline conditions. *Chemistry of Materials*, *14*(5), 2033–2043.

Berry, L., Agarwal, V., Galvin, J., & Safarzadeh, M. S. (2018). Decomposition of monazite concentrate in sulphuric acid. *Canadian Metallurgical Quarterly*, *57*(4), 422–433. https://doi.org/10.1080/00084433.2018.1478490

Blickwedel, T. W. (1949). *Decomposition of Monazite-Effect of a Number of Variables on the Decomposition of Monazite Sand with Sulfuric Acid*. Iowa State Coll., Ames.

Colina, F. G., Abellan, M. N., & Caballero, I. (2006). High-temperature reaction of kaolin with ammonium sulfate. *Industrial and Engineering Chemistry Research*, *45*(2), 495–502. https://doi.org/10.1021/ie050872f

Colina, F. G., Esplugas, S., & Costa, J. (2001). High temperature reaction of kaolin with inorganic acids. *British Ceramic Transactions*, *100*(5), 203–206. https://doi.org/10.1179/096797801681468

Colina, F. G., Esplugas, S., & Costa, J. (2002). High-temperature reaction of kaolin with sulfuric acid. *Industrial and Engineering Chemistry Research*, *41*(17), 4168–4173. https://doi.org/10.1021/ie010886v

Corgnale, C., Gorensek, M. B., & Summers, W. A. (2020). Review of sulfuric acid decomposition processes for sulfur-based thermochemical hydrogen production cycles. In *Processes* (Vol. 8, Issue 11, pp. 1–22). MDPI AG. https://doi.org/10.3390/pr8111383

Criado, J. M., Ortega, A., Real, C., & de Torres, E. T. (1984). Re-examination of the kinetics of the thermal dehydroxylation of kaolinite. *Clay Minerals*, *19*(4), 653–661.

Cychosz, K. A., & Thommes, M. (2018). Progress in the Physisorption Characterization of Nanoporous Gas Storage Materials. *Engineering*, *4*(4), 559–566. https://doi.org/10.1016/J.ENG.2018.06.001

Dash, S., Kamruddin, M., Ajikumar, P. K., Tyagi, A. K., & Raj, B. (2000). Nanocrystalline and metastable phase formation in vacuum thermal decomposition of calcium carbonate. *Thermochimica Acta*, *363*(1–2), 129–135.

Demol, J., Ho, E., & Senanayake, G. (2018a). *Sulfuric Acid Baking and Leaching of Rare Earth Elements, Thorium and Phosphate from a Monazite Concentrate* (pp. 2343–2351). https://doi.org/10.1007/978-3-319-95022-8_197

Demol, J., Ho, E., & Senanayake, G. (2018b). Sulfuric acid baking and leaching of rare earth elements, thorium and phosphate from a monazite concentrate: Effect of bake temperature from 200 to 800 °C. *Hydrometallurgy*, *179*, 254–267. https://doi.org/10.1016/J.HYDROMET.2018.06.002

Demol, J., Ho, E., Soldenhoff, K., & Senanayake, G. (2019). The sulfuric acid bake and leach route for processing of rare earth ores and concentrates: A review. *Hydrometallurgy*, *188*, 123–139. https://doi.org/10.1016/j.hydromet.2019.05.015

Demortier, A., Gobeltz, N., Lelieur, J. P., & Duhayon, C. (1999). Infrared evidence for the formation of an intermediate compound during the synthesis of zeolite Na–A from metakaolin. *International Journal of Inorganic Materials*, *1*(2), 129–134. https://doi.org/10.1016/S1466-6049(99)00020-3

Dutrizac, J. E. (2017). The behaviour of the rare earth elements during gypsum (CaSO4·2H2O) precipitation. *Hydrometallurgy*, *174*, 38–46. https://doi.org/10.1016/j.hydromet.2017.09.013

Earnest, C. M. (1991). Thermal analysis of selected illite and smectite clay minerals. Part I. Illite clay specimens. In *Thermal analysis in the geosciences* (pp. 270–286). Springer.

EDOMWONYI-OTU, L. C., Aderemi, B. O., Ahmed, A. S., Coville, N. J., & Maaza, M. (2013). Influence of Thermal Treatment on Kankara Kaolinite. *Opticon1826 (15) (2013)* , *0*(15). https://doi.org/10.5334/OPT.BC

Erdogan, S., Baysal, A., Akba, O., & Hamamci, C. (2007). Interaction of metals with humic acid isolated from oxidized coal. *Polish Journal of Environmental Studies*, *16*(5).

Gao, W., Zhao, S., Wu, H., Deligeer, W., & Asuha, S. (2016). Direct acid activation of kaolinite and its effects on the adsorption of methylene blue. *Applied Clay Science*, *126*, 98–106.

Gerold, E., Luidold, S., & Antrekowitsch, H. (2020). Selective precipitation of metal oxalates from lithium ion battery leach solutions. *Metals*, *10*(11), 1435.

Ghosh, S. N. (n.d.). *Advances in cement technology: critical reviews and case studies on manufacturing, quality control, optimization and use.* (2014th ed.). Elsevier.

Groen, J. C., Peffer, L. A. A., & Pérez-Ramírez, J. (2003). Pore size determination in modified micro- and mesoporous materials. Pitfalls and limitations in gas adsorption data analysis. *Microporous and Mesoporous Materials*, *60*(1–3), 1–17. https://doi.org/10.1016/S1387-1811(03)00339-1

Gupta, T. (2021). *OXIDATION PRETREATMENT FOR ENHANCED LEACHABILITY OF RARE EARTH ELEMENTS FROM BITUMINOUS COAL SOURCES*.

Gupta, T., Nawab, A., & Honaker, R. (2023). Removal of Iron from Pyrite-Rich Coal Refuse by Calcination and Magnetic Separation for Hydrometallurgical Extraction of Rare Earth Elements. *Minerals*, *13*(3), 327. https://doi.org/10.3390/min13030327

Hadley, T., & Catovic, E. (2014). Beneficiation and extraction of REE from northern minerals' Browns Range Heavy Rare Earth project. *COM 2014—Conference of Metallurgists Proceedings*.

Han, K. N. (2019). Effect of anions on the solubility of rare earth element-bearing minerals in acids. *Mining, Metallurgy & Exploration*, *36*(1), 215–225.

Han, K. N. (2020). Characteristics of Precipitation of Rare Earth Elements with Various Precipitants. *Minerals 2020, Vol. 10, Page 178*, *10*(2), 178. https://doi.org/10.3390/MIN10020178

Han, K. N., & Kim, R. (2021). Thermodynamic Analysis of Precipitation Characteristics of Rare Earth Elements with Sulfate in Comparison with Other Common Precipitants Thermodynamic Analysis of Precipitation Characteristics of Rare Earth Elements with Sulfate in. *Comparison with Other Common Precipitants. Minerals*, *11*, 670. https://doi.org/10.3390/min

He, C., Makovicky, E., & Øsbæck, B. (1995a). Thermal stability and pozzolanic activity of calcined illite. *Applied Clay Science*, *9*(5), 337–354.

He, C., Makovicky, E., & Øsbæck, B. (1995b). Thermal stability and pozzolanic activity of calcined illite. *Applied Clay Science*, *9*(5), 337–354.

Ilić, B. R., Mitrović, A. A., & Miličić, L. R. (2010). Thermal treatment of kaolin clay to obtain metakaolin. *Hemijska Industrija*, *64*(4), 351–356.

Jha, M. K., Kumari, A., Panda, R., Kumar, J. R., Yoo, K., & Lee, J. Y. (2016). Review on hydrometallurgical recovery of rare earth metals. *Hydrometallurgy*, *165*, 2–26.

Ji, B., Li, Q., Tang, H., & Zhang, W. (2022). Rare earth elements (REEs) recovery from coal waste of the Western Kentucky No. 13 and Fire Clay seams. Part II: Re-investigation on the effect of calcination. *Fuel*, *315*, 123145. https://doi.org/10.1016/j.fuel.2022.123145

Ji, B., Li, Q., & Zhang, W. (2022a). *Fuel Rare Earth Elements Recovery from Coal Waste of the Western Kentucky No. 13 and Fire Clay Seams. Part I: Mineralogical Characterization Using SEM-EDS*. https://doi.org/10.1016/j.fuel.2021.121854

Ji, B., Li, Q., & Zhang, W. (2022b). Rare earth elements (REEs) recovery from coal waste of the Western Kentucky No. 13 and Fire Clay Seams. Part I: Mineralogical characterization using SEM-EDS and TEM-EDS. *Fuel*, *307*, 121854. https://doi.org/10.1016/J.FUEL.2021.121854

Kim, E., & Osseo-Asare, K. (2012). Aqueous stability of thorium and rare earth metals in monazite hydrometallurgy: Eh-pH diagrams for the systems Th-, Ce-, La-, Nd- (PO 4)-(SO 4)-H 2O at 25 °c. *Hydrometallurgy*, *113–114*, 67–78. https://doi.org/10.1016/j.hydromet.2011.12.007

Kim, R., Cho, H., Jeong, J., Kim, J., Lee, S., Chung, K. W., Yoon, H.-S., & Kim, C.-J. (2020). Effect of Sulfuric Acid Baking and Caustic Digestion on Enhancing the Recovery of Rare Earth Elements from a Refractory Ore. *Minerals*, *10*(6), 532. https://doi.org/10.3390/min10060532

Kuila, U., & Prasad, M. (2013). Specific surface area and pore-size distribution in clays and shales. *Geophysical Prospecting*, *61*(2), 341–362. https://doi.org/10.1111/1365-2478.12028

Kumari, A., Panda, R., Jha, M. K., Lee, J. Y., Kumar, J. R., & Kumar, V. (2015). Thermal treatment for the separation of phosphate and recovery of rare earth metals (REMs) from Korean monazite. *Journal of Industrial and Engineering Chemistry*, *21*, 696–703.

Kuppusamy, V. K., & Holuszko, M. (2022). Sulfuric acid baking and water leaching of rare earth elements from coal tailings. *Fuel*, *319*, 123738. https://doi.org/10.1016/j.fuel.2022.123738

Lenarda, M., Storaro, L., Talon, A., Moretti, E., & Riello, P. (2007). Solid acid catalysts from clays: Preparation of mesoporous catalysts by chemical activation of metakaolin under acid conditions. *Journal of Colloid and Interface Science*, *311*(2), 537–543.

Li, Q., Ji, B., Honaker, R., Noble, A., & Zhang, W. (2022). Partitioning behavior and mechanisms of rare earth elements during precipitation in acid mine drainage. *Colloids and Surfaces A: Physicochemical and Engineering Aspects*, *641*. https://doi.org/10.1016/j.colsurfa.2022.128563

Lide, D. R. (2004). *CRC handbook of chemistry and physics* (Vol. 85). CRC press.

Liu, X., Liu, X., & Hu, Y. (2015). Investigation of the thermal behaviour and decomposition kinetics of kaolinite. *Clay Minerals*, *50*(2), 199–209.

Luo, J., Jiang, T., Li, G., Peng, Z., Rao, M., & Zhang, Y. (2017). Porous materials from thermally activated kaolinite: Preparation, characterization and application. *Materials*, *10*(6). https://doi.org/10.3390/ma10060647

Makó, É., Senkár, Z., Kristóf, J., & Vágvölgyi, V. (2006). Surface modification of mechanochemically activated kaolinites by selective leaching. *Journal of Colloid and Interface Science*, *294*(2), 362–370. https://doi.org/10.1016/J.JCIS.2005.07.033

Malden, P. J., & Meads, R. E. (1967). Substitution by iron in kaolinite. *Nature*, *215*(5103), 844–846.

Meinhold, R. H., Atakul, H., Davies, T. W., & Slade, R. C. T. (1992). Flash calcination of kaolinite studied by DSC, TG and MAS NMR. *Journal of Thermal Analysis and Calorimetry*, *38*(9), 2053–2065.

Mitchell, R. E. (2002). *Mechanisms of pyrite oxidation to non-slagging species*. Stanford University (US).

Murat, M., & Driouche, M. (1988). Chemical reactivity of thermally activated clay minerals estimation by dissolution in hydrofluoric acid. *Cement and Concrete Research*, *18*(2), 221–228.

Nawab, A., Yang, X., & Honaker, R. (2022a). Parametric study and speciation analysis of rare earth precipitation using oxalic acid in a chloride solution system. *Minerals Engineering*, *176*. https://doi.org/10.1016/j.mineng.2021.107352

Nawab, A., Yang, X., & Honaker, R. (2022b). An acid baking approach to enhance heavy rare earth recovery from bituminous coal-based sources. *Minerals Engineering*, *184*, 107610. https://doi.org/10.1016/J.MINENG.2022.107610

Okada, K., Shimai ', A., Takei, T., Hayashi "*," S., Yasumori, A., & Mackenzie ', K. J. D. (n.d.). *Preparation of microporous silica from metakaolinite by selective leaching method*.

Pilkington, E. S., & Wylie, A. W. (1947). Production of rare earth and thorium compounds from monazite. Part I. *Journal of the Society of Chemical Industry*, *66*(11), 387–394.

Rodriguez-Navarro, C., Ruiz-Agudo, E., Luque, A., Rodriguez-Navarro, A. B., & Ortega-Huertas, M. (2009). Thermal decomposition of calcite: Mechanisms of formation and textural evolution of CaO nanocrystals. *American Mineralogist*, *94*(4), 578–593. https://doi.org/10.2138/AM.2009.3021

Sadri, F. (n.d.). *Control of the Adsorption of Rare Earth Elements on Gypsum (in Hydrometallurgical Processes)*.

Sadri, F., Nazari, A. M., & Ghahreman, A. (2017). A review on the cracking, baking and leaching processes of rare earth element concentrates. *Journal of Rare Earths*, *35*(8), 739–752.

Sanders, J. P., & Gallagher, P. K. (2002). Kinetic analyses using simultaneous TG/DSC measurements: Part I: decomposition of calcium carbonate in argon. *Thermochimica Acta*, *388*(1–2), 115–128.

Schwartz, D., Gadiou, R., Brilhac, J. F., Prado, G., & Martinez, G. (2000). A kinetic study of the decomposition of spent sulfuric acids at high temperature. *Industrial and Engineering Chemistry Research*, *39*(7), 2183–2189. https://doi.org/10.1021/ie990801e

Scott, M., Verba, C., Falcon, A., Poston, J., & McKoy, M. (2017). Characterization of Rare Earth Elements in in Clay Deposits Associated with Central Appalachian Coal Seams. *AGU Fall Meeting Abstracts*, *2017*, B13C-1786.

Shaw, K. G. (1953). *A process for separating thorium compounds from monazite sands*. Iowa State University.

Soltani, F., Abdollahy, M., Petersen, J., Ram, R., Becker, M., Javad Koleini, S. M., & Moradkhani, D. (2018). Leaching and recovery of phosphate and rare earth elements from an iron-rich fluorapatite concentrate: Part I: Direct baking of the concentrate. *Hydrometallurgy*, *177*, 66–78. https://doi.org/10.1016/j.hydromet.2018.02.014

Tang, W., Song, L., Zhang, S., Li, H., Sun, J., & Gu, X. (2017). Preparation of thiourea-intercalated kaolinite and its influence on thermostability and flammability of polypropylene composite. *Journal of Materials Science*, *52*(1), 208–217.

Thommes, M., Kaneko, K., Neimark, A. v, Olivier, J. P., Rodriguez-Reinoso, F., Rouquerol, J., & Sing, K. S. W. (2015a). Physisorption of gases, with special reference to the evaluation of

surface area and pore size distribution (IUPAC Technical Report). *Pure and Applied Chemistry*, *87*(9–10), 1051–1069.

Thommes, M., Kaneko, K., Neimark, A. v., Olivier, J. P., Rodriguez-Reinoso, F., Rouquerol, J., & Sing, K. S. W. (2015b). Physisorption of gases, with special reference to the evaluation of surface area and pore size distribution (IUPAC Technical Report). *Pure and Applied Chemistry*, *87*(9–10), 1051–1069. https://doi.org/10.1515/pac-2014-1117

Traoré, K., Kabré, T. S., & Blanchart, P. (2003). Gehlenite and anorthite crystallisation from kaolinite and calcite mix. *Ceramics International*, *29*(4), 377–383. https://doi.org/10.1016/S0272-8842(02)00148-7

Truex, T. J., Hammerle, R. H., & Armstrong, R. A. (1977). The thermal decomposition of aluminium sulfate. *Thermochimica Acta*, *19*(3), 301–304. https://doi.org/10.1016/0040-6031(77)80005-1

Vijayalakshmi, R., Mishra, S. L., Singh, H., & Gupta, C. K. (2001). Processing of xenotime concentrate by sulphuric acid digestion and selective thorium precipitation for separation of rare earths. In *Hydrometallurgy* (Vol. 61). www.elsevier.nlrlocaterhydromet

Yang, N., Zhang, Z.-C., Ma, N., Liu, H.-L., Zhan, X.-Q., Li, B., Gao, W., Tsai, F.-C., Jiang, T., & Chang, C.-J. (2017). Effect of surface modified kaolin on properties of polypropylene grafted maleic anhydride. *Results in Physics*, *7*, 969–974.

Yavors'kyi, V. T., & Helesh, A. B. (2016). Determination of the parameters of evaporation of the solutions of sulfuric acid with low corrosion activity of the phases. *Materials Science*, *51*(5), 691–700. https://doi.org/10.1007/s11003-016-9892-6

Zhang, P., Han, Z., Jia, J., Wei, C., Liu, Q., Wang, X., Zhou, J., Li, F., & Miao, S. (2017). Occurrence and Distribution of Gallium, Scandium, and Rare Earth Elements in Coal Gangue Collected from Junggar Basin, China. *Https://Doi.Org/10.1080/19392699.2017.1334645*, *39*(7), 389–402. https://doi.org/10.1080/19392699.2017.1334645

Zhang, W., & Honaker, R. (2019). Enhanced leachability of rare earth elements from calcined products of bituminous coals. *Minerals Engineering*, *142*, 105935.

Zhang, W., & Honaker, R. (2020). Process development for the recovery of rare earth elements and critical metals from an acid mine leachate. *Minerals Engineering*, *153*, 106382.

Zhang, W., Noble, A., Yang, X., & Honaker, R. (2020). A comprehensive review of rare earth elements recovery from coal-related materials. In *Minerals* (Vol. 10, Issue 5). MDPI AG. https://doi.org/10.3390/min10050451

## 5.1 APPENDIX: TECHNO-ECONMIC ANALYSIS

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | Heap Leach | Date: | 3/18/2023 |
| Case: | PLS3V1 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 30283 | LPM | Project IRR | 63.8% |
| *(unit conversion)* | 8000 | GPM | Project NPV: | 265,934,411 |
| | | | Payback (operating years): | 1.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Heap Leach | Plant Size: | 30283 TPH | Date: | 3/18/2023 |
| Case: | PLS3V1 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 64,239,224 | | | | | |
| Constant Dollar Annual OPEX | $ | 68,212,936 | $ | 25.38 | /ton | $ 236.6 | /kg |
| Constant Dollar Annual REE Rev | $ | 108,069,553 | $ | 40.20 | /ton | $ 374.84 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | |
|---|---|---|---|---|
| Payback Period | 4.5 | years | *1.5* | *operating years* |
| Internal Rate of Return | 64% | | | |
| Net Present Value | $ 265,934,411 | | | |
| Total Severence Tax Paid | $ - | | | |
| Total State and Federal Tax Paid | $ 178,345,505 | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Heap Leach | Plant Size: 30283 TPH | Date: 3/18/2023
Case: PLS3V1 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Heap Leach | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | tph (short tons) | | | 30283 | LPM | | | | | |
| REE Grade: | ppm or g/t or mg/kg | | | 32.9 | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | | 0.00 | 0.00 | 3.02 | 0.99 | 1.81 | | | | 1.81 |
| Yttrium | | 0.00 | 0.00 | 42.29 | 13.92 | 25.29 | | | | 25.29 |
| Lanthanum | | 0.00 | 0.00 | 0.39 | 0.13 | 0.23 | | | | 0.23 |
| Cerium | | 0.00 | 0.00 | 5.06 | 1.67 | 3.03 | | | | 3.03 |
| Praseodymium | | 0.00 | 0.00 | 1.07 | 0.35 | 0.64 | | | | 0.64 |
| Neodymium | | 0.00 | 0.00 | 7.82 | 2.57 | 4.68 | | | | 4.68 |
| Samarium | | 0.00 | 0.00 | 6.57 | 2.16 | 3.93 | | | | 3.93 |
| Europium | | 0.00 | 0.00 | 1.82 | 0.60 | 1.09 | | | | 1.09 |
| Gadolinium | | 0.00 | 0.00 | 11.62 | 3.82 | 6.95 | | | | 6.95 |
| Terbium | | 0.00 | 0.00 | 2.11 | 0.69 | 1.26 | | | | 1.26 |
| Dysprosium | | 0.00 | 0.00 | 9.51 | 3.13 | 5.69 | | | | 5.69 |
| Holmium | | 0.00 | 0.00 | 1.58 | 0.52 | 0.94 | | | | 0.94 |
| Erbium | | 0.00 | 0.00 | 3.66 | 1.21 | 2.19 | | | | 2.19 |
| Thulium | | 0.00 | 0.00 | 0.52 | 0.17 | 0.31 | | | | 0.31 |
| Ytterbium | | 0.00 | 0.00 | 2.59 | 0.85 | 1.55 | | | | 1.55 |
| Lutetium | | 0.00 | 0.00 | 0.36 | 0.12 | 0.21 | | | | 0.21 |
| Manganese | -- | | 0.00 | | 176.62 | 320.91 | -- | | | 320.91 |
| Cobalt | -- | | 0.00 | | 16.78 | 30.49 | -- | | | 30.49 |
| Nickel | -- | | 0.00 | | 59.43 | 107.98 | -- | | | 107.98 |
| Zinc | -- | | 0.00 | | 14.25 | 25.90 | -- | | | 25.90 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 0.00 | 0.00 | 0.00 | 100.00 | 32.91 | 59.80 | 0.00 | 0.00 | 0.00 | 59.8 |
| Total non-REE CM | -- | 0.00 | 0.00 | -- | 267.08 | 485.28 | -- | 0.00 | 0.00 | 485.3 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 1.81 | 0.00 | 1.79 | 0.00 | 0.00 | 0.00 | 1.79 | 0.02 | 99.1% | |
| Yttrium | 25.29 | 9.21 | 0.00 | 8.49 | 0.17 | 0.00 | 17.86 | 7.43 | 70.6% | |
| Lanthanum | 0.23 | 0.06 | 0.00 | 0.07 | 0.01 | 0.00 | 0.15 | 0.08 | 63.9% | |
| Cerium | 3.03 | 0.70 | 0.00 | 0.38 | 0.01 | 0.00 | 1.08 | 1.94 | 35.8% | |
| Praseodymium | 0.64 | 0.07 | 0.00 | 0.18 | 0.00 | 0.00 | 0.25 | 0.39 | 38.9% | |
| Neodymium | 4.68 | 1.65 | 0.00 | 0.51 | 0.01 | 0.00 | 2.18 | 2.50 | 46.6% | |
| Samarium | 3.93 | 1.35 | 0.00 | 0.79 | 0.02 | 0.00 | 2.16 | 1.77 | 55.0% | |
| Europium | 1.09 | 0.39 | 0.00 | 0.27 | 0.01 | 0.00 | 0.67 | 0.41 | 61.9% | |
| Gadolinium | 6.95 | 2.77 | 0.00 | 1.17 | 0.03 | 0.00 | 3.97 | 2.98 | 57.1% | |
| Terbium | 1.26 | 0.40 | 0.00 | 0.30 | 0.00 | 0.00 | 0.69 | 0.57 | 55.0% | |
| Dysprosium | 5.69 | 1.66 | 0.00 | 0.67 | 0.03 | 0.00 | 2.37 | 3.32 | 41.6% | |
| Holmium | 0.94 | 0.28 | 0.00 | 0.30 | 0.01 | 0.00 | 0.58 | 0.36 | 61.4% | |
| Erbium | 2.19 | 0.65 | 0.00 | 0.47 | 0.01 | 0.00 | 1.13 | 1.06 | 51.7% | |
| Thulium | 0.31 | 0.02 | 0.00 | 0.06 | 0.00 | 0.00 | 0.08 | 0.23 | 26.2% | |
| Ytterbium | 1.55 | 0.36 | 0.00 | 0.31 | 0.01 | 0.00 | 0.68 | 0.87 | 43.7% | |
| Lutetium | 0.21 | 0.05 | 0.00 | 0.06 | 0.00 | 0.00 | 0.11 | 0.11 | 50.7% | |
| Manganese | 320.91 | 0.01 | 0.00 | 113.24 | 5.61 | 0.00 | 118.86 | 202.05 | 37.0% | |
| Cobalt | 30.49 | 0.01 | 0.00 | 0.96 | 18.22 | 0.00 | 19.20 | 11.30 | 63.0% | |
| Nickel | 107.98 | 0.07 | 0.00 | 4.93 | 60.74 | 0.00 | 65.75 | 42.24 | 60.9% | |
| Zinc | 25.90 | 0.04 | 0.00 | 0.46 | 6.99 | 0.00 | 7.49 | 18.41 | 28.9% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 59.80 | 19.61 | 1.79 | 14.03 | 0.32 | 0.00 | 35.75 | 24.05 | 59.8% | <---- |
| Total non-REE CM | 485.28 | 0.13 | 0.00 | 119.59 | 91.57 | 0.00 | 211.29 | 273.99 | 43.5% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | - | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 40 | ton | $ | 1.50 |
| Dust/Volatiles | - | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | |
|---|---|
| 0.47 | TPD REE |
| 0.043 | TPD Sc |
| 5.07 | TPD CM |
| 0.33 | REE |
| 99.0% | Sc |
| 35.3% | Mn |
| 52.3% | Metal-S |
| 30283 | Feed |
| 33 | Grade |
| 4.5 | PP |
| 64% | IRR |
| 265934410.5 | NPV |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Heap Leach | Plant Size: 30283 TPH | Date: 3/18/2023
Case: PLS3V1 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 0.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 2.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | Based on Design |
| Dissolved Solids in Leach | 32.2% | % | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Leaching Retention Time | 45 | min | Based on Design |
| Thickener UF % Solids | 46% | % | SHXP-C3 Samples |
| Residue Settling Rate | 15.0 | ft/hr | Not Available |
| **Reagent Use** | | | |
| Acid Consumption | 264.1 | kg/ton | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Bioleaching Sol'n Consumption | 0 | L/ton | |
| Polymer Dosage | 0.016 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 7.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | Based on Design |
| Precip Solid Yield | 19.78 | g/L | PLS3/PLS3-MB2 |
| Precip Settling Rate | 2.24 | ft/hr | Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 10.0% | % | VT Override from 1.2% |
| **Reagent Use** | | | |
| NaOH Dose | 9.33 | g/L of Feed | PLS3/PLS3-MB2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 588.56 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 0 | tph | |
| Sorter Product Rate | 0 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 0 | ppm | |
| Grade After Sorter | 0 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 0 | tph | |
| Roaster Product Rate | 0 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 0 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 0 | tph | |
| Leaching Residue Rate | 0 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 8,000 | gpm | |
| Thickener Feed % Solids (eff) | 14% | % | |
| Liquid Density | #DIV/0! | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | #DIV/0! | g/L | |
| Leaching Vessel Size | 0.00 | gal | |
| Leaching Vessel Size | 0.00 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 0.00 | tph | 70 to 200 tph |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Liquid Balance | #DIV/0! | gpm | must be > 0; adjust liquid density |
| Slurry Balance | #DIV/0! | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,000 | gpm | |
| Slurry Feed Rate | 30,283 | lpm | |
| Precip Mass Flow | 39.62 | tph | |
| Thickener Feed % Solids (eff) | 1.98% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 0.986 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 194.68 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 186.65 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 46.54 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 3,331 | ft^2 | |
| Thickener Diameter | 65.14 | ft | ideal < 100' |

*PM5: REE Precipitation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 7.0 | -- | VT: Process Design |

**Process Parameters**

| | | | |
|---|---|---|---|
| Precip Conditioning Time | 45 | min | Based on Design |
| Precip Solid Yield | 5.28 | g/L | PLS3/PLS3-MB2 |
| Precip Settling Rate | 0.044 | ft/hr | Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.55% | % | VT Override from 0.33% |

**Reagent Use**

| | | | |
|---|---|---|---|
| NaOH Dose | 2.85 | g/L of Feed | PLS3/PLS3-MB2 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |

**Equipment Specifications**

| | | | |
|---|---|---|---|
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 3639.16 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Slurry Feed Rate | 7953 | gpm | |
| Slurry Feed Rate | 30,107 | lpm | |
| Precip Mass Flow | 10.51 | tph | |
| Thickener Feed % Solids (eff) | 0.5% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 0.982 | g/cc | |

**Other Indicators**

| | | | |
|---|---|---|---|
| Conditioning Vessel Size | 193.55 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 49.56 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 12.35 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 5466 | ft^2 | |
| Thickener Diameter | 83.44 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | - | m/m | target = 6.35 |

*PM6: RE-Redissolution*

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 10.7% | % | Based on Design |
| Dissolved Solids in Re-Leach | 13% | % | PLS3/PLS3-MB2 |
| Leaching Retention Time | 120 | min | |
| **Reagent Use** | | | |
| Acid Consumption | 209.4 | kg/ton | PLS3/PLS3-MB2 |
| | 0.0 | | PLS3/PLS3-MB2 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| Material Balance | | | |
|---|---|---|---|
| Solid Feed Rate | 10.51 | tph | |
| Slurry Flow (feed) | 363 | gpm | |
| Product Flow | 363 | gpm | |
| Liquid Density | 1.082 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 41.3 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 22.02 | v/v | target = 10.0, based on data |
| Liquid Balance | 12.35 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 115.6 | g/L | |

*PM7: REE Oxalate Precipitation*

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | Based on Design |
| Precip Solid Yield | 0.2 | g/L | PLS3/PLS3-MB2 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Thickener UF %Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| NaOH Dose | 0.00 | g/L | PLS3/PLS3-MB2 |
| Oxalic Acid Dose | 15.66 | g/L | PLS3/PLS3-MB2 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| Material Balance | | | |
|---|---|---|---|
| Slurry Feed Rate | 363.35 | gpm | |
| Slurry Feed Rate | 1375.44 | lpm | |
| Precip Mass Flow | 0.02 | tph | |
| Thickener Feed % Solids (eff) | 0.02% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.01 | tph | |
| Liquid Density | 1.081 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 6.88 | m^3 | |
| Filter Press Area | 0.09 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.02 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 0.00 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.77 | m/m | |

*PM8: Sc SX Separation*

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | UK: Design |
| Settler Residence Time | 15.00 | min | UK: Design |
| Loading: Number of Stages | 2.00 | - | UK: Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Loading O:A Advance Ratio | 0.03 | v/v | UK: Design |
| Scrubbing: Number of Stages | | - | N/A |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | N/A |
| Scrubbing O:A Advance Ratio | | v/v | N/A |
| Stripping: Number of Stages | 2.00 | - | UK: Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | UK: Design |
| Sc2O3 Grade | 2% | % | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Precip Solid Yield | 0.4 | g/L | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| Material Balance (SX) | | |
|---|---|---|
| PLS Flow | 8000.0 | gpm |
| Raffinate Rate: | 8000.0 | gpm |
| Load Org Recycle | 7780.1 | gpm |
| Load Org Feed | 8000.0 | gpm |
| Loag Org Advance | 219.9 | gpm |
| Scrub Aq Product | 0.0 | gpm |
| Scrub Aq Recycle | 0.0 | gpm |
| Scrub Aq Feed | 0.0 | gpm |
| Scurb Org Product | 219.9 | gpm |
| Strip Aq Product | 17.5 | gpm |
| Strip Aq Recycle | 202.4 | gpm |
| Strip Aq Feed | 219.9 | gpm |
| Strip Org Product | 219.9 | gpm |
| Extraction Feed (O+A): | 16000.0 | gpm |
| Scrub Feed (O+A): | 219.9 | gpm |
| Strip Feed (O+A): | 439.9 | gpm |
| **Material Balance (Precip)** | | |
| Strip Solution Flow | 17.5 | gpm |
| Solid Density | 3.4 | g/cc |
| Solid Product Rate | 0.00 | tph |
| Liquid Density (after precip) | 1.13 | g/cc |
| Thickener Feed % Solids (eff) | 0.03% | % |
| **Total Volumes** | | |
| Total Extraction Volume | 512000 | gal |
| Total Scrubbing Volume | 0 | gal |
| Total Stripping Volume | 14077 | gal |
| Extraction Org Volume | 255999.96 | gal |
| Scrubbing Org Volume | 0.00 | gal |
| Stripping Org Volume | 7038.36 | gal |
| Total Org Volume | 263038.32 | gal |

| Other Indicators | | | |
|---|---|---|---|
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 9.96 | m^3 | |
| Filter Press Area | 0.02 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM9: CM Precipitation*

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 30 | min | PLS3/PLS3-MB2 |
| Precip Solid Yield | 1.33 | g/L | PLS3/PLS3-MB2 |
| Precip Settlng Rate | 10 | ft/hr | |
| Thickener UF % Solids | 25% | % | Not Available |
| **Reagent Use** | | | |
| NaOH Dose | 0.37 | g/L of Feed | PLS3/PLS3-MB2 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 0.00 | min | N/A |
| Thickener Capacity | 2398.96 | ft^2/tph | N/A |

| Material Balance | | | |
|---|---|---|---|
| Slurry Feed Rate | 7941 | gpm | |
| Slurry Feed Rate | 30,060 | lpm | |
| Precip Mass Flow | 2.64 | tph | |
| Thickener Feed % Solids (eff) | 0.1% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 0.981 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 180.36 | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | 3.10 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 1269 | ft^2 | |
| Thickener Diameter | 40.20 | ft | ideal < 100' |

## PM10: CM Redissolution

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |

**Process Parameters**

| | | | |
|---|---|---|---|
| Re-Leach %Solids | 20.9% | % | PLS3/PLS3-MB2 |
| Dissolved Solids in Re-Leach | 13% | % | PLS3/PLS3-MB2 |
| Leaching Retention Time | 60 | min | PLS3/PLS3-MB2 |

**Reagent Use**

| | | | |
|---|---|---|---|
| Acid Consumption | 1636.051 | kg/ton | PLS3/PLS3-MB2 |

**Equipment Specification**

| | | | |
|---|---|---|---|
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 2.64 | tph | |
| Slurry Flow (feed) | 43 | gpm | |
| Product Flow | 43 | gpm | |
| Liquid Density | 1.173 | g/cc | |

**Other Indicators**

| | | | |
|---|---|---|---|
| Releach Vessel Size | 2.4 | m^3 | |
| Releach Conc. Factor | 184.06 | v/v | target = 102, based on data |
| Liquid Balance | 3.11 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 244.8 | g/L | |

## PM11: CM Recovery

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |

**Process Parameters**

| | | | |
|---|---|---|---|
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 0.36 | g/L | PLS3/PLS3-MB2 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | 0% | % | PLS3/PLS3-MB2 |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 5.64 | g/L | PLS3/PLS3-MB2 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | 1% | % | PLS3/PLS3-MB2 |

**Reagent Use**

| | | | |
|---|---|---|---|
| Na2S Dose | 11.36 | g/L of Feed | PLS3/PLS3-MB2 |
| NaOH Dose | 13.86 | g/L of Feed | PLS3/PLS3-MB2 |
| HCL Dose | 0.00 | g/L of Feed | PLS3/PLS3-MB2 |

**Equipment Specifications**

| | | | |
|---|---|---|---|
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 406.48 | gpm | |
| Co Slurry Feed Rate | 1538.68 | lpm | |
| Co Precip Mass Flow | 0.04 | tph | |
| Co Thickener Feed % Solids (eff) | 0.03% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 406.43 | gpm | |
| Mn Slurry Feed Rate | 1538.51 | lpm | |
| Mn Precip Mass Flow | 0.57 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.52% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.09 | g/cc | |
| Mn Liquid Density | 1.09 | g/cc | |

**Other Indicators**

| | | | |
|---|---|---|---|
| Co Conditioning Vessel Size | 23.1 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 0.17 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.04 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 23.1 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 2.7 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.67 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

## PM12: Biooxidation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |

**Process Parameters**

| | | | |
|---|---|---|---|
| Raw Leaching Solution Requirec | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |

**Reagent Use**

| | | | |
|---|---|---|---|
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |

**Equipment Specifications**

| | | | |
|---|---|---|---|
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |

**Other Indicators**

| | | | |
|---|---|---|---|
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |

**Cost Indicators**

| | | | |
|---|---|---|---|
| Annual OPEX | $ | - | $/yr |
| Solution Cost (OPEX) | $ | - | $/L |
| Solution Cost (OPEX) | $ | - | $/mt |
| Acid Cost (OPEX) | $ | - | $/kg acid equivalent |
| Total CAPEX (Installed) | $ | - | $ |
| Annualized CAPEX | $ | - | $/yr |
| Solution Cost (CAPEX) | $ | - | $/L |
| Solution Cost (CAPEX) | $ | - | $/mt |
| Acid Cost (CAPEX) | $ | - | $/kg acid equivalent |
| Solution Cost (Total) | $ | - | $/L |
| Solution Cost (Total) | $ | - | $/mt |
| Acid Cost (Total) | $ | - | $/kg acid equivalent |

# CAPITAL COST ESTIMATE

Feedstock: Heap Leach | Plant Size: 30283 TPH | Date: 3/18/2023
Case: PLS3V1 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM4 | Al Precipitation | $ 4,932,923 | $ - | $ 1,233,231 | $ 5,426,215 | $ 6,755,638 | $ 18,348,007 | IIIIIIIIIIIIIIIIIII |
| PM5 | REE Precipitation | $ 5,634,807 | $ - | $ 1,408,702 | $ 6,198,288 | $ 7,716,869 | $ 20,958,666 | IIIIIIIIIIIIIIIIIIIII |
| PM6 | RE-Redissolution | $ 427,774 | $ - | $ 106,943 | $ 470,551 | $ 585,836 | $ 1,591,105 | I |
| PM7 | REE Oxalate Precipitation | $ 418,023 | $ - | $ 104,506 | $ 459,825 | $ 572,482 | $ 1,554,836 | I |
| PM8 | Sc SX Separation | $ 2,128,670 | $ 4,660,086 | $ 532,167 | $ 2,341,537 | $ 2,915,213 | $ 12,577,673 | IIIIIIIIIIII |
| PM9 | CM Precipitation | $ 2,039,498 | $ - | $ 509,874 | $ 2,243,447 | $ 2,793,092 | $ 7,585,911 | IIIIIIII |
| PM10 | CM Redissolution | $ 31,107 | $ - | $ 7,777 | $ 34,218 | $ 42,601 | $ 115,703 | |
| PM11 | CM Recovery | $ 405,249 | $ - | $ 101,312 | $ 445,774 | $ 554,988 | $ 1,507,323 | I |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| | TOTAL | $ 16,018,050 | $ 4,660,086 | $ 4,004,513 | $ 17,619,855 | $ 21,936,720 | $ 64,239,224 | |

## Module Summary

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| Total Lang Factor (=1+sum % Base) | | 372% | |

Year of Construction: 2025

# OPERATING COST ESTIMATE

Feedstock: Heap Leach | Plant Size: 30283 TPH | Date: 3/18/2023
Case: PLS3V1 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 10.66 | $ 1.14 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 4.09 | $ 0.44 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 64,239,224 | $ Total | $ 637.29 | $ 1,911.88 | $ 642,392 | $ 2.23 | $ 0.24 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 1.07 | $ 0.11 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 108,069,553 | $ Total | $ 536.06 | $ 1,608.18 | $ 540,348 | $ 1.87 | $ 0.20 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 1.60 | $ 0.17 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 64,239,224 | $ Total | $ 637.29 | $ 1,911.88 | $ 642,392 | $ 2.23 | $ 0.24 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 6,792.10** | **$ 20,376.29** | **$ 6,846,434** | **$ 23.75** | **$ 2.55** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | PR | 829 | | $ 397.20 | $ 1,191.61 | $ 400,382 | $ 1.39 | $ 0.15 |
| PM5 | REE Precipitation | PR | 809 | | $ 387.69 | $ 1,163.06 | $ 390,788 | $ 1.36 | $ 0.15 |
| PM6 | RE-Redissolution | PR | 74 | | $ 35.71 | $ 107.12 | $ 35,991 | $ 0.12 | $ 0.01 |
| PM7 | REE Oxalate Precipitation | PR | 31 | | $ 14.81 | $ 44.44 | $ 14,932 | $ 0.05 | $ 0.01 |
| **Sub-Total** | | | **2900.8** | | **$ 1,390.51** | **$ 4,171.54** | **$ 1,401,637.38** | **$ 4.86** | **$ 0.52** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | CR | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | CR | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | CR | | $ 16,952.59 | $ 50,857.76 | $ 17,088,206 | $ 59.27 | $ 6.36 |
| PM5 | REE Precipitation | CR | | $ 5,148.32 | $ 15,444.95 | $ 5,189,504 | $ 18.00 | $ 1.93 |
| PM6 | RE-Redissolution | CR | | $ 4,754.54 | $ 14,263.61 | $ 4,792,571 | $ 16.62 | $ 1.78 |
| PM7 | REE Oxalate Precipitation | CR | | $ 6,203.96 | $ 18,611.87 | $ 6,253,589 | $ 21.69 | $ 2.33 |
| **Sub-Total** | | | | **$ 50,186.82** | **$ 150,560.46** | **$ 50,588,314.20** | **$ 175.47** | **$ 18.82** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | 0 |
| Non-Hazardous Solid Waste | WD | Tons | - | $ 0.10 | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Precipitate | WD | Tons | 951 | $ 1.50 | $ 475.41 | $ 1,426.22 | $ 479,211 | $ 1.66 | $ 0.18 |
| Dust/Volatiles | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 475.41** | **$ 1,426.22** | **$ 479,210.70** | **$ 1.66** | **$ 0.18** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 52,053** | **$ 156,158** | **$ 52,469,162** | **$ 182** | **$ 20** |

## Overhead Costs

| Allocation | Code | Rate | Basis | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% | % of Direct Cost | $ 20,376 Daily Direct Cost | $ 1,018.81 | $ 3,056.44 | $ 1,026,965 | $ 3.56 | $ 0.38 |
| Variable Portion | OH | 15% | % of Direct Cost | $ 156,158 Daily Direct Cost | $ 7,807.91 | $ 23,423.73 | $ 7,870,374 | $ 27.30 | $ 2.93 |
| **TOTAL OVERHEAD COSTS** | | | | | **$ 8,826.73** | **$ 26,480.18** | **$ 8,897,339.42** | **$ 30.86** | **$ 3.31** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Heap Leach | Plant Size: | 8000 TPD | Date: | 3/18/2023 |
| Case: | PLS3V1 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 6,792.10 | $ 20,376.29 | $ 6,846,434 | $ 24 | $ 2.55 | I |
| Power | $ 1,390.51 | $ 4,171.54 | $ 1,401,637 | $ 5 | $ 0.52 | |
| Consumables | $ 50,186.82 | $ 150,560.46 | $ 50,588,314 | $ 175 | $ 18.82 | IIIIIIIII |
| Leasing/Tolling | $ - | $ - | $ - | $ - | $ - | |
| Waste Disposal | $ 475.41 | $ 1,426.22 | $ 479,211 | $ 2 | $ 0.18 | |
| General Overhead | $ 8,826.73 | $ 26,480.18 | $ 8,897,339 | $ 31 | $ 3.31 | I |
| TOTAL | $ 67,672 | $ 203,015 | $ 68,212,936 | $ 237 | $ 25.38 | |

## Summary by Process Module

| | Process Module | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ - | $ - | $ - | $ - | $ - | |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ - | $ - | $ - | $ - | $ - | |
| PM4 | Al Precipitation | $ 17,350 | $ 52,049 | $ 17,488,588 | $ 60.66 | $ 6.51 | III |
| PM5 | REE Precipitation | $ 5,536 | $ 16,608 | $ 5,580,292 | $ 19.36 | $ 2.08 | I |
| PM6 | RE-Redissolution | $ 4,790 | $ 14,371 | $ 4,828,563 | $ 16.75 | $ 1.80 | |
| PM7 | REE Oxalate Precipitation | $ 6,219 | $ 18,656 | $ 6,268,521 | $ 21.74 | $ 2.33 | I |
| PM8 | Sc SX Separation | $ 3,051 | $ 9,152 | $ 3,074,905 | $ 10.67 | $ 1.14 | |
| PM9 | CM Precipitation | $ 959 | $ 2,877 | $ 966,730 | $ 3.35 | $ 0.36 | |
| PM10 | CM Redissolution | $ 9,346 | $ 28,037 | $ 9,420,360 | $ 32.67 | $ 3.50 | I |
| PM11 | CM Recovery | $ 4,327 | $ 12,982 | $ 4,361,992 | $ 15.13 | $ 1.62 | |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| NA | Unallocated | $ 16,094 | $ 48,283 | $ 16,222,984 | $ 56 | $ 6.04 | III |
| | TOTAL | $ 67,672 | $ 203,015 | $ 68,212,936 | $ 237 | $ 25.38 | |

## Summary by Cost Center

| Cost Center | Code | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| Operating Labor | OL | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 11 | $ 1.14 | |
| Technical Labor | TL | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 4 | $ 0.44 | |
| Power/Utilities | PR | $ 1,390.51 | $ 4,171.54 | $ 1,401,637 | $ 5 | $ 0.52 | |
| Lease Agreements | LS | $ - | $ - | $ - | $ - | $ - | |
| Consumables/Reagents | CR | $ 50,186.82 | $ 150,560.46 | $ 50,588,314 | $ 175 | $ 18.82 | IIIIIIIII |
| Waste Disposal | WD | $ 475.41 | $ 1,426.22 | $ 479,211 | $ 2 | $ 0.18 | |
| Plant Overhead | OH | $ 9,768.88 | $ 29,306.64 | $ 9,847,031 | $ 34 | $ 3.66 | I |
| General Sales, and Administration | GSA | $ 1,630.64 | $ 4,891.93 | $ 1,643,688 | $ 6 | $ 0.61 | |
| Reserved | | | | | | | |
| TOTAL | | $ 67,672 | $ 203,014.69 | $ 68,212,936 | $ 237 | $ 25.38 | |

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Heap Leach | Plant Size: | 30283 TPH | Date: | 3/18/2023 |
| Case: | PLS3V1 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 108,069,553

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 0 | 0 |
| Contained Value | ($/ton) | $ - | $ - |
| Feed Basket Price | ($/kg) | $ 56.3 | $ 249.0 |
| Product Basket Price | ($/kg) | $ 46.9 | $ 366.8 |
| Production Rate | (kg/day) | 815.14 | 858.06 |
| | (t/day) | 0.82 | 0.86 |
| | (t/yr) | 273.89 | 288.31 |
| | (% of US Demand) | 2.7% | |

## Revenue Contribution Table

| | | Product Revenue ($/hr) | | | | | | |
|---|---|---|---|---|---|---|---|---|
| **Element** | **REE/CM Price ($/kg) [1]** | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | **REE Revenue ($/day)** | |
| Scandium | $ 4,200.00 | $ 0.0 | $ 11,513.1 | $ - | $ - | $ - | $ 276,314 | IIII

## PRO FORMA CASH FLOW

Feedstock: Heap Leach | Plant Size: 30283 TPH | Date: 3/18/2023
Case: PLS3V1 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 118,090,518 | $ 121,633,234 | $ 125,282,231 | $ 129,040,698 | $ 132,911,919 | $ 136,899,276 | $ 141,006,255 | $ 145,236,442 | $ 149,593,536 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 118,090,518 | $ 121,633,234 | $ 125,282,231 | $ 129,040,698 | $ 132,911,919 | $ 136,899,276 | $ 141,006,255 | $ 145,236,442 | $ 149,593,536 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 192,718 | $ 1,376,030 | $ 76,428,292 | $ 78,504,591 | $ 80,638,384 | $ 82,831,108 | $ 85,084,225 | $ 87,399,224 | $ 89,777,620 | $ 92,220,949 | $ 94,730,777 |
| *- Operating Costs* | $ - | $ - | $ - | $ 74,538,116 | $ 76,774,260 | $ 79,077,488 | $ 81,449,812 | $ 83,893,307 | $ 86,410,106 | $ 89,002,409 | $ 91,672,481 | $ 94,422,656 |
| *- Loan #1 Interest* | $ - | $ 192,718 | $ 178,097 | $ 162,598 | $ 146,170 | $ 128,756 | $ 110,297 | $ 90,731 | $ 69,991 | $ 48,006 | $ 24,702 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 1,197,933 | $ 1,107,048 | $ 1,010,710 | $ 908,592 | $ 800,347 | $ 685,607 | $ 563,983 | $ 435,061 | $ 298,404 | $ 153,548 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 620,529 | $ 573,451 | $ 523,548 | $ 470,651 | $ 414,580 | $ 355,145 | $ 292,143 | $ 225,362 | $ 154,573 |
| Gross Profit | $ - | $ (192,718) | $ (1,376,030) | $ 41,662,226 | $ 43,128,642 | $ 44,643,847 | $ 46,209,590 | $ 47,827,694 | $ 49,500,052 | $ 51,228,635 | $ 53,015,493 | $ 54,862,758 |
| **Book Deductions** | $ - | $ 240,897 | $ 1,738,522 | $ 2,516,774 | $ 2,328,016 | $ 2,153,415 | $ 1,991,909 | $ 1,842,515 | $ 1,704,327 | $ 1,576,502 | $ 1,458,265 | $ 1,348,895 |
| *- CAPEX #1 Depreciation* | $ - | $ 240,897 | $ 222,830 | $ 206,118 | $ 190,659 | $ 176,359 | $ 163,132 | $ 150,897 | $ 139,580 | $ 129,112 | $ 119,428 | $ 110,471 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 1,515,693 | $ 1,402,016 | $ 1,296,865 | $ 1,199,600 | $ 1,109,630 | $ 1,026,408 | $ 949,427 | $ 878,220 | $ 812,353 | $ 751,427 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 908,641 | $ 840,492 | $ 777,456 | $ 719,146 | $ 665,210 | $ 615,320 | $ 569,171 | $ 526,483 | $ 486,997 |
| Taxable Before Depletion | $ - | $ (433,615) | $ (3,114,552) | $ 39,145,452 | $ 40,800,627 | $ 42,490,432 | $ 44,217,682 | $ 45,985,179 | $ 47,795,725 | $ 49,652,133 | $ 51,557,228 | $ 53,513,864 |
| **Depletion** | $ - | $ - | $ - | $ 16,532,673 | $ 17,028,653 | $ 17,539,512 | $ 18,065,698 | $ 18,607,669 | $ 19,165,899 | $ 19,740,876 | $ 20,333,102 | $ 20,943,095 |
| *- 50% Limit* | $ - | $ (216,807) | $ (1,557,276) | $ 19,572,726 | $ 20,400,313 | $ 21,245,216 | $ 22,108,841 | $ 22,992,589 | $ 23,897,863 | $ 24,826,066 | $ 25,778,614 | $ 26,756,932 |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 16,532,673 | $ 17,028,653 | $ 17,539,512 | $ 18,065,698 | $ 18,607,669 | $ 19,165,899 | $ 19,740,876 | $ 20,333,102 | $ 20,943,095 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (433,615) | $ (3,114,552) | $ 22,612,780 | $ 23,771,974 | $ 24,950,920 | $ 26,151,984 | $ 27,377,510 | $ 28,629,827 | $ 29,911,257 | $ 31,224,126 | $ 32,570,769 |
| *- Income Tax* | $ - | $ (112,740) | $ (809,784) | $ 5,879,323 | $ 6,180,713 | $ 6,487,239 | $ 6,799,516 | $ 7,118,153 | $ 7,443,755 | $ 7,776,927 | $ 8,118,273 | $ 8,468,400 |
| Net Profit/Loss | $ - | $ (320,875) | $ (2,304,769) | $ 16,733,457 | $ 17,591,261 | $ 18,463,681 | $ 19,352,468 | $ 20,259,357 | $ 21,186,072 | $ 22,134,330 | $ 23,105,853 | $ 24,102,369 |
| **Book Deductions** | $ - | $ 240,897 | $ 1,738,522 | $ 19,049,446 | $ 19,356,669 | $ 19,692,927 | $ 20,057,606 | $ 20,450,184 | $ 20,870,225 | $ 21,317,378 | $ 21,791,366 | $ 22,291,990 |
| *+ CAPEX #1 Depreciation* | $ - | $ 240,897 | $ 222,830 | $ 206,118 | $ 190,659 | $ 176,359 | $ 163,132 | $ 150,897 | $ 139,580 | $ 129,112 | $ 119,428 | $ 110,471 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 1,515,693 | $ 1,402,016 | $ 1,296,865 | $ 1,199,600 | $ 1,109,630 | $ 1,026,408 | $ 949,427 | $ 878,220 | $ 812,353 | $ 751,427 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 908,641 | $ 840,492 | $ 777,456 | $ 719,146 | $ 665,210 | $ 615,320 | $ 569,171 | $ 526,483 | $ 486,997 |
| *+ Depletion* | $ - | $ - | $ - | $ 16,532,673 | $ 17,028,653 | $ 17,539,512 | $ 18,065,698 | $ 18,607,669 | $ 19,165,899 | $ 19,740,876 | $ 20,333,102 | $ 20,943,095 |
| Cash Flow | $ - | $ (79,978) | $ (566,246) | $ 35,782,903 | $ 36,947,929 | $ 38,156,608 | $ 39,410,074 | $ 40,709,541 | $ 42,056,297 | $ 43,451,708 | $ 44,897,220 | $ 46,394,359 |
| **Captial Cost** | $ 3,211,961 | $ 20,209,236 | $ 12,115,207 | $ 2,664,073 | $ 2,823,917 | $ 2,993,352 | $ 3,172,953 | $ 3,363,331 | $ 3,565,131 | $ 3,779,038 | $ 4,005,781 | $ 3,809,725 |
| *- Fixed Capital Costs* | $ 6,423,922 | $ 39,931,102 | $ 20,684,311 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 3,211,961 | $ 19,965,551 | $ 10,342,155 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 243,685 | $ 258,306 | $ 273,804 | $ 290,233 | $ 307,647 | $ 326,105 | $ 345,672 | $ 366,412 | $ 388,397 | $ 411,701 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 1,514,746 | $ 1,605,630 | $ 1,701,968 | $ 1,804,086 | $ 1,912,331 | $ 2,027,071 | $ 2,148,696 | $ 2,277,617 | $ 2,414,274 | $ 2,559,131 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 784,638 | $ 831,717 | $ 881,619 | $ 934,517 | $ 990,588 | $ 1,050,023 | $ 1,113,024 | $ 1,179,806 | $ 1,250,594 |
| **Net Cash Flow** | $ (3,211,961) | $ (20,289,214) | $ (12,681,453) | $ 33,118,830 | $ 34,124,012 | $ 35,163,255 | $ 36,237,121 | $ 37,346,211 | $ 38,491,167 | $ 39,672,670 | $ 40,891,439 | $ 42,584,634 |
| Discounted Cash Flow | $ (3,211,961) | $ (18,444,740) | $ (10,480,540) | $ 24,882,667 | $ 23,307,159 | $ 21,833,615 | $ 20,454,910 | $ 19,164,511 | $ 17,956,413 | $ 16,825,085 | $ 15,765,420 | $ 14,925,654 |
| **Cumulative DCF** | $ (3,211,961) | $ (21,656,701) | $ (32,137,241) | $ (7,254,573) | $ 16,052,586 | $ 37,886,201 | $ 58,341,111 | $ 77,505,622 | $ 95,462,036 | $ 112,287,121 | $ 128,052,541 | $ 142,978,195 |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 64,239,224 | Payback Period | 4.5 |
| Constant Dollar OPEX | $ 68,212,936 | Internal Rate of Return | 64% |
| Constant Dollar REE Rev | $ 108,069,553 | Net Present Value | $ 265,934,411 |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ 178,345,505 |

## PRO FORMA CASH FLOW

Feedstock: Heap Leach | Plant Size: 30283 TPH | Date: 3/18/2023
Case: PLS3V1 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 154,081,342** | **$ 158,703,782** | **$ 163,464,895** | **$ 168,368,842** | **$ 173,419,907** | **$ 178,622,505** | **$ 183,981,180** | **$ 189,500,615** | **$ 195,185,634** | **$ 201,041,203** | **$ 207,072,439** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 154,081,342 | $ 158,703,782 | $ 163,464,895 | $ 168,368,842 | $ 173,419,907 | $ 178,622,505 | $ 183,981,180 | $ 189,500,615 | $ 195,185,634 | $ 201,041,203 | $ 207,072,439 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 97,334,873** | **$ 100,172,996** | **$ 103,178,185** | **$ 106,273,531** | **$ 109,461,737** | **$ 112,745,589** | **$ 116,127,957** | **$ 119,611,795** | **$ 123,200,149** | **$ 126,896,154** | **$ 130,703,038** |
| *- Operating Costs* | $ 97,255,335 | $ 100,172,996 | $ 103,178,185 | $ 106,273,531 | $ 109,461,737 | $ 112,745,589 | $ 116,127,957 | $ 119,611,795 | $ 123,200,149 | $ 126,896,154 | $ 130,703,038 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 79,538 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ 56,746,468 | $ 58,530,786 | $ 60,286,710 | $ 62,095,311 | $ 63,958,171 | $ 65,876,916 | $ 67,853,223 | $ 69,888,820 | $ 71,985,484 | $ 74,145,049 | $ 76,369,400 |
| **Book Deductions** | **$ 1,247,728** | **$ 1,154,148** | **$ 1,067,587** | **$ 987,518** | **$ 913,454** | **$ 844,945** | **$ 781,574** | **$ 722,956** | **$ 668,734** | **$ 567,919** | **$ 206,578** |
| *- CAPEX #1 Depreciation* | $ 102,186 | $ 94,522 | $ 87,433 | $ 80,875 | $ 74,810 | $ 69,199 | $ 64,009 | $ 59,208 | $ 54,768 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 695,070 | $ 642,940 | $ 594,719 | $ 550,115 | $ 508,857 | $ 470,692 | $ 435,390 | $ 402,736 | $ 372,531 | $ 344,591 | $ - |
| *- CAPEX #3 Depreciation* | $ 450,472 | $ 416,686 | $ 385,435 | $ 356,527 | $ 329,788 | $ 305,054 | $ 282,175 | $ 261,012 | $ 241,436 | $ 223,328 | $ 206,578 |
| Taxable Before Depletion | $ 55,498,741 | $ 57,376,638 | $ 59,219,123 | $ 61,107,793 | $ 63,044,716 | $ 65,031,971 | $ 67,071,649 | $ 69,165,864 | $ 71,316,750 | $ 73,577,130 | $ 76,162,822 |
| **Depletion** | $ 21,571,388 | $ 22,218,529 | $ 22,885,085 | $ 23,571,638 | $ 24,278,787 | $ 25,007,151 | $ 25,757,365 | $ 26,530,086 | $ 27,325,989 | $ 28,145,768 | $ 28,990,141 |
| *- 50% Limit* | $ 27,749,370 | $ 28,688,319 | $ 29,609,561 | $ 30,553,897 | $ 31,522,358 | $ 32,515,985 | $ 33,535,824 | $ 34,582,932 | $ 35,658,375 | $ 36,788,565 | $ 38,081,411 |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 21,571,388 | $ 22,218,529 | $ 22,885,085 | $ 23,571,638 | $ 24,278,787 | $ 25,007,151 | $ 25,757,365 | $ 26,530,086 | $ 27,325,989 | $ 28,145,768 | $ 28,990,141 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ 33,927,353** | **$ 35,158,109** | **$ 36,334,038** | **$ 37,536,155** | **$ 38,765,929** | **$ 40,024,820** | **$ 41,314,284** | **$ 42,635,778** | **$ 43,990,761** | **$ 45,431,361** | **$ 47,172,681** |
| *- Income Tax* | $ 8,821,112 | $ 9,141,108 | $ 9,446,850 | $ 9,759,400 | $ 10,079,142 | $ 10,406,453 | $ 10,741,714 | $ 11,085,302 | $ 11,437,598 | $ 11,812,154 | $ 12,264,897 |
| Net Profit/Loss | $ 25,106,241 | $ 26,017,001 | $ 26,887,188 | $ 27,776,755 | $ 28,686,788 | $ 29,618,367 | $ 30,572,570 | $ 31,550,475 | $ 32,553,163 | $ 33,619,207 | $ 34,907,784 |
| **Book Deductions** | **$ 22,819,115** | **$ 23,372,677** | **$ 23,952,672** | **$ 24,559,156** | **$ 25,192,241** | **$ 25,852,096** | **$ 26,538,939** | **$ 27,253,042** | **$ 27,994,723** | **$ 28,713,688** | **$ 29,196,720** |
| *+ CAPEX #1 Depreciation* | $ 102,186 | $ 94,522 | $ 87,433 | $ 80,875 | $ 74,810 | $ 69,199 | $ 64,009 | $ 59,208 | $ 54,768 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 695,070 | $ 642,940 | $ 594,719 | $ 550,115 | $ 508,857 | $ 470,692 | $ 435,390 | $ 402,736 | $ 372,531 | $ 344,591 | $ - |
| *+ CAPEX #3 Depreciation* | $ 450,472 | $ 416,686 | $ 385,435 | $ 356,527 | $ 329,788 | $ 305,054 | $ 282,175 | $ 261,012 | $ 241,436 | $ 223,328 | $ 206,578 |
| *+ Depletion* | $ 21,571,388 | $ 22,218,529 | $ 22,885,085 | $ 23,571,638 | $ 24,278,787 | $ 25,007,151 | $ 25,757,365 | $ 26,530,086 | $ 27,325,989 | $ 28,145,768 | $ 28,990,141 |
| Cash Flow | $ 47,925,357 | $ 49,389,678 | $ 50,839,860 | $ 52,335,911 | $ 53,879,029 | $ 55,470,462 | $ 57,111,509 | $ 58,803,518 | $ 60,547,886 | $ 62,332,895 | $ 64,104,503 |
| **Captial Cost** | **$ 1,325,630** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 1,325,630 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ 46,599,727 | $ 49,389,678 | $ 50,839,860 | $ 52,335,911 | $ 53,879,029 | $ 55,470,462 | $ 57,111,509 | $ 58,803,518 | $ 60,547,886 | $ 62,332,895 | $ 64,104,503 |
| Discounted Cash Flow | $ 14,848,109 | $ 14,306,430 | $ 13,387,724 | $ 12,528,801 | $ 11,725,646 | $ 10,974,535 | $ 10,272,007 | $ 9,614,845 | $ 9,000,058 | $ 8,423,080 | $ 7,874,980 |
| **Cumulative DCF** | $ 157,826,304 | $ 172,132,734 | $ 185,520,459 | $ 198,049,259 | $ 209,774,906 | $ 220,749,441 | $ 231,021,448 | $ 240,636,293 | $ 249,636,351 | $ 258,059,430 | $ 265,934,411 |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 64,239,224 | Payback Period | 0 | 4.5 |
| Constant Dollar OPEX | $ 68,212,936 | Internal Rate of Return | 0 | 64% |
| Constant Dollar REE Rev | $ 108,069,553 | Net Present Value | 0 | $ 265,934,411 |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ 178,345,505 |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS3V1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 7.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8000 | gpm |
| Feed Rate | 30283 | lpm |
| Precip Mass Flow | 39.62 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 588.56 | ft^2/tpd |
| NaOH Dose | 9.33 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 7 | 100% | 51,429 | gal | $ 120,146 | $ 841,025 | $ - | |
| 2 | Tank Mixer | 7 | 100% | 26 | HP | $ 99,998 | $ 699,989 | $ - | |
| 3 | Process Pump | 21 | 100% | 1,143 | gpm | $ 47,513 | $ 997,775 | $ - | |
| 4 | Filter Press | 1 | 100% | 187 | ft^3 | $ 394,358 | $ 394,358 | $ - | |
| 5 | Conveyor | 1 | 100% | 40 | TPH | $ 23,749 | $ 23,749 | $ - | |
| 6 | Thickener | 7 | 100% | 3,331 | ft^2 | $ 282,289 | $ 1,976,026 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 4,932,923** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 180.00 | Effiicency: | 0.85 | | $ 86.28 | $ 258.85 | $ 86,974 | $ 0.30 | $ 0.03 |
| 3 | Process Pump | 600.00 | | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 1.01 | $ 0.11 |
| 4 | Filter Press | 31.34 | | | | $ 15.02 | $ 45.07 | $ 15,142 | $ 0.05 | $ 0.01 |
| 5 | Conveyor | 2.46 | | | | $ 1.18 | $ 3.53 | $ 1,187 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 14.83 | | | | $ 7.11 | $ 21.33 | $ 7,166 | $ 0.02 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **829** | | | | **$ 397.20** | **$ 1,191.61** | **$ 400,382** | **$ 1.39** | **$ 0.15** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 16.95 | mt/hr | $ 125.000 | / mt | $ 16,952.59 | $ 50,857.76 | $ 17,088,206 | $ 59.27 | $ 6.36 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 16,952.59** | **$ 50,857.76** | **$ 17,088,206** | **$ 59.27** | **$ 6.36** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 17,350** **$ 52,049** **$ 17,488,588** **$ 60.66** **$ 6.51**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS3V1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 7.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 7953 | gpm |
| Feed Rate | 30107 | lpm |
| Precip Mass Flow | 10.51 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 3639.16 | ft^2/tpd |
| NaOH Dose | 2.85 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 7 | 100% | 51,129 | gal | $ 119,753 | $ 838,270 | $ - | |
| 2 | Tank Mixer | 7 | 100% | 26 | HP | $ 99,670 | $ 697,690 | $ - | |
| 3 | Process Pump | 21 | 100% | 1,136 | gpm | $ 47,407 | $ 995,557 | $ - | |
| 4 | Filter Press | 1 | 100% | 50 | ft^3 | $ 151,782 | $ 151,782 | $ - | |
| 5 | Conveyor | 1 | 100% | 11 | TPH | $ 11,463 | $ 11,463 | $ - | |
| 6 | Thickener | 7 | 100% | 5,466 | ft^2 | $ 420,006 | $ 2,940,045 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 5,634,807** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 178.95 | Effiicency: | 0.85 | | | $ 85.78 | $ 257.34 | $ 86,468 | $ 0.30 | $ 0.03 |
| 3 | Process Pump | 596.51 | | | | | $ 285.94 | $ 857.82 | $ 288,226 | $ 1.00 | $ 0.11 |
| 4 | Filter Press | 8.32 | | | | | $ 3.99 | $ 11.97 | $ 4,020 | $ 0.01 | $ 0.00 |
| 5 | Conveyor | 0.65 | | | | | $ 0.31 | $ 0.94 | $ 315 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 24.33 | | | | | $ 11.66 | $ 34.99 | $ 11,758 | $ 0.04 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **809** | | | | | **$ 387.69** | **$ 1,163.06** | **$ 390,788** | **$ 1.36** | **$ 0.15** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 5.15 | mt/hr | $ 125.000 | / mt | $ 5,148.32 | $ 15,444.95 | $ 5,189,504 | $ 18.00 | $ 1.93 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 5,148.32** | **$ 15,444.95** | **$ 5,189,504** | **$ 18.00** | **$ 1.93** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 5,536** **$ 16,608** **$ 5,580,292** **$ 19.36** **$ 2.08**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS3V1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 10.51 | tph |
| Slurry Feed Rate | 363 | gpm |
| Leaching Time | 120 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 209.36 | kg/ton |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 10,901 | gal | $ 14,136 | $ 56,543 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 16 | HP | $ 77,463 | $ 309,850 | $ - | |
| 3 | Process Pump | 1 | 100% | 363 | gpm | $ 61,381 | $ 61,381 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 427,774** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 65.40 | Effiicency: | 0.85 | | | $ 31.35 | $ 94.05 | $ 31,602 | $ 0.11 | $ 0.01 |
| 3 | Process Pump | 9.08 | | | | | $ 4.35 | $ 13.06 | $ 4,389 | $ 0.02 | $ 0.00 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **74** | | | | | **$ 35.71** | **$ 107.12** | **$ 35,991** | **$ 0.12** | **$ 0.01** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 2.20 | mt/hr | $ 270.000 | / mt | $ 4,754.54 | $ 14,263.61 | $ 4,792,571 | $ 16.62 | $ 1.78 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 4,754.54** | **$ 14,263.61** | **$ 4,792,571** | **$ 16.62** | **$ 1.78** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 4,790** **$ 14,371** **$ 4,828,563** **$ 16.75** **$ 1.80**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS3V1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 363 | gpm |
| Feed Rate | 1375 | lpm |
| Precip Mass Flow | 0.02 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 0.00 | g/L of Feed |
| Oxalic Acid Dose | 15.66 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.01 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 1,817 | gal | $ 2,580 | $ 10,322 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 1 | HP | $ 15,175 | $ 60,699 | $ - | |
| 3 | Process Pump | 3 | 100% | 363 | gpm | $ 61,381 | $ 184,143 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 0.09 | ft^3 | $ 1,558 | $ 1,558 | $ - | |
| 6 | Roaster | 1 | 100% | 0.03 | MBTU/hr | $ 161,301 | $ 161,301 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 418,023** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 3.63 | Effiicency: | 0.85 | | | $ 1.74 | $ 5.23 | $ 1,756 | $ 0.01 | $ 0.00 |
| 3 | Process Pump | 27.25 | | | | | $ 13.06 | $ 39.19 | $ 13,168 | $ 0.05 | $ 0.00 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.01 | | | | | $ 0.01 | $ 0.02 | $ 7 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.00 | | | | | $ 0.00 | $ 0.01 | $ 2 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **31** | | | | | **$ 14.81** | **$ 44.44** | **$ 14,932** | **$ 0.05** | **$ 0.01** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 2 | Oxalic Acid | 1,292.36 | kg/hr | $ 0.600 | / kg | $ 6,203.34 | $ 18,610.01 | $ 6,252,963 | $ 21.69 | $ 2.33 |
| 3 | Natural Gas | 0.03 | MCF/hr | $ 2.860 | / MCF | $ 0.62 | $ 1.86 | $ 626 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 6,203.96** | **$ 18,611.87** | **$ 6,253,589** | **$ 21.69** | **$ 2.33** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 6,219** **$ 18,656** **$ 6,268,521** **$ 21.74** **$ 2.33**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS3V1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8000 | gpm |
| Raffinate Rate: | 8000 | gpm |
| Extraction Feed (O+A) | 16000 | gpm |
| Scrub Feed (O+A) | 220 | gpm |
| Strip Feed (O+A) | 440 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 220 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 263038 | gal |
| Precip Mixing Intensity | 0.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 60.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,000 | gal | $ 93,458 | $ 186,917 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 40 | HP | $ 128,297 | $ 256,594 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 240,000 | gal | $ 428,599 | $ 857,199 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,000 | gpm | $ 199,601 | $ 399,203 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 220 | gal | $ 8,386 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,433 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,299 | gal | $ 38,457 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 440 | gal | $ 12,383 | $ 24,767 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 16,902 | $ 33,804 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,598 | gal | $ 56,790 | $ 113,581 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,322 | $ 19,322 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 57,253 | kg | $ 3,584,582 | $ - | $ 3,584,582 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 993,219 | kg | $ 1,075,504 | $ - | $ 1,075,504 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,632 | gal | $ 5,503 | $ 22,010 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 1.32 | HP | $ 18,702 | $ 74,809 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,322 | $ 38,644 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.02 | ft^3 | $ 449 | $ 449 | $ - | |
| 20 | Process Pump | 2 | 100% | 220 | gpm | $ 50,686 | $ 101,372 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,128,670** | **$ 4,660,086** | |

## Power/Utilities

Power Cost: $ 0.07 $/kWhr
Effiicency: 0.85

| ID | Unit Operation | Total HP | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 80.00 | $ 38.35 | $ 115.04 | $ 38,655 | $ 0.13 | $ 0.01 |
| 3 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 400.00 | $ 191.74 | $ 575.22 | $ 193,275 | $ 0.67 | $ 0.07 |
| 5 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.20 | $ 1.05 | $ 3.16 | $ 1,063 | $ 0.00 | $ 0.00 |
| 11 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.44 | $ 0.21 | $ 0.63 | $ 212 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 5.26 | $ 2.52 | $ 7.57 | $ 2,543 | $ 0.01 | $ 0.00 |
| 17 | Thickener | - | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.88 | $ 0.42 | $ 1.26 | $ 424 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.00 | $ 5.27 | $ 15.81 | $ 5,314 | $ 0.02 | $ 0.00 |
| | **TOTAL:** | **500** | **$ 239.57** | **$ 718.71** | **$ 241,487** | **$ 0.84** | **$ 0.09** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 103.57 | kg/hr | $ 0.940 | / kg | $ 778.84 | $ 2,336.51 | $ 785,068 | $ 2.72 | $ 0.29 |
| 2 | D2EHPA | 5.45 | kg/hr | $ 5.000 | / kg | $ 218.04 | $ 654.12 | $ 219,784 | $ 0.76 | $ 0.08 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.84 | mt/hr | $ 270.000 | / mt | $ 1,814.05 | $ 5,442.16 | $ 1,828,566 | $ 6.34 | $ 0.68 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,810.93** | **$ 8,432.79** | **$ 2,833,418** | **$ 9.83** | **$ 1.05** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,051** **$ 9,152** **$ 3,074,905** **$ 10.67** **$ 1.14**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS3V1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM9
Module Name: CM Precipitation
Include? Yes
Number of Parallel PMs: 5.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 7941 | gpm |
| Feed Rate | 30060 | lpm |
| Precip Mass Flow | 2.64 | tph |
| Precip Conditioning Time | 30 | min |
| Mixing Power Intensity | 0.00 | HP/1000 gal |
| Filter Press Cycle Time | 0.00 | min |
| Thickener Capacity | 2398.96 | ft^2/tpd |
| NaOH Dose | 0.37 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 5 | 100% | 47,647 | gal | $ 115,095 | $ 575,473 | $ - | |
| 2 | Tank Mixer | 5 | 100% | - | HP | $ - | $ - | $ - | |
| 3 | Process Pump | 15 | 100% | 1,588 | gpm | $ 53,867 | $ 808,005 | $ - | |
| 4 | Filter Press | 1 | 100% | - | ft^3 | $ - | $ - | $ - | |
| 5 | Conveyor | 1 | 100% | 3 | TPH | $ 5,372 | $ 5,372 | $ - | |
| 6 | Thickener | 5 | 100% | 1,269 | ft^2 | $ 130,129 | $ 650,647 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,039,498** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | - | Effiicency: | 0.85 | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Process Pump | 595.58 | | | | | $ 285.49 | $ 856.48 | $ 287,779 | $ 1.00 | $ 0.11 |
| 4 | Filter Press | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Conveyor | 0.16 | | | | | $ 0.08 | $ 0.24 | 79 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 4.03 | | | | | $ 1.93 | $ 5.80 | $ 1,950 | $ 0.01 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **600** | | | | | **$ 287.51** | **$ 862.52** | **$ 289,808** | **$ 1.01** | **$ 0.11** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 0.67 | mt/hr | $ 125.000 | / mt | $ 671.55 | $ 2,014.65 | $ 676,922 | $ 2.35 | $ 0.25 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 671.55** | **$ 2,014.65** | **$ 676,922** | **$ 2.35** | **$ 0.25** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 959** | **$ 2,877** | **$ 966,730** | **$ 3.35** | **$ 0.36** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS3V1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM10
Module Name: CM Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 2.64 | tph |
| Slurry Feed Rate | 43 | gpm |
| Leaching Time | 60 | min |
| Mixing Power Intensity | 0.0 | HP/1000 gal |
| Acid Consumption | 1636.05 | kg/ton |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 647 | gal | $ 969 | $ 3,875 | $ - | |
| 2 | Tank Mixer | 4 | 100% | - | HP | $ - | $ - | $ - | |
| 3 | Process Pump | 1 | 100% | 43 | gpm | $ 27,232 | $ 27,232 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 31,107** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | - | Effiicency: | 0.85 | $ - | $ - | $ - | $ - | $ - |
| 3 | Process Pump | 1.08 | | | $ 0.52 | $ 1.55 | $ 521 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1** | | | **$ 0.52** | **$ 1.55** | **$ 521** | **$ 0.00** | **$ 0.00** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 4.33 | mt/hr | $ 270.000 | / mt | $ 9,345.08 | $ 28,035.24 | $ 9,419,839 | $ 32.67 | $ 3.50 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 9,345.08** | **$ 28,035.24** | **$ 9,419,839** | **$ 32.67** | **$ 3.50** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 9,346** **$ 28,037** **$ 9,420,360** **$ 32.67** **$ 3.50**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS3V1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Co Feed Rate | 406.48 | gpm |
| Co Feed Rate | 1538.68 | lpm |
| Mn Feed Rate | 406.43 | gpm |
| Mn Feed Rate | 1538.51 | lpm |
| Co Precip Mass Flow | 0.04 | tph |
| Mn Precip Mass Flow | 0.57 | tph |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Co Precip Conditioning Time | 30.00 | min |
| Mn Precip Conditioning Time | 30.00 | min |
| Co Filter Press Cycle Time | 30.00 | min |
| Mn Filter Press Cycle Time | 30.00 | min |
| Co Thickener Capacity | 0.00 | ft^2/tph |
| Mn Thickener Capacity | 0.00 | ft^2/tph |
| Na2S Dose | 11.36 | g/L of Feed |
| NaOH Dose | 13.86 | g/L of Feed |

| | | |
|---|---|---|
| Co Solid Density | 3.40 | g/cc |
| Mn Solid Density | 3.40 | g/cc |
| HCL Dose | 0.00 | g/L of Feed |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 6,097 | gal | $ 8,144 | $ 16,287 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 9 | HP | $ 55,819 | $ 111,638 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 406 | gpm | $ 64,064 | $ 64,064 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 0.17 | ft^3 | $ 2,580 | $ 2,580 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 6,096 | gal | $ 8,143 | $ 16,285 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 9 | HP | $ 55,816 | $ 111,631 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 406 | gpm | $ 64,061 | $ 64,061 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 3 | ft^3 | $ 18,703 | $ 18,703 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 405,249** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 18.29 | Effiicency: | 0.85 | | | $ 8.77 | $ 26.30 | $ 8,838 | $ 0.03 | $ 0.00 |
| 3 | Process Pump | 10.16 | | | | | $ 4.87 | $ 14.61 | $ 4,910 | $ 0.02 | $ 0.00 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.03 | | | | | $ 0.01 | $ 0.04 | $ 14 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 18.29 | | | | | $ 8.77 | $ 26.30 | $ 8,837 | $ 0.03 | $ 0.00 |
| 8 | Process Pump | 10.16 | | | | | $ 4.87 | $ 14.61 | $ 4,910 | $ 0.02 | $ 0.00 |
| 9 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 0.45 | | | | | $ 0.22 | $ 0.65 | $ 219 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **57** | | | | | **$ 27.51** | **$ 82.53** | **$ 27,729** | **$ 0.10** | **$ 0.01** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 1.05 | mt/hr | $ 360.000 | / mt | $ 3,020.44 | $ 9,061.31 | $ 3,044,599 | $ 10.56 | $ 1.13 |
| 2 | NaOH | 1.28 | mt/hr | $ 125.000 | / mt | $ 1,279.43 | $ 3,838.29 | $ 1,289,664 | $ 4.47 | $ 0.48 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 4,299.86** | **$ 12,899.59** | **$ 4,334,264** | **$ 15.03** | **$ 1.61** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 4,327** **$ 12,982** **$ 4,361,992** **$ 15.13** **$ 1.62**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | Heap Leach | Date: | 3/20/2023 |
| Case: | PLS4V1 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 30283.29 | LPM | Project IRR | N/A |
| *(unit conversion)* | 8000 | GPM | Project NPV: | (422,475,072) |
| | | | Payback (operating years): | 22.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Heap Leach | Plant Size: | 30283 TPH | Date: | 3/20/2023 |
| Case: | PLS4V1 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 44,837,068 | | | | | |
| Constant Dollar Annual OPEX | $ | 72,365,630 | $ | 26.92 | /ton | $ 1,798.7 | /kg |
| Constant Dollar Annual REE Rev | $ | 16,823,684 | $ | 6.26 | /ton | $ 418.17 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | | |
|---|---|---|---|---|---|
| Payback Period | | 25.5 | years | *22.5* | *operating years* |
| Internal Rate of Return | | N/A | | | |
| Net Present Value | $ | (422,475,072) | | | |
| Total Severence Tax Paid | $ | - | | | |
| Total State and Federal Tax Paid | $ | - | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Heap Leach | Plant Size: 30283 TPH | Date: 3/20/2023
Case: PLS4V1 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | Reserved | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | tph (short tons) | | | 30283.3 | LPM | | | | | |
| REE Grade: | ppm or g/t or mg/kg | | | 12.7 | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | | | 0.00 | 2.73 | 0.35 | 0.30 | | | | 0.30 |
| Yttrium | | | 0.00 | 38.72 | 4.91 | 4.22 | | | | 4.22 |
| Lanthanum | | | 0.00 | 1.02 | 0.13 | 0.04 | | | | 0.04 |
| Cerium | | | 0.00 | 7.83 | 0.99 | 0.50 | | | | 0.50 |
| Praseodymium | | | 0.00 | 1.41 | 0.18 | 0.11 | | | | 0.11 |
| Neodymium | | | 0.00 | 9.47 | 1.20 | 0.78 | | | | 0.78 |
| Samarium | | | 0.00 | 6.30 | 0.80 | 0.66 | | | | 0.66 |
| Europium | | | 0.00 | 1.72 | 0.22 | 0.18 | | | | 0.18 |
| Gadolinium | | | 0.00 | 11.03 | 1.40 | 1.16 | | | | 1.16 |
| Terbium | | | 0.00 | 1.62 | 0.21 | 0.21 | | | | 0.21 |
| Dysprosium | | | 0.00 | 9.40 | 1.19 | 0.95 | | | | 0.95 |
| Holmium | | | 0.00 | 1.62 | 0.20 | 0.16 | | | | 0.16 |
| Erbium | | | 0.00 | 3.83 | 0.49 | 0.37 | | | | 0.37 |
| Thulium | | | 0.00 | 0.47 | 0.06 | 0.05 | | | | 0.05 |
| Ytterbium | | | 0.00 | 2.44 | 0.31 | 0.26 | | | | 0.26 |
| Lutetium | | | 0.00 | 0.39 | 0.05 | 0.04 | | | | 0.04 |
| Manganese | | | 0.00 | | 68.85 | 53.49 | -- | | | 53.49 |
| Cobalt | | | 0.00 | | 7.21 | 5.08 | -- | | | 5.08 |
| Nickel | | | 0.00 | | 20.16 | 18.00 | -- | | | 18.00 |
| Zinc | | | 0.00 | | 20.48 | 4.32 | -- | | | 4.32 |
| Reserved | | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 0.00 | 0.00 | 0.00 | 100.00 | 12.68 | 9.97 | 0.00 | 0.00 | 0.00 | 10.0 |
| Total non-REE CM | -- | 0.00 | 0.00 | -- | 116.70 | 80.88 | -- | 0.00 | 0.00 | 80.9 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 0.30 | 0.00 | 0.30 | 0.00 | 0.00 | 0.00 | 0.30 | 0.00 | 99.0% | |
| Yttrium | 4.22 | 1.25 | 0.00 | 0.66 | 0.01 | 0.00 | 1.91 | 2.30 | 45.3% | |
| Lanthanum | 0.04 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.04 | 5.6% | |
| Cerium | 0.50 | 0.21 | 0.00 | 0.05 | 0.00 | 0.00 | 0.26 | 0.25 | 50.9% | |
| Praseodymium | 0.11 | 0.04 | 0.00 | 0.01 | 0.00 | 0.00 | 0.04 | 0.06 | 40.1% | |
| Neodymium | 0.78 | 0.45 | 0.00 | 0.04 | 0.00 | 0.00 | 0.49 | 0.29 | 62.8% | |
| Samarium | 0.66 | 0.24 | 0.00 | 0.01 | 0.00 | 0.00 | 0.26 | 0.40 | 39.5% | |
| Europium | 0.18 | 0.12 | 0.00 | 0.01 | 0.00 | 0.00 | 0.12 | 0.06 | 67.8% | |
| Gadolinium | 1.16 | 0.69 | 0.00 | 0.05 | 0.00 | 0.00 | 0.75 | 0.41 | 64.4% | |
| Terbium | 0.21 | 0.11 | 0.00 | 0.01 | 0.00 | 0.00 | 0.12 | 0.09 | 58.3% | |
| Dysprosium | 0.95 | 0.36 | 0.00 | 0.07 | 0.00 | 0.00 | 0.43 | 0.52 | 44.9% | |
| Holmium | 0.16 | 0.05 | 0.00 | 0.01 | 0.00 | 0.00 | 0.07 | 0.09 | 41.4% | |
| Erbium | 0.37 | 0.10 | 0.00 | 0.04 | 0.00 | 0.00 | 0.14 | 0.22 | 38.6% | |
| Thulium | 0.05 | 0.01 | 0.00 | 0.01 | 0.00 | 0.00 | 0.01 | 0.04 | 22.8% | |
| Ytterbium | 0.26 | 0.06 | 0.00 | 0.03 | 0.00 | 0.00 | 0.09 | 0.17 | 34.8% | |
| Lutetium | 0.04 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.01 | 0.03 | 14.1% | |
| Manganese | 53.49 | 0.00 | 0.00 | 0.01 | 0.00 | 0.00 | 0.01 | 53.48 | 0.0% | |
| Cobalt | 5.08 | 0.00 | 0.00 | 0.00 | 0.01 | 0.00 | 0.01 | 5.07 | 0.2% | |
| Nickel | 18.00 | 0.00 | 0.00 | 0.01 | 0.02 | 0.00 | 0.03 | 17.97 | 0.2% | |
| Zinc | 4.32 | 0.00 | 0.00 | 0.00 | 0.02 | 0.00 | 0.02 | 4.30 | 0.4% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 9.97 | 3.68 | 0.30 | 1.00 | 0.01 | 0.00 | 4.99 | 4.98 | 50.1% | <---- |
| Total non-REE CM | 80.88 | 0.00 | 0.00 | 0.03 | 0.04 | 0.00 | 0.07 | 80.81 | 0.1% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | (30,283) | ton | $ | - |
| Non-Hazardous Solid Waste | - | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 12 | ton | $ | 1.50 |
| Dust/Volatiles | 30,283 | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | |
|---|---|
| 0.09 | TPD REE |
| 0.007 | TPD Sc |
| 0.00 | TPD CM |
| 0.37 | REE |
| 99.0% | Sc |
| 0.0% | Mn |
| 0.2% | Metal-S |
| 30283 | Feed |
| 13 | Grade |
| 25.5 | PP |
| N/A | IRR |
| -422475072 | NPV |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Heap Leach | Plant Size: 30283 TPH | Date: 3/20/2023
Case: PLS4V1 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 4.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 0% | % | UK: Design |
| Dissolved Solids in Leach | 0.0% | % | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Leaching Retention Time | 45 | min | UK: Design |
| Thickener UF % Solids | 0% | % | UK: SHXP-C4/SHXP-C4 Samples |
| Residue Settling Rate | 15.0 | ft/hr | Not Available, Assumed |
| **Reagent Use** | | | |
| Acid Consumption | 0.0 | kg/ton | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Bioleaching Sol'n Consumption | 0 | L/ton | N/A |
| Polymer Dosage | 0.016 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 7.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | Based on Design |
| Precip Solid Yield | 5.93 | g/L | PLS4/PLS4 Mass Balance-V1 |
| Precip Settling Rate | 2.24 | ft/hr | Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 0.8% | % | VT Override from 0.6% \| PLS4/PLS4 Mass Balance-V1 |
| **Reagent Use** | | | |
| NaOH Dose | 6.50 | g/L of Feed | PLS4/PLS4 Mass Balance-V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | |
| Thickener Capacity | 660.57 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 0 | tph | |
| Sorter Product Rate | 0 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 0 | ppm | |
| Grade After Sorter | 0 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 0 | tph | |
| Roaster Product Rate | 0 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 0 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 0 | tph | |
| Leaching Residue Rate | 0 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 8,000 | gpm | |
| Thickener Feed % Solids (eff) | 0% | % | |
| Liquid Density | #DIV/0! | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | #DIV/0! | g/L | |
| Leaching Vessel Size | 0.00 | gal | |
| Leaching Vessel Size | 0.00 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 0.00 | tph | 70 to 200 tph |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Liquid Balance | 8000.00 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | #DIV/0! | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,000 | gpm | |
| Slurry Feed Rate | 30,283 | lpm | |
| Precip Mass Flow | 11.88 | tph | |
| Thickener Feed % Solids (eff) | 0.58% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.016 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 194.68 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 55.99 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 13.96 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 1,121 | ft^2 | |
| Thickener Diameter | 37.80 | ft | ideal < 100' |

*PM5: REE Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 7.0 | -- | VT: Process Design |
| | | | |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45 | min | Based on Design |
| Precip Solid Yield | 2.51 | g/L | PLS4/PLS4 Mass Balance-V1 |
| Precip Settling Rate | 0.044 | ft/hr | Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.25% | % | VT Override from 0.37% \| PLS4/PLS4 Mass Balance-V1 |
| | | | |
| **Reagent Use** | | | |
| NaOH Dose | 4.57 | g/L of Feed | PLS4/PLS4 Mass Balance-V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| | | | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | |
| Filter Press Cycle Time | 60.00 | min | |
| Thickener Capacity | 3582.45 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 7986 | gpm | |
| Slurry Feed Rate | 30,230 | lpm | |
| Precip Mass Flow | 5.01 | tph | |
| Thickener Feed % Solids (eff) | 0.2% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.014 | g/cc | |
| | | | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 194.34 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 47.27 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 5.89 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 2567 | ft^2 | |
| Thickener Diameter | 57.18 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | - | m/m | target = 6.35 |

## PM6: RE-Redissolution

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 10.7% | % | PLS4/PLS4 Mass Balance-V1 |
| Dissolved Solids in Re-Leach | 50% | % | PLS4/PLS4 Mass Balance-V1 |
| Leaching Retention Time | 120 | min | Based on Design |
| **Reagent Use** | | | |
| Acid Consumption | 2861.0 | kg/ton | PLS4/PLS4 Mass Balance-V1 |
| | 0.0 | | PLS4/PLS4 Mass Balance-V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| Material Balance | | | |
|---|---|---|---|
| Solid Feed Rate | 5.01 | tph | |
| Slurry Flow (feed) | 173 | gpm | |
| Product Flow | 173 | gpm | |
| Liquid Density | 1.082 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 19.6 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 46.31 | v/v | target = 3.8 |
| Liquid Balance | 5.89 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 116.0 | g/L | |

## PM7: REE Oxalate Precipitation

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | Based on Design |
| Precip Solid Yield | 0.2 | g/L | PLS4/PLS4 Mass Balance-V1 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Thickener UF %Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| NaOH Dose | 7.67 | g/L | PLS4/PLS4 Mass Balance-V1 |
| Oxalic Acid Dose | 3.11 | g/L | PLS4/PLS4 Mass Balance-V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| Material Balance | | | |
|---|---|---|---|
| Slurry Feed Rate | 172.73 | gpm | |
| Slurry Feed Rate | 653.86 | lpm | |
| Precip Mass Flow | 0.01 | tph | |
| Thickener Feed % Solids (eff) | 0.01% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.00 | tph | |
| Liquid Density | 1.082 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 3.27 | m^3 | |
| Filter Press Area | 0.03 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.01 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 0.00 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.74 | m/m | |

## PM8: Sc SX Separation

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | UK: Design |
| Settler Residence Time | 15.00 | min | UK: Design |
| Loading: Number of Stages | 2.00 | - | UK: Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Loading O:A Advance Ratio | 0.03 | v/v | UK: Design |
| Scrubbing: Number of Stages | | - | N/A |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | N/A |
| Scrubbing O:A Advance Ratio | | v/v | N/A |
| Stripping: Number of Stages | 2.00 | - | UK: Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | UK: Design |
| Sc2O3 Grade | 2% | % | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Precip Solid Yield | 0.4 | g/L | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37 | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50 | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| Material Balance (SX) | | |
|---|---|---|
| PLS Flow | 8000.0 | gpm |
| Raffinate Rate: | 8000.0 | gpm |
| Load Org Recycle | 7780.1 | gpm |
| Load Org Feed | 8000.0 | gpm |
| Loag Org Advance | 219.9 | gpm |
| Scrub Aq Product | 0.0 | gpm |
| Scrub Aq Recycle | 0.0 | gpm |
| Scrub Aq Feed | 0.0 | gpm |
| Scurb Org Product | 219.9 | gpm |
| Strip Aq Product | 17.5 | gpm |
| Strip Aq Recycle | 202.4 | gpm |
| Strip Aq Feed | 219.9 | gpm |
| Strip Org Product | 219.9 | gpm |
| Extraction Feed (O+A): | 16000.0 | gpm |
| Scrub Feed (O+A): | 219.9 | gpm |
| Strip Feed (O+A): | 439.9 | gpm |
| **Material Balance (Precip)** | | |
| Strip Solution Flow | 17.5 | gpm |
| Solid Density | 3.4 | g/cc |
| Solid Product Rate | 0.00 | tph |
| Liquid Density (after precip) | 1.13 | g/cc |
| Thickener Feed % Solids (eff) | 0.03% | % |
| **Total Volumes** | | |
| Total Extraction Volume | 512000 | gal |
| Total Scrubbing Volume | 0 | gal |
| Total Stripping Volume | 14077 | gal |
| Extraction Org Volume | 255999.96 | gal |
| Scrubbing Org Volume | 0.00 | gal |
| Stripping Org Volume | 7038.36 | gal |
| Total Org Volume | 263038.32 | gal |

| Other Indicators | | | |
|---|---|---|---|
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 9.96 | m^3 | |
| Filter Press Area | 0.02 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

## PM9: CM Precipitation

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 0 | min | N/A |
| Precip Solid Yield | 0 | g/L | N/A |
| Precip Settlng Rate | 10 | ft/hr | N/A |
| Thickener UF % Solids | 25% | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 7.24 | g/L of Feed | N/A |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 0.00 | min | N/A |
| Thickener Capacity | 0.00 | ft^2/tph | N/A |

| Material Balance | | | |
|---|---|---|---|
| Slurry Feed Rate | 7980 | gpm | |
| Slurry Feed Rate | 30,208 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.014 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

## PM10: CM Redissolution

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 2.0% | % | N/A |
| Dissolved Solids in Re-Leach | 100% | % | N/A |
| Leaching Retention Time | 45 | min | N/A |
| **Reagent Use** | | | |
| Acid Consumption | 441 | kg/ton | N/A |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 0.00 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 0 | gpm | |
| Liquid Density | 1.000 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | |
| Releach Conc. Factor | #DIV/0! | v/v | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | - | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

## PM11: CM Recovery

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 0.60 | g/L | PLS4/PLS4 Mass Balance-V1 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | | % | Not Applicable |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 24.02 | g/L | PLS4/PLS4 Mass Balance-V1 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Na2S Dose | 0.72 | g/L of Feed | PLS4/PLS4 Mass Balance-V1 |
| NaOH Dose | 10.86 | g/L of Feed | PLS4/PLS4 Mass Balance-V1 |
| HCL Dose | 14.07 | g/L of Feed | PLS4/PLS4 Mass Balance-V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 172.72 | gpm | |
| Co Slurry Feed Rate | 653.83 | lpm | |
| Co Precip Mass Flow | 0.03 | tph | |
| Co Thickener Feed % Solids (eff) | 0.06% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 172.69 | gpm | |
| Mn Slurry Feed Rate | 653.72 | lpm | |
| Mn Precip Mass Flow | 1.04 | tph | |
| Mn Thickener Feed % Solids (eff) | 2.22% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.08 | g/cc | |
| Mn Liquid Density | 1.06 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 9.8 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 0.12 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.03 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 9.8 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 4.9 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 1.22 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

## PM12: Biooxidation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Requirec | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: Heap Leach | Plant Size: 30283 TPH | Date: 3/20/2023
Case: PLS4V1 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM4 | Al Precipitation | $ 3,541,829 | $ - | $ 885,457 | $ 3,896,012 | $ 4,850,534 | $ 13,173,832 | IIIIIIIIIIIIII |
| PM5 | REE Precipitation | $ 4,294,009 | $ - | $ 1,073,502 | $ 4,723,410 | $ 5,880,645 | $ 15,971,566 | IIIIIIIIIIIIIIII |
| PM6 | RE-Redissolution | $ 277,844 | $ - | $ 69,461 | $ 305,629 | $ 380,508 | $ 1,033,442 | I |
| PM7 | REE Oxalate Precipitation | $ 284,015 | $ - | $ 71,004 | $ 312,417 | $ 388,959 | $ 1,056,395 | I |
| PM8 | Sc SX Separation | $ 2,128,670 | $ 4,660,086 | $ 532,167 | $ 2,341,537 | $ 2,915,213 | $ 12,577,673 | IIIIIIIIIIII |
| PM9 | CM Precipitation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM10 | CM Redissolution | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM11 | CM Recovery | $ 275,349 | $ - | $ 68,837 | $ 302,883 | $ 377,090 | $ 1,024,159 | I |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| | TOTAL | $ 10,801,716 | $ 4,660,086 | $ 2,700,429 | $ 11,881,887 | $ 14,792,950 | $ 44,837,068 | |

## Module Summary

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| Total Lang Factor (=1+sum % Base) | | 372% | |

Year of Construction: 2025

# OPERATING COST ESTIMATE

Feedstock: Heap Leach | Plant Size: 30283 TPH | Date: 3/20/2023
Case: PLS4V1 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 76.38 | $ 1.14 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 29.33 | $ 0.44 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 44,837,068 | $ Total | $ 444.81 | $ 1,334.44 | $ 448,371 | $ 11.14 | $ 0.17 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 7.64 | $ 0.11 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 16,823,684 | $ Total | $ 83.45 | $ 250.35 | $ 84,118 | $ 2.09 | $ 0.03 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 11.46 | $ 0.17 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 44,837,068 | $ Total | $ 444.81 | $ 1,334.44 | $ 448,371 | $ 11.14 | $ 0.17 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 5,954.53** | **$ 17,863.58** | **$ 6,002,161** | **$ 149.19** | **$ 2.23** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | PR | 795 | | $ 381.15 | $ 1,143.45 | $ 384,198 | $ 9.55 | $ 0.14 |
| PM5 | REE Precipitation | PR | 798 | | $ 382.67 | $ 1,148.02 | $ 385,736 | $ 9.59 | $ 0.14 |
| PM6 | RE-Redissolution | PR | 35 | | $ 16.97 | $ 50.92 | $ 17,110 | $ 0.43 | $ 0.01 |
| PM7 | REE Oxalate Precipitation | PR | 15 | | $ 7.04 | $ 21.12 | $ 7,098 | $ 0.18 | $ 0.00 |
| **Sub-Total** | | | **2168.3** | | **$ 1,039.40** | **$ 3,118.21** | **$ 1,047,718.12** | **$ 26.04** | **$ 0.39** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | CR | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | CR | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | CR | | $ 11,808.33 | $ 35,424.99 | $ 11,902,796 | $ 295.86 | $ 4.43 |
| PM5 | REE Precipitation | CR | | $ 8,286.17 | $ 24,858.50 | $ 8,352,458 | $ 207.61 | $ 3.11 |
| PM6 | RE-Redissolution | CR | | $ 30,990.99 | $ 92,972.96 | $ 31,238,915 | $ 776.48 | $ 11.62 |
| PM7 | REE Oxalate Precipitation | CR | | $ 886.40 | $ 2,659.20 | $ 893,492 | $ 22.21 | $ 0.33 |
| **Sub-Total** | | | | **$ 55,290.68** | **$ 165,872.03** | **$ 55,733,002.68** | **$ 1,385.31** | **$ 20.73** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | (726,799) | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | - | $ 0.10 | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Precipitate | WD | Tons | 285 | $ 1.50 | $ 142.61 | $ 427.84 | $ 143,753 | $ 3.57 | $ 0.05 |
| Dust/Volatiles | WD | Tons | 43,607,938 | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 142.61** | **$ 427.84** | **$ 143,752.63** | **$ 3.57** | **$ 0.05** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 56,473** | **$ 169,418** | **$ 56,924,473** | **$ 1,415** | **$ 21** |

## Overhead Costs

| Allocation | Code | Rate | Basis | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% % of Direct Cost | $ 17,864 Daily Direct Cost | $ 893.18 | $ 2,679.54 | $ 900,324 | $ 22.38 | $ 0.33 |
| Variable Portion | OH | 15% % of Direct Cost | $ 169,418 Daily Direct Cost | $ 8,470.90 | $ 25,412.71 | $ 8,538,671 | $ 212.24 | $ 3.18 |
| **TOTAL OVERHEAD COSTS** | | | | **$ 9,364.08** | **$ 28,092.25** | **$ 9,438,995.22** | **$ 234.62** | **$ 3.51** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Heap Leach | Plant Size: | 8000 TPD | Date: | 3/20/2023 |
| Case: | PLS4V1 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 5,954.53 | $ 17,863.58 | $ 6,002,161 | $ 149 | $ 2.23 | I |
| Power | $ 1,039.40 | $ 3,118.21 | $ 1,047,718 | $ 26 | $ 0.39 | |
| Consumables | $ 55,290.68 | $ 165,872.03 | $ 55,733,003 | $ 1,385 | $ 20.73 | IIIIIIIII |
| Leasing/Tolling | $ - | $ - | $ - | $ - | $ - | |
| Waste Disposal | $ 142.61 | $ 427.84 | $ 143,753 | $ 4 | $ 0.05 | |
| General Overhead | $ 9,364.08 | $ 28,092.25 | $ 9,438,995 | $ 235 | $ 3.51 | I |
| TOTAL | $ 71,791 | $ 215,374 | $ 72,365,630 | $ 1,799 | $ 26.92 | |

## Summary by Process Module

| | Process Module | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ - | $ - | $ - | $ - | $ - | |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ - | $ - | $ - | $ - | $ - | |
| PM4 | Al Precipitation | $ 12,189 | $ 36,568 | $ 12,286,993 | $ 305.41 | $ 4.57 | II |
| PM5 | REE Precipitation | $ 8,669 | $ 26,007 | $ 8,738,193 | $ 217.20 | $ 3.25 | I |
| PM6 | RE-Redissolution | $ 31,008 | $ 93,024 | $ 31,256,024 | $ 776.90 | $ 11.63 | IIIII |
| PM7 | REE Oxalate Precipitation | $ 893 | $ 2,680 | $ 900,589 | $ 22.39 | $ 0.34 | |
| NA | Unallocated | $ 15,461 | $ 46,384 | $ 15,584,909 | $ 387 | $ 5.80 | II |
| | TOTAL | $ 71,791 | $ 215,374 | $ 72,365,630 | $ 1,799 | $ 26.92 | |

## Summary by Cost Center

| Cost Center | Code | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| Operating Labor | OL | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 76 | $ 1.14 | |
| Technical Labor | TL | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 29 | $ 0.44 | |
| Power/Utilities | PR | $ 1,039.40 | $ 3,118.21 | $ 1,047,718 | $ 26 | $ 0.39 | |
| Lease Agreements | LS | $ - | $ - | $ - | $ - | $ - | |
| Consumables/Reagents | CR | $ 55,290.68 | $ 165,872.03 | $ 55,733,003 | $ 1,385 | $ 20.73 | IIIIIIIII |
| Waste Disposal | WD | $ 142.61 | $ 427.84 | $ 143,753 | $ 4 | $ 0.05 | |
| Plant Overhead | OH | $ 10,113.75 | $ 30,341.26 | $ 10,194,665 | $ 253 | $ 3.79 | I |
| General Sales, and Administration | GSA | $ 985.55 | $ 2,956.66 | $ 993,437 | $ 25 | $ 0.37 | |
| Reserved | | | | | | | |
| TOTAL | | $ 71,791 | $ 215,373.90 | $ 72,365,630 | $ 1,799 | $ 26.92 | |

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Heap Leach | Plant Size: | 30283 TPH | Date: | 3/20/2023 |
| Case: | PLS4V1 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 16,823,684

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 12 | 13 |
| Contained Value | ($/ton) | $ 0.6 | $ 2.6 |
| Feed Basket Price | ($/kg) | $ 54.4 | $ 228.9 |
| Product Basket Price | ($/kg) | $ 55.9 | $ 437.2 |
| Production Rate | (kg/day) | 112.59 | 119.74 |
| | (t/day) | 0.11 | 0.12 |
| | (t/yr) | 37.83 | 40.23 |
| | (% of US Demand) | 0.4% | |

## Revenue Contribution Table

| Element | REE/CM Price ($/kg) [1] | Product Revenue ($/hr) MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | REE Revenue ($/day) |
|---|---|---|---|---|---|---|---|
| Scandium | $ 4,200.00 | $ 0.0 | $ 1,918.9 | $ - | $ - | $ - | $ 46,054 |
| Yttrium | $ 6.00 | $ 7.1 | $ - | $ - | $ - | $ - | $ 171 |
| Lanthanum | $ 2.00 | $ 0.0 | $ - | $ - | $ - | $ - | $ 0 |
| Cerium | $ 2.00 | $ 0.4 | $ - | $ - | $ - | $ - | $ 9 |
| Praseodymium | $ 52.00 | $ 1.7 | $ - | $ - | $ - | $ - | $ 42 |
| Neodymium | $ 42.00 | $ 16.5 | $ - | $ - | $ - | $ - | $ 395 |
| Samarium | $ 2.11 | $ 0.4 | $ - | $ - | $ - | $ - | $ 11 |
| Europium | $ 34.00 | $ 3.5 | $ - | $ - | $ - | $ - | $ 83 |
| Gadolinium | $ 32.00 | $ 19.2 | $ - | $ - | $ - | $ - | $ 461 |
| Terbium | $ 400.00 | $ 39.1 | $ - | $ - | $ - | $ - | $ 938 I |
| Dysprosium | $ 230.00 | $ 70.9 | $ - | $ - | $ - | $ - | $ 1,700 II |
| Holmium | $ 52.96 | $ 2.3 | $ - | $ - | $ - | $ - | $ 56 |
| Erbium | $ 34.00 | $ 3.0 | $ - | $ - | $ - | $ - | $ 71 |
| Thulium | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Ytterbium | $ 28.72 | $ 1.4 | $ - | $ - | $ - | $ - | $ 33 |
| Lutetium | $ 796.64 | $ 1.8 | $ - | $ - | $ - | $ - | $ 43 |
| Manganese | $ 0.005 | $ - | $ - | $ 0.0 | $ - | $ - | $ 0 |
| Cobalt | $ 22.00 | $ - | $ - | $ - | $ 0.1 | $ - | $ 3 |
| Nickel | $ 18.00 | $ - | $ - | $ - | $ 1,572.1 | $ - | $ 37,731 |
| Zinc | $ 3.02 | $ - | $ - | $ - | $ 97.3 | $ - | $ 2,334 |
| Reserved | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Total: | | $ 167.3 | $ 1,918.9 | $ 0.0 | $ 0.1 | $ - | $ 50,070 |

[1] REEs listed as $/kg REO; Mn listed as $/kg 44% Mn Ore; Co listed as $/kg metal content

## Coal Revenue

List Coal Revenue Sources Here:

**Annual Coal Revenue:** $ -

## Other Revenue

List Other Revenue Sources Here:

**Annual Other Revenue:** $ -

## PRO FORMA CASH FLOW

Feedstock: Heap Leach | Plant Size: 30283 TPH | Date: 3/20/2023
Case: PLS4V1 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 18,383,693 | $ 18,935,204 | $ 19,503,260 | $ 20,088,358 | $ 20,691,009 | $ 21,311,739 | $ 21,951,091 | $ 22,609,624 | $ 23,287,913 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 18,383,693 | $ 18,935,204 | $ 19,503,260 | $ 20,088,358 | $ 20,691,009 | $ 21,311,739 | $ 21,951,091 | $ 22,609,624 | $ 23,287,913 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 134,511 | $ 960,428 | $ 80,395,164 | $ 82,655,874 | $ 84,981,058 | $ 87,372,450 | $ 89,831,823 | $ 92,360,990 | $ 94,961,807 | $ 97,636,170 | $ 100,386,016 |
| *- Operating Costs* | $ - | $ - | $ - | $ 79,075,878 | $ 81,448,154 | $ 83,891,599 | $ 86,408,347 | $ 89,000,597 | $ 91,670,615 | $ 94,420,733 | $ 97,253,355 | $ 100,170,956 |
| *- Loan #1 Interest* | $ - | $ 134,511 | $ 124,306 | $ 113,489 | $ 102,022 | $ 89,868 | $ 76,984 | $ 63,327 | $ 48,851 | $ 33,507 | $ 17,241 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 836,122 | $ 772,687 | $ 705,446 | $ 634,170 | $ 558,619 | $ 478,534 | $ 393,643 | $ 303,660 | $ 208,277 | $ 107,172 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 433,111 | $ 400,252 | $ 365,421 | $ 328,500 | $ 289,364 | $ 247,880 | $ 203,907 | $ 157,296 | $ 107,888 |
| Gross Profit | $ - | $ (134,511) | $ (960,428) | $ (62,011,471) | $ (63,720,670) | $ (65,477,798) | $ (67,284,092) | $ (69,140,814) | $ (71,049,251) | $ (73,010,716) | $ (75,026,546) | $ (77,098,103) |
| **Book Deductions** | $ - | $ 168,139 | $ 1,213,437 | $ 1,756,633 | $ 1,624,886 | $ 1,503,019 | $ 1,390,293 | $ 1,286,021 | $ 1,189,569 | $ 1,100,352 | $ 1,017,825 | $ 941,488 |
| *- CAPEX #1 Depreciation* | $ - | $ 168,139 | $ 155,529 | $ 143,864 | $ 133,074 | $ 123,094 | $ 113,862 | $ 105,322 | $ 97,423 | $ 90,116 | $ 83,357 | $ 77,106 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 1,057,908 | $ 978,565 | $ 905,173 | $ 837,285 | $ 774,489 | $ 716,402 | $ 662,672 | $ 612,971 | $ 566,999 | $ 524,474 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 634,204 | $ 586,639 | $ 542,641 | $ 501,943 | $ 464,297 | $ 429,475 | $ 397,264 | $ 367,469 | $ 339,909 |
| Taxable Before Depletion | $ - | $ (302,650) | $ (2,173,865) | $ (63,768,104) | $ (65,345,556) | $ (66,980,817) | $ (68,674,384) | $ (70,426,835) | $ (72,238,820) | $ (74,111,068) | $ (76,044,371) | $ (78,039,591) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ - | $ (151,325) | $ (1,086,932) | $ (31,884,052) | $ (32,672,778) | $ (33,490,409) | $ (34,337,192) | $ (35,213,417) | $ (36,119,410) | $ (37,055,534) | $ (38,022,186) | $ (39,019,796) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 2,573,717 | $ 2,650,929 | $ 2,730,456 | $ 2,812,370 | $ 2,896,741 | $ 2,983,643 | $ 3,073,153 | $ 3,165,347 | $ 3,260,308 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (302,650) | $ (2,173,865) | $ (63,768,104) | $ (65,345,556) | $ (66,980,817) | $ (68,674,384) | $ (70,426,835) | $ (72,238,820) | $ (74,111,068) | $ (76,044,371) | $ (78,039,591) |
| *- Income Tax* | $ - | $ (78,689) | $ (565,205) | $ (16,579,707) | $ (16,989,844) | $ (17,415,012) | $ (17,855,340) | $ (18,310,977) | $ (18,782,093) | $ (19,268,878) | $ (19,771,536) | $ (20,290,294) |
| Net Profit/Loss | $ - | $ (223,961) | $ (1,608,660) | $ (47,188,397) | $ (48,355,711) | $ (49,565,805) | $ (50,819,044) | $ (52,115,858) | $ (53,456,727) | $ (54,842,190) | $ (56,272,835) | $ (57,749,297) |
| **Book Deductions** | $ - | $ 168,139 | $ 1,213,437 | $ 1,756,633 | $ 1,624,886 | $ 1,503,019 | $ 1,390,293 | $ 1,286,021 | $ 1,189,569 | $ 1,100,352 | $ 1,017,825 | $ 941,488 |
| *+ CAPEX #1 Depreciation* | $ - | $ 168,139 | $ 155,529 | $ 143,864 | $ 133,074 | $ 123,094 | $ 113,862 | $ 105,322 | $ 97,423 | $ 90,116 | $ 83,357 | $ 77,106 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 1,057,908 | $ 978,565 | $ 905,173 | $ 837,285 | $ 774,489 | $ 716,402 | $ 662,672 | $ 612,971 | $ 566,999 | $ 524,474 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 634,204 | $ 586,639 | $ 542,641 | $ 501,943 | $ 464,297 | $ 429,475 | $ 397,264 | $ 367,469 | $ 339,909 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ - | $ (55,822) | $ (395,223) | $ (45,431,764) | $ (46,730,825) | $ (48,062,785) | $ (49,428,752) | $ (50,829,837) | $ (52,267,158) | $ (53,741,838) | $ (55,255,009) | $ (56,807,809) |
| **Captial Cost** | $ 2,241,853 | $ 14,105,445 | $ 8,456,054 | $ 1,859,444 | $ 1,971,010 | $ 2,089,271 | $ 2,214,627 | $ 2,347,505 | $ 2,488,355 | $ 2,637,656 | $ 2,795,916 | $ 2,659,075 |
| *- Fixed Capital Costs* | $ 4,483,707 | $ 27,870,721 | $ 14,437,034 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 2,241,853 | $ 13,935,361 | $ 7,218,517 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 170,085 | $ 180,290 | $ 191,107 | $ 202,574 | $ 214,728 | $ 227,612 | $ 241,269 | $ 255,745 | $ 271,089 | $ 287,355 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 1,057,247 | $ 1,120,682 | $ 1,187,923 | $ 1,259,199 | $ 1,334,750 | $ 1,414,835 | $ 1,499,726 | $ 1,589,709 | $ 1,685,092 | $ 1,786,197 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 547,654 | $ 580,513 | $ 615,344 | $ 652,265 | $ 691,401 | $ 732,885 | $ 776,858 | $ 823,469 | $ 872,877 |
| **Net Cash Flow** | $ (2,241,853) | $ (14,161,268) | $ (8,851,277) | $ (47,291,207) | $ (48,701,836) | $ (50,152,056) | $ (51,643,379) | $ (53,177,341) | $ (54,755,513) | $ (56,379,495) | $ (58,050,925) | $ (59,466,884) |
| Discounted Cash Flow | $ (2,241,853) | $ (12,873,880) | $ (7,315,105) | $ (35,530,584) | $ (33,264,009) | $ (31,140,481) | $ (29,151,341) | $ (27,288,384) | $ (25,543,851) | $ (23,910,409) | $ (22,381,145) | $ (20,842,780) |
| **Cumulative DCF** | $ (2,241,853) | $ (15,115,733) | $ (22,430,838) | $ (57,961,422) | $ (91,225,431) | $ (122,365,912) | $ (151,517,253) | $ (178,805,638) | $ (204,349,488) | $ (228,259,898) | $ (250,641,042) | $ (271,483,822) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 44,837,068 | Payback Period | 25.5 |
| Constant Dollar OPEX | $ 72,365,630 | Internal Rate of Return | N/A |
| Constant Dollar REE Rev | $ 16,823,684 | Net Present Value | $ (422,475,072) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ - |

## PRO FORMA CASH FLOW

Feedstock: Heap Leach | Plant Size: 30283 TPH | Date: 3/20/2023
Case: PLS4V1 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 23,986,550** | **$ 24,706,147** | **$ 25,447,331** | **$ 26,210,751** | **$ 26,997,074** | **$ 27,806,986** | **$ 28,641,195** | **$ 29,500,431** | **$ 30,385,444** | **$ 31,297,008** | **$ 32,235,918** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 23,986,550 | $ 24,706,147 | $ 25,447,331 | $ 26,210,751 | $ 26,997,074 | $ 27,806,986 | $ 28,641,195 | $ 29,500,431 | $ 30,385,444 | $ 31,297,008 | $ 32,235,918 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 103,231,600** | **$ 106,271,367** | **$ 109,459,508** | **$ 112,743,294** | **$ 116,125,592** | **$ 119,609,360** | **$ 123,197,641** | **$ 126,893,570** | **$ 130,700,377** | **$ 134,621,389** | **$ 138,660,030** |
| *- Operating Costs* | $ 103,176,085 | $ 106,271,367 | $ 109,459,508 | $ 112,743,294 | $ 116,125,592 | $ 119,609,360 | $ 123,197,641 | $ 126,893,570 | $ 130,700,377 | $ 134,621,389 | $ 138,660,030 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 55,515 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ (79,245,050) | $ (81,565,221) | $ (84,012,177) | $ (86,532,543) | $ (89,128,519) | $ (91,802,374) | $ (94,556,446) | $ (97,393,139) | $ (100,314,933) | $ (103,324,381) | $ (106,424,113) |
| **Book Deductions** | **$ 870,877** | **$ 805,561** | **$ 745,144** | **$ 689,258** | **$ 637,564** | **$ 589,746** | **$ 545,515** | **$ 504,602** | **$ 466,757** | **$ 396,391** | **$ 144,186** |
| *- CAPEX #1 Depreciation* | $ 71,323 | $ 65,973 | $ 61,025 | $ 56,449 | $ 52,215 | $ 48,299 | $ 44,676 | $ 41,326 | $ 38,226 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 485,138 | $ 448,753 | $ 415,096 | $ 383,964 | $ 355,167 | $ 328,529 | $ 303,890 | $ 281,098 | $ 260,016 | $ 240,514 | $ - |
| *- CAPEX #3 Depreciation* | $ 314,416 | $ 290,835 | $ 269,022 | $ 248,846 | $ 230,182 | $ 212,918 | $ 196,950 | $ 182,178 | $ 168,515 | $ 155,876 | $ 144,186 |
| Taxable Before Depletion | $ (80,115,926) | $ (82,370,782) | $ (84,757,321) | $ (87,221,801) | $ (89,766,083) | $ (92,392,121) | $ (95,101,961) | $ (97,897,741) | $ (100,781,690) | $ (103,720,772) | $ (106,568,298) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ (40,057,963) | $ (41,185,391) | $ (42,378,661) | $ (43,610,900) | $ (44,883,041) | $ (46,196,060) | $ (47,550,981) | $ (48,948,870) | $ (50,390,845) | $ (51,860,386) | $ (53,284,149) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 3,358,117 | $ 3,458,861 | $ 3,562,626 | $ 3,669,505 | $ 3,779,590 | $ 3,892,978 | $ 4,009,767 | $ 4,130,060 | $ 4,253,962 | $ 4,381,581 | $ 4,513,028 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ (80,115,926)** | **$ (82,370,782)** | **$ (84,757,321)** | **$ (87,221,801)** | **$ (89,766,083)** | **$ (92,392,121)** | **$ (95,101,961)** | **$ (97,897,741)** | **$ (100,781,690)** | **$ (103,720,772)** | **$ (106,568,298)** |
| *- Income Tax* | $ (20,830,141) | $ (21,416,403) | $ (22,036,904) | $ (22,677,668) | $ (23,339,181) | $ (24,021,951) | $ (24,726,510) | $ (25,453,413) | $ (26,203,239) | $ (26,967,401) | $ (27,707,758) |
| Net Profit/Loss | $ (59,285,786) | $ (60,954,378) | $ (62,720,418) | $ (64,544,133) | $ (66,426,901) | $ (68,370,169) | $ (70,375,451) | $ (72,444,328) | $ (74,578,451) | $ (76,753,371) | $ (78,860,541) |
| **Book Deductions** | **$ 870,877** | **$ 805,561** | **$ 745,144** | **$ 689,258** | **$ 637,564** | **$ 589,746** | **$ 545,515** | **$ 504,602** | **$ 466,757** | **$ 396,391** | **$ 144,186** |
| *+ CAPEX #1 Depreciation* | $ 71,323 | $ 65,973 | $ 61,025 | $ 56,449 | $ 52,215 | $ 48,299 | $ 44,676 | $ 41,326 | $ 38,226 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 485,138 | $ 448,753 | $ 415,096 | $ 383,964 | $ 355,167 | $ 328,529 | $ 303,890 | $ 281,098 | $ 260,016 | $ 240,514 | $ - |
| *+ CAPEX #3 Depreciation* | $ 314,416 | $ 290,835 | $ 269,022 | $ 248,846 | $ 230,182 | $ 212,918 | $ 196,950 | $ 182,178 | $ 168,515 | $ 155,876 | $ 144,186 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ (58,414,909) | $ (60,148,817) | $ (61,975,274) | $ (63,854,874) | $ (65,789,337) | $ (67,780,423) | $ (69,829,936) | $ (71,939,726) | $ (74,111,694) | $ (76,356,981) | $ (78,716,355) |
| **Captial Cost** | **$ 925,250** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 925,250 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ (59,340,159) | $ (60,148,817) | $ (61,975,274) | $ (63,854,874) | $ (65,789,337) | $ (67,780,423) | $ (69,829,936) | $ (71,939,726) | $ (74,111,694) | $ (76,356,981) | $ (78,716,355) |
| Discounted Cash Flow | $ (18,907,603) | $ (17,422,970) | $ (16,320,027) | $ (15,286,349) | $ (14,317,677) | $ (13,409,995) | $ (12,559,528) | $ (11,762,720) | $ (11,016,231) | $ (10,318,162) | $ (9,669,987) |
| **Cumulative DCF** | $ (290,391,426) | $ (307,814,396) | $ (324,134,422) | $ (339,420,771) | $ (353,738,448) | $ (367,148,443) | $ (379,707,971) | $ (391,470,691) | $ (402,486,922) | $ (412,805,085) | $ (422,475,072) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 44,837,068 | Payback Period | 0 | 25.5 |
| Constant Dollar OPEX | $ 72,365,630 | Internal Rate of Return | 0 | N/A |
| Constant Dollar REE Rev | $ 16,823,684 | Net Present Value | 0 | $ (422,475,072) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ - |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS4V1
Date: 3/20/2023
By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 7.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8000 | gpm |
| Feed Rate | 30283 | lpm |
| Precip Mass Flow | 11.88 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 660.57 | ft^2/tpd |
| NaOH Dose | 6.50 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 7 | 100% | 51,429 | gal | $ 120,146 | $ 841,025 | $ - | |
| 2 | Tank Mixer | 7 | 100% | 26 | HP | $ 99,998 | $ 699,989 | $ - | |
| 3 | Process Pump | 21 | 100% | 1,143 | gpm | $ 47,513 | $ 997,775 | $ - | |
| 4 | Filter Press | 1 | 100% | 56 | ft^3 | $ 165,728 | $ 165,728 | $ - | |
| 5 | Conveyor | 1 | 100% | 12 | TPH | $ 12,261 | $ 12,261 | $ - | |
| 6 | Thickener | 7 | 100% | 1,121 | ft^2 | $ 117,864 | $ 825,051 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 3,541,829** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 180.00 | Effiicency: | 0.85 | | | $ 86.28 | $ 258.85 | $ 86,974 | $ 2.16 | $ 0.03 |
| 3 | Process Pump | 600.00 | | | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 7.21 | $ 0.11 |
| 4 | Filter Press | 9.40 | | | | | $ 4.51 | $ 13.52 | $ 4,542 | $ 0.11 | $ 0.00 |
| 5 | Conveyor | 0.74 | | | | | $ 0.35 | $ 1.06 | $ 356 | $ 0.01 | $ 0.00 |
| 6 | Thickener | 4.99 | | | | | $ 2.39 | $ 7.18 | $ 2,413 | $ 0.06 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **795** | | | | | **$ 381.15** | **$ 1,143.45** | **$ 384,198** | **$ 9.55** | **$ 0.14** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 11.81 | mt/hr | $ 125.000 | / mt | $ 11,808.33 | $ 35,424.99 | $ 11,902,796 | $ 295.86 | $ 4.43 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 11,808.33** | **$ 35,424.99** | **$ 11,902,796** | **$ 295.86** | **$ 4.43** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 12,189** | **$ 36,568** | **$ 12,286,993** | **$ 305.41** | **$ 4.57** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS4V1
Date: 3/20/2023
By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 7.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 7986 | gpm |
| Feed Rate | 30230 | lpm |
| Precip Mass Flow | 5.01 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 60.00 | min |
| Thickener Capacity | 3582.45 | ft^2/tpd |
| NaOH Dose | 4.57 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 7 | 100% | 51,339 | gal | $ 120,028 | $ 840,199 | $ - | |
| 2 | Tank Mixer | 7 | 100% | 26 | HP | $ 99,900 | $ 699,300 | $ - | |
| 3 | Process Pump | 21 | 100% | 1,141 | gpm | $ 47,481 | $ 997,110 | $ - | |
| 4 | Filter Press | 1 | 100% | 47 | ft^3 | $ 146,719 | $ 146,719 | $ - | |
| 5 | Conveyor | 1 | 100% | 5 | TPH | $ 7,634 | $ 7,634 | $ - | |
| 6 | Thickener | 7 | 100% | 2,567 | ft^2 | $ 229,007 | $ 1,603,046 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 4,294,009** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 179.69 | Effiicency: | 0.85 | | | $ 86.13 | $ 258.40 | $ 86,822 | $ 2.16 | $ 0.03 |
| 3 | Process Pump | 598.95 | | | | | $ 287.11 | $ 861.33 | $ 289,407 | $ 7.19 | $ 0.11 |
| 4 | Filter Press | 7.94 | | | | | $ 3.80 | $ 11.41 | $ 3,835 | $ 0.10 | $ 0.00 |
| 5 | Conveyor | 0.31 | | | | | $ 0.15 | $ 0.45 | $ 150 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 11.43 | | | | | $ 5.48 | $ 16.43 | $ 5,521 | $ 0.14 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **798** | | | | | **$ 382.67** | **$ 1,148.02** | **$ 385,736** | **$ 9.59** | **$ 0.14** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 8.29 | mt/hr | $ 125.000 | / mt | $ 8,286.17 | $ 24,858.50 | $ 8,352,458 | $ 207.61 | $ 3.11 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 8,286.17** | **$ 24,858.50** | **$ 8,352,458** | **$ 207.61** | **$ 3.11** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 8,669** **$ 26,007** **$ 8,738,193** **$ 217.20** **$ 3.25**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS4V1
Date: 3/20/2023
By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 5.01 | tph |
| Slurry Feed Rate | 173 | gpm |
| Leaching Time | 120 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 2861.01 | kg/ton |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 5,182 | gal | $ 6,979 | $ 27,914 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 8 | HP | $ 50,927 | $ 203,707 | $ - | |
| 3 | Process Pump | 1 | 100% | 173 | gpm | $ 46,223 | $ 46,223 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 277,844** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 31.09 | Effiicency: | 0.85 | | | $ 14.90 | $ 44.71 | $ 15,023 | $ 0.37 | $ 0.01 |
| 3 | Process Pump | 4.32 | | | | | $ 2.07 | $ 6.21 | $ 2,087 | $ 0.05 | $ 0.00 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **35** | | | | | **$ 16.97** | **$ 50.92** | **$ 17,110** | **$ 0.43** | **$ 0.01** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 14.35 | mt/hr | $ 270.000 | / mt | $ 30,990.99 | $ 92,972.96 | $ 31,238,915 | $ 776.48 | $ 11.62 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 30,990.99** | **$ 92,972.96** | **$ 31,238,915** | **$ 776.48** | **$ 11.62** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 31,008** **$ 93,024** **$ 31,256,024** **$ 776.90** **$ 11.63**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS4V1
Date: 3/20/2023
By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 173 | gpm |
| Feed Rate | 654 | lpm |
| Precip Mass Flow | 0.01 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 7.67 | g/L of Feed |
| Oxalic Acid Dose | 3.11 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.00 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 864 | gal | $ 1,274 | $ 5,096 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 0 | HP | $ 9,976 | $ 39,905 | $ - | |
| 3 | Process Pump | 3 | 100% | 173 | gpm | $ 46,223 | $ 138,669 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 0.03 | ft^3 | $ 756 | $ 756 | $ - | |
| 6 | Roaster | 1 | 100% | 0.01 | MBTU/hr | $ 99,589 | $ 99,589 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 284,015** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 1.73 | Effiicency: | 0.85 | | $ 0.83 | $ 2.48 | $ 835 | $ 0.02 | $ 0.00 |
| 3 | Process Pump | 12.95 | | | | $ 6.21 | $ 18.63 | $ 6,260 | $ 0.16 | $ 0.00 |
| 4 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.01 | | | | $ 0.00 | $ 0.01 | $ 3 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.00 | | | | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **15** | | | | **$ 7.04** | **$ 21.12** | **$ 7,098** | **$ 0.18** | **$ 0.00** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 0.30 | mt/hr | $ 125.000 | / mt | $ 300.73 | $ 902.20 | $ 303,140 | $ 7.53 | $ 0.11 |
| 2 | Oxalic Acid | 121.97 | kg/hr | $ 0.600 | / kg | $ 585.44 | $ 1,756.32 | $ 590,122 | $ 14.67 | $ 0.22 |
| 3 | Natural Gas | 0.01 | MCF/hr | $ 2.860 | / MCF | $ 0.23 | $ 0.68 | 229 | $ 0.01 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 886.40** | **$ 2,659.20** | **$ 893,492** | **$ 22.21** | **$ 0.33** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 893** **$ 2,680** **$ 900,589** **$ 22.39** **$ 0.34**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS4V1
Date: 3/20/2023
By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8000 | gpm |
| Raffinate Rate: | 8000 | gpm |
| Extraction Feed (O+A) | 16000 | gpm |
| Scrub Feed (O+A) | 220 | gpm |
| Strip Feed (O+A) | 440 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 220 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 37.00 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 50.00 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 263038 | gal |
| Precip Mixing Intensity | 0.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 60.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,000 | gal | $ 93,458 | $ 186,917 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 40 | HP | $ 128,297 | $ 256,594 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 240,000 | gal | $ 428,599 | $ 857,199 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,000 | gpm | $ 199,601 | $ 399,203 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 220 | gal | $ 8,386 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,433 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,299 | gal | $ 38,457 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 440 | gal | $ 12,383 | $ 24,767 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 16,902 | $ 33,804 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,598 | gal | $ 56,790 | $ 113,581 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,322 | $ 19,322 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 57,253 | kg | $ 3,584,582 | $ - | $ 3,584,582 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 993,219 | kg | $ 1,075,504 | $ - | $ 1,075,504 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,632 | gal | $ 5,503 | $ 22,010 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 1.32 | HP | $ 18,702 | $ 74,809 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,322 | $ 38,644 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.02 | ft^3 | $ 449 | $ 449 | $ - | |
| 20 | Process Pump | 2 | 100% | 220 | gpm | $ 50,686 | $ 101,372 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,128,670** | **$ 4,660,086** | |

## Power/Utilities

Power Cost: $ 0.07 $/kWhr
Effiicency: 0.85

| ID | Unit Operation | Total HP | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 80.00 | $ 38.35 | $ 115.04 | $ 38,655 | $ 0.96 | $ 0.01 |
| 3 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 400.00 | $ 191.74 | $ 575.22 | $ 193,275 | $ 4.80 | $ 0.07 |
| 5 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.20 | $ 1.05 | $ 3.16 | $ 1,063 | $ 0.03 | $ 0.00 |
| 11 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.44 | $ 0.21 | $ 0.63 | $ 212 | $ 0.01 | $ 0.00 |
| 13 | Cyanex 272 | - | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 5.26 | $ 2.52 | $ 7.57 | $ 2,543 | $ 0.06 | $ 0.00 |
| 17 | Thickener | - | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.88 | $ 0.42 | $ 1.26 | $ 424 | $ 0.01 | $ 0.00 |
| 19 | Filter Press | 0.00 | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.00 | $ 5.27 | $ 15.81 | $ 5,314 | $ 0.13 | $ 0.00 |
| | **TOTAL:** | **500** | **$ 239.57** | **$ 718.71** | **$ 241,487** | **$ 6.00** | **$ 0.09** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 103.57 | kg/hr | $ 0.940 | / kg | $ 778.84 | $ 2,336.51 | $ 785,068 | $ 19.51 | $ 0.29 |
| 2 | D2EHPA | 5.45 | kg/hr | $ 5.000 | / kg | $ 218.04 | $ 654.12 | $ 219,784 | $ 5.46 | $ 0.08 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.84 | mt/hr | $ 270.000 | / mt | $ 1,814.05 | $ 5,442.16 | $ 1,828,566 | $ 45.45 | $ 0.68 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,810.93** | **$ 8,432.79** | **$ 2,833,418** | **$ 70.43** | **$ 1.05** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,051** **$ 9,152** **$ 3,074,905** **$ 76.43** **$ 1.14**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Heap Leach
Case: PLS4V1
Date: 3/20/2023
By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 172.72 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 653.83 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 172.69 | gpm | HCL Dose | 14.07 | g/L of Feed |
| Mn Feed Rate | 653.72 | lpm | | | |
| Co Precip Mass Flow | 0.03 | tph | | | |
| Mn Precip Mass Flow | 1.04 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 0.72 | g/L of Feed | | | |
| NaOH Dose | 10.86 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 2,591 | gal | $ 3,614 | $ 7,228 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 4 | HP | $ 34,447 | $ 68,894 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 173 | gpm | $ 46,222 | $ 46,222 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 0.12 | ft^3 | $ 2,005 | $ 2,005 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 2,590 | gal | $ 3,614 | $ 7,227 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 4 | HP | $ 34,444 | $ 68,888 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 173 | gpm | $ 46,219 | $ 46,219 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 5 | ft^3 | $ 28,665 | $ 28,665 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 275,349** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 7.77 | Effiicency: | 0.85 | | | $ 3.73 | $ 11.18 | $ 3,756 | $ 0.09 | $ 0.00 |
| 3 | Process Pump | 4.32 | | | | | $ 2.07 | $ 6.21 | $ 2,086 | $ 0.05 | $ 0.00 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.02 | | | | | $ 0.01 | $ 0.03 | $ 10 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 7.77 | | | | | $ 3.73 | $ 11.18 | $ 3,755 | $ 0.09 | $ 0.00 |
| 8 | Process Pump | 4.32 | | | | | $ 2.07 | $ 6.21 | $ 2,086 | $ 0.05 | $ 0.00 |
| 9 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 0.82 | | | | | $ 0.39 | $ 1.18 | $ 397 | $ 0.01 | $ 0.00 |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **25** | | | | | **$ 11.99** | **$ 35.98** | **$ 12,090** | **$ 0.30** | **$ 0.00** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 0.03 | mt/hr | $ 360.000 | / mt | $ 81.76 | $ 245.27 | $ 82,411 | $ 2.05 | $ 0.03 |
| 2 | NaOH | 0.43 | mt/hr | $ 125.000 | / mt | $ 426.06 | $ 1,278.17 | $ 429,466 | $ 10.67 | $ 0.16 |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 0.05 | $ 0.14 | $ 47 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 507.86** | **$ 1,523.59** | **$ 511,925** | **$ 12.72** | **$ 0.19** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 520** **$ 1,560** **$ 524,015** **$ 13.02** **$ 0.19**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | Shakespeare Lignite | Date: | 3/18/2023 |
| Case: | SHXP-C1 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | N/A |
| *(unit conversion)* | 12000 | TPD | Project NPV: | (1,673,705,781) |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 22.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C1 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 101,426,081 | | | | | |
| Constant Dollar Annual OPEX | $ | 358,599,332 | $ | 88.94 | /ton | $ 254.3 | /kg |
| Constant Dollar Annual REE Rev | $ | 130,145,495 | $ | 32.28 | /ton | $ 92.29 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | | |
|---|---|---|---|---|---|
| Payback Period | 25.5 | years | *22.5* | *operating years* | |
| Internal Rate of Return | N/A | | | | |
| Net Present Value | $ (1,673,705,781) | | | | |
| Total Severence Tax Paid | $ - | | | | |
| Total State and Federal Tax Paid | $ - | | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C1 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 1,250 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 2.61 | 32.63 | 14.80 | | | | | | | 14.80 |
| Yttrium | 16.93 | 211.68 | 96.02 | | | | | | | 96.02 |
| Lanthanum | 15.39 | 192.37 | 87.26 | | | | | | | 87.26 |
| Cerium | 30.86 | 385.81 | 175.00 | | | | | | | 175.00 |
| Praseodymium | 4.24 | 53.03 | 24.05 | | | | | | | 24.05 |
| Neodymium | 15.21 | 190.07 | 86.22 | | | | | | | 86.22 |
| Samarium | 3.18 | 39.81 | 18.06 | | | | | | | 18.06 |
| Europium | 0.78 | 9.72 | 4.41 | | | | | | | 4.41 |
| Gadolinium | 3.52 | 43.96 | 19.94 | | | | | | | 19.94 |
| Terbium | 0.28 | 3.48 | 1.58 | | | | | | | 1.58 |
| Dysprosium | 2.85 | 35.68 | 16.19 | | | | | | | 16.19 |
| Holmium | 0.53 | 6.64 | 3.01 | | | | | | | 3.01 |
| Erbium | 1.46 | 18.19 | 8.25 | | | | | | | 8.25 |
| Thulium | 0.24 | 3.02 | 1.37 | | | | | | | 1.37 |
| Ytterbium | 1.59 | 19.85 | 9.01 | | | | | | | 9.01 |
| Lutetium | 0.32 | 4.03 | 1.83 | | | | | | | 1.83 |
| Manganese | -- | 260.10 | 117.98 | | | 0.00 | -- | | | 117.98 |
| Cobalt | -- | 134.30 | 60.92 | | | 0.00 | -- | | | 60.92 |
| Nickel | -- | 337.90 | 153.27 | | | 0.00 | -- | | | 153.27 |
| Zinc | -- | 136.80 | 62.05 | | | 0.00 | -- | | | 62.05 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 1250.00 | 566.99 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 567.0 |
| Total non-REE CM | -- | 869.10 | 394.22 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 394.2 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 14.80 | 0.00 | 2.21 | 0.00 | 0.00 | 0.00 | 2.21 | 12.59 | 14.9% | |
| Yttrium | 96.02 | 15.68 | 0.00 | 19.29 | 4.87 | 0.00 | 39.83 | 56.18 | 41.5% | |
| Lanthanum | 87.26 | 6.61 | 0.00 | 15.53 | 7.76 | 0.00 | 29.90 | 57.36 | 34.3% | |
| Cerium | 175.00 | 17.86 | 0.00 | 10.86 | 22.66 | 0.00 | 51.37 | 123.63 | 29.4% | |
| Praseodymium | 24.05 | 2.08 | 0.00 | 2.60 | 1.91 | 0.00 | 6.59 | 17.46 | 27.4% | |
| Neodymium | 86.22 | 9.24 | 0.00 | 3.03 | 7.93 | 0.00 | 20.20 | 66.02 | 23.4% | |
| Samarium | 18.06 | 2.15 | 0.00 | 1.48 | 1.53 | 0.00 | 5.16 | 12.90 | 28.6% | |
| Europium | 4.41 | 0.49 | 0.00 | 0.30 | 0.25 | 0.00 | 1.04 | 3.37 | 23.6% | |
| Gadolinium | 19.94 | 2.91 | 0.00 | 2.10 | 1.89 | 0.00 | 6.90 | 13.04 | 34.6% | |
| Terbium | 1.58 | 0.28 | 0.00 | 0.19 | 0.13 | 0.00 | 0.60 | 0.98 | 37.7% | |
| Dysprosium | 16.19 | 1.77 | 0.00 | 0.67 | 1.45 | 0.00 | 3.89 | 12.30 | 24.0% | |
| Holmium | 3.01 | 0.46 | 0.00 | 0.44 | 0.21 | 0.00 | 1.11 | 1.90 | 36.8% | |
| Erbium | 8.25 | 1.36 | 0.00 | 1.38 | 0.50 | 0.00 | 3.24 | 5.01 | 39.3% | |
| Thulium | 1.37 | 0.15 | 0.00 | 0.20 | 0.04 | 0.00 | 0.38 | 0.99 | 27.9% | |
| Ytterbium | 9.01 | 0.98 | 0.00 | 0.84 | 0.26 | 0.00 | 2.08 | 6.93 | 23.0% | |
| Lutetium | 1.83 | 0.13 | 0.00 | 0.11 | 0.13 | 0.00 | 0.37 | 1.45 | 20.5% | |
| Manganese | 117.98 | 6.32 | 0.00 | 8.99 | 2.17 | 0.00 | 17.48 | 100.50 | 14.8% | |
| Cobalt | 60.92 | 6.13 | 0.00 | 3.32 | 15.11 | 0.00 | 24.56 | 36.36 | 40.3% | |
| Nickel | 153.27 | 10.29 | 0.00 | 7.20 | 23.66 | 0.00 | 41.14 | 112.13 | 26.8% | |
| Zinc | 62.05 | 5.20 | 0.00 | 5.64 | 21.11 | 0.00 | 31.94 | 30.11 | 51.5% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 566.99 | 62.15 | 2.21 | 59.01 | 51.51 | 0.00 | 174.88 | 392.11 | 30.8% | <---- |
| Total non-REE CM | 394.22 | 27.94 | 0.00 | 25.15 | 62.04 | 0.00 | 115.12 | 279.09 | 29.2% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 384 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 9 | ton | $ | 1.50 |
| Dust/Volatiles | - | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | | |
|---|---|---|
| 1.49 | TPD REE | |
| 0.053 | TPD Sc | |
| 2.09 | TPD CM | |
| 0.11 | REE | |
| 14.9% | Sc | |
| 7.6% | Mn | |
| 21.7% | Metal-S | |
| 500 | Feed | |
| 1,250 | Grade | |
| 25.5 | PP | |
| N/A | IRR | |
| -1673705781 | NPV | |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C1 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 0.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | Based on Design |
| Dissolved Solids in Leach | 23.2% | % | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Leaching Retention Time | 45 | min | Based on Design |
| Thickener UF % Solids | 46% | % | SHXP-C1 Samples |
| Residue Settling Rate | 15.0 | ft/hr | Not Available |
| **Reagent Use** | | | |
| Acid Consumption | 194.9 | kg/ton | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Bioleaching Sol'n Consumption | 0 | L/ton | |
| Polymer Dosage | 0.016 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 9.03 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | Based on Design |
| Precip Solid Yield | 4.44 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Precip Settling Rate | 2.24 | ft/hr | Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 0.6% | % | VT Override from 0.3% |
| **Reagent Use** | | | |
| NaOH Dose | 20.39 | g/L of Feed | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 887.99 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 1250 | ppm | |
| Grade After Sorter | 1250 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 500 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 0 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 500 | tph | |
| Leaching Residue Rate | 383.85 | tph | |
| Slurry Flow (feed) | 9250 | gpm | |
| Product Flow | 8,292 | gpm | |
| Thickener Feed % Solids (eff) | 15% | % | |
| Liquid Density | 1.021 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 57.98 | g/L | |
| Leaching Vessel Size | 17343.75 | gal | |
| Leaching Vessel Size | 65.65 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 191.93 | tph | 70 to 200 tph |
| Thickener Area | 577.78 | ft^2 | |
| Thickener Diameter | 27.13 | ft | ideal < 100' |
| Liquid Balance | 289.96 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,292 | gpm | |
| Slurry Feed Rate | 31,388 | lpm | |
| Precip Mass Flow | 9.20 | tph | |
| Thickener Feed % Solids (eff) | 0.44% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.018 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 235.41 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 43.34 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 10.81 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 1,362 | ft^2 | |
| Thickener Diameter | 41.65 | ft | ideal < 100' |

*PM5: REE Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |
| | | | |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45 | min | Based on Design |
| Precip Solid Yield | 3.87 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Precip Settling Rate | 0.044 | ft/hr | Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.39% | % | VT Override from 0.16% |
| | | | |
| **Reagent Use** | | | |
| NaOH Dose | 3.66 | g/L of Feed | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| | | | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 4928.65 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 8263 | gpm | |
| Slurry Feed Rate | 31,278 | lpm | |
| Precip Mass Flow | 8.00 | tph | |
| Thickener Feed % Solids (eff) | 0.4% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.015 | g/cc | |
| | | | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 234.59 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 37.69 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 9.39 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 6568 | ft^2 | |
| Thickener Diameter | 91.47 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | 62.53 | m/m | target = 6.35 |

## *PM6: RE-Redissolution*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 10.7% | % | Based on Design |
| Dissolved Solids in Re-Leach | 32% | % | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Leaching Retention Time | | min | |
| **Reagent Use** | | | |
| Acid Consumption | 1473.4 | kg/ton | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| | 0.0 | | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| **Material Balance** | | | |
|---|---|---|---|
| Solid Feed Rate | 8.00 | tph | |
| Slurry Flow (feed) | 275 | gpm | |
| Product Flow | 275 | gpm | |
| Liquid Density | 1.082 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 30.11 | v/v | target = 3.8 |
| Liquid Balance | 9.39 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 116.0 | g/L | |

## *PM7: REE Oxalate Precipitation*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | Based on Design |
| Precip Solid Yield | 0.6 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Thickener UF %Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| NaOH Dose | 0.00 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Oxalic Acid Dose | 24.98 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| **Material Balance** | | | |
|---|---|---|---|
| Slurry Feed Rate | 371.92 | gpm | |
| Slurry Feed Rate | 1407.87 | lpm | |
| Precip Mass Flow | 0.05 | tph | |
| Thickener Feed % Solids (eff) | 0.05% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.03 | tph | |
| Liquid Density | 1.105 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 7.04 | m^3 | |
| Filter Press Area | 0.26 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.06 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 15.81 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.72 | m/m | |

## *PM8: Sc SX Separation*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | Based on Design |
| Settler Residence Time | 15.00 | min | Based on Design |
| Loading: Number of Stages | 2.00 | - | Based on Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | Based on Design |
| Loading O:A Advance Ratio | 0.03 | v/v | Based on Design |
| Scrubbing: Number of Stages | | - | Not Applicable |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | Not Applicable |
| Scrubbing O:A Advance Ratio | | v/v | Not Applicable |
| Stripping: Number of Stages | 2.00 | - | Based on Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | Based on Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | Based on Design |
| Sc2O3 Grade | 2% | % | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | Not Availabe |
| Precip Solid Yield | 0.4 | g/L | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| **Material Balance (SX)** | | |
|---|---|---|
| PLS Flow | 8291.8 | gpm |
| Raffinate Rate: | 8291.8 | gpm |
| Load Org Recycle | 8063.8 | gpm |
| Load Org Feed | 8291.8 | gpm |
| Loag Org Advance | 228.0 | gpm |
| Scrub Aq Product | 0.0 | gpm |
| Scrub Aq Recycle | 0.0 | gpm |
| Scrub Aq Feed | 0.0 | gpm |
| Scurb Org Product | 228.0 | gpm |
| Strip Aq Product | 18.2 | gpm |
| Strip Aq Recycle | 209.8 | gpm |
| Strip Aq Feed | 228.0 | gpm |
| Strip Org Product | 228.0 | gpm |
| Extraction Feed (O+A): | 16583.5 | gpm |
| Scrub Feed (O+A): | 228.0 | gpm |
| Strip Feed (O+A): | 455.9 | gpm |
| **Material Balance (Precip)** | | |
| Strip Solution Flow | 18.2 | gpm |
| Solid Density | 3.4 | g/cc |
| Solid Product Rate | 0.00 | tph |
| Liquid Density (after precip) | 1.13 | g/cc |
| Thickener Feed % Solids (eff) | 0.03% | % |
| **Total Volumes** | | |
| Total Extraction Volume | 530673 | gal |
| Total Scrubbing Volume | 0 | gal |
| Total Stripping Volume | 14590 | gal |
| Extraction Org Volume | 265336.47 | gal |
| Scrubbing Org Volume | 0.00 | gal |
| Stripping Org Volume | 7295.05 | gal |
| Total Org Volume | 272631.52 | gal |

| **Other Indicators** | | | |
|---|---|---|---|
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 10.33 | m^3 | |
| Filter Press Area | 0.01 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

## *PM9: CM Precipitation*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 30 | min | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Precip Solid Yield | 2.87 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Precip Settlng Rate | 10 | ft/hr | |
| Thickener UF % Solids | 25% | % | Not Available |
| **Reagent Use** | | | |
| NaOH Dose | 3.56 | g/L of Feed | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 30.00 | min | N/A |
| Thickener Capacity | 1107.76 | ft^2/tph | N/A |

| **Material Balance** | | | |
|---|---|---|---|
| Slurry Feed Rate | 8253 | gpm | |
| Slurry Feed Rate | 31,242 | lpm | |
| Precip Mass Flow | 5.92 | tph | |
| Thickener Feed % Solids (eff) | 0.3% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.013 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 234.32 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 27.92 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 6.96 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 1640 | ft^2 | |
| Thickener Diameter | 45.71 | ft | ideal < 100' |

## PM10: CM Redissolution

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 20.9% | % | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Dissolved Solids in Re-Leach | 14% | % | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Leaching Retention Time | 60 | min | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Reagent Use** | | | |
| Acid Consumption | 2684.53 | kg/ton | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 5.92 | tph | |
| Slurry Flow (feed) | 97 | gpm | |
| Product Flow | 97 | gpm | |
| Liquid Density | 1.173 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 5.5 | m^3 | |
| Releach Conc. Factor | 85.51 | v/v | |
| Liquid Balance | 6.96 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 245.1 | g/L | |

## PM11: CM Recovery

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 1.16 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | 0% | % | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 1.86 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | 1% | % | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Reagent Use** | | | |
| Na2S Dose | 6.88 | g/L of Feed | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| NaOH Dose | 0.00 | g/L of Feed | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| HCL Dose | 58.96 | g/L of Feed | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 371.86 | gpm | |
| Co Slurry Feed Rate | 1407.63 | lpm | |
| Co Precip Mass Flow | 0.11 | tph | |
| Co Thickener Feed % Solids (eff) | 0.10% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 371.73 | gpm | |
| Mn Slurry Feed Rate | 1407.15 | lpm | |
| Mn Precip Mass Flow | 0.17 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.17% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.10 | g/cc | |
| Mn Liquid Density | 1.10 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 21.1 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 0.51 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.13 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 21.1 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 0.8 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.20 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

## PM12: Biooxidation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Require | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C1 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 8,914,552 | $ - | $ 2,228,638 | $ 9,806,007 | $ 12,208,479 | $ 33,157,675 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 3,364,244 | $ - | $ 841,061 | $ 3,700,669 | $ 4,607,333 | $ 12,513,307 | IIIIIIIIIIII |
| PM5 | REE Precipitation | $ 5,440,016 | $ - | $ 1,360,004 | $ 5,984,017 | $ 7,450,102 | $ 20,234,138 | IIIIIIIIIIIIIIIIIIII |
| PM6 | RE-Redissolution | $ 55,224 | $ - | $ 13,806 | $ 60,746 | $ 75,629 | $ 205,405 | |
| PM7 | REE Oxalate Precipitation | $ 534,222 | $ - | $ 133,556 | $ 587,644 | $ 731,617 | $ 1,987,039 | II |
| PM8 | Sc SX Separation | $ 2,233,975 | $ 4,830,043 | $ 558,494 | $ 2,457,372 | $ 3,059,428 | $ 13,139,312 | IIIIIIIIIIIII |
| PM9 | CM Precipitation | $ 2,440,090 | $ - | $ 610,022 | $ 2,684,099 | $ 3,341,703 | $ 9,075,913 | IIIIIIIIII |
| PM10 | CM Redissolution | $ 144,580 | $ - | $ 36,145 | $ 159,038 | $ 198,002 | $ 537,764 | |
| PM11 | CM Recovery | $ 379,606 | $ - | $ 94,902 | $ 417,567 | $ 519,871 | $ 1,411,945 | I |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| | TOTAL | $ 25,970,167 | $ 4,830,043 | $ 6,492,542 | $ 28,567,184 | $ 35,566,144 | $ 101,426,081 | |

## Module Summary

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| Total Lang Factor (=1+sum % Base) | | 372% | |

Year of Construction: 2025

# OPERATING COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C1 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 2.18 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 0.84 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 101,426,081 | $ Total | $ 1,006.21 | $ 3,018.63 | $ 1,014,261 | $ 0.72 | $ 0.25 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 0.22 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 130,145,495 | $ Total | $ 645.56 | $ 1,936.69 | $ 650,727 | $ 0.46 | $ 0.16 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 0.33 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 101,426,081 | $ Total | $ 1,006.21 | $ 3,018.63 | $ 1,014,261 | $ 0.72 | $ 0.25 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 7,639.44** | **$ 22,918.31** | **$ 7,700,551** | **$ 5.46** | **$ 1.91** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 0.37 | $ 0.13 |
| PM2 | Roasting | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | PR | 1326 | | $ 635.46 | $ 1,906.37 | $ 640,540 | $ 0.45 | $ 0.16 |
| PM4 | Al Precipitation | PR | 821 | | $ 393.78 | $ 1,181.35 | $ 396,934 | $ 0.28 | $ 0.10 |
| PM5 | REE Precipitation | PR | 838 | | $ 401.46 | $ 1,204.39 | $ 404,674 | $ 0.29 | $ 0.10 |
| PM6 | RE-Redissolution | PR | 7 | | $ 3.30 | $ 9.90 | $ 3,327 | $ 0.00 | $ 0.00 |
| PM7 | REE Oxalate Precipitation | PR | 32 | | $ 15.18 | $ 45.54 | $ 15,302 | $ 0.01 | $ 0.00 |
| **Sub-Total** | | | **5432.9** | | **$ 2,604.27** | **$ 7,812.80** | **$ 2,625,100.75** | **$ 1.86** | **$ 0.65** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.07 | $ 0.03 |
| PM2 | Roasting | CR | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | CR | | $ 172,156.42 | $ 516,469.25 | $ 173,533,667 | $ 123.05 | $ 43.04 |
| PM4 | Al Precipitation | CR | | $ 38,390.82 | $ 115,172.47 | $ 38,697,949 | $ 27.44 | $ 9.60 |
| PM5 | REE Precipitation | CR | | $ 6,867.21 | $ 20,601.63 | $ 6,922,149 | $ 4.91 | $ 1.72 |
| PM6 | RE-Redissolution | CR | | $ 25,446.77 | $ 76,340.31 | $ 25,650,345 | $ 18.19 | $ 6.36 |
| PM7 | REE Oxalate Precipitation | CR | | $ 10,129.27 | $ 30,387.81 | $ 10,210,304 | $ 7.24 | $ 2.53 |
| **Sub-Total** | | | | **$ 298,689.52** | **$ 896,068.57** | **$ 301,079,037.98** | **$ 213.50** | **$ 74.67** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 9,212 | $ 0.10 | $ 307.08 | $ 921.24 | $ 309,537 | $ 0.22 | $ 0.08 |
| Non-Hazardous Precipitate | WD | Tons | 221 | $ 1.50 | $ 110.40 | $ 331.19 | $ 111,280 | $ 0.08 | $ 0.03 |
| Dust/Volatiles | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 417.48** | **$ 1,252.43** | **$ 420,816.53** | **$ 0.30** | **$ 0.10** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 301,711** | **$ 905,134** | **$ 304,124,955** | **$ 216** | **$ 75** |

## Overhead Costs

| Allocation | Code | Rate | Basis | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% | % of Direct Cost | $ 22,918 Daily Direct Cost | $ 1,145.92 | $ 3,437.75 | $ 1,155,083 | $ 0.82 | $ 0.29 |
| Variable Portion | OH | 15% | % of Direct Cost | $ 905,134 Daily Direct Cost | $ 45,256.69 | $ 135,770.07 | $ 45,618,743 | $ 32.35 | $ 11.31 |
| **TOTAL OVERHEAD COSTS** | | | | | **$ 46,402.61** | **$ 139,207.82** | **$ 46,773,825.89** | **$ 33.17** | **$ 11.60** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | SHXP-C1 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 7,639.44 | $ 22,918.31 | $ 7,700,551 | $ 5 | $ 1.91 | |
| Power | $ 2,604.27 | $ 7,812.80 | $ 2,625,101 | $ 2 | $ 0.65 | |
| Consumables | $ 298,689.52 | $ 896,068.57 | $ 301,079,038 | $ 213 | $ 74.67 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| Leasing/Tolling | $ - | $ - | $ - | $ - | $ - | |
| Waste Disposal | $ 417.48 | $ 1,252.43 | $ 420,817 | $ 0 | $ 0.10 | |
| General Overhead | $ 46,402.61 | $ 139,207.82 | $ 46,773,826 | $ 33 | $ 11.60 | IIIII |
| TOTAL | $ 355,753 | $ 1,067,260 | $ 358,599,332 | $ 254 | $ 88.94 | |

## Summary by Process Module

| | Process Module | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 611 | $ 1,834 | $ 616,370 | $ 0.44 | $ 0.15 | |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 172,792 | $ 518,376 | $ 174,174,207 | $ 123.51 | $ 43.20 | IIIIIIIIIIIIIIIIII |
| PM4 | AI Precipitation | $ 38,785 | $ 116,354 | $ 39,094,884 | $ 27.72 | $ 9.70 | IIIII |
| PM5 | REE Precipitation | $ 7,269 | $ 21,806 | $ 7,326,823 | $ 5.20 | $ 1.82 | |
| PM6 | RE-Redissolution | $ 25,450 | $ 76,350 | $ 25,653,672 | $ 18.19 | $ 6.36 | III |
| PM7 | REE Oxalate Precipitation | $ 10,144 | $ 30,433 | $ 10,225,606 | $ 7.25 | $ 2.54 | I |
| PM8 | Sc SX Separation | $ 3,167 | $ 9,501 | $ 3,192,321 | $ 2.26 | $ 0.79 | |
| PM9 | CM Precipitation | $ 7,027 | $ 21,082 | $ 7,083,715 | $ 5.02 | $ 1.76 | |
| PM10 | CM Redissolution | $ 34,349 | $ 103,047 | $ 34,623,958 | $ 24.55 | $ 8.59 | IIIII |
| PM11 | CM Recovery | $ 1,699 | $ 5,097 | $ 1,712,583 | $ 1.21 | $ 0.42 | |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| NA | Unallocated | $ 54,460 | $ 163,379 | $ 54,895,193 | $ 39 | $ 13.61 | IIIIII |
| | TOTAL | $ 355,753 | $ 1,067,260 | $ 358,599,332 | $ 254 | $ 88.94 | |

## Summary by Cost Center

| Cost Center | Code | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| Operating Labor | OL | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 2 | $ 0.76 | |
| Technical Labor | TL | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 1 | $ 0.29 | |
| Power/Utilities | PR | $ 2,604.27 | $ 7,812.80 | $ 2,625,101 | $ 2 | $ 0.65 | |
| Lease Agreements | LS | $ - | $ - | $ - | $ - | $ - | |
| Consumables/Reagents | CR | $ 298,689.52 | $ 896,068.57 | $ 301,079,038 | $ 213 | $ 74.67 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| Waste Disposal | WD | $ 417.48 | $ 1,252.43 | $ 420,817 | $ 0 | $ 0.10 | |
| Plant Overhead | OH | $ 47,713.68 | $ 143,141.03 | $ 48,095,386 | $ 34 | $ 11.93 | IIIII |
| General Sales, and Administration | GSA | $ 2,109.06 | $ 6,327.19 | $ 2,125,937 | $ 2 | $ 0.53 | |
| Reserved | | | | | | | |
| TOTAL | | $ 355,753 | $ 1,067,259.92 | $ 358,599,332 | $ 254 | $ 88.94 | |

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C1 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 130,145,495

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 1217 | 1250 |
| Contained Value | ($/ton) | $ 31.1 | $ 221.8 |
| Feed Basket Price | ($/kg) | $ 28.1 | $ 195.6 |
| Product Basket Price | ($/kg) | $ 23.9 | $ 105.0 |
| Production Rate | (kg/day) | 4144.01 | 4197.09 |
| | (t/day) | 4.14 | 4.20 |
| | (t/yr) | 1392.39 | 1410.22 |
| | (% of US Demand) | 13.9% | |

## Revenue Contribution Table

| | | Product Revenue ($/hr) | | | | | | |
|---|---|---|---|---|---|---|---|---|
| **Element** | **REE/CM Price ($/kg) [1]** | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | **REE Revenue ($/day)** | |
| Scandium | $ 4,200.00 | $ 5.8 | $ 14,222.1 | $ - | $ - | $ - | $ 341,471 | IIII

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C1 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 142,213,496 | $ 146,479,901 | $ 150,874,298 | $ 155,400,527 | $ 160,062,543 | $ 164,864,419 | $ 169,810,352 | $ 174,904,662 | $ 180,151,802 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 142,213,496 | $ 146,479,901 | $ 150,874,298 | $ 155,400,527 | $ 160,062,543 | $ 164,864,419 | $ 169,810,352 | $ 174,904,662 | $ 180,151,802 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 304,278 | $ 2,172,587 | $ 394,835,535 | $ 406,338,694 | $ 418,179,378 | $ 430,367,256 | $ 442,912,264 | $ 455,824,605 | $ 469,114,757 | $ 482,793,481 | $ 496,871,828 |
| *- Operating Costs* | $ - | $ - | $ - | $ 391,851,172 | $ 403,606,707 | $ 415,714,908 | $ 428,186,356 | $ 441,031,946 | $ 454,262,905 | $ 467,890,792 | $ 481,927,516 | $ 496,385,341 |
| *- Loan #1 Interest* | $ - | $ 304,278 | $ 281,193 | $ 256,723 | $ 230,785 | $ 203,290 | $ 174,146 | $ 143,253 | $ 110,507 | $ 75,796 | $ 39,002 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 1,891,394 | $ 1,747,897 | $ 1,595,791 | $ 1,434,559 | $ 1,263,653 | $ 1,082,492 | $ 890,462 | $ 686,910 | $ 471,145 | $ 242,434 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 979,742 | $ 905,411 | $ 826,620 | $ 743,102 | $ 654,572 | $ 560,731 | $ 461,259 | $ 355,819 | $ 244,053 |
| Gross Profit | $ - | $ (304,278) | $ (2,172,587) | $ (252,622,038) | $ (259,858,793) | $ (267,305,080) | $ (274,966,729) | $ (282,849,721) | $ (290,960,186) | $ (299,304,405) | $ (307,888,819) | $ (316,720,026) |
| **Book Deductions** | $ - | $ 380,348 | $ 2,744,920 | $ 3,973,686 | $ 3,675,660 | $ 3,399,985 | $ 3,144,986 | $ 2,909,112 | $ 2,690,929 | $ 2,489,109 | $ 2,302,426 | $ 2,129,744 |
| *- CAPEX #1 Depreciation* | $ - | $ 380,348 | $ 351,822 | $ 325,435 | $ 301,027 | $ 278,450 | $ 257,567 | $ 238,249 | $ 220,380 | $ 203,852 | $ 188,563 | $ 174,421 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 2,393,098 | $ 2,213,616 | $ 2,047,595 | $ 1,894,025 | $ 1,751,973 | $ 1,620,575 | $ 1,499,032 | $ 1,386,605 | $ 1,282,609 | $ 1,186,414 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,434,635 | $ 1,327,037 | $ 1,227,510 | $ 1,135,446 | $ 1,050,288 | $ 971,516 | $ 898,653 | $ 831,254 | $ 768,910 |
| Taxable Before Depletion | $ - | $ (684,626) | $ (4,917,507) | $ (256,595,724) | $ (263,534,453) | $ (270,705,065) | $ (278,111,716) | $ (285,758,834) | $ (293,651,114) | $ (301,793,514) | $ (310,191,245) | $ (318,849,770) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ - | $ (342,313) | $ (2,458,753) | $ (128,297,862) | $ (131,767,226) | $ (135,352,532) | $ (139,055,858) | $ (142,879,417) | $ (146,825,557) | $ (150,896,757) | $ (155,095,623) | $ (159,424,885) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 19,909,889 | $ 20,507,186 | $ 21,122,402 | $ 21,756,074 | $ 22,408,756 | $ 23,081,019 | $ 23,773,449 | $ 24,486,653 | $ 25,221,252 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (684,626) | $ (4,917,507) | $ (256,595,724) | $ (263,534,453) | $ (270,705,065) | $ (278,111,716) | $ (285,758,834) | $ (293,651,114) | $ (301,793,514) | $ (310,191,245) | $ (318,849,770) |
| *- Income Tax* | $ - | $ (178,003) | $ (1,278,552) | $ (66,714,888) | $ (68,518,958) | $ (70,383,317) | $ (72,309,046) | $ (74,297,297) | $ (76,349,290) | $ (78,466,314) | $ (80,649,724) | $ (82,900,940) |
| Net Profit/Loss | $ - | $ (506,623) | $ (3,638,955) | $ (189,880,836) | $ (195,015,495) | $ (200,321,748) | $ (205,802,670) | $ (211,461,537) | $ (217,301,825) | $ (223,327,201) | $ (229,541,521) | $ (235,948,830) |
| **Book Deductions** | $ - | $ 380,348 | $ 2,744,920 | $ 3,973,686 | $ 3,675,660 | $ 3,399,985 | $ 3,144,986 | $ 2,909,112 | $ 2,690,929 | $ 2,489,109 | $ 2,302,426 | $ 2,129,744 |
| *+ CAPEX #1 Depreciation* | $ - | $ 380,348 | $ 351,822 | $ 325,435 | $ 301,027 | $ 278,450 | $ 257,567 | $ 238,249 | $ 220,380 | $ 203,852 | $ 188,563 | $ 174,421 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 2,393,098 | $ 2,213,616 | $ 2,047,595 | $ 1,894,025 | $ 1,751,973 | $ 1,620,575 | $ 1,499,032 | $ 1,386,605 | $ 1,282,609 | $ 1,186,414 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,434,635 | $ 1,327,037 | $ 1,227,510 | $ 1,135,446 | $ 1,050,288 | $ 971,516 | $ 898,653 | $ 831,254 | $ 768,910 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ - | $ (126,275) | $ (894,035) | $ (185,907,150) | $ (191,339,836) | $ (196,921,763) | $ (202,657,683) | $ (208,552,425) | $ (214,610,896) | $ (220,838,091) | $ (227,239,096) | $ (233,819,086) |
| **Captial Cost** | $ 5,071,304 | $ 31,907,975 | $ 19,128,468 | $ 4,206,254 | $ 4,458,629 | $ 4,726,147 | $ 5,009,715 | $ 5,310,298 | $ 5,628,916 | $ 5,966,651 | $ 6,324,650 | $ 6,015,102 |
| *- Fixed Capital Costs* | $ 10,142,608 | $ 63,046,452 | $ 32,658,062 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 5,071,304 | $ 31,523,226 | $ 16,329,031 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 384,749 | $ 407,834 | $ 432,305 | $ 458,243 | $ 485,737 | $ 514,882 | $ 545,774 | $ 578,521 | $ 613,232 | $ 650,026 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 2,391,603 | $ 2,535,099 | $ 2,687,205 | $ 2,848,437 | $ 3,019,343 | $ 3,200,504 | $ 3,392,534 | $ 3,596,086 | $ 3,811,851 | $ 4,040,563 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 1,238,850 | $ 1,313,181 | $ 1,391,972 | $ 1,475,490 | $ 1,564,020 | $ 1,657,861 | $ 1,757,333 | $ 1,862,773 | $ 1,974,539 |
| **Net Cash Flow** | $ (5,071,304) | $ (32,034,251) | $ (20,022,503) | $ (190,113,404) | $ (195,798,465) | $ (201,647,910) | $ (207,667,399) | $ (213,862,723) | $ (220,239,812) | $ (226,804,743) | $ (233,563,746) | $ (239,834,187) |
| Discounted Cash Flow | $ (5,071,304) | $ (29,122,046) | $ (16,547,523) | $ (142,835,014) | $ (133,732,986) | $ (125,207,487) | $ (117,222,833) | $ (109,745,392) | $ (102,743,498) | $ (96,187,351) | $ (90,048,935) | $ (84,060,420) |
| **Cumulative DCF** | $ (5,071,304) | $ (34,193,350) | $ (50,740,874) | $ (193,575,888) | $ (327,308,874) | $ (452,516,360) | $ (569,739,193) | $ (679,484,586) | $ (782,228,083) | $ (878,415,435) | $ (968,464,369) | $ (1,052,524,789) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 101,426,081 | Payback Period | 25.5 |
| Constant Dollar OPEX | $ 358,599,332 | Internal Rate of Return | N/A |
| Constant Dollar REE Rev | $ 130,145,495 | Net Present Value | $ (1,673,705,781) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ - |

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C1 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 185,556,356** | **$ 191,123,047** | **$ 196,856,738** | **$ 202,762,440** | **$ 208,845,314** | **$ 215,110,673** | **$ 221,563,993** | **$ 228,210,913** | **$ 235,057,240** | **$ 242,108,958** | **$ 249,372,226** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 185,556,356 | $ 191,123,047 | $ 196,856,738 | $ 202,762,440 | $ 208,845,314 | $ 215,110,673 | $ 221,563,993 | $ 228,210,913 | $ 235,057,240 | $ 242,108,958 | $ 249,372,226 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 511,402,482** | **$ 526,615,208** | **$ 542,413,665** | **$ 558,686,075** | **$ 575,446,657** | **$ 592,710,057** | **$ 610,491,358** | **$ 628,806,099** | **$ 647,670,282** | **$ 667,100,390** | **$ 687,113,402** |
| *- Operating Costs* | $ 511,276,901 | $ 526,615,208 | $ 542,413,665 | $ 558,686,075 | $ 575,446,657 | $ 592,710,057 | $ 610,491,358 | $ 628,806,099 | $ 647,670,282 | $ 667,100,390 | $ 687,113,402 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 125,581 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ (325,846,126) | $ (335,492,162) | $ (345,556,926) | $ (355,923,634) | $ (366,601,343) | $ (377,599,384) | $ (388,927,365) | $ (400,595,186) | $ (412,613,042) | $ (424,991,433) | $ (437,741,176) |
| **Book Deductions** | **$ 1,970,013** | **$ 1,822,262** | **$ 1,685,593** | **$ 1,559,173** | **$ 1,442,235** | **$ 1,334,067** | **$ 1,234,012** | **$ 1,141,462** | **$ 1,055,852** | **$ 896,677** | **$ 326,163** |
| *- CAPEX #1 Depreciation* | $ 161,339 | $ 149,239 | $ 138,046 | $ 127,692 | $ 118,116 | $ 109,257 | $ 101,063 | $ 93,483 | $ 86,472 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 1,097,433 | $ 1,015,125 | $ 938,991 | $ 868,566 | $ 803,424 | $ 743,167 | $ 687,430 | $ 635,872 | $ 588,182 | $ 544,068 | $ - |
| *- CAPEX #3 Depreciation* | $ 711,241 | $ 657,898 | $ 608,556 | $ 562,914 | $ 520,696 | $ 481,644 | $ 445,520 | $ 412,106 | $ 381,198 | $ 352,608 | $ 326,163 |
| Taxable Before Depletion | $ (327,816,139) | $ (337,314,424) | $ (347,242,519) | $ (357,482,807) | $ (368,043,578) | $ (378,933,451) | $ (390,161,378) | $ (401,736,648) | $ (413,668,894) | $ (425,888,110) | $ (438,067,339) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ (163,908,070) | $ (168,657,212) | $ (173,621,260) | $ (178,741,404) | $ (184,021,789) | $ (189,466,726) | $ (195,080,689) | $ (200,868,324) | $ (206,834,447) | $ (212,944,055) | $ (219,033,669) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 25,977,890 | $ 26,757,227 | $ 27,559,943 | $ 28,386,742 | $ 29,238,344 | $ 30,115,494 | $ 31,018,959 | $ 31,949,528 | $ 32,908,014 | $ 33,895,254 | $ 34,912,112 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ (327,816,139)** | **$ (337,314,424)** | **$ (347,242,519)** | **$ (357,482,807)** | **$ (368,043,578)** | **$ (378,933,451)** | **$ (390,161,378)** | **$ (401,736,648)** | **$ (413,668,894)** | **$ (425,888,110)** | **$ (438,067,339)** |
| *- Income Tax* | $ (85,232,196) | $ (87,701,750) | $ (90,283,055) | $ (92,945,530) | $ (95,691,330) | $ (98,522,697) | $ (101,441,958) | $ (104,451,528) | $ (107,553,912) | $ (110,730,908) | $ (113,897,508) |
| Net Profit/Loss | $ (242,583,943) | $ (249,612,674) | $ (256,959,464) | $ (264,537,277) | $ (272,352,248) | $ (280,410,754) | $ (288,719,419) | $ (297,285,119) | $ (306,114,981) | $ (315,157,201) | $ (324,169,831) |
| **Book Deductions** | **$ 1,970,013** | **$ 1,822,262** | **$ 1,685,593** | **$ 1,559,173** | **$ 1,442,235** | **$ 1,334,067** | **$ 1,234,012** | **$ 1,141,462** | **$ 1,055,852** | **$ 896,677** | **$ 326,163** |
| *+ CAPEX #1 Depreciation* | $ 161,339 | $ 149,239 | $ 138,046 | $ 127,692 | $ 118,116 | $ 109,257 | $ 101,063 | $ 93,483 | $ 86,472 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 1,097,433 | $ 1,015,125 | $ 938,991 | $ 868,566 | $ 803,424 | $ 743,167 | $ 687,430 | $ 635,872 | $ 588,182 | $ 544,068 | $ - |
| *+ CAPEX #3 Depreciation* | $ 711,241 | $ 657,898 | $ 608,556 | $ 562,914 | $ 520,696 | $ 481,644 | $ 445,520 | $ 412,106 | $ 381,198 | $ 352,608 | $ 326,163 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ (240,613,930) | $ (247,790,411) | $ (255,273,872) | $ (262,978,104) | $ (270,910,013) | $ (279,076,686) | $ (287,485,407) | $ (296,143,658) | $ (305,059,129) | $ (314,260,524) | $ (323,843,668) |
| **Captial Cost** | **$ 2,093,011** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 2,093,011 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ (242,706,941) | $ (247,790,411) | $ (255,273,872) | $ (262,978,104) | $ (270,910,013) | $ (279,076,686) | $ (287,485,407) | $ (296,143,658) | $ (305,059,129) | $ (314,260,524) | $ (323,843,668) |
| Discounted Cash Flow | $ (77,333,911) | $ (71,776,056) | $ (67,221,589) | $ (62,954,867) | $ (58,957,912) | $ (55,213,835) | $ (51,706,777) | $ (48,421,854) | $ (45,345,096) | $ (42,466,204) | $ (39,782,891) |
| **Cumulative DCF** | $ (1,129,858,700) | $ (1,201,634,756) | $ (1,268,856,345) | $ (1,331,811,212) | $ (1,390,769,124) | $ (1,445,982,959) | $ (1,497,689,736) | $ (1,546,111,591) | $ (1,591,456,686) | $ (1,633,922,890) | $ (1,673,705,781) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 101,426,081 | Payback Period | 0 | 25.5 |
| Constant Dollar OPEX | $ 358,599,332 | Internal Rate of Return | 0 | N/A |
| Constant Dollar REE Rev | $ 130,145,495 | Net Present Value | 0 | $ (1,673,705,781) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ - |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 0.21 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 0.13 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.01 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 7 | Conveyor | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 0.37** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.07 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.07** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 611** | **$ 1,834** | **$ 616,370** | **$ 0.44** | **$ 0.15** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 6.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 500 | tph |
| Residue Rate | 384 | tph |
| Slurry Flow | 9250 | gpm |
| Leaching Time | 45 | min |
| Acid Consumption | 194.92 | kg/ton |
| Bioleaching Sol'n Consumption | 0 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 2 | -- |
| Thickener Capacity | 9.03 | ft^2/tph |
| Polymer Dose | 0.016 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 24 | 100% | 17,344 | gal | $ 97,795 | $ 2,347,070 | $ - | |
| 2 | Tank Mixer | 24 | 100% | 26 | HP | $ 100,658 | $ 2,415,786 | $ - | |
| 3 | Process Pump | 12 | 100% | 1,542 | gpm | $ 106,518 | $ 1,278,221 | $ - | |
| 4 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 5 | Belt Filter | 2 | 100% | 192 | TPH | $ 1,159,434 | $ 2,318,867 | $ - | |
| 6 | Solution Heater | 1 | 100% | 1 | MBTU/hr | $ 43,673 | $ 43,673 | $ - | |
| 7 | Thickener | 6 | 100% | 578 | ft^2 | $ 69,230 | $ 415,378 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 8,914,552** | **$ -** | |

## Power/Utilities

Power Cost: $ 0.07 $/kWhr
Efficency: 0.85

| ID | Unit Operation | Total HP | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 624.38 | $ 299.30 | $ 897.89 | $ 301,691 | $ 0.21 | $ 0.07 |
| 3 | Process Pump | 462.50 | $ 221.70 | $ 665.10 | $ 223,475 | $ 0.16 | $ 0.06 |
| 4 | Conveyor | 31.00 | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 5 | Belt Filter | 93.08 | $ 44.62 | $ 133.86 | $ 44,977 | $ 0.03 | $ 0.01 |
| 6 | Solution Heater | 112.49 | $ 53.92 | $ 161.77 | $ 54,354 | $ 0.04 | $ 0.01 |
| 7 | Thickener | 2.20 | $ 1.06 | $ 3.17 | $ 1,065 | $ 0.00 | $ 0.00 |
| 8 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1326** | **$ 635.46** | **$ 1,906.37** | **$ 640,540** | **$ 0.45** | **$ 0.16** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | - | L/hr | $ 0.008 | / L | $ - | $ - | $ - | $ - | $ - |
| 2 | H2SO4 | 97.46 | mt/hr | $ 220.000 | / mt | $ 171,526.54 | $ 514,579.61 | $ 172,898,748 | $ 122.60 | $ 42.88 |
| 3 | Natural Gas | 1.08 | MCF/hr | $ 2.860 | / MCF | $ 24.82 | $ 74.46 | $ 25,018 | $ 0.02 | $ 0.01 |
| 4 | Polymer | 33.61 | kg/hr | $ 2.250 | / kg | $ 605.06 | $ 1,815.18 | $ 609,901 | $ 0.43 | $ 0.15 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 172,156.42** | **$ 516,469.25** | **$ 173,533,667** | **$ 123.05** | **$ 43.04** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 172,792** **$ 518,376** **$ 174,174,207** **$ 123.51** **$ 43.20**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 6.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8292 | gpm |
| Feed Rate | 31388 | lpm |
| Precip Mass Flow | 9.20 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 887.99 | ft^2/tpd |
| NaOH Dose | 20.39 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 6 | 100% | 62,188 | gal | $ 133,694 | $ 802,162 | $ - | |
| 2 | Tank Mixer | 6 | 100% | 31 | HP | $ 111,308 | $ 667,847 | $ - | |
| 3 | Process Pump | 18 | 100% | 1,382 | gpm | $ 51,083 | $ 919,502 | $ - | |
| 4 | Filter Press | 1 | 100% | 43 | ft^3 | $ 137,826 | $ 137,826 | $ - | |
| 5 | Conveyor | 1 | 100% | 9 | TPH | $ 10,653 | $ 10,653 | $ - | |
| 6 | Thickener | 6 | 100% | 1,362 | ft^2 | $ 137,709 | $ 826,254 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 3,364,244** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 186.56 | Effiicency: | 0.85 | | | $ 89.43 | $ 268.29 | $ 90,146 | $ 0.06 | $ 0.02 |
| 3 | Process Pump | 621.88 | | | | | $ 298.10 | $ 894.30 | $ 300,486 | $ 0.21 | $ 0.07 |
| 4 | Filter Press | 7.28 | | | | | $ 3.49 | $ 10.47 | $ 3,516 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 0.57 | | | | | $ 0.27 | $ 0.82 | $ 276 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 5.20 | | | | | $ 2.49 | $ 7.47 | $ 2,510 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **821** | | | | | **$ 393.78** | **$ 1,181.35** | **$ 396,934** | **$ 0.28** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 38.39 | mt/hr | $ 125.000 | / mt | $ 38,390.82 | $ 115,172.47 | $ 38,697,949 | $ 27.44 | $ 9.60 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 38,390.82** | **$ 115,172.47** | **$ 38,697,949** | **$ 27.44** | **$ 9.60** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 38,785** | **$ 116,354** | **$ 39,094,884** | **$ 27.72** | **$ 9.70** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 6.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8263 | gpm |
| Feed Rate | 31278 | lpm |
| Precip Mass Flow | 8.00 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 4928.65 | ft^2/tpd |
| NaOH Dose | 3.66 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 6 | 100% | 61,971 | gal | $ 133,431 | $ 800,584 | $ - | |
| 2 | Tank Mixer | 6 | 100% | 31 | HP | $ 111,088 | $ 666,529 | $ - | |
| 3 | Process Pump | 18 | 100% | 1,377 | gpm | $ 51,015 | $ 918,275 | $ - | |
| 4 | Filter Press | 1 | 100% | 38 | ft^3 | $ 124,626 | $ 124,626 | $ - | |
| 5 | Conveyor | 1 | 100% | 8 | TPH | $ 9,863 | $ 9,863 | $ - | |
| 6 | Thickener | 6 | 100% | 6,568 | ft^2 | $ 486,690 | $ 2,920,140 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 5,440,016** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 185.91 | Effiicency: | 0.85 | | | $ 89.12 | $ 267.35 | $ 89,831 | $ 0.06 | $ 0.02 |
| 3 | Process Pump | 619.71 | | | | | $ 297.06 | $ 891.18 | $ 299,436 | $ 0.21 | $ 0.07 |
| 4 | Filter Press | 6.33 | | | | | $ 3.03 | $ 9.10 | $ 3,057 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 0.50 | | | | | $ 0.24 | $ 0.71 | $ 240 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 25.06 | | | | | $ 12.01 | $ 36.04 | $ 12,110 | $ 0.01 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **838** | | | | | **$ 401.46** | **$ 1,204.39** | **$ 404,674** | **$ 0.29** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 6.87 | mt/hr | $ 125.000 | / mt | $ 6,867.21 | $ 20,601.63 | $ 6,922,149 | $ 4.91 | $ 1.72 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 6,867.21** | **$ 20,601.63** | **$ 6,922,149** | **$ 4.91** | **$ 1.72** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 7,269** **$ 21,806** **$ 7,326,823** **$ 5.20** **$ 1.82**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 8.00 | tph |
| Slurry Feed Rate | 275 | gpm |
| Leaching Time | 0 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 1473.42 | kg/ton |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | - | gal | $ - | $ - | $ - | |
| 2 | Tank Mixer | 4 | 100% | - | HP | $ - | $ - | $ - | |
| 3 | Process Pump | 1 | 100% | 275 | gpm | $ 55,224 | $ 55,224 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 55,224** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | - | Effiicency: | 0.85 | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Process Pump | 6.88 | | | | | $ 3.30 | $ 9.90 | $ 3,327 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **7** | | | | | **$ 3.30** | **$ 9.90** | **$ 3,327** | **$ 0.00** | **$ 0.00** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 11.78 | mt/hr | $ 270.000 | / mt | $ 25,446.77 | $ 76,340.31 | $ 25,650,345 | $ 18.19 | $ 6.36 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 25,446.77** | **$ 76,340.31** | **$ 25,650,345** | **$ 18.19** | **$ 6.36** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 25,450** | **$ 76,350** | **$ 25,653,672** | **$ 18.19** | **$ 6.36** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 372 | gpm |
| Feed Rate | 1408 | lpm |
| Precip Mass Flow | 0.05 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 0.00 | g/L of Feed |
| Oxalic Acid Dose | 24.98 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.03 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 1,860 | gal | $ 2,638 | $ 10,553 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 1 | HP | $ 15,375 | $ 61,502 | $ - | |
| 3 | Process Pump | 3 | 100% | 372 | gpm | $ 61,929 | $ 185,787 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 0.26 | ft^3 | $ 3,431 | $ 3,431 | $ - | |
| 6 | Roaster | 1 | 100% | 0.08 | MBTU/hr | $ 272,950 | $ 272,950 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 534,222** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 3.72 | Effiicency: | 0.85 | | $ 1.78 | $ 5.35 | $ 1,797 | $ 0.00 | $ 0.00 |
| 3 | Process Pump | 27.89 | | | | $ 13.37 | $ 40.11 | $ 13,478 | $ 0.01 | $ 0.00 |
| 4 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.04 | | | | $ 0.02 | $ 0.06 | $ 21 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.01 | | | | $ 0.01 | $ 0.02 | $ 6 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **32** | | | | **$ 15.18** | **$ 45.54** | **$ 15,302** | **$ 0.01** | **$ 0.00** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 2 | Oxalic Acid | 2,109.88 | kg/hr | $ 0.600 | / kg | $ 10,127.41 | $ 30,382.23 | $ 10,208,431 | $ 7.24 | $ 2.53 |
| 3 | Natural Gas | 0.08 | MCF/hr | $ 2.860 | / MCF | $ 1.86 | $ 5.58 | $ 1,874 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 10,129.27** | **$ 30,387.81** | **$ 10,210,304** | **$ 7.24** | **$ 2.53** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 10,144** **$ 30,433** **$ 10,225,606** **$ 7.25** **$ 2.54**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8292 | gpm |
| Raffinate Rate: | 8292 | gpm |
| Extraction Feed (O+A) | 16584 | gpm |
| Scrub Feed (O+A) | 228 | gpm |
| Strip Feed (O+A) | 456 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 228 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 272632 | gal |
| Precip Mixing Intensity | 1.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 30.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,584 | gal | $ 95,360 | $ 190,720 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 41 | HP | $ 130,915 | $ 261,831 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 248,753 | gal | $ 437,322 | $ 874,643 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,292 | gpm | $ 202,347 | $ 404,694 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 228 | gal | $ 8,556 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,666 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,420 | gal | $ 39,240 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 456 | gal | $ 12,635 | $ 25,271 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 17,247 | $ 34,494 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,839 | gal | $ 57,946 | $ 115,892 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,588 | $ 19,588 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 59,341 | kg | $ 3,715,315 | $ - | $ 3,715,315 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 1,029,443 | kg | $ 1,114,728 | $ - | $ 1,114,728 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,728 | gal | $ 5,693 | $ 22,771 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 4.09 | HP | $ 35,462 | $ 141,849 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,588 | $ 39,175 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.01 | ft^3 | $ 279 | $ 279 | $ - | |
| 20 | Process Pump | 2 | 100% | 228 | gpm | $ 51,383 | $ 102,766 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,233,975** | **$ 4,830,043** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 82.92 | Effiicency: | 0.85 | | $ 39.75 | $ 119.24 | $ 40,065 | $ 0.03 | $ 0.01 |
| 3 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 414.59 | | | | $ 198.73 | $ 596.20 | $ 200,324 | $ 0.14 | $ 0.05 |
| 5 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.28 | | | | $ 1.09 | $ 3.28 | $ 1,102 | $ 0.00 | $ 0.00 |
| 11 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.45 | | | | $ 0.22 | $ 0.65 | $ 220 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 16.37 | | | | $ 7.85 | $ 23.54 | $ 7,908 | $ 0.01 | $ 0.00 |
| 17 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.91 | | | | $ 0.44 | $ 1.31 | $ 439 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | | | | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.40 | | | | $ 5.46 | $ 16.39 | $ 5,508 | $ 0.00 | $ 0.00 |
| | **TOTAL:** | **529** | | | | **$ 253.54** | **$ 760.61** | **$ 255,566** | **$ 0.18** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 107.35 | kg/hr | $ 0.940 | / kg | $ 807.24 | $ 2,421.73 | $ 813,700 | $ 0.58 | $ 0.20 |
| 2 | D2EHPA | 5.65 | kg/hr | $ 5.000 | / kg | $ 225.99 | $ 677.98 | $ 227,800 | $ 0.16 | $ 0.06 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.87 | mt/hr | $ 270.000 | / mt | $ 1,880.21 | $ 5,640.64 | $ 1,895,255 | $ 1.34 | $ 0.47 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,913.45** | **$ 8,740.34** | **$ 2,936,755** | **$ 2.08** | **$ 0.73** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,167** **$ 9,501** **$ 3,192,321** **$ 2.26** **$ 0.79**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 371.86 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 1407.63 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 371.73 | gpm | HCL Dose | 58.96 | g/L of Feed |
| Mn Feed Rate | 1407.15 | lpm | | | |
| Co Precip Mass Flow | 0.11 | tph | | | |
| Mn Precip Mass Flow | 0.17 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 6.88 | g/L of Feed | | | |
| NaOH Dose | 0.00 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 5,578 | gal | $ 7,484 | $ 14,967 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 8 | HP | $ 53,086 | $ 106,171 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 372 | gpm | $ 61,925 | $ 61,925 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 0.51 | ft^3 | $ 5,606 | $ 5,606 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 5,576 | gal | $ 7,481 | $ 14,963 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 8 | HP | $ 53,076 | $ 106,151 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 372 | gpm | $ 61,917 | $ 61,917 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 1 | ft^3 | $ 7,905 | $ 7,905 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 379,606** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 16.73 | Effiicency: | 0.85 | | | $ 8.02 | $ 24.06 | $ 8,085 | $ 0.01 | $ 0.00 |
| 3 | Process Pump | 9.30 | | | | | $ 4.46 | $ 13.37 | $ 4,492 | $ 0.00 | $ 0.00 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.09 | | | | | $ 0.04 | $ 0.12 | $ 41 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 16.73 | | | | | $ 8.02 | $ 24.06 | $ 8,083 | $ 0.01 | $ 0.00 |
| 8 | Process Pump | 9.29 | | | | | $ 4.45 | $ 13.36 | $ 4,490 | $ 0.00 | $ 0.00 |
| 9 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 0.14 | | | | | $ 0.07 | $ 0.20 | $ 66 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **52** | | | | | **$ 25.06** | **$ 75.17** | **$ 25,258** | **$ 0.02** | **$ 0.01** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 0.58 | mt/hr | $ 360.000 | / mt | $ 1,673.11 | $ 5,019.33 | $ 1,686,496 | $ 1.20 | $ 0.42 |
| 2 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 0.82 | $ 2.47 | $ 829 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 1,673.93** | **$ 5,021.80** | **$ 1,687,325** | **$ 1.20** | **$ 0.42** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 1,699** **$ 5,097** **$ 1,712,583** **$ 1.21** **$ 0.42**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | Shakespeare Lignite | Date: | 3/18/2023 |
| Case: | SHXP-C1 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | 192.3% |
| *(unit conversion)* | 12000 | TPD | Project NPV: | 2,534,795,739 |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 0.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C1 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 101,426,081 | | | | | |
| Constant Dollar Annual OPEX | $ | 361,884,789 | $ | 89.75 | /ton | $ 108.8 | /kg |
| Constant Dollar Annual REE Rev | $ | 701,529,411 | $ | 173.99 | /ton | $ 211.00 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | |
|---|---|---|---|---|
| Payback Period | 3.5 | years | *0.5* | *operating years* |
| Internal Rate of Return | 192% | | | |
| Net Present Value | $ 2,534,795,739 | | | |
| Total Severence Tax Paid | $ - | | | |
| Total State and Federal Tax Paid | $ 1,826,652,752 | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C1 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 1,250 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 2.61 | 32.63 | 14.80 | | | | | | | 14.80 |
| Yttrium | 16.93 | 211.68 | 96.02 | | | | | | | 96.02 |
| Lanthanum | 15.39 | 192.37 | 87.26 | | | | | | | 87.26 |
| Cerium | 30.86 | 385.81 | 175.00 | | | | | | | 175.00 |
| Praseodymium | 4.24 | 53.03 | 24.05 | | | | | | | 24.05 |
| Neodymium | 15.21 | 190.07 | 86.22 | | | | | | | 86.22 |
| Samarium | 3.18 | 39.81 | 18.06 | | | | | | | 18.06 |
| Europium | 0.78 | 9.72 | 4.41 | | | | | | | 4.41 |
| Gadolinium | 3.52 | 43.96 | 19.94 | | | | | | | 19.94 |
| Terbium | 0.28 | 3.48 | 1.58 | | | | | | | 1.58 |
| Dysprosium | 2.85 | 35.68 | 16.19 | | | | | | | 16.19 |
| Holmium | 0.53 | 6.64 | 3.01 | | | | | | | 3.01 |
| Erbium | 1.46 | 18.19 | 8.25 | | | | | | | 8.25 |
| Thulium | 0.24 | 3.02 | 1.37 | | | | | | | 1.37 |
| Ytterbium | 1.59 | 19.85 | 9.01 | | | | | | | 9.01 |
| Lutetium | 0.32 | 4.03 | 1.83 | | | | | | | 1.83 |
| Manganese | -- | 260.10 | 117.98 | | | 0.00 | -- | | | 117.98 |
| Cobalt | -- | 134.30 | 60.92 | | | 0.00 | -- | | | 60.92 |
| Nickel | -- | 337.90 | 153.27 | | | 0.00 | -- | | | 153.27 |
| Zinc | -- | 136.80 | 62.05 | | | 0.00 | -- | | | 62.05 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 1250.00 | 566.99 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 567.0 |
| Total non-REE CM | -- | 869.10 | 394.22 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 394.2 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed Rate (kg/hr) | MREO Product 1 Rate (kg/hr) | Sc2O3 Product 2 Rate (kg/hr) | Mn Product 3 Rate (kg/hr) | CoS Product 4 Rate (kg/hr) | Reserved Product 5 Rate (kg/hr) | Total Product (kg/hr) | Calc. Loss Rate (kg/hr) | Overall Recovery (%) | |
|---|---|---|---|---|---|---|---|---|---|---|
| Scandium | 14.80 | 0.01 | 12.82 | 0.02 | 0.00 | 0.00 | 12.84 | 1.96 | 86.8% | |
| Yttrium | 96.02 | 29.84 | 0.00 | 36.71 | 9.26 | 0.00 | 75.82 | 20.20 | 79.0% | |
| Lanthanum | 87.26 | 13.84 | 0.00 | 32.53 | 16.25 | 0.00 | 62.62 | 24.64 | 71.8% | |
| Cerium | 175.00 | 43.98 | 0.00 | 26.74 | 55.80 | 0.00 | 126.52 | 48.48 | 72.3% | |
| Praseodymium | 24.05 | 5.55 | 0.00 | 6.92 | 5.09 | 0.00 | 17.56 | 6.50 | 73.0% | |
| Neodymium | 86.22 | 26.82 | 0.00 | 8.78 | 23.00 | 0.00 | 58.60 | 27.61 | 68.0% | |
| Samarium | 18.06 | 5.25 | 0.00 | 3.62 | 3.72 | 0.00 | 12.58 | 5.47 | 69.7% | |
| Europium | 4.41 | 1.28 | 0.00 | 0.78 | 0.66 | 0.00 | 2.72 | 1.69 | 61.8% | |
| Gadolinium | 19.94 | 6.44 | 0.00 | 4.65 | 4.19 | 0.00 | 15.27 | 4.67 | 76.6% | |
| Terbium | 1.58 | 0.37 | 0.00 | 0.25 | 0.17 | 0.00 | 0.79 | 0.79 | 50.3% | |
| Dysprosium | 16.19 | 5.68 | 0.00 | 2.16 | 4.66 | 0.00 | 12.50 | 3.69 | 77.2% | |
| Holmium | 3.01 | 0.79 | 0.00 | 0.75 | 0.35 | 0.00 | 1.89 | 1.12 | 62.7% | |
| Erbium | 8.25 | 2.61 | 0.00 | 2.66 | 0.97 | 0.00 | 6.24 | 2.02 | 75.6% | |
| Thulium | 1.37 | 0.26 | 0.00 | 0.34 | 0.07 | 0.00 | 0.67 | 0.70 | 48.8% | |
| Ytterbium | 9.01 | 2.20 | 0.00 | 1.90 | 0.58 | 0.00 | 4.67 | 4.33 | 51.9% | |
| Lutetium | 1.83 | 0.35 | 0.00 | 0.29 | 0.36 | 0.00 | 1.00 | 0.83 | 54.9% | |
| Manganese | 117.98 | 11.52 | 0.00 | 16.38 | 3.95 | 0.00 | 31.85 | 86.13 | 27.0% | |
| Cobalt | 60.92 | 8.54 | 0.00 | 4.62 | 21.03 | 0.00 | 34.19 | 26.72 | 56.1% | |
| Nickel | 153.27 | 17.40 | 0.00 | 12.17 | 40.00 | 0.00 | 69.57 | 83.70 | 45.4% | |
| Zinc | 62.05 | 7.21 | 0.00 | 7.82 | 29.30 | 0.00 | 44.33 | 17.72 | 71.4% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 566.99 | 145.27 | 12.82 | 129.08 | 125.14 | 0.00 | 412.30 | 154.69 | 72.7% | <---- |
| Total non-REE CM | 394.22 | 44.67 | 0.00 | 41.00 | 94.28 | 0.00 | 179.95 | 214.27 | 45.6% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 384 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 9 | ton | $ | 1.50 |
| Dust/Volatiles | - | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | |
|---|---|
| 3.49 | TPD REE |
| 0.308 | TPD Sc |
| 3.25 | TPD CM |
| 0.26 | REE |
| 86.6% | Sc |
| 13.9% | Mn |
| 32.7% | Metal-S |
| 500 | Feed |
| 1,250 | Grade |
| 3.5 | PP |
| 192% | IRR |
| 2534795739 | NPV |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C1 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 0.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | Based on Design |
| Dissolved Solids in Leach | 23.2% | % | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Leaching Retention Time | 45 | min | Based on Design |
| Thickener UF % Solids | 46% | % | SHXP-C1 Samples |
| Residue Settling Rate | 15.0 | ft/hr | Not Available |
| **Reagent Use** | | | |
| Acid Consumption | 194.9 | kg/ton | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Bioleaching Sol'n Consumption | 0 | L/ton | |
| Polymer Dosage | 0.016 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 9.03 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | Based on Design |
| Precip Solid Yield | 4.44 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Precip Settling Rate | 2.24 | ft/hr | Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 0.6% | % | VT Override from 0.3% |
| **Reagent Use** | | | |
| NaOH Dose | 20.39 | g/L of Feed | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 887.99 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 1250 | ppm | |
| Grade After Sorter | 1250 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 500 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 0 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 500 | tph | |
| Leaching Residue Rate | 383.85 | tph | |
| Slurry Flow (feed) | 9250 | gpm | |
| Product Flow | 8,292 | gpm | |
| Thickener Feed % Solids (eff) | 15% | % | |
| Liquid Density | 1.021 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 57.98 | g/L | |
| Leaching Vessel Size | 17343.75 | gal | |
| Leaching Vessel Size | 65.65 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 191.93 | tph | 70 to 200 tph |
| Thickener Area | 577.78 | ft^2 | |
| Thickener Diameter | 27.13 | ft | ideal < 100' |
| Liquid Balance | 289.96 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,292 | gpm | |
| Slurry Feed Rate | 31,388 | lpm | |
| Precip Mass Flow | 9.20 | tph | |
| Thickener Feed % Solids (eff) | 0.44% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.018 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 235.41 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 43.34 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 10.81 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 1,362 | ft^2 | |
| Thickener Diameter | 41.65 | ft | ideal < 100' |

*PM5: REE Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |
| | | | |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45 | min | Based on Design |
| Precip Solid Yield | 3.87 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Precip Settling Rate | 0.044 | ft/hr | Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.39% | % | VT Override from 0.16% |
| | | | |
| **Reagent Use** | | | |
| NaOH Dose | 3.66 | g/L of Feed | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| | | | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 4928.65 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 8263 | gpm | |
| Slurry Feed Rate | 31,278 | lpm | |
| Precip Mass Flow | 8.00 | tph | |
| Thickener Feed % Solids (eff) | 0.4% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.015 | g/cc | |
| | | | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 234.59 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 37.69 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 9.39 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 6568 | ft^2 | |
| Thickener Diameter | 91.47 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | 62.53 | m/m | target = 6.35 |

## *PM6: RE-Redissolution*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 10.7% | % | Based on Design |
| Dissolved Solids in Re-Leach | 32% | % | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Leaching Retention Time | | min | |
| **Reagent Use** | | | |
| Acid Consumption | 1473.4 | kg/ton | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| | 0.0 | | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| **Material Balance** | | | |
|---|---|---|---|
| Solid Feed Rate | 8.00 | tph | |
| Slurry Flow (feed) | 275 | gpm | |
| Product Flow | 275 | gpm | |
| Liquid Density | 1.082 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 30.11 | v/v | target = 3.8 |
| Liquid Balance | 9.39 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 116.0 | g/L | |

## *PM7: REE Oxalate Precipitation*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | Based on Design |
| Precip Solid Yield | 0.6 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Thickener UF %Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| NaOH Dose | 0.00 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Oxalic Acid Dose | 24.98 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| **Material Balance** | | | |
|---|---|---|---|
| Slurry Feed Rate | 371.92 | gpm | |
| Slurry Feed Rate | 1407.87 | lpm | |
| Precip Mass Flow | 0.05 | tph | |
| Thickener Feed % Solids (eff) | 0.05% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.03 | tph | |
| Liquid Density | 1.105 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 7.04 | m^3 | |
| Filter Press Area | 0.26 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.06 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 15.71 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.71 | m/m | |

## *PM8: Sc SX Separation*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | Based on Design |
| Settler Residence Time | 15.00 | min | Based on Design |
| Loading: Number of Stages | 2.00 | - | Based on Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | Based on Design |
| Loading O:A Advance Ratio | 0.03 | v/v | Based on Design |
| Scrubbing: Number of Stages | | - | Not Applicable |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | Not Applicable |
| Scrubbing O:A Advance Ratio | | v/v | Not Applicable |
| Stripping: Number of Stages | 2.00 | - | Based on Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | Based on Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | Based on Design |
| Sc2O3 Grade | 2% | % | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | Not Availabe |
| Precip Solid Yield | 0.4 | g/L | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| **Material Balance (SX)** | | |
|---|---|---|
| PLS Flow | 8291.8 | gpm |
| Raffinate Rate: | 8291.8 | gpm |
| Load Org Recycle | 8063.8 | gpm |
| Load Org Feed | 8291.8 | gpm |
| Loag Org Advance | 228.0 | gpm |
| Scrub Aq Product | 0.0 | gpm |
| Scrub Aq Recycle | 0.0 | gpm |
| Scrub Aq Feed | 0.0 | gpm |
| Scurb Org Product | 228.0 | gpm |
| Strip Aq Product | 18.2 | gpm |
| Strip Aq Recycle | 209.8 | gpm |
| Strip Aq Feed | 228.0 | gpm |
| Strip Org Product | 228.0 | gpm |
| Extraction Feed (O+A): | 16583.5 | gpm |
| Scrub Feed (O+A): | 228.0 | gpm |
| Strip Feed (O+A): | 455.9 | gpm |
| **Material Balance (Precip)** | | |
| Strip Solution Flow | 18.2 | gpm |
| Solid Density | 3.4 | g/cc |
| Solid Product Rate | 0.00 | tph |
| Liquid Density (after precip) | 1.13 | g/cc |
| Thickener Feed % Solids (eff) | 0.03% | % |
| **Total Volumes** | | |
| Total Extraction Volume | 530673 | gal |
| Total Scrubbing Volume | 0 | gal |
| Total Stripping Volume | 14590 | gal |
| Extraction Org Volume | 265336.47 | gal |
| Scrubbing Org Volume | 0.00 | gal |
| Stripping Org Volume | 7295.05 | gal |
| Total Org Volume | 272631.52 | gal |

| **Other Indicators** | | | |
|---|---|---|---|
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 10.33 | m^3 | |
| Filter Press Area | 0.01 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

## *PM9: CM Precipitation*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 30 | min | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Precip Solid Yield | 2.87 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Precip Settlng Rate | 10 | ft/hr | |
| Thickener UF % Solids | 25% | % | Not Available |
| **Reagent Use** | | | |
| NaOH Dose | 3.56 | g/L of Feed | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 30.00 | min | N/A |
| Thickener Capacity | 1107.76 | ft^2/tph | N/A |

| **Material Balance** | | | |
|---|---|---|---|
| Slurry Feed Rate | 8253 | gpm | |
| Slurry Feed Rate | 31,242 | lpm | |
| Precip Mass Flow | 5.92 | tph | |
| Thickener Feed % Solids (eff) | 0.3% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.013 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 234.32 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 27.92 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 6.96 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 1640 | ft^2 | |
| Thickener Diameter | 45.71 | ft | ideal < 100' |

## PM10: CM Redissolution

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 20.9% | % | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Dissolved Solids in Re-Leach | 14% | % | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Leaching Retention Time | 60 | min | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Reagent Use** | | | |
| Acid Consumption | 2684.53 | kg/ton | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 5.92 | tph | |
| Slurry Flow (feed) | 97 | gpm | |
| Product Flow | 97 | gpm | |
| Liquid Density | 1.173 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 5.5 | m^3 | |
| Releach Conc. Factor | 85.51 | v/v | |
| Liquid Balance | 6.96 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 245.1 | g/L | |

## PM11: CM Recovery

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 1.16 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | 0% | % | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 1.86 | g/L | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | 1% | % | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Reagent Use** | | | |
| Na2S Dose | 6.88 | g/L of Feed | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| NaOH Dose | 0.00 | g/L of Feed | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| HCL Dose | 58.96 | g/L of Feed | SHXP-C1/SHXP-C1 MASS BALANCE V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 371.86 | gpm | |
| Co Slurry Feed Rate | 1407.63 | lpm | |
| Co Precip Mass Flow | 0.11 | tph | |
| Co Thickener Feed % Solids (eff) | 0.10% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 371.73 | gpm | |
| Mn Slurry Feed Rate | 1407.15 | lpm | |
| Mn Precip Mass Flow | 0.17 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.17% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.10 | g/cc | |
| Mn Liquid Density | 1.10 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 21.1 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 0.51 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.13 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 21.1 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 0.8 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.20 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

## PM12: Biooxidation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Requirec | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C1 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 8,914,552 | $ - | $ 2,228,638 | $ 9,806,007 | $ 12,208,479 | $ 33,157,675 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 3,364,244 | $ - | $ 841,061 | $ 3,700,669 | $ 4,607,333 | $ 12,513,307 | IIIIIIIIIIII |
| PM5 | REE Precipitation | $ 5,440,016 | $ - | $ 1,360,004 | $ 5,984,017 | $ 7,450,102 | $ 20,234,138 | IIIIIIIIIIIIIIIIIIII |
| PM6 | RE-Redissolution | $ 55,224 | $ - | $ 13,806 | $ 60,746 | $ 75,629 | $ 205,405 | |
| PM7 | REE Oxalate Precipitation | $ 534,222 | $ - | $ 133,556 | $ 587,644 | $ 731,617 | $ 1,987,039 | II |
| PM8 | Sc SX Separation | $ 2,233,975 | $ 4,830,043 | $ 558,494 | $ 2,457,372 | $ 3,059,428 | $ 13,139,312 | IIIIIIIIIIIII |
| PM9 | CM Precipitation | $ 2,440,090 | $ - | $ 610,022 | $ 2,684,099 | $ 3,341,703 | $ 9,075,913 | IIIIIIIIII |
| PM10 | CM Redissolution | $ 144,580 | $ - | $ 36,145 | $ 159,038 | $ 198,002 | $ 537,764 | |
| PM11 | CM Recovery | $ 379,606 | $ - | $ 94,902 | $ 417,567 | $ 519,871 | $ 1,411,945 | I |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| | TOTAL | $ 25,970,167 | $ 4,830,043 | $ 6,492,542 | $ 28,567,184 | $ 35,566,144 | $ 101,426,081 | |

## Module Summary

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| Total Lang Factor (=1+sum % Base) | | 372% | |

Year of Construction: 2025

# OPERATING COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C1 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 0.92 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 0.35 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 101,426,081 | $ Total | $ 1,006.21 | $ 3,018.63 | $ 1,014,261 | $ 0.31 | $ 0.25 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 0.09 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 701,529,411 | $ Total | $ 3,479.81 | $ 10,439.43 | $ 3,507,647 | $ 1.05 | $ 0.87 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 0.14 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 101,426,081 | $ Total | $ 1,006.21 | $ 3,018.63 | $ 1,014,261 | $ 0.31 | $ 0.25 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 10,473.68** | **$ 31,421.04** | **$ 10,557,470** | **$ 3.18** | **$ 2.62** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 0.15 | $ 0.13 |
| PM2 | Roasting | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | PR | 1326 | | $ 635.46 | $ 1,906.37 | $ 640,540 | $ 0.19 | $ 0.16 |
| PM4 | Al Precipitation | PR | 821 | | $ 393.78 | $ 1,181.35 | $ 396,934 | $ 0.12 | $ 0.10 |
| PM5 | REE Precipitation | PR | 838 | | $ 401.46 | $ 1,204.39 | $ 404,674 | $ 0.12 | $ 0.10 |
| PM6 | RE-Redissolution | PR | 7 | | $ 3.30 | $ 9.90 | $ 3,327 | $ 0.00 | $ 0.00 |
| PM7 | REE Oxalate Precipitation | PR | 32 | | $ 15.18 | $ 45.54 | $ 15,302 | $ 0.00 | $ 0.00 |
| **Sub-Total** | | | **5432.9** | | **$ 2,604.27** | **$ 7,812.80** | **$ 2,625,100.75** | **$ 0.79** | **$ 0.65** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.03 | $ 0.03 |
| PM2 | Roasting | CR | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | CR | | $ 172,156.42 | $ 516,469.25 | $ 173,533,667 | $ 52.19 | $ 43.04 |
| PM4 | Al Precipitation | CR | | $ 38,390.82 | $ 115,172.47 | $ 38,697,949 | $ 11.64 | $ 9.60 |
| PM5 | REE Precipitation | CR | | $ 6,867.21 | $ 20,601.63 | $ 6,922,149 | $ 2.08 | $ 1.72 |
| PM6 | RE-Redissolution | CR | | $ 25,446.77 | $ 76,340.31 | $ 25,650,345 | $ 7.71 | $ 6.36 |
| PM7 | REE Oxalate Precipitation | CR | | $ 10,129.27 | $ 30,387.81 | $ 10,210,304 | $ 3.07 | $ 2.53 |
| **Sub-Total** | | | | **$ 298,689.52** | **$ 896,068.57** | **$ 301,079,037.98** | **$ 90.56** | **$ 74.67** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 9,212 | $ 0.10 | $ 307.08 | $ 921.24 | $ 309,537 | $ 0.09 | $ 0.08 |
| Non-Hazardous Precipitate | WD | Tons | 221 | $ 1.50 | $ 110.40 | $ 331.19 | $ 111,280 | $ 0.03 | $ 0.03 |
| Dust/Volatiles | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 417.48** | **$ 1,252.43** | **$ 420,816.53** | **$ 0.13** | **$ 0.10** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 301,711** | **$ 905,134** | **$ 304,124,955** | **$ 91** | **$ 75** |

## Overhead Costs

| Allocation | Code | Rate | Basis | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% | % of Direct Cost | $ 31,421 Daily Direct Cost | $ 1,571.05 | $ 4,713.16 | $ 1,583,621 | $ 0.48 | $ 0.39 |
| Variable Portion | OH | 15% | % of Direct Cost | $ 905,134 Daily Direct Cost | $ 45,256.69 | $ 135,770.07 | $ 45,618,743 | $ 13.72 | $ 11.31 |
| **TOTAL OVERHEAD COSTS** | | | | | **$ 46,827.74** | **$ 140,483.23** | **$ 47,202,363.83** | **$ 14.20** | **$ 11.71** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | SHXP-C1 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 10,473.68 | $ 31,421.04 | $ 10,557,470 | $ 3 | $ 2.62 | I |
| Power | $ 2,604.27 | $ 7,812.80 | $ 2,625,101 | $ 1 | $ 0.65 | |
| Consumables | $ 298,689.52 | $ 896,068.57 | $ 301,079,038 | $ 91 | $ 74.67 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| Leasing/Tolling | $ - | $ - | $ - | $ - | $ - | |
| Waste Disposal | $ 417.48 | $ 1,252.43 | $ 420,817 | $ 0 | $ 0.10 | |
| General Overhead | $ 46,827.74 | $ 140,483.23 | $ 47,202,364 | $ 14 | $ 11.71 | IIIII |
| TOTAL | $ 359,013 | $ 1,077,038 | $ 361,884,789 | $ 109 | $ 89.75 | |

## Summary by Process Module

| | Process Module | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 611 | $ 1,834 | $ 616,370 | $ 0.19 | $ 0.15 | |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 172,792 | $ 518,376 | $ 174,174,207 | $ 52.39 | $ 43.20 | IIIIIIIIIIIIIIIIII |
| PM4 | AI Precipitation | $ 38,785 | $ 116,354 | $ 39,094,884 | $ 11.76 | $ 9.70 | IIII |
| PM5 | REE Precipitation | $ 7,269 | $ 21,806 | $ 7,326,823 | $ 2.20 | $ 1.82 | |
| PM6 | RE-Redissolution | $ 25,450 | $ 76,350 | $ 25,653,672 | $ 7.72 | $ 6.36 | III |
| PM7 | REE Oxalate Precipitation | $ 10,144 | $ 30,433 | $ 10,225,606 | $ 3.08 | $ 2.54 | I |
| PM8 | Sc SX Separation | $ 3,167 | $ 9,501 | $ 3,192,321 | $ 0.96 | $ 0.79 | |
| PM9 | CM Precipitation | $ 7,027 | $ 21,082 | $ 7,083,715 | $ 2.13 | $ 1.76 | |
| PM10 | CM Redissolution | $ 34,349 | $ 103,047 | $ 34,623,958 | $ 10.41 | $ 8.59 | IIII |
| PM11 | CM Recovery | $ 1,699 | $ 5,097 | $ 1,712,583 | $ 0.52 | $ 0.42 | |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| NA | Unallocated | $ 57,719 | $ 173,157 | $ 58,180,651 | $ 17 | $ 14.43 | IIIIIII |
| | TOTAL | $ 359,013 | $ 1,077,038 | $ 361,884,789 | $ 109 | $ 89.75 | |

## Summary by Cost Center

| Cost Center | Code | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| Operating Labor | OL | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 1 | $ 0.76 | |
| Technical Labor | TL | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 0 | $ 0.29 | |
| Power/Utilities | PR | $ 2,604.27 | $ 7,812.80 | $ 2,625,101 | $ 1 | $ 0.65 | |
| Lease Agreements | LS | $ - | $ - | $ - | $ - | $ - | |
| Consumables/Reagents | CR | $ 298,689.52 | $ 896,068.57 | $ 301,079,038 | $ 91 | $ 74.67 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| Waste Disposal | WD | $ 417.48 | $ 1,252.43 | $ 420,817 | $ 0 | $ 0.10 | |
| Plant Overhead | OH | $ 48,138.81 | $ 144,416.44 | $ 48,523,924 | $ 15 | $ 12.03 | IIIIII |
| General Sales, and Administration | GSA | $ 4,943.31 | $ 14,829.93 | $ 4,982,856 | $ 1 | $ 1.24 | |
| Reserved | | | | | | | |
| TOTAL | | $ 359,013 | $ 1,077,038.06 | $ 361,884,789 | $ 109 | $ 89.75 | |

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C1 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 701,529,411

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 1217 | 1250 |
| Contained Value | ($/ton) | $ 31.1 | $ 221.8 |
| Feed Basket Price | ($/kg) | $ 28.1 | $ 195.6 |
| Product Basket Price | ($/kg) | $ 27.1 | $ 226.9 |
| Production Rate | (kg/day) | 9586.99 | 9895.27 |
| | (t/day) | 9.59 | 9.90 |
| | (t/yr) | 3221.23 | 3324.81 |
| | (% of US Demand) | 32.2% | |

## Revenue Contribution Table

| | | Product Revenue ($/hr) | | | | | |
|---|---|---|---|---|---|---|---|
| **Element** | **REE/CM Price ($/kg) [1]** | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | **REE Revenue ($/day)** |
| Scandium | $ 4,200.00 | $ 33.8 | $ 82,594.6 | $ - | $ - | $ - | $ 1,983,083 |
| Yttrium | $ 6.00 | $ 170.5 | $ - | $ - | $ - | $ - | $ 4,093 |
| Lanthanum | $ 2.00 | $ 24.3 | $ - | $ - | $ - | $ - | $ 584 |
| Cerium | $ 2.00 | $ 81.0 | $ - | $ - | $ - | $ - | $ 1,945 |
| Praseodymium | $ 52.00 | $ 261.6 | $ - | $ - | $ - | $ - | $ 6,279 |
| Neodymium | $ 42.00 | $ 985.6 | $ - | $ - | $ - | $ - | $ 23,654 |
| Samarium | $ 2.11 | $ 9.6 | $ - | $ - | $ - | $ - | $ 231 |
| Europium | $ 34.00 | $ 37.9 | $ - | $ - | $ - | $ - | $ 909 |
| Gadolinium | $ 32.00 | $ 178.1 | $ - | $ - | $ - | $ - | $ 4,274 |
| Terbium | $ 400.00 | $ 131.6 | $ - | $ - | $ - | $ - | $ 3,159 |
| Dysprosium | $ 230.00 | $ 1,126.1 | $ - | $ - | $ - | $ - | $ 27,026 |
| Holmium | $ 52.96 | $ 36.0 | $ - | $ - | $ - | $ - | $ 864 |
| Erbium | $ 34.00 | $ 76.1 | $ - | $ - | $ - | $ - | $ 1,825 |
| Thulium | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Ytterbium | $ 28.72 | $ 53.9 | $ - | $ - | $ - | $ - | $ 1,294 |
| Lutetium | $ 796.64 | $ 240.8 | $ - | $ - | $ - | $ - | $ 5,780 |
| Manganese | $ 0.005 | $ - | $ - | $ 0.1 | $ - | $ - | $ 3 |
| Cobalt | $ 22.00 | $ - | $ - | $ - | $ 347.0 | $ - | $ 8,329 |
| Nickel | $ 18.00 | $ - | $ - | $ - | $ 540.0 | $ - | $ 12,961 |
| Zinc | $ 3.02 | $ - | $ - | $ - | $ 66.4 | $ - | $ 1,594 |
| Reserved | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Total: | | $ 3,447.0 | $ 82,594.6 | $ 0.1 | $ 953.5 | $ - | $ 2,087,885 |

[1] REEs listed as $/kg REO; Mn listed as $/kg 44% Mn Ore; Co listed as $/kg metal content

## Coal Revenue

List Coal Revenue Sources Here:

**Annual Coal Revenue:** $ -

## Other Revenue

List Other Revenue Sources Here:

**Annual Other Revenue:** $ -

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C1 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 766,580,128 | $ 789,577,532 | $ 813,264,858 | $ 837,662,804 | $ 862,792,688 | $ 888,676,469 | $ 915,336,763 | $ 942,796,866 | $ 971,080,772 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 766,580,128 | $ 789,577,532 | $ 813,264,858 | $ 837,662,804 | $ 862,792,688 | $ 888,676,469 | $ 915,336,763 | $ 942,796,866 | $ 971,080,772 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 304,278 | $ 2,172,587 | $ 398,425,643 | $ 410,036,506 | $ 421,988,124 | $ 434,290,264 | $ 446,952,962 | $ 459,986,524 | $ 473,401,534 | $ 487,208,862 | $ 501,419,670 |
| *- Operating Costs* | $ - | $ - | $ - | $ 395,441,280 | $ 407,304,519 | $ 419,523,654 | $ 432,109,364 | $ 445,072,645 | $ 458,424,824 | $ 472,177,569 | $ 486,342,896 | $ 500,933,183 |
| *- Loan #1 Interest* | $ - | $ 304,278 | $ 281,193 | $ 256,723 | $ 230,785 | $ 203,290 | $ 174,146 | $ 143,253 | $ 110,507 | $ 75,796 | $ 39,002 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 1,891,394 | $ 1,747,897 | $ 1,595,791 | $ 1,434,559 | $ 1,263,653 | $ 1,082,492 | $ 890,462 | $ 686,910 | $ 471,145 | $ 242,434 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 979,742 | $ 905,411 | $ 826,620 | $ 743,102 | $ 654,572 | $ 560,731 | $ 461,259 | $ 355,819 | $ 244,053 |
| Gross Profit | $ - | $ (304,278) | $ (2,172,587) | $ 368,154,486 | $ 379,541,026 | $ 391,276,735 | $ 403,372,540 | $ 415,839,726 | $ 428,689,945 | $ 441,935,229 | $ 455,588,004 | $ 469,661,102 |
| **Book Deductions** | $ - | $ 380,348 | $ 2,744,920 | $ 3,973,686 | $ 3,675,660 | $ 3,399,985 | $ 3,144,986 | $ 2,909,112 | $ 2,690,929 | $ 2,489,109 | $ 2,302,426 | $ 2,129,744 |
| *- CAPEX #1 Depreciation* | $ - | $ 380,348 | $ 351,822 | $ 325,435 | $ 301,027 | $ 278,450 | $ 257,567 | $ 238,249 | $ 220,380 | $ 203,852 | $ 188,563 | $ 174,421 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 2,393,098 | $ 2,213,616 | $ 2,047,595 | $ 1,894,025 | $ 1,751,973 | $ 1,620,575 | $ 1,499,032 | $ 1,386,605 | $ 1,282,609 | $ 1,186,414 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,434,635 | $ 1,327,037 | $ 1,227,510 | $ 1,135,446 | $ 1,050,288 | $ 971,516 | $ 898,653 | $ 831,254 | $ 768,910 |
| Taxable Before Depletion | $ - | $ (684,626) | $ (4,917,507) | $ 364,180,800 | $ 375,865,367 | $ 387,876,750 | $ 400,227,553 | $ 412,930,613 | $ 425,999,016 | $ 439,446,120 | $ 453,285,578 | $ 467,531,358 |
| **Depletion** | $ - | $ - | $ - | $ 107,321,218 | $ 110,540,855 | $ 113,857,080 | $ 117,272,793 | $ 120,790,976 | $ 124,414,706 | $ 128,147,147 | $ 131,991,561 | $ 135,951,308 |
| *- 50% Limit* | $ - | $ (342,313) | $ (2,458,753) | $ 182,090,400 | $ 187,932,683 | $ 193,938,375 | $ 200,113,777 | $ 206,465,307 | $ 212,999,508 | $ 219,723,060 | $ 226,642,789 | $ 233,765,679 |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 107,321,218 | $ 110,540,855 | $ 113,857,080 | $ 117,272,793 | $ 120,790,976 | $ 124,414,706 | $ 128,147,147 | $ 131,991,561 | $ 135,951,308 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (684,626) | $ (4,917,507) | $ 256,859,582 | $ 265,324,512 | $ 274,019,669 | $ 282,954,761 | $ 292,139,637 | $ 301,584,310 | $ 311,298,973 | $ 321,294,017 | $ 331,580,050 |
| *- Income Tax* | $ - | $ (178,003) | $ (1,278,552) | $ 66,783,491 | $ 68,984,373 | $ 71,245,114 | $ 73,568,238 | $ 75,956,306 | $ 78,411,921 | $ 80,937,733 | $ 83,536,444 | $ 86,210,813 |
| Net Profit/Loss | $ - | $ (506,623) | $ (3,638,955) | $ 190,076,091 | $ 196,340,139 | $ 202,774,555 | $ 209,386,523 | $ 216,183,331 | $ 223,172,390 | $ 230,361,240 | $ 237,757,573 | $ 245,369,237 |
| **Book Deductions** | $ - | $ 380,348 | $ 2,744,920 | $ 111,294,904 | $ 114,216,514 | $ 117,257,065 | $ 120,417,779 | $ 123,700,089 | $ 127,105,634 | $ 130,636,256 | $ 134,293,987 | $ 138,081,052 |
| *+ CAPEX #1 Depreciation* | $ - | $ 380,348 | $ 351,822 | $ 325,435 | $ 301,027 | $ 278,450 | $ 257,567 | $ 238,249 | $ 220,380 | $ 203,852 | $ 188,563 | $ 174,421 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 2,393,098 | $ 2,213,616 | $ 2,047,595 | $ 1,894,025 | $ 1,751,973 | $ 1,620,575 | $ 1,499,032 | $ 1,386,605 | $ 1,282,609 | $ 1,186,414 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,434,635 | $ 1,327,037 | $ 1,227,510 | $ 1,135,446 | $ 1,050,288 | $ 971,516 | $ 898,653 | $ 831,254 | $ 768,910 |
| *+ Depletion* | $ - | $ - | $ - | $ 107,321,218 | $ 110,540,855 | $ 113,857,080 | $ 117,272,793 | $ 120,790,976 | $ 124,414,706 | $ 128,147,147 | $ 131,991,561 | $ 135,951,308 |
| Cash Flow | $ - | $ (126,275) | $ (894,035) | $ 301,370,995 | $ 310,556,653 | $ 320,031,621 | $ 329,804,302 | $ 339,883,420 | $ 350,278,024 | $ 360,997,496 | $ 372,051,560 | $ 383,450,289 |
| **Captial Cost** | $ 5,071,304 | $ 31,907,975 | $ 19,128,468 | $ 4,206,254 | $ 4,458,629 | $ 4,726,147 | $ 5,009,715 | $ 5,310,298 | $ 5,628,916 | $ 5,966,651 | $ 6,324,650 | $ 6,015,102 |
| *- Fixed Capital Costs* | $ 10,142,608 | $ 63,046,452 | $ 32,658,062 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 5,071,304 | $ 31,523,226 | $ 16,329,031 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 384,749 | $ 407,834 | $ 432,305 | $ 458,243 | $ 485,737 | $ 514,882 | $ 545,774 | $ 578,521 | $ 613,232 | $ 650,026 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 2,391,603 | $ 2,535,099 | $ 2,687,205 | $ 2,848,437 | $ 3,019,343 | $ 3,200,504 | $ 3,392,534 | $ 3,596,086 | $ 3,811,851 | $ 4,040,563 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 1,238,850 | $ 1,313,181 | $ 1,391,972 | $ 1,475,490 | $ 1,564,020 | $ 1,657,861 | $ 1,757,333 | $ 1,862,773 | $ 1,974,539 |
| **Net Cash Flow** | $ (5,071,304) | $ (32,034,251) | $ (20,022,503) | $ 297,164,741 | $ 306,098,024 | $ 315,305,474 | $ 324,794,586 | $ 334,573,122 | $ 344,649,108 | $ 355,030,845 | $ 365,726,909 | $ 377,435,188 |
| Discounted Cash Flow | $ (5,071,304) | $ (29,122,046) | $ (16,547,523) | $ 223,264,268 | $ 209,069,069 | $ 195,779,892 | $ 183,338,077 | $ 171,688,914 | $ 160,781,352 | $ 150,567,736 | $ 141,003,556 | $ 132,288,731 |
| **Cumulative DCF** | $ (5,071,304) | $ (34,193,350) | $ (50,740,874) | $ 172,523,395 | $ 381,592,464 | $ 577,372,356 | $ 760,710,432 | $ 932,399,346 | $ 1,093,180,698 | $ 1,243,748,434 | $ 1,384,751,990 | $ 1,517,040,721 |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 101,426,081 | Payback Period | 3.5 |
| Constant Dollar OPEX | $ 361,884,789 | Internal Rate of Return | 192% |
| Constant Dollar REE Rev | $ 701,529,411 | Net Present Value | $ 2,534,795,739 |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ 1,826,652,752 |

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C1 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 1,000,213,195** | **$ 1,030,219,591** | **$ 1,061,126,178** | **$ 1,092,959,964** | **$ 1,125,748,763** | **$ 1,159,521,226** | **$ 1,194,306,862** | **$ 1,230,136,068** | **$ 1,267,040,150** | **$ 1,305,051,355** | **$ 1,344,202,895** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 1,000,213,195 | $ 1,030,219,591 | $ 1,061,126,178 | $ 1,092,959,964 | $ 1,125,748,763 | $ 1,159,521,226 | $ 1,194,306,862 | $ 1,230,136,068 | $ 1,267,040,150 | $ 1,305,051,355 | $ 1,344,202,895 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 516,086,759** | **$ 531,440,014** | **$ 547,383,214** | **$ 563,804,710** | **$ 580,718,852** | **$ 598,140,417** | **$ 616,084,630** | **$ 634,567,169** | **$ 653,604,184** | **$ 673,212,309** | **$ 693,408,679** |
| *- Operating Costs* | $ 515,961,178 | $ 531,440,014 | $ 547,383,214 | $ 563,804,710 | $ 580,718,852 | $ 598,140,417 | $ 616,084,630 | $ 634,567,169 | $ 653,604,184 | $ 673,212,309 | $ 693,408,679 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 125,581 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ 484,126,436 | $ 498,779,577 | $ 513,742,965 | $ 529,155,253 | $ 545,029,911 | $ 561,380,808 | $ 578,222,233 | $ 595,568,900 | $ 613,435,967 | $ 631,839,046 | $ 650,794,217 |
| **Book Deductions** | **$ 1,970,013** | **$ 1,822,262** | **$ 1,685,593** | **$ 1,559,173** | **$ 1,442,235** | **$ 1,334,067** | **$ 1,234,012** | **$ 1,141,462** | **$ 1,055,852** | **$ 896,677** | **$ 326,163** |
| *- CAPEX #1 Depreciation* | $ 161,339 | $ 149,239 | $ 138,046 | $ 127,692 | $ 118,116 | $ 109,257 | $ 101,063 | $ 93,483 | $ 86,472 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 1,097,433 | $ 1,015,125 | $ 938,991 | $ 868,566 | $ 803,424 | $ 743,167 | $ 687,430 | $ 635,872 | $ 588,182 | $ 544,068 | $ - |
| *- CAPEX #3 Depreciation* | $ 711,241 | $ 657,898 | $ 608,556 | $ 562,914 | $ 520,696 | $ 481,644 | $ 445,520 | $ 412,106 | $ 381,198 | $ 352,608 | $ 326,163 |
| Taxable Before Depletion | $ 482,156,423 | $ 496,957,315 | $ 512,057,372 | $ 527,596,080 | $ 543,587,676 | $ 560,046,741 | $ 576,988,220 | $ 594,427,438 | $ 612,380,115 | $ 630,942,369 | $ 650,468,054 |
| **Depletion** | $ 140,029,847 | $ 144,230,743 | $ 148,557,665 | $ 153,014,395 | $ 157,604,827 | $ 162,332,972 | $ 167,202,961 | $ 172,219,050 | $ 177,385,621 | $ 182,707,190 | $ 188,188,405 |
| *- 50% Limit* | $ 241,078,211 | $ 248,478,657 | $ 256,028,686 | $ 263,798,040 | $ 271,793,838 | $ 280,023,370 | $ 288,494,110 | $ 297,213,719 | $ 306,190,057 | $ 315,471,184 | $ 325,234,027 |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 140,029,847 | $ 144,230,743 | $ 148,557,665 | $ 153,014,395 | $ 157,604,827 | $ 162,332,972 | $ 167,202,961 | $ 172,219,050 | $ 177,385,621 | $ 182,707,190 | $ 188,188,405 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ 342,126,576** | **$ 352,726,572** | **$ 363,499,707** | **$ 374,581,685** | **$ 385,982,849** | **$ 397,713,769** | **$ 409,785,259** | **$ 422,208,389** | **$ 434,994,494** | **$ 448,235,179** | **$ 462,279,649** |
| *- Income Tax* | $ 88,952,910 | $ 91,708,909 | $ 94,509,924 | $ 97,391,238 | $ 100,355,541 | $ 103,405,580 | $ 106,544,167 | $ 109,774,181 | $ 113,098,568 | $ 116,541,147 | $ 120,192,709 |
| Net Profit/Loss | $ 253,173,666 | $ 261,017,663 | $ 268,989,783 | $ 277,190,447 | $ 285,627,308 | $ 294,308,189 | $ 303,241,092 | $ 312,434,208 | $ 321,895,925 | $ 331,694,033 | $ 342,086,940 |
| **Book Deductions** | **$ 141,999,861** | **$ 146,053,005** | **$ 150,243,258** | **$ 154,573,568** | **$ 159,047,062** | **$ 163,667,039** | **$ 168,436,973** | **$ 173,360,511** | **$ 178,441,473** | **$ 183,603,866** | **$ 188,514,568** |
| *+ CAPEX #1 Depreciation* | $ 161,339 | $ 149,239 | $ 138,046 | $ 127,692 | $ 118,116 | $ 109,257 | $ 101,063 | $ 93,483 | $ 86,472 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 1,097,433 | $ 1,015,125 | $ 938,991 | $ 868,566 | $ 803,424 | $ 743,167 | $ 687,430 | $ 635,872 | $ 588,182 | $ 544,068 | $ - |
| *+ CAPEX #3 Depreciation* | $ 711,241 | $ 657,898 | $ 608,556 | $ 562,914 | $ 520,696 | $ 481,644 | $ 445,520 | $ 412,106 | $ 381,198 | $ 352,608 | $ 326,163 |
| *+ Depletion* | $ 140,029,847 | $ 144,230,743 | $ 148,557,665 | $ 153,014,395 | $ 157,604,827 | $ 162,332,972 | $ 167,202,961 | $ 172,219,050 | $ 177,385,621 | $ 182,707,190 | $ 188,188,405 |
| Cash Flow | $ 395,173,526 | $ 407,070,668 | $ 419,233,041 | $ 431,764,015 | $ 444,674,370 | $ 457,975,228 | $ 471,678,065 | $ 485,794,719 | $ 500,337,398 | $ 515,297,899 | $ 530,601,508 |
| **Captial Cost** | **$ 2,093,011** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 2,093,011 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ 393,080,515 | $ 407,070,668 | $ 419,233,041 | $ 431,764,015 | $ 444,674,370 | $ 457,975,228 | $ 471,678,065 | $ 485,794,719 | $ 500,337,398 | $ 515,297,899 | $ 530,601,508 |
| Discounted Cash Flow | $ 125,247,566 | $ 117,913,873 | $ 110,397,162 | $ 103,360,872 | $ 96,774,099 | $ 90,607,957 | $ 84,835,446 | $ 79,431,318 | $ 74,371,966 | $ 69,632,499 | $ 65,182,259 |
| **Cumulative DCF** | $ 1,642,288,286 | $ 1,760,202,159 | $ 1,870,599,322 | $ 1,973,960,194 | $ 2,070,734,293 | $ 2,161,342,250 | $ 2,246,177,696 | $ 2,325,609,015 | $ 2,399,980,981 | $ 2,469,613,480 | $ 2,534,795,739 |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 101,426,081 | Payback Period | 0 | 3.5 |
| Constant Dollar OPEX | $ 361,884,789 | Internal Rate of Return | 0 | 192% |
| Constant Dollar REE Rev | $ 701,529,411 | Net Present Value | 0 | $ 2,534,795,739 |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ 1,826,652,752 |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 0.09 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 0.05 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.00 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.00 | $ 0.00 |
| 7 | Conveyor | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 0.15** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.03 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.03** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 611** | **$ 1,834** | **$ 616,370** | **$ 0.19** | **$ 0.15** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 6.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 500 | tph |
| Residue Rate | 384 | tph |
| Slurry Flow | 9250 | gpm |
| Leaching Time | 45 | min |
| Acid Consumption | 194.92 | kg/ton |
| Bioleaching Sol'n Consumption | 0 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 2 | -- |
| Thickener Capacity | 9.03 | ft^2/tph |
| Polymer Dose | 0.016 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 24 | 100% | 17,344 | gal | $ 97,795 | $ 2,347,070 | $ - | |
| 2 | Tank Mixer | 24 | 100% | 26 | HP | $ 100,658 | $ 2,415,786 | $ - | |
| 3 | Process Pump | 12 | 100% | 1,542 | gpm | $ 106,518 | $ 1,278,221 | $ - | |
| 4 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 5 | Belt Filter | 2 | 100% | 192 | TPH | $ 1,159,434 | $ 2,318,867 | $ - | |
| 6 | Solution Heater | 1 | 100% | 1 | MBTU/hr | $ 43,673 | $ 43,673 | $ - | |
| 7 | Thickener | 6 | 100% | 578 | ft^2 | $ 69,230 | $ 415,378 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 8,914,552** | **$ -** | |

## Power/Utilities

Power Cost: $ 0.07 $/kWhr
Efficency: 0.85

| ID | Unit Operation | Total HP | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 624.38 | $ 299.30 | $ 897.89 | $ 301,691 | $ 0.09 | $ 0.07 |
| 3 | Process Pump | 462.50 | $ 221.70 | $ 665.10 | $ 223,475 | $ 0.07 | $ 0.06 |
| 4 | Conveyor | 31.00 | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.00 | $ 0.00 |
| 5 | Belt Filter | 93.08 | $ 44.62 | $ 133.86 | $ 44,977 | $ 0.01 | $ 0.01 |
| 6 | Solution Heater | 112.49 | $ 53.92 | $ 161.77 | $ 54,354 | $ 0.02 | $ 0.01 |
| 7 | Thickener | 2.20 | $ 1.06 | $ 3.17 | $ 1,065 | $ 0.00 | $ 0.00 |
| 8 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1326** | **$ 635.46** | **$ 1,906.37** | **$ 640,540** | **$ 0.19** | **$ 0.16** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | - | L/hr | $ 0.008 | / L | $ - | $ - | $ - | $ - | $ - |
| 2 | H2SO4 | 97.46 | mt/hr | $ 220.000 | / mt | $ 171,526.54 | $ 514,579.61 | $ 172,898,748 | $ 52.00 | $ 42.88 |
| 3 | Natural Gas | 1.08 | MCF/hr | $ 2.860 | / MCF | $ 24.82 | $ 74.46 | $ 25,018 | $ 0.01 | $ 0.01 |
| 4 | Polymer | 33.61 | kg/hr | $ 2.250 | / kg | $ 605.06 | $ 1,815.18 | $ 609,901 | $ 0.18 | $ 0.15 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 172,156.42** | **$ 516,469.25** | **$ 173,533,667** | **$ 52.19** | **$ 43.04** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| | **Total Operating Cost:** | **$ 172,792** | **$ 518,376** | **$ 174,174,207** | **$ 52.39** | **$ 43.20** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 6.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8292 | gpm |
| Feed Rate | 31388 | lpm |
| Precip Mass Flow | 9.20 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 887.99 | ft^2/tpd |
| NaOH Dose | 20.39 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 6 | 100% | 62,188 | gal | $ 133,694 | $ 802,162 | $ - | |
| 2 | Tank Mixer | 6 | 100% | 31 | HP | $ 111,308 | $ 667,847 | $ - | |
| 3 | Process Pump | 18 | 100% | 1,382 | gpm | $ 51,083 | $ 919,502 | $ - | |
| 4 | Filter Press | 1 | 100% | 43 | ft^3 | $ 137,826 | $ 137,826 | $ - | |
| 5 | Conveyor | 1 | 100% | 9 | TPH | $ 10,653 | $ 10,653 | $ - | |
| 6 | Thickener | 6 | 100% | 1,362 | ft^2 | $ 137,709 | $ 826,254 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 3,364,244** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 186.56 | Effiicency: | 0.85 | | | $ 89.43 | $ 268.29 | $ 90,146 | $ 0.03 | $ 0.02 |
| 3 | Process Pump | 621.88 | | | | | $ 298.10 | $ 894.30 | $ 300,486 | $ 0.09 | $ 0.07 |
| 4 | Filter Press | 7.28 | | | | | $ 3.49 | $ 10.47 | $ 3,516 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 0.57 | | | | | $ 0.27 | $ 0.82 | $ 276 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 5.20 | | | | | $ 2.49 | $ 7.47 | $ 2,510 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **821** | | | | | **$ 393.78** | **$ 1,181.35** | **$ 396,934** | **$ 0.12** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 38.39 | mt/hr | $ 125.000 | / mt | $ 38,390.82 | $ 115,172.47 | $ 38,697,949 | $ 11.64 | $ 9.60 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 38,390.82** | **$ 115,172.47** | **$ 38,697,949** | **$ 11.64** | **$ 9.60** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 38,785** | **$ 116,354** | **$ 39,094,884** | **$ 11.76** | **$ 9.70** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 6.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8263 | gpm |
| Feed Rate | 31278 | lpm |
| Precip Mass Flow | 8.00 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 4928.65 | ft^2/tpd |
| NaOH Dose | 3.66 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 6 | 100% | 61,971 | gal | $ 133,431 | $ 800,584 | $ - | |
| 2 | Tank Mixer | 6 | 100% | 31 | HP | $ 111,088 | $ 666,529 | $ - | |
| 3 | Process Pump | 18 | 100% | 1,377 | gpm | $ 51,015 | $ 918,275 | $ - | |
| 4 | Filter Press | 1 | 100% | 38 | ft^3 | $ 124,626 | $ 124,626 | $ - | |
| 5 | Conveyor | 1 | 100% | 8 | TPH | $ 9,863 | $ 9,863 | $ - | |
| 6 | Thickener | 6 | 100% | 6,568 | ft^2 | $ 486,690 | $ 2,920,140 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 5,440,016** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 185.91 | Effiicency: | 0.85 | | | $ 89.12 | $ 267.35 | $ 89,831 | $ 0.03 | $ 0.02 |
| 3 | Process Pump | 619.71 | | | | | $ 297.06 | $ 891.18 | $ 299,436 | $ 0.09 | $ 0.07 |
| 4 | Filter Press | 6.33 | | | | | $ 3.03 | $ 9.10 | $ 3,057 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 0.50 | | | | | $ 0.24 | $ 0.71 | $ 240 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 25.06 | | | | | $ 12.01 | $ 36.04 | $ 12,110 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **838** | | | | | **$ 401.46** | **$ 1,204.39** | **$ 404,674** | **$ 0.12** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 6.87 | mt/hr | $ 125.000 | / mt | $ 6,867.21 | $ 20,601.63 | $ 6,922,149 | $ 2.08 | $ 1.72 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 6,867.21** | **$ 20,601.63** | **$ 6,922,149** | **$ 2.08** | **$ 1.72** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 7,269** | **$ 21,806** | **$ 7,326,823** | **$ 2.20** | **$ 1.82** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 8.00 | tph |
| Slurry Feed Rate | 275 | gpm |
| Leaching Time | 0 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 1473.42 | kg/ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | - | gal | $ - | $ - | $ - | |
| 2 | Tank Mixer | 4 | 100% | - | HP | $ - | $ - | $ - | |
| 3 | Process Pump | 1 | 100% | 275 | gpm | $ 55,224 | $ 55,224 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 55,224** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | - | Effiicency: | 0.85 | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Process Pump | 6.88 | | | | | $ 3.30 | $ 9.90 | $ 3,327 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **7** | | | | | **$ 3.30** | **$ 9.90** | **$ 3,327** | **$ 0.00** | **$ 0.00** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 11.78 | mt/hr | $ 270.000 | / mt | $ 25,446.77 | $ 76,340.31 | $ 25,650,345 | $ 7.71 | $ 6.36 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 25,446.77** | **$ 76,340.31** | **$ 25,650,345** | **$ 7.71** | **$ 6.36** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 25,450** **$ 76,350** **$ 25,653,672** **$ 7.72** **$ 6.36**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 372 | gpm |
| Feed Rate | 1408 | lpm |
| Precip Mass Flow | 0.05 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 0.00 | g/L of Feed |
| Oxalic Acid Dose | 24.98 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.03 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 1,860 | gal | $ 2,638 | $ 10,553 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 1 | HP | $ 15,375 | $ 61,502 | $ - | |
| 3 | Process Pump | 3 | 100% | 372 | gpm | $ 61,929 | $ 185,787 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 0.26 | ft^3 | $ 3,431 | $ 3,431 | $ - | |
| 6 | Roaster | 1 | 100% | 0.08 | MBTU/hr | $ 272,950 | $ 272,950 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 534,222** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 3.72 | Effiicency: | 0.85 | | | $ 1.78 | $ 5.35 | $ 1,797 | $ 0.00 | $ 0.00 |
| 3 | Process Pump | 27.89 | | | | | $ 13.37 | $ 40.11 | $ 13,478 | $ 0.00 | $ 0.00 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.04 | | | | | $ 0.02 | $ 0.06 | $ 21 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.01 | | | | | $ 0.01 | $ 0.02 | $ 6 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **32** | | | | | **$ 15.18** | **$ 45.54** | **$ 15,302** | **$ 0.00** | **$ 0.00** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 2 | Oxalic Acid | 2,109.88 | kg/hr | $ 0.600 | / kg | $ 10,127.41 | $ 30,382.23 | $ 10,208,431 | $ 3.07 | $ 2.53 |
| 3 | Natural Gas | 0.08 | MCF/hr | $ 2.860 | / MCF | $ 1.86 | $ 5.58 | $ 1,874 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 10,129.27** | **$ 30,387.81** | **$ 10,210,304** | **$ 3.07** | **$ 2.53** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 10,144** **$ 30,433** **$ 10,225,606** **$ 3.08** **$ 2.54**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8292 | gpm |
| Raffinate Rate: | 8292 | gpm |
| Extraction Feed (O+A) | 16584 | gpm |
| Scrub Feed (O+A) | 228 | gpm |
| Strip Feed (O+A) | 456 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 228 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 272632 | gal |
| Precip Mixing Intensity | 1.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 30.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,584 | gal | $ 95,360 | $ 190,720 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 41 | HP | $ 130,915 | $ 261,831 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 248,753 | gal | $ 437,322 | $ 874,643 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,292 | gpm | $ 202,347 | $ 404,694 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 228 | gal | $ 8,556 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,666 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,420 | gal | $ 39,240 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 456 | gal | $ 12,635 | $ 25,271 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 17,247 | $ 34,494 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,839 | gal | $ 57,946 | $ 115,892 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,588 | $ 19,588 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 59,341 | kg | $ 3,715,315 | $ - | $ 3,715,315 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 1,029,443 | kg | $ 1,114,728 | $ - | $ 1,114,728 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,728 | gal | $ 5,693 | $ 22,771 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 4.09 | HP | $ 35,462 | $ 141,849 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,588 | $ 39,175 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.01 | ft^3 | $ 279 | $ 279 | $ - | |
| 20 | Process Pump | 2 | 100% | 228 | gpm | $ 51,383 | $ 102,766 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,233,975** | **$ 4,830,043** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 82.92 | Effiicency: | 0.85 | | $ 39.75 | $ 119.24 | $ 40,065 | $ 0.01 | $ 0.01 |
| 3 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 414.59 | | | | $ 198.73 | $ 596.20 | $ 200,324 | $ 0.06 | $ 0.05 |
| 5 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.28 | | | | $ 1.09 | $ 3.28 | $ 1,102 | $ 0.00 | $ 0.00 |
| 11 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.45 | | | | $ 0.22 | $ 0.65 | $ 220 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 16.37 | | | | $ 7.85 | $ 23.54 | $ 7,908 | $ 0.00 | $ 0.00 |
| 17 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.91 | | | | $ 0.44 | $ 1.31 | $ 439 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | | | | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.40 | | | | $ 5.46 | $ 16.39 | $ 5,508 | $ 0.00 | $ 0.00 |
| | **TOTAL:** | **529** | | | | **$ 253.54** | **$ 760.61** | **$ 255,566** | **$ 0.08** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 107.35 | kg/hr | $ 0.940 | / kg | $ 807.24 | $ 2,421.73 | $ 813,700 | $ 0.24 | $ 0.20 |
| 2 | D2EHPA | 5.65 | kg/hr | $ 5.000 | / kg | $ 225.99 | $ 677.98 | $ 227,800 | $ 0.07 | $ 0.06 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.87 | mt/hr | $ 270.000 | / mt | $ 1,880.21 | $ 5,640.64 | $ 1,895,255 | $ 0.57 | $ 0.47 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,913.45** | **$ 8,740.34** | **$ 2,936,755** | **$ 0.88** | **$ 0.73** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 3,167** | **$ 9,501** | **$ 3,192,321** | **$ 0.96** | **$ 0.79** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C1
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Co Feed Rate | 371.86 | gpm |
| Co Feed Rate | 1407.63 | lpm |
| Mn Feed Rate | 371.73 | gpm |
| Mn Feed Rate | 1407.15 | lpm |
| Co Precip Mass Flow | 0.11 | tph |
| Mn Precip Mass Flow | 0.17 | tph |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Co Precip Conditioning Time | 30.00 | min |
| Mn Precip Conditioning Time | 30.00 | min |
| Co Filter Press Cycle Time | 30.00 | min |
| Mn Filter Press Cycle Time | 30.00 | min |
| Co Thickener Capacity | 0.00 | ft^2/tph |
| Mn Thickener Capacity | 0.00 | ft^2/tph |
| Na2S Dose | 6.88 | g/L of Feed |
| NaOH Dose | 0.00 | g/L of Feed |

| | | |
|---|---|---|
| Co Solid Density | 3.40 | g/cc |
| Mn Solid Density | 3.40 | g/cc |
| HCL Dose | 58.96 | g/L of Feed |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 5,578 | gal | $ 7,484 | $ 14,967 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 8 | HP | $ 53,086 | $ 106,171 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 372 | gpm | $ 61,925 | $ 61,925 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 0.51 | ft^3 | $ 5,606 | $ 5,606 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 5,576 | gal | $ 7,481 | $ 14,963 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 8 | HP | $ 53,076 | $ 106,151 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 372 | gpm | $ 61,917 | $ 61,917 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 1 | ft^3 | $ 7,905 | $ 7,905 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 379,606** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 16.73 | Effiicency: | 0.85 | | | $ 8.02 | $ 24.06 | $ 8,085 | $ 0.00 | $ 0.00 |
| 3 | Process Pump | 9.30 | | | | | $ 4.46 | $ 13.37 | $ 4,492 | $ 0.00 | $ 0.00 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.09 | | | | | $ 0.04 | $ 0.12 | $ 41 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 16.73 | | | | | $ 8.02 | $ 24.06 | $ 8,083 | $ 0.00 | $ 0.00 |
| 8 | Process Pump | 9.29 | | | | | $ 4.45 | $ 13.36 | $ 4,490 | $ 0.00 | $ 0.00 |
| 9 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 0.14 | | | | | $ 0.07 | $ 0.20 | $ 66 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **52** | | | | | **$ 25.06** | **$ 75.17** | **$ 25,258** | **$ 0.01** | **$ 0.01** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 0.58 | mt/hr | $ 360.000 | / mt | $ 1,673.11 | $ 5,019.33 | $ 1,686,496 | $ 0.51 | $ 0.42 |
| 2 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 0.82 | $ 2.47 | $ 829 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 1,673.93** | **$ 5,021.80** | **$ 1,687,325** | **$ 0.51** | **$ 0.42** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 1,699** **$ 5,097** **$ 1,712,583** **$ 0.52** **$ 0.42**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | Shakespeare Lignite | Date: | 3/18/2023 |
| Case: | SHXP-C2 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | an |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | 110.1% |
| *(unit conversion)* | 12000 | TPD | Project NPV: | 800,345,392 |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 0.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C2 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 82,324,754 | | | | | |
| Constant Dollar Annual OPEX | $ | 592,715,503 | $ | 147.00 | /ton | $ 148.2 | /kg |
| Constant Dollar Annual REE Rev | $ | 698,280,043 | $ | 173.18 | /ton | $ 174.58 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | |
|---|---|---|---|---|
| Payback Period | 3.5 | years | *0.5* | *operating years* |
| Internal Rate of Return | 110% | | | |
| Net Present Value | $ 800,345,392 | | | |
| Total Severence Tax Paid | $ - | | | |
| Total State and Federal Tax Paid | $ 395,676,533 | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C2 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 1,250 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 2.39 | 29.84 | 13.54 | | | | | | | 13.54 |
| Yttrium | 19.38 | 242.23 | 109.87 | | | | | | | 109.87 |
| Lanthanum | 14.92 | 186.47 | 84.58 | | | | | | | 84.58 |
| Cerium | 29.52 | 369.01 | 167.38 | | | | | | | 167.38 |
| Praseodymium | 3.72 | 46.50 | 21.09 | | | | | | | 21.09 |
| Neodymium | 14.47 | 180.92 | 82.06 | | | | | | | 82.06 |
| Samarium | 3.22 | 40.26 | 18.26 | | | | | | | 18.26 |
| Europium | 0.72 | 9.02 | 4.09 | | | | | | | 4.09 |
| Gadolinium | 3.59 | 44.88 | 20.36 | | | | | | | 20.36 |
| Terbium | 0.37 | 4.63 | 2.10 | | | | | | | 2.10 |
| Dysprosium | 3.35 | 41.87 | 18.99 | | | | | | | 18.99 |
| Holmium | 0.59 | 7.40 | 3.36 | | | | | | | 3.36 |
| Erbium | 1.52 | 18.97 | 8.61 | | | | | | | 8.61 |
| Thulium | 0.24 | 3.01 | 1.36 | | | | | | | 1.36 |
| Ytterbium | 1.68 | 21.05 | 9.55 | | | | | | | 9.55 |
| Lutetium | 0.31 | 3.93 | 1.78 | | | | | | | 1.78 |
| Manganese | -- | 637.00 | 288.94 | | | 0.00 | -- | | | 288.94 |
| Cobalt | -- | 304.30 | 138.03 | | | 0.00 | -- | | | 138.03 |
| Nickel | -- | 760.00 | 344.73 | | | 0.00 | -- | | | 344.73 |
| Zinc | -- | 234.70 | 106.46 | | | 0.00 | -- | | | 106.46 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 1250.00 | 566.99 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 567.0 |
| Total non-REE CM | -- | 1936.00 | 878.16 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 878.2 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 13.54 | 0.01 | 11.73 | 0.01 | 0.02 | 0.00 | 11.77 | 1.77 | 86.9% | |
| Yttrium | 109.87 | 76.22 | 0.00 | 1.24 | 24.23 | 0.00 | 101.70 | 8.18 | 92.6% | |
| Lanthanum | 84.58 | 57.20 | 0.00 | 0.21 | 17.47 | 0.00 | 74.89 | 9.70 | 88.5% | |
| Cerium | 167.38 | 124.04 | 0.00 | 1.15 | 21.92 | 0.00 | 147.11 | 20.27 | 87.9% | |
| Praseodymium | 21.09 | 13.29 | 0.00 | 0.11 | 2.35 | 0.00 | 15.75 | 5.34 | 74.7% | |
| Neodymium | 82.06 | 67.85 | 0.00 | 0.30 | 0.97 | 0.00 | 69.13 | 12.94 | 84.2% | |
| Samarium | 18.26 | 15.39 | 0.00 | 0.10 | 0.06 | 0.00 | 15.55 | 2.71 | 85.2% | |
| Europium | 4.09 | 3.08 | 0.00 | 0.00 | 0.02 | 0.00 | 3.10 | 1.00 | 75.6% | |
| Gadolinium | 20.36 | 15.50 | 0.00 | 0.16 | 2.19 | 0.00 | 17.86 | 2.50 | 87.7% | |
| Terbium | 2.10 | 1.37 | 0.00 | 0.03 | 0.05 | 0.00 | 1.44 | 0.66 | 68.7% | |
| Dysprosium | 18.99 | 14.95 | 0.00 | 0.26 | 1.95 | 0.00 | 17.17 | 1.83 | 90.4% | |
| Holmium | 3.36 | 2.48 | 0.00 | 0.03 | 0.57 | 0.00 | 3.08 | 0.27 | 91.8% | |
| Erbium | 8.61 | 5.56 | 0.00 | 0.07 | 1.71 | 0.00 | 7.35 | 1.25 | 85.4% | |
| Thulium | 1.36 | 0.80 | 0.00 | 0.02 | 0.25 | 0.00 | 1.08 | 0.29 | 78.9% | |
| Ytterbium | 9.55 | 5.32 | 0.00 | 0.13 | 2.27 | 0.00 | 7.72 | 1.83 | 80.9% | |
| Lutetium | 1.78 | 0.95 | 0.00 | 0.01 | 0.35 | 0.00 | 1.32 | 0.46 | 73.9% | |
| Manganese | 288.94 | 1.03 | 0.00 | 95.89 | 173.50 | 0.00 | 270.42 | 18.52 | 93.6% | |
| Cobalt | 138.03 | 0.62 | 0.00 | 76.50 | 6.69 | 0.00 | 83.81 | 54.22 | 60.7% | |
| Nickel | 344.73 | 2.51 | 0.00 | 158.55 | 128.63 | 0.00 | 289.69 | 55.04 | 84.0% | |
| Zinc | 106.46 | 0.70 | 0.00 | 2.80 | 89.21 | 0.00 | 92.72 | 13.74 | 87.1% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 566.99 | 404.03 | 11.73 | 3.84 | 76.40 | 0.00 | 496.00 | 70.99 | 87.5% | <---- |
| Total non-REE CM | 878.16 | 4.86 | 0.00 | 333.74 | 398.03 | 0.00 | 736.63 | 141.52 | 83.9% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 440 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 9 | ton | $ | 1.50 |
| Dust/Volatiles | - | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | |
|---|---|
| 9.70 | TPD REE |
| 0.281 | TPD Sc |
| 17.56 | TPD CM |
| 0.71 | REE |
| 86.6% | Sc |
| 33.2% | Mn |
| 38.1% | Metal-S |
| 500 | Feed |
| 1,250 | Grade |
| 3.5 | PP |
| 110% | IRR |
| 800345391.8 | NPV |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C2 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 0.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 2.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | UK: Design |
| Dissolved Solids in Leach | 12.1% | % | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Leaching Retention Time | 45 | min | UK: Design |
| Thickener UF % Solids | 40% | % | UK: SHXP-C4/SHXP-C4 Samples |
| Residue Settling Rate | 15.0 | ft/hr | Not Available, Assumed |
| **Reagent Use** | | | |
| Acid Consumption | 345.0 | kg/ton | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Bioleaching Sol'n Consumption | 0 | L/ton | N/A |
| Polymer Dosage | 0.016 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 6.72 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 8.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | UK: Design |
| Precip Solid Yield | 4.43 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 2.24 | ft/hr | UK: Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 0.5% | % | VT Override to 2% \| UK: SHXP-C4/SHXP-C4 Samples 0.5% |
| **Reagent Use** | | | |
| NaOH Dose | 44.29 | g/L of Feed | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 398.04 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 1250 | ppm | |
| Grade After Sorter | 1250 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 500 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 0 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 500 | tph | |
| Leaching Residue Rate | 439.5 | tph | |
| Slurry Flow (feed) | 9250 | gpm | |
| Product Flow | 8,153 | gpm | |
| Thickener Feed % Solids (eff) | 18% | % | |
| Liquid Density | 1.011 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 30.20 | g/L | |
| Leaching Vessel Size | 52031.25 | gal | |
| Leaching Vessel Size | 196.96 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 219.75 | tph | 70 to 200 tph |
| Thickener Area | 1477.76 | ft^2 | |
| Thickener Diameter | 43.39 | ft | ideal < 100' |
| Liquid Balance | 151.03 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,153 | gpm | |
| Slurry Feed Rate | 30,862 | lpm | |
| Precip Mass Flow | 9.05 | tph | |
| Thickener Feed % Solids (eff) | 0.44% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.008 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 173.60 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 42.64 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 10.63 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 450 | ft^2 | |
| Thickener Diameter | 23.95 | ft | ideal < 100' |

*PM5: REE Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| | | | |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45 | min | UK: Based on Design |
| Precip Solid Yield | | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 0.044 | ft/hr | UK: Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.00% | % | UK: SHXP-C4/SHXP-C4 Samples |
| | | | |
| **Reagent Use** | | | |
| NaOH Dose | | g/L of Feed | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| | | | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 8142 | gpm | |
| Slurry Feed Rate | 30,822 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.008 | g/cc | |
| | | | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^3 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | #DIV/0! | m/m | target = 6.35 |

## *PM6: RE-Redissolution*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 5.0% | % | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Dissolved Solids in Re-Leach | 5% | % | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Leaching Retention Time | 5 | min | UK: Design |
| **Reagent Use** | | | |
| Acid Consumption | | kg/ton | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | - | tph | |
| Slurry Flow (feed) | - | gpm | |
| Product Flow | #DIV/0! | gpm | |
| Liquid Density | #DIV/0! | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | #DIV/0! | v/v | target = 3.8 |
| Liquid Balance | #DIV/0! | gpm | must be > 0; adjust slurry density |
| Slurry Balance | #DIV/0! | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

## *PM7: REE Oxalate Precipitation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | UK: Design |
| Precip Solid Yield | 0.8 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Thickener UF %Solids | | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 0.00 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Oxalic Acid Dose | 6.65 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

**Material Balance**

| | | | |
|---|---|---|---|
| Slurry Feed Rate | 8142.21 | gpm | |
| Slurry Feed Rate | 30821.63 | lpm | |
| Precip Mass Flow | 1.59 | tph | |
| Thickener Feed % Solids (eff) | 0.08% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.93 | tph | |
| Liquid Density | 1.007 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 154.11 | m^3 | |
| Filter Press Area | 7.47 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 1.86 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 0.54 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.71 | m/m | |

## *PM8: Sc SX Separation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | Based on Design |
| Settler Residence Time | 15.00 | min | Based on Design |
| Loading: Number of Stages | 2.00 | - | Based on Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | Based on Design |
| Loading O:A Advance Ratio | 0.03 | v/v | Based on Design |
| Scrubbing: Number of Stages | | - | Not Applicable |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | Not Applicable |
| Scrubbing O:A Advance Ratio | | v/v | Not Applicable |
| Stripping: Number of Stages | 2.00 | - | Based on Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | Based on Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | Based on Design |
| Sc2O3 Grade | 2% | % | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | Not Availabe |
| Precip Solid Yield | 0.4 | g/L | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance (SX)**

| | | | |
|---|---|---|---|
| PLS Flow | 8152.8 | gpm | |
| Raffinate Rate: | 8152.8 | gpm | |
| Load Org Recycle | 7928.7 | gpm | |
| Load Org Feed | 8152.8 | gpm | |
| Loag Org Advance | 224.2 | gpm | |
| Scrub Aq Product | 0.0 | gpm | |
| Scrub Aq Recycle | 0.0 | gpm | |
| Scrub Aq Feed | 0.0 | gpm | |
| Scurb Org Product | 224.2 | gpm | |
| Strip Aq Product | 17.9 | gpm | |
| Strip Aq Recycle | 206.3 | gpm | |
| Strip Aq Feed | 224.2 | gpm | |
| Strip Org Product | 224.2 | gpm | |
| Extraction Feed (O+A): | 16305.7 | gpm | |
| Scrub Feed (O+A): | 224.2 | gpm | |
| Strip Feed (O+A): | 448.3 | gpm | |

**Material Balance (Precip)**

| | | | |
|---|---|---|---|
| Strip Solution Flow | 17.9 | gpm | |
| Solid Density | 3.4 | g/cc | |
| Solid Product Rate | 0.00 | tph | |
| Liquid Density (after precip) | 1.13 | g/cc | |
| Thickener Feed % Solids (eff) | 0.03% | % | |

**Total Volumes**

| | | | |
|---|---|---|---|
| Total Extraction Volume | 521782 | gal | |
| Total Scrubbing Volume | 0 | gal | |
| Total Stripping Volume | 14346 | gal | |
| Extraction Org Volume | 260890.90 | gal | |
| Scrubbing Org Volume | 0.00 | gal | |
| Stripping Org Volume | 7172.83 | gal | |
| Total Org Volume | 268063.73 | gal | |
| **Other Indicators** | | | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 10.15 | m^3 | |
| Filter Press Area | 0.01 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

## *PM9: CM Precipitation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 0 | min | N/A |
| Precip Solid Yield | - | g/L | N/A |
| Precip Settlng Rate | 10 | ft/hr | N/A |
| Thickener UF % Solids | 25% | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 7.24 | g/L of Feed | N/A |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 30.00 | min | N/A |
| Thickener Capacity | 0.00 | ft^2/tph | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Slurry Feed Rate | 8142 | gpm | |
| Slurry Feed Rate | 30,822 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.008 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM10: CM Redissolution*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 2.0% | % | N/A |
| Dissolved Solids in Re-Leach | 100% | % | N/A |
| Leaching Retention Time | 45 | min | N/A |
| **Reagent Use** | | | |
| Acid Consumption | 441 | kg/ton | N/A |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 0.00 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 0 | gpm | |
| Liquid Density | 1.000 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | |
| Releach Conc. Factor | #DIV/0! | v/v | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | - | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

*PM11: CM Recovery*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 3.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | UK: Design |
| Co Precip Solid Yield | 2.02 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Co Precip Settling Rate | | ft/hr | N/A |
| Co Thickener UF % Solids | 0% | % | N/A |
| Mn Precip Conditioning Time | 30.00 | min | UK: Design |
| Mn Precip Solid Yield | 2.68 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Mn Precip Settling Rate | | ft/hr | N/A |
| Mn Thickener UF % Solids | 1% | % | N/A |
| **Reagent Use** | | | |
| Na2S Dose | 8.76 | g/L of Feed | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| NaOH Dose | 1.72 | g/L of Feed | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| HCL Dose | 3.12 | g/L of Feed | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 8140.35 | gpm | |
| Co Slurry Feed Rate | 30814.59 | lpm | |
| Co Precip Mass Flow | 4.12 | tph | |
| Co Thickener Feed % Solids (eff) | 0.20% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 8135.52 | gpm | |
| Mn Slurry Feed Rate | 30796.29 | lpm | |
| Mn Precip Mass Flow | 5.45 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.27% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.01 | g/cc | |
| Mn Liquid Density | 1.00 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 154.1 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 19.40 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 4.83 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 154.0 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 25.7 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 6.40 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

*PM12: Biooxidation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Requirec | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C2 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 6,414,323 | $ - | $ 1,603,581 | $ 7,055,756 | $ 8,784,416 | $ 23,858,076 | IIIIIIIIIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 3,342,824 | $ - | $ 835,706 | $ 3,677,107 | $ 4,577,998 | $ 12,433,636 | IIIIIIIIIIIII |
| PM5 | REE Precipitation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM6 | RE-Redissolution | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM7 | REE Oxalate Precipitation | $ 2,566,853 | $ - | $ 641,713 | $ 2,823,538 | $ 3,515,305 | $ 9,547,410 | IIIIIIIIII |
| PM8 | Sc SX Separation | $ 2,214,405 | $ 4,749,118 | $ 553,601 | $ 2,435,846 | $ 3,032,628 | $ 12,985,599 | IIIIIIIIIIIII |
| PM9 | CM Precipitation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM10 | CM Redissolution | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM11 | CM Recovery | $ 3,854,403 | $ - | $ 963,601 | $ 4,239,844 | $ 5,278,605 | $ 14,336,453 | IIIIIIIIIIIIII |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| | TOTAL | $ 20,856,469 | $ 4,749,118 | $ 5,214,117 | $ 22,942,116 | $ 28,562,934 | $ 82,324,754 | |

## Module Summary

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| Total Lang Factor (=1+sum % Base) | | 372% | |

Year of Construction: 2025

# OPERATING COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C2 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 0.77 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 0.30 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 82,324,754 | $ Total | $ 816.71 | $ 2,450.14 | $ 823,248 | $ 0.21 | $ 0.20 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 0.08 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 698,280,043 | $ Total | $ 3,463.69 | $ 10,391.07 | $ 3,491,400 | $ 0.87 | $ 0.87 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 0.12 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 82,324,754 | $ Total | $ 816.71 | $ 2,450.14 | $ 823,248 | $ 0.21 | $ 0.20 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 10,078.57** | **$ 30,235.71** | **$ 10,159,197** | **$ 2.54** | **$ 2.52** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 0.13 | $ 0.13 |
| PM2 | Roasting | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | PR | 1304 | | $ 625.23 | $ 1,875.70 | $ 630,237 | $ 0.16 | $ 0.16 |
| PM4 | Al Precipitation | PR | 805 | | $ 385.84 | $ 1,157.51 | $ 388,924 | $ 0.10 | $ 0.10 |
| PM5 | REE Precipitation | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | PR | 694 | | $ 332.53 | $ 997.60 | $ 335,193 | $ 0.08 | $ 0.08 |
| **Sub-Total** | | | **5550.4** | | **$ 2,660.59** | **$ 7,981.78** | **$ 2,681,876.75** | **$ 0.67** | **$ 0.67** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.03 | $ 0.03 |
| PM2 | Roasting | CR | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | CR | | $ 304,183.33 | $ 912,549.99 | $ 306,616,798 | $ 76.66 | $ 76.05 |
| PM4 | Al Precipitation | CR | | $ 82,021.09 | $ 246,063.27 | $ 82,677,259 | $ 20.67 | $ 20.51 |
| PM5 | REE Precipitation | CR | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | CR | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | CR | | $ 59,100.57 | $ 177,301.70 | $ 59,573,371 | $ 14.89 | $ 14.78 |
| **Sub-Total** | | | | **$ 498,114.91** | **$ 1,494,344.73** | **$ 502,099,830.25** | **$ 125.53** | **$ 124.53** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 10,548 | $ 0.10 | $ 351.60 | $ 1,054.80 | $ 354,413 | $ 0.09 | $ 0.09 |
| Non-Hazardous Precipitate | WD | Tons | 217 | $ 1.50 | $ 108.60 | $ 325.80 | $ 109,469 | $ 0.03 | $ 0.03 |
| Dust/Volatiles | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 460.20** | **$ 1,380.60** | **$ 463,881.52** | **$ 0.12** | **$ 0.12** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 501,236** | **$ 1,503,707** | **$ 505,245,589** | **$ 126** | **$ 125** |

## Overhead Costs

| Allocation | Code | Rate | Basis | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% | % of Direct Cost | $ 30,236 Daily Direct Cost | $ 1,511.79 | $ 4,535.36 | $ 1,523,880 | $ 0.38 | $ 0.38 |
| Variable Portion | OH | 15% | % of Direct Cost | $ 1,503,707 Daily Direct Cost | $ 75,185.36 | $ 225,556.07 | $ 75,786,838 | $ 18.95 | $ 18.80 |
| **TOTAL OVERHEAD COSTS** | | | | | **$ 76,697.14** | **$ 230,091.42** | **$ 77,310,717.81** | **$ 19.33** | **$ 19.17** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | SHXP-C2 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 10,078.57 | $ 30,235.71 | $ 10,159,197 | $ 3 | $ 2.52 | I |
| Power | $ 2,660.59 | $ 7,981.78 | $ 2,681,877 | $ 1 | $ 0.67 | |
| Consumables | $ 498,114.91 | $ 1,494,344.73 | $ 502,099,830 | $ 126 | $ 124.53 | IIII

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C2 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 698,280,043

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 1220 | 1250 |
| Contained Value | ($/ton) | $ 32.5 | $ 206.9 |
| Feed Basket Price | ($/kg) | $ 29.3 | $ 182.4 |
| Product Basket Price | ($/kg) | $ 28.2 | $ 180.3 |
| Production Rate | (kg/day) | 11621.51 | 11903.92 |
| | (t/day) | 11.62 | 11.90 |
| | (t/yr) | 3904.83 | 3999.72 |
| | (% of US Demand) | 39.0% | |

## Revenue Contribution Table

| Element | REE/CM Price ($/kg) [1] | Product Revenue ($/hr) MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | REE Revenue ($/day) | |
|---|---|---|---|---|---|---|---|---|
| Scandium | $ 4,200.00 | $ 71.0 | $ 75,535.7 | $ - | $ - | $ - | $ 1,814,560 | IIII

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C2 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 763,029,457 | $ 785,920,341 | $ 809,497,951 | $ 833,782,889 | $ 858,796,376 | $ 884,560,267 | $ 911,097,075 | $ 938,429,988 | $ 966,582,887 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 763,029,457 | $ 785,920,341 | $ 809,497,951 | $ 833,782,889 | $ 858,796,376 | $ 884,560,267 | $ 911,097,075 | $ 938,429,988 | $ 966,582,887 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 246,974 | $ 1,763,429 | $ 650,098,558 | $ 669,323,999 | $ 689,120,058 | $ 709,503,485 | $ 730,491,509 | $ 752,101,855 | $ 774,352,753 | $ 797,262,954 | $ 820,851,743 |
| *- Operating Costs* | $ - | $ - | $ - | $ 647,676,234 | $ 667,106,521 | $ 687,119,716 | $ 707,733,308 | $ 728,965,307 | $ 750,834,266 | $ 773,359,294 | $ 796,560,073 | $ 820,456,875 |
| *- Loan #1 Interest* | $ - | $ 246,974 | $ 228,237 | $ 208,375 | $ 187,322 | $ 165,005 | $ 141,350 | $ 116,275 | $ 89,695 | $ 61,521 | $ 31,656 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 1,535,192 | $ 1,418,720 | $ 1,295,260 | $ 1,164,392 | $ 1,025,672 | $ 878,629 | $ 722,764 | $ 557,546 | $ 382,415 | $ 196,777 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 795,229 | $ 734,897 | $ 670,945 | $ 603,155 | $ 531,298 | $ 455,130 | $ 374,392 | $ 288,809 | $ 198,091 |
| Gross Profit | $ - | $ (246,974) | $ (1,763,429) | $ 112,930,899 | $ 116,596,341 | $ 120,377,893 | $ 124,279,405 | $ 128,304,867 | $ 132,458,412 | $ 136,744,323 | $ 141,167,034 | $ 145,731,144 |
| **Book Deductions** | $ - | $ 308,718 | $ 2,227,976 | $ 3,225,331 | $ 2,983,432 | $ 2,759,674 | $ 2,552,699 | $ 2,361,246 | $ 2,184,153 | $ 2,020,341 | $ 1,868,816 | $ 1,728,655 |
| *- CAPEX #1 Depreciation* | $ - | $ 308,718 | $ 285,564 | $ 264,147 | $ 244,336 | $ 226,011 | $ 209,060 | $ 193,380 | $ 178,877 | $ 165,461 | $ 153,051 | $ 141,573 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 1,942,412 | $ 1,796,731 | $ 1,661,976 | $ 1,537,328 | $ 1,422,028 | $ 1,315,376 | $ 1,216,723 | $ 1,125,469 | $ 1,041,059 | $ 962,979 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,164,454 | $ 1,077,120 | $ 996,336 | $ 921,611 | $ 852,490 | $ 788,553 | $ 729,412 | $ 674,706 | $ 624,103 |
| Taxable Before Depletion | $ - | $ (555,692) | $ (3,991,405) | $ 109,705,567 | $ 113,612,910 | $ 117,618,219 | $ 121,726,706 | $ 125,943,621 | $ 130,274,260 | $ 134,723,981 | $ 139,298,218 | $ 144,002,490 |
| **Depletion** | $ - | $ - | $ - | $ 54,852,784 | $ 56,806,455 | $ 58,809,109 | $ 60,863,353 | $ 62,971,810 | $ 65,137,130 | $ 67,361,991 | $ 69,649,109 | $ 72,001,245 |
| *- 50% Limit* | $ - | $ (277,846) | $ (1,995,702) | $ 54,852,784 | $ 56,806,455 | $ 58,809,109 | $ 60,863,353 | $ 62,971,810 | $ 65,137,130 | $ 67,361,991 | $ 69,649,109 | $ 72,001,245 |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 106,824,124 | $ 110,028,848 | $ 113,329,713 | $ 116,729,605 | $ 120,231,493 | $ 123,838,437 | $ 127,553,591 | $ 131,380,198 | $ 135,321,604 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (555,692) | $ (3,991,405) | $ 54,852,784 | $ 56,806,455 | $ 58,809,109 | $ 60,863,353 | $ 62,971,810 | $ 65,137,130 | $ 67,361,991 | $ 69,649,109 | $ 72,001,245 |
| *- Income Tax* | $ - | $ (144,480) | $ (1,037,765) | $ 14,261,724 | $ 14,769,678 | $ 15,290,368 | $ 15,824,472 | $ 16,372,671 | $ 16,935,654 | $ 17,514,118 | $ 18,108,768 | $ 18,720,324 |
| Net Profit/Loss | $ - | $ (411,212) | $ (2,953,639) | $ 40,591,060 | $ 42,036,777 | $ 43,518,741 | $ 45,038,881 | $ 46,599,140 | $ 48,201,476 | $ 49,847,873 | $ 51,540,341 | $ 53,280,921 |
| **Book Deductions** | $ - | $ 308,718 | $ 2,227,976 | $ 58,078,115 | $ 59,789,886 | $ 61,568,783 | $ 63,416,052 | $ 65,333,057 | $ 67,321,283 | $ 69,382,332 | $ 71,517,925 | $ 73,729,899 |
| *+ CAPEX #1 Depreciation* | $ - | $ 308,718 | $ 285,564 | $ 264,147 | $ 244,336 | $ 226,011 | $ 209,060 | $ 193,380 | $ 178,877 | $ 165,461 | $ 153,051 | $ 141,573 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 1,942,412 | $ 1,796,731 | $ 1,661,976 | $ 1,537,328 | $ 1,422,028 | $ 1,315,376 | $ 1,216,723 | $ 1,125,469 | $ 1,041,059 | $ 962,979 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,164,454 | $ 1,077,120 | $ 996,336 | $ 921,611 | $ 852,490 | $ 788,553 | $ 729,412 | $ 674,706 | $ 624,103 |
| *+ Depletion* | $ - | $ - | $ - | $ 54,852,784 | $ 56,806,455 | $ 58,809,109 | $ 60,863,353 | $ 62,971,810 | $ 65,137,130 | $ 67,361,991 | $ 69,649,109 | $ 72,001,245 |
| Cash Flow | $ - | $ (102,494) | $ (725,664) | $ 98,669,175 | $ 101,826,663 | $ 105,087,524 | $ 108,454,933 | $ 111,932,196 | $ 115,522,759 | $ 119,230,205 | $ 123,058,266 | $ 127,010,821 |
| **Captial Cost** | $ 4,116,238 | $ 25,898,824 | $ 15,526,050 | $ 3,414,100 | $ 3,618,946 | $ 3,836,083 | $ 4,066,248 | $ 4,310,223 | $ 4,568,836 | $ 4,842,966 | $ 5,133,544 | $ 4,882,292 |
| *- Fixed Capital Costs* | $ 8,232,475 | $ 51,173,067 | $ 26,507,649 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 4,116,238 | $ 25,586,533 | $ 13,253,824 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 312,291 | $ 331,028 | $ 350,890 | $ 371,943 | $ 394,260 | $ 417,915 | $ 442,990 | $ 469,570 | $ 497,744 | $ 527,608 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 1,941,198 | $ 2,057,670 | $ 2,181,130 | $ 2,311,998 | $ 2,450,718 | $ 2,597,761 | $ 2,753,627 | $ 2,918,844 | $ 3,093,975 | $ 3,279,613 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 1,005,541 | $ 1,065,873 | $ 1,129,825 | $ 1,197,615 | $ 1,269,472 | $ 1,345,640 | $ 1,426,379 | $ 1,511,961 | $ 1,602,679 |
| **Net Cash Flow** | $ (4,116,238) | $ (26,001,318) | $ (16,251,714) | $ 95,255,075 | $ 98,207,717 | $ 101,251,441 | $ 104,388,685 | $ 107,621,974 | $ 110,953,923 | $ 114,387,239 | $ 117,924,721 | $ 122,128,528 |
| Discounted Cash Flow | $ (4,116,238) | $ (23,637,562) | $ (13,431,169) | $ 71,566,547 | $ 67,077,192 | $ 62,869,179 | $ 58,924,691 | $ 55,227,089 | $ 51,760,824 | $ 48,511,356 | $ 45,465,085 | $ 42,805,304 |
| **Cumulative DCF** | $ (4,116,238) | $ (27,753,800) | $ (41,184,968) | $ 30,381,579 | $ 97,458,771 | $ 160,327,950 | $ 219,252,641 | $ 274,479,731 | $ 326,240,554 | $ 374,751,910 | $ 420,216,995 | $ 463,022,299 |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 82,324,754 | Payback Period | 3.5 |
| Constant Dollar OPEX | $ 592,715,503 | Internal Rate of Return | 110% |
| Constant Dollar REE Rev | $ 698,280,043 | Net Present Value | $ 800,345,392 |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ 395,676,533 |

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C2 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 995,580,374** | **$ 1,025,447,785** | **$ 1,056,211,219** | **$ 1,087,897,555** | **$ 1,120,534,482** | **$ 1,154,150,516** | **$ 1,188,775,032** | **$ 1,224,438,283** | **$ 1,261,171,431** | **$ 1,299,006,574** | **$ 1,337,976,772** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 995,580,374 | $ 1,025,447,785 | $ 1,056,211,219 | $ 1,087,897,555 | $ 1,120,534,482 | $ 1,154,150,516 | $ 1,188,775,032 | $ 1,224,438,283 | $ 1,261,171,431 | $ 1,299,006,574 | $ 1,337,976,772 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 845,172,512** | **$ 870,422,699** | **$ 896,535,380** | **$ 923,431,441** | **$ 951,134,385** | **$ 979,668,416** | **$ 1,009,058,469** | **$ 1,039,330,223** | **$ 1,070,510,129** | **$ 1,102,625,433** | **$ 1,135,704,196** |
| *- Operating Costs* | $ 845,070,582 | $ 870,422,699 | $ 896,535,380 | $ 923,431,441 | $ 951,134,385 | $ 979,668,416 | $ 1,009,058,469 | $ 1,039,330,223 | $ 1,070,510,129 | $ 1,102,625,433 | $ 1,135,704,196 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 101,930 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ 150,407,862 | $ 155,025,086 | $ 159,675,839 | $ 164,466,114 | $ 169,400,097 | $ 174,482,100 | $ 179,716,563 | $ 185,108,060 | $ 190,661,302 | $ 196,381,141 | $ 202,272,575 |
| **Book Deductions** | **$ 1,599,005** | **$ 1,479,080** | **$ 1,368,149** | **$ 1,265,538** | **$ 1,170,623** | **$ 1,082,826** | **$ 1,001,614** | **$ 926,493** | **$ 857,006** | **$ 727,808** | **$ 264,737** |
| *- CAPEX #1 Depreciation* | $ 130,955 | $ 121,133 | $ 112,048 | $ 103,644 | $ 95,871 | $ 88,681 | $ 82,030 | $ 75,877 | $ 70,187 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 890,756 | $ 823,949 | $ 762,153 | $ 704,991 | $ 652,117 | $ 603,208 | $ 557,968 | $ 516,120 | $ 477,411 | $ 441,605 | $ - |
| *- CAPEX #3 Depreciation* | $ 577,295 | $ 533,998 | $ 493,948 | $ 456,902 | $ 422,634 | $ 390,937 | $ 361,617 | $ 334,495 | $ 309,408 | $ 286,203 | $ 264,737 |
| Taxable Before Depletion | $ 148,808,857 | $ 153,546,006 | $ 158,307,690 | $ 163,200,576 | $ 168,229,475 | $ 173,399,275 | $ 178,714,950 | $ 184,181,567 | $ 189,804,296 | $ 195,653,333 | $ 202,007,838 |
| **Depletion** | $ 74,404,428 | $ 76,773,003 | $ 79,153,845 | $ 81,600,288 | $ 84,114,737 | $ 86,699,637 | $ 89,357,475 | $ 92,090,784 | $ 94,902,148 | $ 97,826,667 | $ 101,003,919 |
| *- 50% Limit* | $ 74,404,428 | $ 76,773,003 | $ 79,153,845 | $ 81,600,288 | $ 84,114,737 | $ 86,699,637 | $ 89,357,475 | $ 92,090,784 | $ 94,902,148 | $ 97,826,667 | $ 101,003,919 |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 139,381,252 | $ 143,562,690 | $ 147,869,571 | $ 152,305,658 | $ 156,874,827 | $ 161,581,072 | $ 166,428,504 | $ 171,421,360 | $ 176,564,000 | $ 181,860,920 | $ 187,316,748 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ 74,404,428** | **$ 76,773,003** | **$ 79,153,845** | **$ 81,600,288** | **$ 84,114,737** | **$ 86,699,637** | **$ 89,357,475** | **$ 92,090,784** | **$ 94,902,148** | **$ 97,826,667** | **$ 101,003,919** |
| *- Income Tax* | $ 19,345,151 | $ 19,960,981 | $ 20,580,000 | $ 21,216,075 | $ 21,869,832 | $ 22,541,906 | $ 23,232,943 | $ 23,943,604 | $ 24,674,559 | $ 25,434,933 | $ 26,261,019 |
| Net Profit/Loss | $ 55,059,277 | $ 56,812,022 | $ 58,573,845 | $ 60,384,213 | $ 62,244,906 | $ 64,157,732 | $ 66,124,531 | $ 68,147,180 | $ 70,227,590 | $ 72,391,733 | $ 74,742,900 |
| **Book Deductions** | **$ 76,003,434** | **$ 78,252,083** | **$ 80,521,994** | **$ 82,865,826** | **$ 85,285,360** | **$ 87,782,463** | **$ 90,359,089** | **$ 93,017,277** | **$ 95,759,154** | **$ 98,554,474** | **$ 101,268,656** |
| *+ CAPEX #1 Depreciation* | $ 130,955 | $ 121,133 | $ 112,048 | $ 103,644 | $ 95,871 | $ 88,681 | $ 82,030 | $ 75,877 | $ 70,187 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 890,756 | $ 823,949 | $ 762,153 | $ 704,991 | $ 652,117 | $ 603,208 | $ 557,968 | $ 516,120 | $ 477,411 | $ 441,605 | $ - |
| *+ CAPEX #3 Depreciation* | $ 577,295 | $ 533,998 | $ 493,948 | $ 456,902 | $ 422,634 | $ 390,937 | $ 361,617 | $ 334,495 | $ 309,408 | $ 286,203 | $ 264,737 |
| *+ Depletion* | $ 74,404,428 | $ 76,773,003 | $ 79,153,845 | $ 81,600,288 | $ 84,114,737 | $ 86,699,637 | $ 89,357,475 | $ 92,090,784 | $ 94,902,148 | $ 97,826,667 | $ 101,003,919 |
| Cash Flow | $ 131,062,711 | $ 135,064,105 | $ 139,095,839 | $ 143,250,039 | $ 147,530,266 | $ 151,940,195 | $ 156,483,620 | $ 161,164,457 | $ 165,986,744 | $ 170,946,208 | $ 176,011,556 |
| **Captial Cost** | **$ 1,698,840** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 1,698,840 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ 129,363,871 | $ 135,064,105 | $ 139,095,839 | $ 143,250,039 | $ 147,530,266 | $ 151,940,195 | $ 156,483,620 | $ 161,164,457 | $ 165,986,744 | $ 170,946,208 | $ 176,011,556 |
| Discounted Cash Flow | $ 41,219,316 | $ 39,123,260 | $ 36,628,282 | $ 34,292,920 | $ 32,106,884 | $ 30,060,558 | $ 28,144,955 | $ 26,351,676 | $ 24,672,872 | $ 23,100,059 | $ 21,622,311 |
| **Cumulative DCF** | $ 504,241,615 | $ 543,364,875 | $ 579,993,157 | $ 614,286,078 | $ 646,392,962 | $ 676,453,519 | $ 704,598,474 | $ 730,950,150 | $ 755,623,022 | $ 778,723,081 | $ 800,345,392 |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 82,324,754 | Payback Period | 0 | 3.5 |
| Constant Dollar OPEX | $ 592,715,503 | Internal Rate of Return | 0 | 110% |
| Constant Dollar REE Rev | $ 698,280,043 | Net Present Value | 0 | $ 800,345,392 |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ 395,676,533 |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C2
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 0.07 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 0.05 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.00 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.00 | $ 0.00 |
| 7 | Conveyor | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 0.13** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.03 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.03** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 611** | **$ 1,834** | **$ 616,370** | **$ 0.15** | **$ 0.15** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C2
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 2.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 500 | tph |
| Residue Rate | 440 | tph |
| Slurry Flow | 9250 | gpm |
| Leaching Time | 45 | min |
| Acid Consumption | 344.96 | kg/ton |
| Bioleaching Sol'n Consumption | 0 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 2 | -- |
| Thickener Capacity | 6.72 | ft^2/tph |
| Polymer Dose | 0.016 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 8 | 100% | 52,031 | gal | $ 181,404 | $ 1,451,235 | $ - | |
| 2 | Tank Mixer | 8 | 100% | 78 | HP | $ 187,044 | $ 1,496,351 | $ - | |
| 3 | Process Pump | 4 | 100% | 4,625 | gpm | $ 161,957 | $ 647,827 | $ - | |
| 4 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 5 | Belt Filter | 2 | 100% | 220 | TPH | $ 1,199,210 | $ 2,398,421 | $ - | |
| 6 | Solution Heater | 1 | 100% | 1 | MBTU/hr | $ 30,807 | $ 30,807 | $ - | |
| 7 | Thickener | 2 | 100% | 1,478 | ft^2 | $ 147,063 | $ 294,127 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 6,414,323** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 624.38 | Effiicency: | 0.85 | | $ 299.30 | $ 897.89 | $ 301,691 | $ 0.08 | $ 0.07 |
| 3 | Process Pump | 462.50 | | | | $ 221.70 | $ 665.10 | $ 223,475 | $ 0.06 | $ 0.06 |
| 4 | Conveyor | 31.00 | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.00 | $ 0.00 |
| 5 | Belt Filter | 106.58 | | | | $ 51.09 | $ 153.27 | $ 51,498 | $ 0.01 | $ 0.01 |
| 6 | Solution Heater | 78.00 | | | | $ 37.39 | $ 112.16 | $ 37,687 | $ 0.01 | $ 0.01 |
| 7 | Thickener | 1.88 | | | | $ 0.90 | $ 2.70 | $ 908 | $ 0.00 | $ 0.00 |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1304** | | | | **$ 625.23** | **$ 1,875.70** | **$ 630,237** | **$ 0.16** | **$ 0.16** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | - | L/hr | $ 0.008 | / L | $ - | $ - | $ - | $ - | $ - |
| 2 | H2SO4 | 172.48 | mt/hr | $ 220.000 | / mt | $ 303,561.06 | $ 910,683.19 | $ 305,989,551 | $ 76.50 | $ 75.89 |
| 3 | Natural Gas | 0.75 | MCF/hr | $ 2.860 | / MCF | $ 17.21 | $ 51.63 | $ 17,346 | $ 0.00 | $ 0.00 |
| 4 | Polymer | 33.61 | kg/hr | $ 2.250 | / kg | $ 605.06 | $ 1,815.18 | $ 609,901 | $ 0.15 | $ 0.15 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 304,183.33** | **$ 912,549.99** | **$ 306,616,798** | **$ 76.66** | **$ 76.05** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 304,809** **$ 914,426** **$ 307,247,035** **$ 76.82** **$ 76.20**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C2
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 8.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8153 | gpm |
| Feed Rate | 30862 | lpm |
| Precip Mass Flow | 9.05 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 398.04 | ft^2/tpd |
| NaOH Dose | 44.29 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 8 | 100% | 45,860 | gal | $ 112,647 | $ 901,174 | $ - | |
| 2 | Tank Mixer | 8 | 100% | 23 | HP | $ 93,739 | $ 749,914 | $ - | |
| 3 | Process Pump | 24 | 100% | 1,019 | gpm | $ 45,481 | $ 1,091,544 | $ - | |
| 4 | Filter Press | 1 | 100% | 43 | ft^3 | $ 136,207 | $ 136,207 | $ - | |
| 5 | Conveyor | 1 | 100% | 9 | TPH | $ 10,557 | $ 10,557 | $ - | |
| 6 | Thickener | 8 | 100% | 450 | ft^2 | $ 56,679 | $ 453,428 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 3,342,824** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 183.44 | Effiicency: | 0.85 | | $ 87.93 | $ 263.80 | $ 88,636 | $ 0.02 | $ 0.02 |
| 3 | Process Pump | 611.46 | | | | $ 293.11 | $ 879.32 | $ 295,452 | $ 0.07 | $ 0.07 |
| 4 | Filter Press | 7.16 | | | | $ 3.43 | $ 10.29 | $ 3,459 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 0.56 | | | | $ 0.27 | $ 0.81 | $ 271 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 2.29 | | | | $ 1.10 | $ 3.29 | $ 1,107 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **805** | | | | **$ 385.84** | **$ 1,157.51** | **$ 388,924** | **$ 0.10** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 82.02 | mt/hr | $ 125.000 | / mt | $ 82,021.09 | $ 246,063.27 | $ 82,677,259 | $ 20.67 | $ 20.51 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 82,021.09** | **$ 246,063.27** | **$ 82,677,259** | **$ 20.67** | **$ 20.51** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| | **Total Operating Cost:** | **$ 82,407** | **$ 247,221** | **$ 83,066,183** | **$ 20.77** | **$ 20.60** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C2
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8142 | gpm |
| Feed Rate | 30822 | lpm |
| Precip Mass Flow | 1.59 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 0.00 | g/L of Feed |
| Oxalic Acid Dose | 6.65 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.93 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 40,711 | gal | $ 49,375 | $ 197,501 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 20 | HP | $ 87,650 | $ 350,600 | $ - | |
| 3 | Process Pump | 3 | 100% | 8,142 | gpm | $ 200,947 | $ 602,842 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 7.47 | ft^3 | $ 38,874 | $ 38,874 | $ - | |
| 6 | Roaster | 1 | 100% | 2.45 | MBTU/hr | $ 1,377,035 | $ 1,377,035 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,566,853** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 81.42 | Effiicency: | 0.85 | | $ 39.03 | $ 117.09 | $ 39,342 | $ 0.01 | $ 0.01 |
| 3 | Process Pump | 610.67 | | | | $ 292.72 | $ 878.17 | $ 295,067 | $ 0.07 | $ 0.07 |
| 4 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 1.25 | | | | $ 0.60 | $ 1.80 | $ 606 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.37 | | | | $ 0.18 | $ 0.53 | $ 178 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **694** | | | | **$ 332.53** | **$ 997.60** | **$ 335,193** | **$ 0.08** | **$ 0.08** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 2 | Oxalic Acid | 12,301.34 | kg/hr | $ 0.600 | / kg | $ 59,046.43 | $ 177,139.28 | $ 59,518,797 | $ 14.88 | $ 14.76 |
| 3 | Natural Gas | 2.37 | MCF/hr | $ 2.860 | / MCF | $ 54.14 | $ 162.42 | $ 54,574 | $ 0.01 | $ 0.01 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 59,100.57** | **$ 177,301.70** | **$ 59,573,371** | **$ 14.89** | **$ 14.78** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 59,433** **$ 178,299** **$ 59,908,564** **$ 14.98** **$ 14.86**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C2
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8153 | gpm |
| Raffinate Rate: | 8153 | gpm |
| Extraction Feed (O+A) | 16306 | gpm |
| Scrub Feed (O+A) | 224 | gpm |
| Strip Feed (O+A) | 448 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 224 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 268064 | gal |
| Precip Mixing Intensity | 1.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 30.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,306 | gal | $ 94,458 | $ 188,917 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 41 | HP | $ 129,674 | $ 259,348 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 244,585 | gal | $ 433,186 | $ 866,371 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,153 | gpm | $ 201,047 | $ 402,095 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 224 | gal | $ 8,475 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,556 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,362 | gal | $ 38,869 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 448 | gal | $ 12,516 | $ 25,032 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 17,084 | $ 34,167 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,725 | gal | $ 57,398 | $ 114,796 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,462 | $ 19,462 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 58,347 | kg | $ 3,653,067 | $ - | $ 3,653,067 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 1,012,195 | kg | $ 1,096,052 | $ - | $ 1,096,052 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,682 | gal | $ 5,602 | $ 22,409 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 4.02 | HP | $ 35,126 | $ 140,503 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,462 | $ 38,924 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.01 | ft^3 | $ 276 | $ 276 | $ - | |
| 20 | Process Pump | 2 | 100% | 224 | gpm | $ 51,053 | $ 102,106 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,214,405** | **$ 4,749,118** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 81.53 | Effiicency: | 0.85 | | | $ 39.08 | $ 117.24 | $ 39,394 | $ 0.01 | $ 0.01 |
| 3 | Steel Tanks | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 407.64 | | | | | $ 195.40 | $ 586.21 | $ 196,968 | $ 0.05 | $ 0.05 |
| 5 | Steel Tanks | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.24 | | | | | $ 1.07 | $ 3.22 | $ 1,083 | $ 0.00 | $ 0.00 |
| 11 | Steel Tanks | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.45 | | | | | $ 0.21 | $ 0.64 | $ 216 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 16.09 | | | | | $ 7.71 | $ 23.14 | $ 7,776 | $ 0.00 | $ 0.00 |
| 17 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.89 | | | | | $ 0.43 | $ 1.29 | $ 432 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | | | | | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.21 | | | | | $ 5.37 | $ 16.12 | $ 5,415 | $ 0.00 | $ 0.00 |
| | **TOTAL:** | **520** | | | | | **$ 249.29** | **$ 747.87** | **$ 251,284** | **$ 0.06** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 105.55 | kg/hr | $ 0.940 | / kg | $ 793.72 | $ 2,381.15 | $ 800,067 | $ 0.20 | $ 0.20 |
| 2 | D2EHPA | 5.56 | kg/hr | $ 5.000 | / kg | $ 222.21 | $ 666.62 | $ 223,983 | $ 0.06 | $ 0.06 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.86 | mt/hr | $ 270.000 | / mt | $ 1,848.71 | $ 5,546.13 | $ 1,863,501 | $ 0.47 | $ 0.46 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,864.63** | **$ 8,593.90** | **$ 2,887,551** | **$ 0.72** | **$ 0.72** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,114** **$ 9,342** **$ 3,138,835** **$ 0.78** **$ 0.78**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C2
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 3.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 8140.35 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 30814.59 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 8135.52 | gpm | HCL Dose | 3.12 | g/L of Feed |
| Mn Feed Rate | 30796.29 | lpm | | | |
| Co Precip Mass Flow | 4.12 | tph | | | |
| Mn Precip Mass Flow | 5.45 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 8.76 | g/L of Feed | | | |
| NaOH Dose | 1.72 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 6 | 100% | 40,702 | gal | $ 49,365 | $ 296,187 | $ - | Co Conditioning |
| 2 | Tank Mixer | 6 | 100% | 61 | HP | $ 162,852 | $ 977,110 | $ - | Co Conditioning |
| 3 | Process Pump | 3 | 100% | 2,713 | gpm | $ 132,151 | $ 396,453 | $ - | |
| 4 | Thickener | 3 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 3 | 100% | 19.40 | ft^3 | $ 77,270 | $ 231,810 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 6 | 100% | 40,678 | gal | $ 49,337 | $ 296,020 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 6 | 100% | 61 | HP | $ 162,797 | $ 976,783 | $ - | Mn Conditioning |
| 8 | Process Pump | 3 | 100% | 2,712 | gpm | $ 132,121 | $ 396,363 | $ - | |
| 9 | Thickener | 3 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 3 | 100% | 26 | ft^3 | $ 94,559 | $ 283,676 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 3,854,403** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 366.32 | Effiicency: | 0.85 | | $ 175.59 | $ 526.78 | $ 177,000 | $ 0.04 | $ 0.04 |
| 3 | Process Pump | 203.51 | | | | $ 97.55 | $ 292.66 | $ 98,333 | $ 0.02 | $ 0.02 |
| 4 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 9.77 | | | | $ 4.68 | $ 14.05 | $ 4,722 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 366.10 | | | | $ 175.49 | $ 526.47 | $ 176,894 | $ 0.04 | $ 0.04 |
| 8 | Process Pump | 203.39 | | | | $ 97.49 | $ 292.48 | $ 98,275 | $ 0.02 | $ 0.02 |
| 9 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 12.94 | | | | $ 6.20 | $ 18.60 | $ 6,251 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1162** | | | | **$ 557.02** | **$ 1,671.06** | **$ 561,475** | **$ 0.14** | **$ 0.14** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 16.20 | mt/hr | $ 360.000 | / mt | $ 46,665.16 | $ 139,995.47 | $ 47,038,477 | $ 11.76 | $ 11.67 |
| 2 | NaOH | 3.18 | mt/hr | $ 125.000 | / mt | $ 3,177.67 | $ 9,533.01 | $ 3,203,091 | $ 0.80 | $ 0.79 |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 1.66 | $ 4.99 | $ 1,678 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 49,844.49** | **$ 149,533.47** | **$ 50,243,246** | **$ 12.56** | **$ 12.46** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 50,402** **$ 151,205** **$ 50,804,720** **$ 12.70** **$ 12.60**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | Shakespeare Lignite | Date: | 3/18/2023 |
| Case: | SHXP-C2 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | an |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | N/A |
| *(unit conversion)* | 12000 | TPD | Project NPV: | (1,895,498,034) |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 22.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C2 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 82,324,754 | | | | | |
| Constant Dollar Annual OPEX | $ | 590,587,927 | $ | 146.48 | /ton | $ 236.8 | /kg |
| Constant Dollar Annual REE Rev | $ | 328,266,793 | $ | 81.42 | /ton | $ 131.62 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | | |
|---|---|---|---|---|---|
| Payback Period | | 25.5 | years | *22.5* | *operating years* |
| Internal Rate of Return | | N/A | | | |
| Net Present Value | $ | (1,895,498,034) | | | |
| Total Severence Tax Paid | $ | - | | | |
| Total State and Federal Tax Paid | $ | - | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C2 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 1,250 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 2.39 | 29.84 | 13.54 | | | | | | | 13.54 |
| Yttrium | 19.38 | 242.23 | 109.87 | | | | | | | 109.87 |
| Lanthanum | 14.92 | 186.47 | 84.58 | | | | | | | 84.58 |
| Cerium | 29.52 | 369.01 | 167.38 | | | | | | | 167.38 |
| Praseodymium | 3.72 | 46.50 | 21.09 | | | | | | | 21.09 |
| Neodymium | 14.47 | 180.92 | 82.06 | | | | | | | 82.06 |
| Samarium | 3.22 | 40.26 | 18.26 | | | | | | | 18.26 |
| Europium | 0.72 | 9.02 | 4.09 | | | | | | | 4.09 |
| Gadolinium | 3.59 | 44.88 | 20.36 | | | | | | | 20.36 |
| Terbium | 0.37 | 4.63 | 2.10 | | | | | | | 2.10 |
| Dysprosium | 3.35 | 41.87 | 18.99 | | | | | | | 18.99 |
| Holmium | 0.59 | 7.40 | 3.36 | | | | | | | 3.36 |
| Erbium | 1.52 | 18.97 | 8.61 | | | | | | | 8.61 |
| Thulium | 0.24 | 3.01 | 1.36 | | | | | | | 1.36 |
| Ytterbium | 1.68 | 21.05 | 9.55 | | | | | | | 9.55 |
| Lutetium | 0.31 | 3.93 | 1.78 | | | | | | | 1.78 |
| Manganese | -- | 637.00 | 288.94 | | | 0.00 | -- | | | 288.94 |
| Cobalt | -- | 304.30 | 138.03 | | | 0.00 | -- | | | 138.03 |
| Nickel | -- | 760.00 | 344.73 | | | 0.00 | -- | | | 344.73 |
| Zinc | -- | 234.70 | 106.46 | | | 0.00 | -- | | | 106.46 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 1250.00 | 566.99 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 567.0 |
| Total non-REE CM | -- | 1936.00 | 878.16 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 878.2 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 13.54 | 0.01 | 5.23 | 0.00 | 0.01 | 0.00 | 5.25 | 8.28 | 38.8% | |
| Yttrium | 109.87 | 54.19 | 0.00 | 0.88 | 17.23 | 0.00 | 72.31 | 37.56 | 65.8% | |
| Lanthanum | 84.58 | 36.66 | 0.00 | 0.14 | 11.20 | 0.00 | 47.99 | 36.59 | 56.7% | |
| Cerium | 167.38 | 73.24 | 0.00 | 0.68 | 12.94 | 0.00 | 86.85 | 80.52 | 51.9% | |
| Praseodymium | 21.09 | 7.75 | 0.00 | 0.07 | 1.37 | 0.00 | 9.18 | 11.91 | 43.5% | |
| Neodymium | 82.06 | 38.10 | 0.00 | 0.17 | 0.55 | 0.00 | 38.81 | 43.25 | 47.3% | |
| Samarium | 18.26 | 9.53 | 0.00 | 0.06 | 0.04 | 0.00 | 9.63 | 8.63 | 52.7% | |
| Europium | 4.09 | 1.85 | 0.00 | 0.00 | 0.01 | 0.00 | 1.87 | 2.23 | 45.6% | |
| Gadolinium | 20.36 | 9.74 | 0.00 | 0.10 | 1.38 | 0.00 | 11.22 | 9.14 | 55.1% | |
| Terbium | 2.10 | 1.15 | 0.00 | 0.02 | 0.04 | 0.00 | 1.21 | 0.88 | 57.8% | |
| Dysprosium | 18.99 | 9.30 | 0.00 | 0.16 | 1.22 | 0.00 | 10.68 | 8.32 | 56.2% | |
| Holmium | 3.36 | 1.70 | 0.00 | 0.02 | 0.39 | 0.00 | 2.11 | 1.25 | 62.9% | |
| Erbium | 8.61 | 4.15 | 0.00 | 0.06 | 1.28 | 0.00 | 5.48 | 3.13 | 63.7% | |
| Thulium | 1.36 | 0.53 | 0.00 | 0.01 | 0.17 | 0.00 | 0.71 | 0.65 | 52.1% | |
| Ytterbium | 9.55 | 3.56 | 0.00 | 0.09 | 1.52 | 0.00 | 5.16 | 4.39 | 54.0% | |
| Lutetium | 1.78 | 0.59 | 0.00 | 0.01 | 0.22 | 0.00 | 0.81 | 0.97 | 45.5% | |
| Manganese | 288.94 | 0.75 | 0.00 | 69.29 | 125.38 | 0.00 | 195.42 | 93.52 | 67.6% | |
| Cobalt | 138.03 | 0.49 | 0.00 | 61.44 | 5.38 | 0.00 | 67.31 | 70.71 | 48.8% | |
| Nickel | 344.73 | 1.69 | 0.00 | 106.65 | 86.52 | 0.00 | 194.87 | 149.86 | 56.5% | |
| Zinc | 106.46 | 0.51 | 0.00 | 2.02 | 64.41 | 0.00 | 66.94 | 39.52 | 62.9% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 566.99 | 252.04 | 5.23 | 2.46 | 49.55 | 0.00 | 309.29 | 257.70 | 54.5% | <---- |
| Total non-REE CM | 878.16 | 3.44 | 0.00 | 239.42 | 281.69 | 0.00 | 524.54 | 353.61 | 59.7% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 440 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 9 | ton | $ | 1.50 |
| Dust/Volatiles | - | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | | |
|---|---|---|
| 6.05 | TPD REE | |
| 0.126 | TPD Sc | |
| 12.51 | TPD CM | |
| 0.44 | REE | |
| 38.7% | Sc | |
| 24.0% | Mn | |
| 26.5% | Metal-S | |
| 500 | Feed | |
| 1,250 | Grade | |
| 25.5 | PP | |
| N/A | IRR | |
| -1895498034 | NPV | |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C2 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 0.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 2.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | UK: Design |
| Dissolved Solids in Leach | 12.1% | % | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Leaching Retention Time | 45 | min | UK: Design |
| Thickener UF % Solids | 40% | % | UK: SHXP-C4/SHXP-C4 Samples |
| Residue Settling Rate | 15.0 | ft/hr | Not Available, Assumed |
| **Reagent Use** | | | |
| Acid Consumption | 345.0 | kg/ton | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Bioleaching Sol'n Consumption | 0 | L/ton | N/A |
| Polymer Dosage | 0.016 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 6.72 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 8.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | UK: Design |
| Precip Solid Yield | 4.43 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 2.24 | ft/hr | UK: Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 0.5% | % | VT Override to 2% \| UK: SHXP-C4/SHXP-C4 Samples 0.5% |
| **Reagent Use** | | | |
| NaOH Dose | 44.29 | g/L of Feed | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 398.04 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 1250 | ppm | |
| Grade After Sorter | 1250 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 500 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 0 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 500 | tph | |
| Leaching Residue Rate | 439.5 | tph | |
| Slurry Flow (feed) | 9250 | gpm | |
| Product Flow | 8,153 | gpm | |
| Thickener Feed % Solids (eff) | 18% | % | |
| Liquid Density | 1.011 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 30.20 | g/L | |
| Leaching Vessel Size | 52031.25 | gal | |
| Leaching Vessel Size | 196.96 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 219.75 | tph | 70 to 200 tph |
| Thickener Area | 1477.76 | ft^2 | |
| Thickener Diameter | 43.39 | ft | ideal < 100' |
| Liquid Balance | 151.03 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,153 | gpm | |
| Slurry Feed Rate | 30,862 | lpm | |
| Precip Mass Flow | 9.05 | tph | |
| Thickener Feed % Solids (eff) | 0.44% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.008 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 173.60 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 42.64 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 10.63 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 450 | ft^2 | |
| Thickener Diameter | 23.95 | ft | ideal < 100' |

*PM5: REE Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| | | | |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45 | min | UK: Based on Design |
| Precip Solid Yield | | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 0.044 | ft/hr | UK: Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.00% | % | UK: SHXP-C4/SHXP-C4 Samples |
| | | | |
| **Reagent Use** | | | |
| NaOH Dose | | g/L of Feed | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| | | | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 8142 | gpm | |
| Slurry Feed Rate | 30,822 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.008 | g/cc | |
| | | | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^3 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | #DIV/0! | m/m | target = 6.35 |

## *PM6: RE-Redissolution*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 5.0% | % | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Dissolved Solids in Re-Leach | 5% | % | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Leaching Retention Time | 5 | min | UK: Design |
| **Reagent Use** | | | |
| Acid Consumption | | kg/ton | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | - | tph | |
| Slurry Flow (feed) | - | gpm | |
| Product Flow | #DIV/0! | gpm | |
| Liquid Density | #DIV/0! | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | #DIV/0! | v/v | target = 3.8 |
| Liquid Balance | #DIV/0! | gpm | must be > 0; adjust slurry density |
| Slurry Balance | #DIV/0! | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

## *PM7: REE Oxalate Precipitation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | UK: Design |
| Precip Solid Yield | 0.8 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Thickener UF %Solids | | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 0.00 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Oxalic Acid Dose | 6.65 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

**Material Balance**

| | | | |
|---|---|---|---|
| Slurry Feed Rate | 8142.21 | gpm | |
| Slurry Feed Rate | 30821.63 | lpm | |
| Precip Mass Flow | 1.59 | tph | |
| Thickener Feed % Solids (eff) | 0.08% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.92 | tph | |
| Liquid Density | 1.007 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 154.11 | m^3 | |
| Filter Press Area | 7.47 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 1.86 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 0.54 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.71 | m/m | |

## *PM8: Sc SX Separation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | Based on Design |
| Settler Residence Time | 15.00 | min | Based on Design |
| Loading: Number of Stages | 2.00 | - | Based on Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | Based on Design |
| Loading O:A Advance Ratio | 0.03 | v/v | Based on Design |
| Scrubbing: Number of Stages | | - | Not Applicable |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | Not Applicable |
| Scrubbing O:A Advance Ratio | | v/v | Not Applicable |
| Stripping: Number of Stages | 2.00 | - | Based on Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | Based on Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | Based on Design |
| Sc2O3 Grade | 2% | % | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | Not Availabe |
| Precip Solid Yield | 0.4 | g/L | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance (SX)**

| | | | |
|---|---|---|---|
| PLS Flow | 8152.8 | gpm | |
| Raffinate Rate: | 8152.8 | gpm | |
| Load Org Recycle | 7928.7 | gpm | |
| Load Org Feed | 8152.8 | gpm | |
| Loag Org Advance | 224.2 | gpm | |
| Scrub Aq Product | 0.0 | gpm | |
| Scrub Aq Recycle | 0.0 | gpm | |
| Scrub Aq Feed | 0.0 | gpm | |
| Scurb Org Product | 224.2 | gpm | |
| Strip Aq Product | 17.9 | gpm | |
| Strip Aq Recycle | 206.3 | gpm | |
| Strip Aq Feed | 224.2 | gpm | |
| Strip Org Product | 224.2 | gpm | |
| Extraction Feed (O+A): | 16305.7 | gpm | |
| Scrub Feed (O+A): | 224.2 | gpm | |
| Strip Feed (O+A): | 448.3 | gpm | |

**Material Balance (Precip)**

| | | | |
|---|---|---|---|
| Strip Solution Flow | 17.9 | gpm | |
| Solid Density | 3.4 | g/cc | |
| Solid Product Rate | 0.00 | tph | |
| Liquid Density (after precip) | 1.13 | g/cc | |
| Thickener Feed % Solids (eff) | 0.03% | % | |

**Total Volumes**

| | | | |
|---|---|---|---|
| Total Extraction Volume | 521782 | gal | |
| Total Scrubbing Volume | 0 | gal | |
| Total Stripping Volume | 14346 | gal | |
| Extraction Org Volume | 260890.90 | gal | |
| Scrubbing Org Volume | 0.00 | gal | |
| Stripping Org Volume | 7172.83 | gal | |
| Total Org Volume | 268063.73 | gal | |
| **Other Indicators** | | | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 10.15 | m^3 | |
| Filter Press Area | 0.01 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

## *PM9: CM Precipitation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 0 | min | N/A |
| Precip Solid Yield | - | g/L | N/A |
| Precip Settlng Rate | 10 | ft/hr | N/A |
| Thickener UF % Solids | 25% | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 7.24 | g/L of Feed | N/A |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 30.00 | min | N/A |
| Thickener Capacity | 0.00 | ft^2/tph | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Slurry Feed Rate | 8142 | gpm | |
| Slurry Feed Rate | 30,822 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.008 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM10: CM Redissolution*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 2.0% | % | N/A |
| Dissolved Solids in Re-Leach | 100% | % | N/A |
| Leaching Retention Time | 45 | min | N/A |
| **Reagent Use** | | | |
| Acid Consumption | 441 | kg/ton | N/A |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 0.00 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 0 | gpm | |
| Liquid Density | 1.000 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | |
| Releach Conc. Factor | #DIV/0! | v/v | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | - | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

*PM11: CM Recovery*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 3.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | UK: Design |
| Co Precip Solid Yield | 2.02 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Co Precip Settling Rate | | ft/hr | N/A |
| Co Thickener UF % Solids | 0% | % | N/A |
| Mn Precip Conditioning Time | 30.00 | min | UK: Design |
| Mn Precip Solid Yield | 2.68 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Mn Precip Settling Rate | | ft/hr | N/A |
| Mn Thickener UF % Solids | 1% | % | N/A |
| **Reagent Use** | | | |
| Na2S Dose | 8.76 | g/L of Feed | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| NaOH Dose | 1.72 | g/L of Feed | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| HCL Dose | 3.12 | g/L of Feed | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 8140.35 | gpm | |
| Co Slurry Feed Rate | 30814.59 | lpm | |
| Co Precip Mass Flow | 4.12 | tph | |
| Co Thickener Feed % Solids (eff) | 0.20% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 8135.52 | gpm | |
| Mn Slurry Feed Rate | 30796.29 | lpm | |
| Mn Precip Mass Flow | 5.45 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.27% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.01 | g/cc | |
| Mn Liquid Density | 1.00 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 154.1 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 19.40 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 4.83 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 154.0 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 25.7 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 6.40 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

*PM12: Biooxidation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Requirec | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C2 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 6,414,323 | $ - | $ 1,603,581 | $ 7,055,756 | $ 8,784,416 | $ 23,858,076 | IIIIIIIIIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 3,342,824 | $ - | $ 835,706 | $ 3,677,107 | $ 4,577,998 | $ 12,433,636 | IIIIIIIIIIIII |
| PM5 | REE Precipitation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM6 | RE-Redissolution | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM7 | REE Oxalate Precipitation | $ 2,566,853 | $ - | $ 641,713 | $ 2,823,538 | $ 3,515,305 | $ 9,547,410 | IIIIIIIIII |
| PM8 | Sc SX Separation | $ 2,214,405 | $ 4,749,118 | $ 553,601 | $ 2,435,846 | $ 3,032,628 | $ 12,985,599 | IIIIIIIIIIIII |
| PM9 | CM Precipitation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM10 | CM Redissolution | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM11 | CM Recovery | $ 3,854,403 | $ - | $ 963,601 | $ 4,239,844 | $ 5,278,605 | $ 14,336,453 | IIIIIIIIIIIIII |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| | TOTAL | $ 20,856,469 | $ 4,749,118 | $ 5,214,117 | $ 22,942,116 | $ 28,562,934 | $ 82,324,754 | |

## Module Summary

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| Total Lang Factor (=1+sum % Base) | | 372% | |

Year of Construction: 2025

# OPERATING COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C2 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 1.23 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 0.47 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 82,324,754 | $ Total | $ 816.71 | $ 2,450.14 | $ 823,248 | $ 0.33 | $ 0.20 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 0.12 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 328,266,793 | $ Total | $ 1,628.31 | $ 4,884.92 | $ 1,641,334 | $ 0.66 | $ 0.41 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 0.18 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 82,324,754 | $ Total | $ 816.71 | $ 2,450.14 | $ 823,248 | $ 0.33 | $ 0.20 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 8,243.19** | **$ 24,729.56** | **$ 8,309,131** | **$ 3.33** | **$ 2.06** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 0.21 | $ 0.13 |
| PM2 | Roasting | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | PR | 1304 | | $ 625.23 | $ 1,875.70 | $ 630,237 | $ 0.25 | $ 0.16 |
| PM4 | Al Precipitation | PR | 805 | | $ 385.84 | $ 1,157.51 | $ 388,924 | $ 0.16 | $ 0.10 |
| PM5 | REE Precipitation | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | PR | 694 | | $ 332.53 | $ 997.60 | $ 335,193 | $ 0.13 | $ 0.08 |
| **Sub-Total** | | | **5550.4** | | **$ 2,660.59** | **$ 7,981.78** | **$ 2,681,876.75** | **$ 1.08** | **$ 0.67** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.04 | $ 0.03 |
| PM2 | Roasting | CR | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | CR | | $ 304,183.33 | $ 912,549.99 | $ 306,616,798 | $ 122.94 | $ 76.05 |
| PM4 | Al Precipitation | CR | | $ 82,021.09 | $ 246,063.27 | $ 82,677,259 | $ 33.15 | $ 20.51 |
| PM5 | REE Precipitation | CR | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | CR | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | CR | | $ 59,100.57 | $ 177,301.70 | $ 59,573,371 | $ 23.89 | $ 14.78 |
| **Sub-Total** | | | | **$ 498,114.91** | **$ 1,494,344.73** | **$ 502,099,830.25** | **$ 201.32** | **$ 124.53** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 10,548 | $ 0.10 | $ 351.60 | $ 1,054.80 | $ 354,413 | $ 0.14 | $ 0.09 |
| Non-Hazardous Precipitate | WD | Tons | 217 | $ 1.50 | $ 108.60 | $ 325.80 | $ 109,469 | $ 0.04 | $ 0.03 |
| Dust/Volatiles | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 460.20** | **$ 1,380.60** | **$ 463,881.52** | **$ 0.19** | **$ 0.12** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 501,236** | **$ 1,503,707** | **$ 505,245,589** | **$ 203** | **$ 125** |

## Overhead Costs

| Allocation | Code | Rate | Basis | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% | % of Direct Cost | $ 24,730 Daily Direct Cost | $ 1,236.48 | $ 3,709.43 | $ 1,246,370 | $ 0.50 | $ 0.31 |
| Variable Portion | OH | 15% | % of Direct Cost | $ 1,503,707 Daily Direct Cost | $ 75,185.36 | $ 225,556.07 | $ 75,786,838 | $ 30.39 | $ 18.80 |
| **TOTAL OVERHEAD COSTS** | | | | | **$ 76,421.83** | **$ 229,265.50** | **$ 77,033,207.87** | **$ 30.89** | **$ 19.11** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | SHXP-C2 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 8,243.19 | $ 24,729.56 | $ 8,309,131 | $ 3 | $ 2.06 | I |
| Power | $ 2,660.59 | $ 7,981.78 | $ 2,681,877 | $ 1 | $ 0.67 | |
| Consumables | $ 498,114.91 | $ 1,494,344.73 | $ 502,099,830 | $ 201 | $ 124.53 | IIII

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C2 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 328,266,793

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 1220 | 1250 |
| Contained Value | ($/ton) | $ 32.5 | $ 206.9 |
| Feed Basket Price | ($/kg) | $ 29.3 | $ 182.4 |
| Product Basket Price | ($/kg) | $ 28.0 | $ 136.9 |
| Production Rate | (kg/day) | 7296.82 | 7422.89 |
| | (t/day) | 7.30 | 7.42 |
| | (t/yr) | 2451.73 | 2494.09 |
| | (% of US Demand) | 24.5% | |

## Revenue Contribution Table

| Element | REE/CM Price ($/kg) [1] | Product Revenue ($/hr) MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | REE Revenue ($/day) | |
|---|---|---|---|---|---|---|---|---|
| Scandium | $ 4,200.00 | $ 31.7 | $ 33,720.3 | $ - | $ - | $ - | $ 810,048 | IIII

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C2 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 358,705,988 | $ 369,467,167 | $ 380,551,183 | $ 391,967,718 | $ 403,726,750 | $ 415,838,552 | $ 428,313,709 | $ 441,163,120 | $ 454,398,013 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 358,705,988 | $ 369,467,167 | $ 380,551,183 | $ 391,967,718 | $ 403,726,750 | $ 415,838,552 | $ 428,313,709 | $ 441,163,120 | $ 454,398,013 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 246,974 | $ 1,763,429 | $ 647,773,698 | $ 666,929,394 | $ 686,653,614 | $ 706,963,048 | $ 727,874,859 | $ 749,406,705 | $ 771,576,748 | $ 794,403,669 | $ 817,906,680 |
| *- Operating Costs* | $ - | $ - | $ - | $ 645,351,374 | $ 664,711,915 | $ 684,653,272 | $ 705,192,871 | $ 726,348,657 | $ 748,139,116 | $ 770,583,290 | $ 793,700,789 | $ 817,511,812 |
| *- Loan #1 Interest* | $ - | $ 246,974 | $ 228,237 | $ 208,375 | $ 187,322 | $ 165,005 | $ 141,350 | $ 116,275 | $ 89,695 | $ 61,521 | $ 31,656 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 1,535,192 | $ 1,418,720 | $ 1,295,260 | $ 1,164,392 | $ 1,025,672 | $ 878,629 | $ 722,764 | $ 557,546 | $ 382,415 | $ 196,777 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 795,229 | $ 734,897 | $ 670,945 | $ 603,155 | $ 531,298 | $ 455,130 | $ 374,392 | $ 288,809 | $ 198,091 |
| Gross Profit | $ - | $ (246,974) | $ (1,763,429) | $ (289,067,711) | $ (297,462,226) | $ (306,102,432) | $ (314,995,330) | $ (324,148,109) | $ (333,568,153) | $ (343,263,040) | $ (353,240,549) | $ (363,508,667) |
| **Book Deductions** | $ - | $ 308,718 | $ 2,227,976 | $ 3,225,331 | $ 2,983,432 | $ 2,759,674 | $ 2,552,699 | $ 2,361,246 | $ 2,184,153 | $ 2,020,341 | $ 1,868,816 | $ 1,728,655 |
| *- CAPEX #1 Depreciation* | $ - | $ 308,718 | $ 285,564 | $ 264,147 | $ 244,336 | $ 226,011 | $ 209,060 | $ 193,380 | $ 178,877 | $ 165,461 | $ 153,051 | $ 141,573 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 1,942,412 | $ 1,796,731 | $ 1,661,976 | $ 1,537,328 | $ 1,422,028 | $ 1,315,376 | $ 1,216,723 | $ 1,125,469 | $ 1,041,059 | $ 962,979 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,164,454 | $ 1,077,120 | $ 996,336 | $ 921,611 | $ 852,490 | $ 788,553 | $ 729,412 | $ 674,706 | $ 624,103 |
| Taxable Before Depletion | $ - | $ (555,692) | $ (3,991,405) | $ (292,293,042) | $ (300,445,658) | $ (308,862,106) | $ (317,548,028) | $ (326,509,355) | $ (335,752,306) | $ (345,283,381) | $ (355,109,365) | $ (365,237,321) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ - | $ (277,846) | $ (1,995,702) | $ (146,146,521) | $ (150,222,829) | $ (154,431,053) | $ (158,774,014) | $ (163,254,678) | $ (167,876,153) | $ (172,641,691) | $ (177,554,683) | $ (182,618,661) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 50,218,838 | $ 51,725,403 | $ 53,277,166 | $ 54,875,481 | $ 56,521,745 | $ 58,217,397 | $ 59,963,919 | $ 61,762,837 | $ 63,615,722 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (555,692) | $ (3,991,405) | $ (292,293,042) | $ (300,445,658) | $ (308,862,106) | $ (317,548,028) | $ (326,509,355) | $ (335,752,306) | $ (345,283,381) | $ (355,109,365) | $ (365,237,321) |
| *- Income Tax* | $ - | $ (144,480) | $ (1,037,765) | $ (75,996,191) | $ (78,115,871) | $ (80,304,148) | $ (82,562,487) | $ (84,892,432) | $ (87,295,600) | $ (89,773,679) | $ (92,328,435) | $ (94,961,704) |
| Net Profit/Loss | $ - | $ (411,212) | $ (2,953,639) | $ (216,296,851) | $ (222,329,787) | $ (228,557,958) | $ (234,985,541) | $ (241,616,923) | $ (248,456,706) | $ (255,509,702) | $ (262,780,930) | $ (270,275,618) |
| **Book Deductions** | $ - | $ 308,718 | $ 2,227,976 | $ 3,225,331 | $ 2,983,432 | $ 2,759,674 | $ 2,552,699 | $ 2,361,246 | $ 2,184,153 | $ 2,020,341 | $ 1,868,816 | $ 1,728,655 |
| *+ CAPEX #1 Depreciation* | $ - | $ 308,718 | $ 285,564 | $ 264,147 | $ 244,336 | $ 226,011 | $ 209,060 | $ 193,380 | $ 178,877 | $ 165,461 | $ 153,051 | $ 141,573 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 1,942,412 | $ 1,796,731 | $ 1,661,976 | $ 1,537,328 | $ 1,422,028 | $ 1,315,376 | $ 1,216,723 | $ 1,125,469 | $ 1,041,059 | $ 962,979 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,164,454 | $ 1,077,120 | $ 996,336 | $ 921,611 | $ 852,490 | $ 788,553 | $ 729,412 | $ 674,706 | $ 624,103 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ - | $ (102,494) | $ (725,664) | $ (213,071,520) | $ (219,346,355) | $ (225,798,284) | $ (232,432,842) | $ (239,255,677) | $ (246,272,554) | $ (253,489,361) | $ (260,912,114) | $ (268,546,963) |
| **Captial Cost** | $ 4,116,238 | $ 25,898,824 | $ 15,526,050 | $ 3,414,100 | $ 3,618,946 | $ 3,836,083 | $ 4,066,248 | $ 4,310,223 | $ 4,568,836 | $ 4,842,966 | $ 5,133,544 | $ 4,882,292 |
| *- Fixed Capital Costs* | $ 8,232,475 | $ 51,173,067 | $ 26,507,649 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 4,116,238 | $ 25,586,533 | $ 13,253,824 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 312,291 | $ 331,028 | $ 350,890 | $ 371,943 | $ 394,260 | $ 417,915 | $ 442,990 | $ 469,570 | $ 497,744 | $ 527,608 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 1,941,198 | $ 2,057,670 | $ 2,181,130 | $ 2,311,998 | $ 2,450,718 | $ 2,597,761 | $ 2,753,627 | $ 2,918,844 | $ 3,093,975 | $ 3,279,613 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 1,005,541 | $ 1,065,873 | $ 1,129,825 | $ 1,197,615 | $ 1,269,472 | $ 1,345,640 | $ 1,426,379 | $ 1,511,961 | $ 1,602,679 |
| **Net Cash Flow** | $ (4,116,238) | $ (26,001,318) | $ (16,251,714) | $ (216,485,620) | $ (222,965,301) | $ (229,634,367) | $ (236,499,090) | $ (243,565,900) | $ (250,841,390) | $ (258,332,327) | $ (266,045,659) | $ (273,429,255) |
| Discounted Cash Flow | $ (4,116,238) | $ (23,637,562) | $ (13,431,169) | $ (162,648,850) | $ (152,288,301) | $ (142,584,875) | $ (133,497,571) | $ (124,987,819) | $ (117,019,360) | $ (109,558,125) | $ (102,572,118) | $ (95,835,286) |
| **Cumulative DCF** | $ (4,116,238) | $ (27,753,800) | $ (41,184,968) | $ (203,833,819) | $ (356,122,120) | $ (498,706,995) | $ (632,204,566) | $ (757,192,384) | $ (874,211,744) | $ (983,769,869) | $ (1,086,341,987) | $ (1,182,177,273) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 82,324,754 | Payback Period | 25.5 |
| Constant Dollar OPEX | $ 590,587,927 | Internal Rate of Return | N/A |
| Constant Dollar REE Rev | $ 328,266,793 | Net Present Value | $ (1,895,498,034) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ - |

# PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C2 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 468,029,954** | **$ 482,070,852** | **$ 496,532,978** | **$ 511,428,967** | **$ 526,771,836** | **$ 542,574,991** | **$ 558,852,241** | **$ 575,617,808** | **$ 592,886,343** | **$ 610,672,933** | **$ 628,993,121** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 468,029,954 | $ 482,070,852 | $ 496,532,978 | $ 511,428,967 | $ 526,771,836 | $ 542,574,991 | $ 558,852,241 | $ 575,617,808 | $ 592,886,343 | $ 610,672,933 | $ 628,993,121 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 842,139,097** | **$ 867,298,282** | **$ 893,317,230** | **$ 920,116,747** | **$ 947,720,249** | **$ 976,151,857** | **$ 1,005,436,413** | **$ 1,035,599,505** | **$ 1,066,667,490** | **$ 1,098,667,515** | **$ 1,131,627,540** |
| *- Operating Costs* | $ 842,037,167 | $ 867,298,282 | $ 893,317,230 | $ 920,116,747 | $ 947,720,249 | $ 976,151,857 | $ 1,005,436,413 | $ 1,035,599,505 | $ 1,066,667,490 | $ 1,098,667,515 | $ 1,131,627,540 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 101,930 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ (374,109,143) | $ (385,227,429) | $ (396,784,252) | $ (408,687,780) | $ (420,948,413) | $ (433,576,865) | $ (446,584,171) | $ (459,981,696) | $ (473,781,147) | $ (487,994,582) | $ (502,634,419) |
| **Book Deductions** | **$ 1,599,005** | **$ 1,479,080** | **$ 1,368,149** | **$ 1,265,538** | **$ 1,170,623** | **$ 1,082,826** | **$ 1,001,614** | **$ 926,493** | **$ 857,006** | **$ 727,808** | **$ 264,737** |
| *- CAPEX #1 Depreciation* | $ 130,955 | $ 121,133 | $ 112,048 | $ 103,644 | $ 95,871 | $ 88,681 | $ 82,030 | $ 75,877 | $ 70,187 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 890,756 | $ 823,949 | $ 762,153 | $ 704,991 | $ 652,117 | $ 603,208 | $ 557,968 | $ 516,120 | $ 477,411 | $ 441,605 | $ - |
| *- CAPEX #3 Depreciation* | $ 577,295 | $ 533,998 | $ 493,948 | $ 456,902 | $ 422,634 | $ 390,937 | $ 361,617 | $ 334,495 | $ 309,408 | $ 286,203 | $ 264,737 |
| Taxable Before Depletion | $ (375,708,149) | $ (386,706,509) | $ (398,152,401) | $ (409,953,317) | $ (422,119,035) | $ (434,659,691) | $ (447,585,785) | $ (460,908,189) | $ (474,638,153) | $ (488,722,390) | $ (502,899,157) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ (187,854,074) | $ (193,353,255) | $ (199,076,201) | $ (204,976,659) | $ (211,059,518) | $ (217,329,846) | $ (223,792,893) | $ (230,454,095) | $ (237,319,077) | $ (244,361,195) | $ (251,449,578) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 65,524,194 | $ 67,489,919 | $ 69,514,617 | $ 71,600,055 | $ 73,748,057 | $ 75,960,499 | $ 78,239,314 | $ 80,586,493 | $ 83,004,088 | $ 85,494,211 | $ 88,059,037 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ (375,708,149)** | **$ (386,706,509)** | **$ (398,152,401)** | **$ (409,953,317)** | **$ (422,119,035)** | **$ (434,659,691)** | **$ (447,585,785)** | **$ (460,908,189)** | **$ (474,638,153)** | **$ (488,722,390)** | **$ (502,899,157)** |
| *- Income Tax* | $ (97,684,119) | $ (100,543,692) | $ (103,519,624) | $ (106,587,863) | $ (109,750,949) | $ (113,011,520) | $ (116,372,304) | $ (119,836,129) | $ (123,405,920) | $ (127,067,821) | $ (130,753,781) |
| Net Profit/Loss | $ (278,024,030) | $ (286,162,817) | $ (294,632,777) | $ (303,365,455) | $ (312,368,086) | $ (321,648,171) | $ (331,213,481) | $ (341,072,060) | $ (351,232,233) | $ (361,654,568) | $ (372,145,376) |
| **Book Deductions** | **$ 1,599,005** | **$ 1,479,080** | **$ 1,368,149** | **$ 1,265,538** | **$ 1,170,623** | **$ 1,082,826** | **$ 1,001,614** | **$ 926,493** | **$ 857,006** | **$ 727,808** | **$ 264,737** |
| *+ CAPEX #1 Depreciation* | $ 130,955 | $ 121,133 | $ 112,048 | $ 103,644 | $ 95,871 | $ 88,681 | $ 82,030 | $ 75,877 | $ 70,187 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 890,756 | $ 823,949 | $ 762,153 | $ 704,991 | $ 652,117 | $ 603,208 | $ 557,968 | $ 516,120 | $ 477,411 | $ 441,605 | $ - |
| *+ CAPEX #3 Depreciation* | $ 577,295 | $ 533,998 | $ 493,948 | $ 456,902 | $ 422,634 | $ 390,937 | $ 361,617 | $ 334,495 | $ 309,408 | $ 286,203 | $ 264,737 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ (276,425,025) | $ (284,683,737) | $ (293,264,628) | $ (302,099,917) | $ (311,197,464) | $ (320,565,346) | $ (330,211,867) | $ (340,145,567) | $ (350,375,228) | $ (360,926,760) | $ (371,880,639) |
| **Captial Cost** | **$ 1,698,840** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 1,698,840 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ (278,123,864) | $ (284,683,737) | $ (293,264,628) | $ (302,099,917) | $ (311,197,464) | $ (320,565,346) | $ (330,211,867) | $ (340,145,567) | $ (350,375,228) | $ (360,926,760) | $ (371,880,639) |
| Discounted Cash Flow | $ (88,618,834) | $ (82,462,738) | $ (77,225,742) | $ (72,320,318) | $ (67,725,635) | $ (63,422,145) | $ (59,391,507) | $ (55,616,518) | $ (52,081,045) | $ (48,772,239) | $ (45,684,039) |
| **Cumulative DCF** | $ (1,270,796,107) | $ (1,353,258,845) | $ (1,430,484,588) | $ (1,502,804,906) | $ (1,570,530,541) | $ (1,633,952,686) | $ (1,693,344,193) | $ (1,748,960,711) | $ (1,801,041,756) | $ (1,849,813,995) | $ (1,895,498,034) |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

## Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 82,324,754 | Payback Period | 0 | 25.5 |
| Constant Dollar OPEX | $ 590,587,927 | Internal Rate of Return | 0 | N/A |
| Constant Dollar REE Rev | $ 328,266,793 | Net Present Value | 0 | $ (1,895,498,034) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ - |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C2
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 0.12 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 0.07 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.01 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 7 | Conveyor | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 0.21** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.04 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.04** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 611** | **$ 1,834** | **$ 616,370** | **$ 0.25** | **$ 0.15** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C2
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 2.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 500 | tph |
| Residue Rate | 440 | tph |
| Slurry Flow | 9250 | gpm |
| Leaching Time | 45 | min |
| Acid Consumption | 344.96 | kg/ton |
| Bioleaching Sol'n Consumption | 0 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 2 | -- |
| Thickener Capacity | 6.72 | ft^2/tph |
| Polymer Dose | 0.016 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 8 | 100% | 52,031 | gal | $ 181,404 | $ 1,451,235 | $ - | |
| 2 | Tank Mixer | 8 | 100% | 78 | HP | $ 187,044 | $ 1,496,351 | $ - | |
| 3 | Process Pump | 4 | 100% | 4,625 | gpm | $ 161,957 | $ 647,827 | $ - | |
| 4 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 5 | Belt Filter | 2 | 100% | 220 | TPH | $ 1,199,210 | $ 2,398,421 | $ - | |
| 6 | Solution Heater | 1 | 100% | 1 | MBTU/hr | $ 30,807 | $ 30,807 | $ - | |
| 7 | Thickener | 2 | 100% | 1,478 | ft^2 | $ 147,063 | $ 294,127 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 6,414,323** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 624.38 | Effiicency: | 0.85 | | $ 299.30 | $ 897.89 | $ 301,691 | $ 0.12 | $ 0.07 |
| 3 | Process Pump | 462.50 | | | | $ 221.70 | $ 665.10 | $ 223,475 | $ 0.09 | $ 0.06 |
| 4 | Conveyor | 31.00 | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 5 | Belt Filter | 106.58 | | | | $ 51.09 | $ 153.27 | $ 51,498 | $ 0.02 | $ 0.01 |
| 6 | Solution Heater | 78.00 | | | | $ 37.39 | $ 112.16 | $ 37,687 | $ 0.02 | $ 0.01 |
| 7 | Thickener | 1.88 | | | | $ 0.90 | $ 2.70 | $ 908 | $ 0.00 | $ 0.00 |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1304** | | | | **$ 625.23** | **$ 1,875.70** | **$ 630,237** | **$ 0.25** | **$ 0.16** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | - | L/hr | $ 0.008 | / L | $ - | $ - | $ - | $ - | $ - |
| 2 | H2SO4 | 172.48 | mt/hr | $ 220.000 | / mt | $ 303,561.06 | $ 910,683.19 | $ 305,989,551 | $ 122.69 | $ 75.89 |
| 3 | Natural Gas | 0.75 | MCF/hr | $ 2.860 | / MCF | $ 17.21 | $ 51.63 | $ 17,346 | $ 0.01 | $ 0.00 |
| 4 | Polymer | 33.61 | kg/hr | $ 2.250 | / kg | $ 605.06 | $ 1,815.18 | $ 609,901 | $ 0.24 | $ 0.15 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 304,183.33** | **$ 912,549.99** | **$ 306,616,798** | **$ 122.94** | **$ 76.05** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | **Total Operating Cost:** | | | | | **$ 304,809** | **$ 914,426** | **$ 307,247,035** | **$ 123.19** | **$ 76.20** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C2

Date: 3/18/2023
By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 8.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8153 | gpm |
| Feed Rate | 30862 | lpm |
| Precip Mass Flow | 9.05 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 398.04 | ft^2/tpd |
| NaOH Dose | 44.29 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 8 | 100% | 45,860 | gal | $ 112,647 | $ 901,174 | $ - | |
| 2 | Tank Mixer | 8 | 100% | 23 | HP | $ 93,739 | $ 749,914 | $ - | |
| 3 | Process Pump | 24 | 100% | 1,019 | gpm | $ 45,481 | $ 1,091,544 | $ - | |
| 4 | Filter Press | 1 | 100% | 43 | ft^3 | $ 136,207 | $ 136,207 | $ - | |
| 5 | Conveyor | 1 | 100% | 9 | TPH | $ 10,557 | $ 10,557 | $ - | |
| 6 | Thickener | 8 | 100% | 450 | ft^2 | $ 56,679 | $ 453,428 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 3,342,824** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 183.44 | Effiicency: | 0.85 | | | $ 87.93 | $ 263.80 | $ 88,636 | $ 0.04 | $ 0.02 |
| 3 | Process Pump | 611.46 | | | | | $ 293.11 | $ 879.32 | $ 295,452 | $ 0.12 | $ 0.07 |
| 4 | Filter Press | 7.16 | | | | | $ 3.43 | $ 10.29 | $ 3,459 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 0.56 | | | | | $ 0.27 | $ 0.81 | $ 271 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 2.29 | | | | | $ 1.10 | $ 3.29 | $ 1,107 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **805** | | | | | **$ 385.84** | **$ 1,157.51** | **$ 388,924** | **$ 0.16** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 82.02 | mt/hr | $ 125.000 | / mt | $ 82,021.09 | $ 246,063.27 | $ 82,677,259 | $ 33.15 | $ 20.51 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 82,021.09** | **$ 246,063.27** | **$ 82,677,259** | **$ 33.15** | **$ 20.51** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 82,407** | **$ 247,221** | **$ 83,066,183** | **$ 33.31** | **$ 20.60** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C2

Date: 3/18/2023
By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8142 | gpm |
| Feed Rate | 30822 | lpm |
| Precip Mass Flow | 1.59 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 0.00 | g/L of Feed |
| Oxalic Acid Dose | 6.65 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.92 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 40,711 | gal | $ 49,375 | $ 197,501 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 20 | HP | $ 87,650 | $ 350,600 | $ - | |
| 3 | Process Pump | 3 | 100% | 8,142 | gpm | $ 200,947 | $ 602,842 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 7.47 | ft^3 | $ 38,874 | $ 38,874 | $ - | |
| 6 | Roaster | 1 | 100% | 2.45 | MBTU/hr | $ 1,377,035 | $ 1,377,035 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,566,853** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 81.42 | Effiicency: | 0.85 | | $ 39.03 | $ 117.09 | $ 39,342 | $ 0.02 | $ 0.01 |
| 3 | Process Pump | 610.67 | | | | $ 292.72 | $ 878.17 | $ 295,067 | $ 0.12 | $ 0.07 |
| 4 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 1.25 | | | | $ 0.60 | $ 1.80 | $ 606 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.37 | | | | $ 0.18 | $ 0.53 | $ 178 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **694** | | | | **$ 332.53** | **$ 997.60** | **$ 335,193** | **$ 0.13** | **$ 0.08** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 2 | Oxalic Acid | 12,301.34 | kg/hr | $ 0.600 | / kg | $ 59,046.43 | $ 177,139.28 | $ 59,518,797 | $ 23.86 | $ 14.76 |
| 3 | Natural Gas | 2.37 | MCF/hr | $ 2.860 | / MCF | $ 54.14 | $ 162.42 | $ 54,574 | $ 0.02 | $ 0.01 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 59,100.57** | **$ 177,301.70** | **$ 59,573,371** | **$ 23.89** | **$ 14.78** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 59,433** **$ 178,299** **$ 59,908,564** **$ 24.02** **$ 14.86**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C2

Date: 3/18/2023
By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8153 | gpm |
| Raffinate Rate: | 8153 | gpm |
| Extraction Feed (O+A) | 16306 | gpm |
| Scrub Feed (O+A) | 224 | gpm |
| Strip Feed (O+A) | 448 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 224 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 268064 | gal |
| Precip Mixing Intensity | 1.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 30.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,306 | gal | $ 94,458 | $ 188,917 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 41 | HP | $ 129,674 | $ 259,348 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 244,585 | gal | $ 433,186 | $ 866,371 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,153 | gpm | $ 201,047 | $ 402,095 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 224 | gal | $ 8,475 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,556 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,362 | gal | $ 38,869 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 448 | gal | $ 12,516 | $ 25,032 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 17,084 | $ 34,167 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,725 | gal | $ 57,398 | $ 114,796 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,462 | $ 19,462 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 58,347 | kg | $ 3,653,067 | $ - | $ 3,653,067 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 1,012,195 | kg | $ 1,096,052 | $ - | $ 1,096,052 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,682 | gal | $ 5,602 | $ 22,409 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 4.02 | HP | $ 35,126 | $ 140,503 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,462 | $ 38,924 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.01 | ft^3 | $ 276 | $ 276 | $ - | |
| 20 | Process Pump | 2 | 100% | 224 | gpm | $ 51,053 | $ 102,106 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,214,405** | **$ 4,749,118** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 81.53 | Effiicency: | 0.85 | | $ 39.08 | $ 117.24 | $ 39,394 | $ 0.02 | $ 0.01 |
| 3 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 407.64 | | | | $ 195.40 | $ 586.21 | $ 196,968 | $ 0.08 | $ 0.05 |
| 5 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.24 | | | | $ 1.07 | $ 3.22 | $ 1,083 | $ 0.00 | $ 0.00 |
| 11 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.45 | | | | $ 0.21 | $ 0.64 | $ 216 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 16.09 | | | | $ 7.71 | $ 23.14 | $ 7,776 | $ 0.00 | $ 0.00 |
| 17 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.89 | | | | $ 0.43 | $ 1.29 | $ 432 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | | | | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.21 | | | | $ 5.37 | $ 16.12 | $ 5,415 | $ 0.00 | $ 0.00 |
| | **TOTAL:** | **520** | | | | **$ 249.29** | **$ 747.87** | **$ 251,284** | **$ 0.10** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 105.55 | kg/hr | $ 0.940 | / kg | $ 793.72 | $ 2,381.15 | $ 800,067 | $ 0.32 | $ 0.20 |
| 2 | D2EHPA | 5.56 | kg/hr | $ 5.000 | / kg | $ 222.21 | $ 666.62 | $ 223,983 | $ 0.09 | $ 0.06 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.86 | mt/hr | $ 270.000 | / mt | $ 1,848.71 | $ 5,546.13 | $ 1,863,501 | $ 0.75 | $ 0.46 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,864.63** | **$ 8,593.90** | **$ 2,887,551** | **$ 1.16** | **$ 0.72** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 3,114** | **$ 9,342** | **$ 3,138,835** | **$ 1.26** | **$ 0.78** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C2
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 3.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 8140.35 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 30814.59 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 8135.52 | gpm | HCL Dose | 3.12 | g/L of Feed |
| Mn Feed Rate | 30796.29 | lpm | | | |
| Co Precip Mass Flow | 4.12 | tph | | | |
| Mn Precip Mass Flow | 5.45 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 8.76 | g/L of Feed | | | |
| NaOH Dose | 1.72 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 6 | 100% | 40,702 | gal | $ 49,365 | $ 296,187 | $ - | Co Conditioning |
| 2 | Tank Mixer | 6 | 100% | 61 | HP | $ 162,852 | $ 977,110 | $ - | Co Conditioning |
| 3 | Process Pump | 3 | 100% | 2,713 | gpm | $ 132,151 | $ 396,453 | $ - | |
| 4 | Thickener | 3 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 3 | 100% | 19.40 | ft^3 | $ 77,270 | $ 231,810 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 6 | 100% | 40,678 | gal | $ 49,337 | $ 296,020 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 6 | 100% | 61 | HP | $ 162,797 | $ 976,783 | $ - | Mn Conditioning |
| 8 | Process Pump | 3 | 100% | 2,712 | gpm | $ 132,121 | $ 396,363 | $ - | |
| 9 | Thickener | 3 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 3 | 100% | 26 | ft^3 | $ 94,559 | $ 283,676 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 3,854,403** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 366.32 | Effiicency: | 0.85 | | | $ 175.59 | $ 526.78 | $ 177,000 | $ 0.07 | $ 0.04 |
| 3 | Process Pump | 203.51 | | | | | $ 97.55 | $ 292.66 | $ 98,333 | $ 0.04 | $ 0.02 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 9.77 | | | | | $ 4.68 | $ 14.05 | $ 4,722 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 366.10 | | | | | $ 175.49 | $ 526.47 | $ 176,894 | $ 0.07 | $ 0.04 |
| 8 | Process Pump | 203.39 | | | | | $ 97.49 | $ 292.48 | $ 98,275 | $ 0.04 | $ 0.02 |
| 9 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 12.94 | | | | | $ 6.20 | $ 18.60 | $ 6,251 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1162** | | | | | **$ 557.02** | **$ 1,671.06** | **$ 561,475** | **$ 0.23** | **$ 0.14** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 16.20 | mt/hr | $ 360.000 | / mt | $ 46,665.16 | $ 139,995.47 | $ 47,038,477 | $ 18.86 | $ 11.67 |
| 2 | NaOH | 3.18 | mt/hr | $ 125.000 | / mt | $ 3,177.67 | $ 9,533.01 | $ 3,203,091 | $ 1.28 | $ 0.79 |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 1.66 | $ 4.99 | $ 1,678 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 49,844.49** | **$ 149,533.47** | **$ 50,243,246** | **$ 20.14** | **$ 12.46** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 50,402** | **$ 151,205** | **$ 50,804,720** | **$ 20.37** | **$ 12.60** |

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | Shakespeare Lignite | Date: | 3/13/2023 |
| Case: | SHXP-C3 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | an |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | 176.8% |
| *(unit conversion)* | 12000 | TPD | Project NPV: | 2,326,320,433 |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 0.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/13/2023 |
| Case: | SHXP-C3 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 108,132,002 | | | | | |
| Constant Dollar Annual OPEX | $ | 415,594,391 | $ | 103.07 | /ton | $ 139.0 | /kg |
| Constant Dollar Annual REE Rev | $ | 724,975,145 | $ | 179.81 | /ton | $ 242.45 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | |
|---|---|---|---|---|
| Payback Period | 3.5 | years | *0.5* | *operating years* |
| Internal Rate of Return | 177% | | | |
| Net Present Value | $ 2,326,320,433 | | | |
| Total Severence Tax Paid | $ - | | | |
| Total State and Federal Tax Paid | $ 1,569,468,650 | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/13/2023
Case: SHXP-C3 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 1,250 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 2.52 | 31.54 | 14.31 | | | | | | | 14.31 |
| Yttrium | 17.93 | 224.14 | 101.67 | | | | | | | 101.67 |
| Lanthanum | 14.92 | 186.53 | 84.61 | | | | | | | 84.61 |
| Cerium | 29.73 | 371.57 | 168.54 | | | | | | | 168.54 |
| Praseodymium | 3.84 | 47.95 | 21.75 | | | | | | | 21.75 |
| Neodymium | 15.16 | 189.50 | 85.96 | | | | | | | 85.96 |
| Samarium | 3.24 | 40.54 | 18.39 | | | | | | | 18.39 |
| Europium | 0.73 | 9.12 | 4.14 | | | | | | | 4.14 |
| Gadolinium | 3.73 | 46.59 | 21.13 | | | | | | | 21.13 |
| Terbium | 0.49 | 6.08 | 2.76 | | | | | | | 2.76 |
| Dysprosium | 3.34 | 41.77 | 18.95 | | | | | | | 18.95 |
| Holmium | 0.62 | 7.76 | 3.52 | | | | | | | 3.52 |
| Erbium | 1.39 | 17.39 | 7.89 | | | | | | | 7.89 |
| Thulium | 0.37 | 4.63 | 2.10 | | | | | | | 2.10 |
| Ytterbium | 1.68 | 21.03 | 9.54 | | | | | | | 9.54 |
| Lutetium | 0.31 | 3.84 | 1.74 | | | | | | | 1.74 |
| Manganese | -- | 505.70 | 229.38 | | | 0.00 | -- | | | 229.38 |
| Cobalt | -- | 258.60 | 117.30 | | | 0.00 | -- | | | 117.30 |
| Nickel | -- | 718.70 | 326.00 | | | 0.00 | -- | | | 326.00 |
| Zinc | -- | 210.80 | 95.62 | | | 0.00 | -- | | | 95.62 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 1250.00 | 566.99 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 567.0 |
| Total non-REE CM | -- | 1693.80 | 768.30 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 768.3 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 14.31 | 0.01 | 12.39 | 0.00 | 0.00 | 0.00 | 12.41 | 1.90 | 86.7% | |
| Yttrium | 101.67 | 57.52 | 0.00 | 15.18 | 1.01 | 0.00 | 73.71 | 27.96 | 72.5% | |
| Lanthanum | 84.61 | 23.50 | 0.00 | 22.40 | 1.60 | 0.00 | 47.50 | 37.11 | 56.1% | |
| Cerium | 168.54 | 77.75 | 0.00 | 30.34 | 2.12 | 0.00 | 110.21 | 58.33 | 65.4% | |
| Praseodymium | 21.75 | 9.14 | 0.00 | 3.49 | 0.17 | 0.00 | 12.80 | 8.95 | 58.8% | |
| Neodymium | 85.96 | 45.55 | 0.00 | 9.04 | 0.68 | 0.00 | 55.27 | 30.68 | 64.3% | |
| Samarium | 18.39 | 11.49 | 0.00 | 0.66 | 0.10 | 0.00 | 12.25 | 6.13 | 66.6% | |
| Europium | 4.14 | 2.27 | 0.00 | 0.20 | 0.02 | 0.00 | 2.49 | 1.65 | 60.1% | |
| Gadolinium | 21.13 | 12.90 | 0.00 | 1.90 | 0.13 | 0.00 | 14.92 | 6.21 | 70.6% | |
| Terbium | 2.76 | 1.16 | 0.00 | 0.00 | 0.00 | 0.00 | 1.16 | 1.60 | 42.1% | |
| Dysprosium | 18.95 | 10.95 | 0.00 | 1.94 | 0.17 | 0.00 | 13.06 | 5.89 | 68.9% | |
| Holmium | 3.52 | 2.06 | 0.00 | 0.55 | 0.03 | 0.00 | 2.63 | 0.89 | 74.8% | |
| Erbium | 7.89 | 4.70 | 0.00 | 0.49 | 0.06 | 0.00 | 5.25 | 2.64 | 66.5% | |
| Thulium | 2.10 | 0.54 | 0.00 | 0.06 | 0.01 | 0.00 | 0.60 | 1.50 | 28.8% | |
| Ytterbium | 9.54 | 5.09 | 0.00 | 0.42 | 0.06 | 0.00 | 5.57 | 3.97 | 58.4% | |
| Lutetium | 1.74 | 0.76 | 0.00 | 0.21 | 0.01 | 0.00 | 0.98 | 0.76 | 56.2% | |
| Manganese | 229.38 | 0.02 | 0.00 | 45.42 | 3.03 | 0.00 | 48.48 | 180.90 | 21.1% | |
| Cobalt | 117.30 | 0.05 | 0.00 | 5.40 | 71.73 | 0.00 | 77.19 | 40.11 | 65.8% | |
| Nickel | 326.00 | 0.29 | 0.00 | 5.46 | 159.75 | 0.00 | 165.49 | 160.50 | 50.8% | |
| Zinc | 95.62 | 0.11 | 0.00 | 0.06 | 69.50 | 0.00 | 69.66 | 25.96 | 72.9% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 566.99 | 265.39 | 12.39 | 86.88 | 6.15 | 0.00 | 370.82 | 196.18 | 65.4% | <---- |
| Total non-REE CM | 768.30 | 0.47 | 0.00 | 56.34 | 304.01 | 0.00 | 360.82 | 407.48 | 47.0% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 339 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 16 | ton | $ | 1.50 |
| Dust/Volatiles | - | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | |
|---|---|
| 6.37 | TPD REE |
| 0.297 | TPD Sc |
| 8.65 | TPD CM |
| 0.47 | REE |
| 86.6% | Sc |
| 19.8% | Mn |
| 55.8% | Metal-S |
| 500 | Feed |
| 1,250 | Grade |
| 3.5 | PP |
| 177% | IRR |
| 2326320433 | NPV |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/13/2023
Case: SHXP-C3 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 0.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 2.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | Based on Design |
| Dissolved Solids in Leach | 32.2% | % | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Leaching Retention Time | 45 | min | Based on Design |
| Thickener UF % Solids | 46% | % | SHXP-C3 Samples |
| Residue Settling Rate | 15.0 | ft/hr | Not Available |
| **Reagent Use** | | | |
| Acid Consumption | 264.1 | kg/ton | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Bioleaching Sol'n Consumption | 0 | L/ton | |
| Polymer Dosage | 0.016 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 10.75 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 7.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | Based on Design |
| Precip Solid Yield | 7.54 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Precip Settling Rate | 2.24 | ft/hr | Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 1.5% | % | VT Override to 2% \| SHXP-C3 Samples |
| **Reagent Use** | | | |
| NaOH Dose | 18.31 | g/L of Feed | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 974.97 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 1250 | ppm | |
| Grade After Sorter | 1250 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 500 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 0 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 500 | tph | |
| Leaching Residue Rate | 339 | tph | |
| Slurry Flow (feed) | 9250 | gpm | |
| Product Flow | 8,404 | gpm | |
| Thickener Feed % Solids (eff) | 14% | % | |
| Liquid Density | 1.029 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 80.37 | g/L | |
| Leaching Vessel Size | 52031.25 | gal | |
| Leaching Vessel Size | 196.96 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 169.50 | tph | 70 to 200 tph |
| Thickener Area | 1822.72 | ft^2 | |
| Thickener Diameter | 48.19 | ft | ideal < 100' |
| Liquid Balance | 401.92 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,404 | gpm | |
| Slurry Feed Rate | 31,812 | lpm | |
| Precip Mass Flow | 15.87 | tph | |
| Thickener Feed % Solids (eff) | 0.73% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.023 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 204.50 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 74.75 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 18.64 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 2,210 | ft^2 | |
| Thickener Diameter | 53.06 | ft | ideal < 100' |

*PM5: REE Precipitation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |

**Process Parameters**

| | | | |
|---|---|---|---|
| Precip Conditioning Time | 45 | min | Based on Design |
| Precip Solid Yield | 8.49 | g/L | VT Override from 1.83 \|\| SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Precip Settling Rate | 0.044 | ft/hr | Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.88% | % | VT Override to 0.88% \| SHXP-C3 Samples |

**Reagent Use**

| | | | |
|---|---|---|---|
| NaOH Dose | 3.67 | g/L of Feed | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |

**Equipment Specifications**

| | | | |
|---|---|---|---|
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 4942.15 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Slurry Feed Rate | 8385 | gpm | |
| Slurry Feed Rate | 31,741 | lpm | |
| Precip Mass Flow | 17.82 | tph | |
| Thickener Feed % Solids (eff) | 0.8% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.017 | g/cc | |

**Other Indicators**

| | | | |
|---|---|---|---|
| Conditioning Vessel Size | 238.06 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 84.01 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 20.94 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 14681 | ft^2 | |
| Thickener Diameter | 136.75 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | 28.05 | m/m | target = 6.35 |

*PM6: RE-Redissolution*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 7.6% | % | VT Override from 10.7% \| Based on Design |
| Dissolved Solids in Re-Leach | 11% | % | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Leaching Retention Time | 120 | min | UK: Design |
| **Reagent Use** | | | |
| Acid Consumption | 1122.3 | kg/ton | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| | 0.0 | | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Solid Feed Rate | 17.82 | tph | |
| Slurry Flow (feed) | 888 | gpm | |
| Product Flow | 888 | gpm | |
| Liquid Density | 1.057 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 100.8 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 9.47 | v/v | target = 10.0, based on data |
| Liquid Balance | 20.94 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 80.2 | g/L | |

*PM7: REE Oxalate Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | Based on Design |
| Precip Solid Yield | 0.7 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Thickener UF %Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| NaOH Dose | 1.86 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Oxalic Acid Dose | 7.58 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 887.74 | gpm | |
| Slurry Feed Rate | 3360.44 | lpm | |
| Precip Mass Flow | 0.15 | tph | |
| Thickener Feed % Solids (eff) | 0.06% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.09 | tph | |
| Liquid Density | 1.056 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 16.80 | m^3 | |
| Filter Press Area | 0.71 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.18 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 5.70 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.71 | m/m | |

*PM8: Sc SX Separation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | UK: Design |
| Settler Residence Time | 15.00 | min | UK: Design |
| Loading: Number of Stages | 2.00 | - | UK: Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Loading O:A Advance Ratio | 0.03 | v/v | UK: Design |
| Scrubbing: Number of Stages | | - | N/A |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | N/A |
| Scrubbing O:A Advance Ratio | | v/v | N/A |
| Stripping: Number of Stages | 2.00 | - | UK: Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | UK: Design |
| Sc2O3 Grade | 2% | % | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Precip Solid Yield | 0.4 | g/L | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance (SX)** | | | |
| PLS Flow | 8403.7 | gpm | |
| Raffinate Rate: | 8403.7 | gpm | |
| Load Org Recycle | 8172.7 | gpm | |
| Load Org Feed | 8403.7 | gpm | |
| Loag Org Advance | 231.0 | gpm | |
| Scrub Aq Product | 0.0 | gpm | |
| Scrub Aq Recycle | 0.0 | gpm | |
| Scrub Aq Feed | 0.0 | gpm | |
| Scurb Org Product | 231.0 | gpm | |
| Strip Aq Product | 18.4 | gpm | |
| Strip Aq Recycle | 212.6 | gpm | |
| Strip Aq Feed | 231.0 | gpm | |
| Strip Org Product | 231.0 | gpm | |
| Extraction Feed (O+A): | 16807.5 | gpm | |
| Scrub Feed (O+A): | 231.0 | gpm | |
| Strip Feed (O+A): | 462.1 | gpm | |
| **Material Balance (Precip)** | | | |
| Strip Solution Flow | 18.4 | gpm | |
| Solid Density | 3.4 | g/cc | |
| Solid Product Rate | 0.00 | tph | |
| Liquid Density (after precip) | 1.13 | g/cc | |
| Thickener Feed % Solids (eff) | 0.03% | % | |
| **Total Volumes** | | | |
| Total Extraction Volume | 537839 | gal | |
| Total Scrubbing Volume | 0 | gal | |
| Total Stripping Volume | 14787 | gal | |
| Extraction Org Volume | 268919.28 | gal | |
| Scrubbing Org Volume | 0.00 | gal | |
| Stripping Org Volume | 7393.56 | gal | |
| Total Org Volume | 276312.84 | gal | |
| **Other Indicators** | | | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 10.47 | m^3 | |
| Filter Press Area | 0.02 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM9: CM Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 30 | min | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Precip Solid Yield | 1.33 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Precip Settlng Rate | 10 | ft/hr | |
| Thickener UF % Solids | 25% | % | Not Available |
| **Reagent Use** | | | |
| NaOH Dose | 2.96 | g/L of Feed | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 0.00 | min | N/A |
| Thickener Capacity | 2399.30 | ft^2/tph | N/A |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 8364 | gpm | |
| Slurry Feed Rate | 31,662 | lpm | |
| Precip Mass Flow | 2.79 | tph | |
| Thickener Feed % Solids (eff) | 0.1% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.016 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 189.97 | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | 3.27 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 1337 | ft^2 | |
| Thickener Diameter | 41.26 | ft | ideal < 100' |

## PM10: CM Redissolution

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |

**Process Parameters**

| | | | |
|---|---|---|---|
| Re-Leach %Solids | 20.9% | % | VT Override from 20.9% \|\| SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Dissolved Solids in Re-Leach | 100% | % | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Leaching Retention Time | 60 | min | SHXP-C3/SHXP-C3 MASS BALANCE V3 |

**Reagent Use**

| | | | |
|---|---|---|---|
| Acid Consumption | 411.78 | kg/ton | SHXP-C3/SHXP-C3 MASS BALANCE V3 |

**Equipment Specification**

| | | | |
|---|---|---|---|
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 2.79 | tph | |
| Slurry Flow (feed) | 45 | gpm | |
| Product Flow | 45 | gpm | |
| Liquid Density | 1.173 | g/cc | |

**Other Indicators**

| | | | |
|---|---|---|---|
| Releach Vessel Size | 2.6 | m^3 | |
| Releach Conc. Factor | 184.06 | v/v | target = 102, based on data |
| Liquid Balance | 3.27 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 244.8 | g/L | |

## PM11: CM Recovery

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |

**Process Parameters**

| | | | |
|---|---|---|---|
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 2.29 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | 0% | % | SHXP-C3 Samples |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 2.91 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | 1% | % | SHXP-C3 Samples |

**Reagent Use**

| | | | |
|---|---|---|---|
| Na2S Dose | 8.04 | g/L of Feed | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| NaOH Dose | 12.59 | g/L of Feed | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| HCL Dose | 7.58 | g/L of Feed | SHXP-C3/SHXP-C3 MASS BALANCE V3 |

**Equipment Specifications**

| | | | |
|---|---|---|---|
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 933.00 | gpm | |
| Co Slurry Feed Rate | 3531.80 | lpm | |
| Co Precip Mass Flow | 0.53 | tph | |
| Co Thickener Feed % Solids (eff) | 0.22% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 932.37 | gpm | |
| Mn Slurry Feed Rate | 3529.42 | lpm | |
| Mn Precip Mass Flow | 0.68 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.27% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.06 | g/cc | |
| Mn Liquid Density | 1.06 | g/cc | |

**Other Indicators**

| | | | |
|---|---|---|---|
| Co Conditioning Vessel Size | 53.0 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 2.52 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.63 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 52.9 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 3.2 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.80 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

## PM12: Biooxidation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |

**Process Parameters**

| | | | |
|---|---|---|---|
| Raw Leaching Solution Requirec | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |

**Reagent Use**

| | | | |
|---|---|---|---|
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |

**Equipment Specifications**

| | | | |
|---|---|---|---|
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |

**Other Indicators**

| | | | |
|---|---|---|---|
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |

**Cost Indicators**

| | | | |
|---|---|---|---|
| Annual OPEX | $ | - | $/yr |
| Solution Cost (OPEX) | $ | - | $/L |
| Solution Cost (OPEX) | $ | - | $/mt |
| Acid Cost (OPEX) | $ | - | $/kg acid equivalent |
| Total CAPEX (Installed) | $ | - | $ |
| Annualized CAPEX | $ | - | $/yr |
| Solution Cost (CAPEX) | $ | - | $/L |
| Solution Cost (CAPEX) | $ | - | $/mt |
| Acid Cost (CAPEX) | $ | - | $/kg acid equivalent |
| Solution Cost (Total) | $ | - | $/L |
| Solution Cost (Total) | $ | - | $/mt |
| Acid Cost (Total) | $ | - | $/kg acid equivalent |

# CAPITAL COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/13/2023
Case: SHXP-C3 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 6,318,002 | $ - | $ 1,579,500 | $ 6,949,802 | $ 8,652,504 | $ 23,499,808 | IIIIIIIIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 4,241,248 | $ - | $ 1,060,312 | $ 4,665,373 | $ 5,808,389 | $ 15,775,322 | IIIIIIIIIIIIIIII |
| PM5 | REE Precipitation | $ 8,207,556 | $ - | $ 2,051,889 | $ 9,028,311 | $ 11,240,248 | $ 30,528,004 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| PM6 | RE-Redissolution | $ 731,126 | $ - | $ 182,781 | $ 804,239 | $ 1,001,277 | $ 2,719,423 | III |
| PM7 | REE Oxalate Precipitation | $ 834,596 | $ - | $ 208,649 | $ 918,056 | $ 1,142,980 | $ 3,104,282 | III |
| PM8 | Sc SX Separation | $ 2,183,808 | $ 4,895,262 | $ 545,952 | $ 2,402,189 | $ 2,990,725 | $ 13,017,937 | IIIIIIIIIIIIII |
| PM9 | CM Precipitation | $ 2,100,595 | $ - | $ 525,149 | $ 2,310,655 | $ 2,876,765 | $ 7,813,164 | IIIIIIII |
| PM10 | CM Redissolution | $ 31,847 | $ - | $ 7,962 | $ 35,032 | $ 43,615 | $ 118,456 | |
| PM11 | CM Recovery | $ 643,104 | $ - | $ 160,776 | $ 707,414 | $ 880,731 | $ 2,392,025 | II |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| | TOTAL | $ 27,755,542 | $ 4,895,262 | $ 6,938,886 | $ 30,531,096 | $ 38,011,215 | $ 108,132,002 | |

## Module Summary

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| Total Lang Factor (=1+sum % Base) | | 372% | |

Year of Construction: 2025

# OPERATING COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/13/2023
Case: SHXP-C3 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 1.03 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 0.39 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 108,132,002 | $ Total | $ 1,072.74 | $ 3,218.21 | $ 1,081,320 | $ 0.36 | $ 0.27 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 0.10 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 724,975,145 | $ Total | $ 3,596.11 | $ 10,788.32 | $ 3,624,876 | $ 1.21 | $ 0.90 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 0.15 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 108,132,002 | $ Total | $ 1,072.74 | $ 3,218.21 | $ 1,081,320 | $ 0.36 | $ 0.27 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 10,723.03** | **$ 32,169.10** | **$ 10,808,817** | **$ 3.61** | **$ 2.68** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 0.17 | $ 0.13 |
| PM2 | Roasting | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | PR | 1280 | | $ 613.76 | $ 1,841.29 | $ 618,673 | $ 0.21 | $ 0.15 |
| PM4 | Al Precipitation | PR | 843 | | $ 403.97 | $ 1,211.91 | $ 407,201 | $ 0.14 | $ 0.10 |
| PM5 | REE Precipitation | PR | 889 | | $ 426.04 | $ 1,278.12 | $ 429,447 | $ 0.14 | $ 0.11 |
| PM6 | RE-Redissolution | PR | 182 | | $ 87.24 | $ 261.71 | $ 87,933 | $ 0.03 | $ 0.02 |
| PM7 | REE Oxalate Precipitation | PR | 76 | | $ 36.24 | $ 108.73 | $ 36,534 | $ 0.01 | $ 0.01 |
| **Sub-Total** | | | **5624.3** | | **$ 2,696.01** | **$ 8,088.03** | **$ 2,717,579.20** | **$ 0.91** | **$ 0.67** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.03 | $ 0.03 |
| PM2 | Roasting | CR | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | CR | | $ 233,034.54 | $ 699,103.63 | $ 234,898,819 | $ 78.55 | $ 58.26 |
| PM4 | Al Precipitation | CR | | $ 34,957.02 | $ 104,871.05 | $ 35,236,673 | $ 11.78 | $ 8.74 |
| PM5 | REE Precipitation | CR | | $ 6,989.37 | $ 20,968.12 | $ 7,045,287 | $ 2.36 | $ 1.75 |
| PM6 | RE-Redissolution | CR | | $ 43,206.79 | $ 129,620.36 | $ 43,552,443 | $ 14.56 | $ 10.80 |
| PM7 | REE Oxalate Precipitation | CR | | $ 7,719.91 | $ 23,159.74 | $ 7,781,673 | $ 2.60 | $ 1.93 |
| **Sub-Total** | | | | **$ 344,637.63** | **$ 1,033,912.89** | **$ 347,394,732.39** | **$ 116.18** | **$ 86.16** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 8,136 | $ 0.10 | $ 271.20 | $ 813.60 | $ 273,370 | $ 0.09 | $ 0.07 |
| Non-Hazardous Precipitate | WD | Tons | 381 | $ 1.50 | $ 190.41 | $ 571.22 | $ 191,929 | $ 0.06 | $ 0.05 |
| Dust/Volatiles | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 461.61** | **$ 1,384.82** | **$ 465,298.38** | **$ 0.16** | **$ 0.12** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 347,795** | **$ 1,043,386** | **$ 350,577,610** | **$ 117** | **$ 87** |

## Overhead Costs

| Allocation | Code | Rate | Basis | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% | % of Direct Cost | $ 32,169 Daily Direct Cost | $ 1,608.45 | $ 4,825.36 | $ 1,621,323 | $ 0.54 | $ 0.40 |
| Variable Portion | OH | 15% | % of Direct Cost | $ 1,043,386 Daily Direct Cost | $ 52,169.29 | $ 156,507.86 | $ 52,586,641 | $ 17.59 | $ 13.04 |
| **TOTAL OVERHEAD COSTS** | | | | | **$ 53,777.74** | **$ 161,333.23** | **$ 54,207,964.10** | **$ 18.13** | **$ 13.44** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 12000 TPD | Date: | 3/13/2023 |
| Case: | SHXP-C3 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 10,723.03 | $ 32,169.10 | $ 10,808,817 | $ 4 | $ 2.68 | I |
| Power | $ 2,696.01 | $ 8,088.03 | $ 2,717,579 | $ 1 | $ 0.67 | |
| Consumables | $ 344,637.63 | $ 1,033,912.89 | $ 347,394,732 | $ 116 | $ 86.16 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| Leasing/Tolling | $ - | $ - | $ - | $ - | $ - | |
| Waste Disposal | $ 461.61 | $ 1,384.82 | $ 465,298 | $ 0 | $ 0.12 | |
| General Overhead | $ 53,777.74 | $ 161,333.23 | $ 54,207,964 | $ 18 | $ 13.44 | IIIIII |
| TOTAL | $ 412,296 | $ 1,236,888 | $ 415,594,391 | $ 139 | $ 103.07 | |

## Summary by Process Module

| | Process Module | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 611 | $ 1,834 | $ 616,370 | $ 0.21 | $ 0.15 | |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 233,648 | $ 700,945 | $ 235,517,492 | $ 78.76 | $ 58.41 | IIIIIIIIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 35,361 | $ 106,083 | $ 35,643,874 | $ 11.92 | $ 8.84 | IIIII |
| PM5 | REE Precipitation | $ 7,415 | $ 22,246 | $ 7,474,734 | $ 2.50 | $ 1.85 | |
| PM6 | RE-Redissolution | $ 43,294 | $ 129,882 | $ 43,640,376 | $ 14.59 | $ 10.82 | IIIII |
| PM7 | REE Oxalate Precipitation | $ 7,756 | $ 23,268 | $ 7,818,207 | $ 2.61 | $ 1.94 | |
| PM8 | Sc SX Separation | $ 3,204 | $ 9,613 | $ 3,230,084 | $ 1.08 | $ 0.80 | |
| PM9 | CM Precipitation | $ 5,926 | $ 17,778 | $ 5,973,361 | $ 2.00 | $ 1.48 | |
| PM10 | CM Redissolution | $ 2,478 | $ 7,434 | $ 2,497,763 | $ 0.84 | $ 0.62 | |
| PM11 | CM Recovery | $ 7,639 | $ 22,917 | $ 7,700,052 | $ 2.58 | $ 1.91 | |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| NA | Unallocated | $ 64,962 | $ 194,887 | $ 65,482,080 | $ 22 | $ 16.24 | IIIIIIII |
| | TOTAL | $ 412,296 | $ 1,236,888 | $ 415,594,391 | $ 139 | $ 103.07 | |

## Summary by Cost Center

| Cost Center | Code | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| Operating Labor | OL | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 1 | $ 0.76 | |
| Technical Labor | TL | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 0 | $ 0.29 | |
| Power/Utilities | PR | $ 2,696.01 | $ 8,088.03 | $ 2,717,579 | $ 1 | $ 0.67 | |
| Lease Agreements | LS | $ - | $ - | $ - | $ - | $ - | |
| Consumables/Reagents | CR | $ 344,637.63 | $ 1,033,912.89 | $ 347,394,732 | $ 116 | $ 86.16 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| Waste Disposal | WD | $ 461.61 | $ 1,384.82 | $ 465,298 | $ 0 | $ 0.12 | |
| Plant Overhead | OH | $ 55,155.34 | $ 165,466.02 | $ 55,596,583 | $ 19 | $ 13.79 | IIIIII |
| General Sales, and Administration | GSA | $ 5,126.13 | $ 15,378.40 | $ 5,167,144 | $ 2 | $ 1.28 | |
| Reserved | | | | | | | |
| TOTAL | | $ 412,296 | $ 1,236,888.07 | $ 415,594,391 | $ 139 | $ 103.07 | |

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/13/2023 |
| Case: | SHXP-C3 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 724,975,145

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 1218 | 1250 |
| Contained Value | ($/ton) | $ 33.3 | $ 217.7 |
| Feed Basket Price | ($/kg) | $ 30.2 | $ 192.0 |
| Product Basket Price | ($/kg) | $ 29.5 | $ 244.0 |
| Production Rate | (kg/day) | 8601.82 | 8899.56 |
| | (t/day) | 8.60 | 8.90 |
| | (t/yr) | 2890.21 | 2990.25 |
| | (% of US Demand) | 28.9% | |

## Revenue Contribution Table

| | | Product Revenue ($/hr) | | | | | |
|---|---|---|---|---|---|---|---|
| **Element** | **REE/CM Price ($/kg) [1]** | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | **REE Revenue ($/day)** |
| Scandium | $ 4,200.00 | $ 61.8 | $ 79,828.3 | $ - | $ - | $ - | $ 1,917,361 |
| Yttrium | $ 6.00 | $ 328.7 | $ - | $ - | $ - | $ - | $ 7,888 |
| Lanthanum | $ 2.00 | $ 41.3 | $ - | $ - | $ - | $ - | $ 992 |
| Cerium | $ 2.00 | $ 143.2 | $ - | $ - | $ - | $ - | $ 3,438 |
| Praseodymium | $ 52.00 | $ 430.6 | $ - | $ - | $ - | $ - | $ 10,335 |
| Neodymium | $ 42.00 | $ 1,673.7 | $ - | $ - | $ - | $ - | $ 40,169 |
| Samarium | $ 2.11 | $ 21.1 | $ - | $ - | $ - | $ - | $ 506 |
| Europium | $ 34.00 | $ 67.0 | $ - | $ - | $ - | $ - | $ 1,608 |
| Gadolinium | $ 32.00 | $ 356.7 | $ - | $ - | $ - | $ - | $ 8,561 |
| Terbium | $ 400.00 | $ 409.0 | $ - | $ - | $ - | $ - | $ 9,817 |
| Dysprosium | $ 230.00 | $ 2,171.2 | $ - | $ - | $ - | $ - | $ 52,110 |
| Holmium | $ 52.96 | $ 93.7 | $ - | $ - | $ - | $ - | $ 2,248 |
| Erbium | $ 34.00 | $ 137.1 | $ - | $ - | $ - | $ - | $ 3,289 |
| Thulium | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Ytterbium | $ 28.72 | $ 124.8 | $ - | $ - | $ - | $ - | $ 2,996 |
| Lutetium | $ 796.64 | $ 516.3 | $ - | $ - | $ - | $ - | $ 12,390 |
| Manganese | $ 0.005 | $ - | $ - | $ 0.4 | $ - | $ - | $ 10 |
| Cobalt | $ 22.00 | $ - | $ - | $ - | $ 1,183.6 | $ - | $ 28,405 |
| Nickel | $ 18.00 | $ - | $ - | $ - | $ 2,156.6 | $ - | $ 51,758 |
| Zinc | $ 3.02 | $ - | $ - | $ - | $ 157.5 | $ - | $ 3,781 |
| Reserved | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Total: | | $ 6,576.3 | $ 79,828.3 | $ 0.4 | $ 3,497.7 | $ - | $ 2,157,664 |

[1] REEs listed as $/kg REO; Mn listed as $/kg 44% Mn Ore; Co listed as $/kg metal content

## Coal Revenue

List Coal Revenue Sources Here:

**Annual Coal Revenue:** $ -

## Other Revenue

List Other Revenue Sources Here:

**Annual Other Revenue:** $ -

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/13/2023
Case: SHXP-C3 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 792,199,915 | $ 815,965,913 | $ 840,444,890 | $ 865,658,237 | $ 891,627,984 | $ 918,376,823 | $ 945,928,128 | $ 974,305,972 | $ 1,003,535,151 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 792,199,915 | $ 815,965,913 | $ 840,444,890 | $ 865,658,237 | $ 891,627,984 | $ 918,376,823 | $ 945,928,128 | $ 974,305,972 | $ 1,003,535,151 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 324,396 | $ 2,316,230 | $ 457,312,890 | $ 470,667,765 | $ 484,415,215 | $ 498,566,531 | $ 513,133,318 | $ 528,127,495 | $ 543,561,306 | $ 559,447,330 | $ 575,798,485 |
| *- Operating Costs* | $ - | $ - | $ - | $ 454,131,213 | $ 467,755,149 | $ 481,787,803 | $ 496,241,438 | $ 511,128,681 | $ 526,462,541 | $ 542,256,417 | $ 558,524,110 | $ 575,279,833 |
| *- Loan #1 Interest* | $ - | $ 324,396 | $ 299,785 | $ 273,697 | $ 246,044 | $ 216,731 | $ 185,660 | $ 152,725 | $ 117,813 | $ 80,807 | $ 41,580 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 2,016,446 | $ 1,863,462 | $ 1,701,299 | $ 1,529,407 | $ 1,347,201 | $ 1,154,063 | $ 949,336 | $ 732,326 | $ 502,295 | $ 258,463 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 1,044,519 | $ 965,273 | $ 881,273 | $ 792,233 | $ 697,850 | $ 597,805 | $ 491,756 | $ 379,345 | $ 260,189 |
| Gross Profit | $ - | $ (324,396) | $ (2,316,230) | $ 334,887,025 | $ 345,298,148 | $ 356,029,675 | $ 367,091,705 | $ 378,494,666 | $ 390,249,329 | $ 402,366,822 | $ 414,858,642 | $ 427,736,666 |
| **Book Deductions** | $ - | $ 405,495 | $ 2,926,404 | $ 4,236,412 | $ 3,918,681 | $ 3,624,780 | $ 3,352,921 | $ 3,101,452 | $ 2,868,843 | $ 2,653,680 | $ 2,454,654 | $ 2,270,555 |
| *- CAPEX #1 Depreciation* | $ - | $ 405,495 | $ 375,083 | $ 346,952 | $ 320,930 | $ 296,861 | $ 274,596 | $ 254,001 | $ 234,951 | $ 217,330 | $ 201,030 | $ 185,953 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 2,551,321 | $ 2,359,972 | $ 2,182,974 | $ 2,019,251 | $ 1,867,807 | $ 1,727,722 | $ 1,598,143 | $ 1,478,282 | $ 1,367,411 | $ 1,264,855 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,529,488 | $ 1,414,776 | $ 1,308,668 | $ 1,210,518 | $ 1,119,729 | $ 1,035,749 | $ 958,068 | $ 886,213 | $ 819,747 |
| Taxable Before Depletion | $ - | $ (729,891) | $ (5,242,634) | $ 330,650,614 | $ 341,379,467 | $ 352,404,896 | $ 363,738,784 | $ 375,393,214 | $ 387,380,485 | $ 399,713,142 | $ 412,403,988 | $ 425,466,112 |
| **Depletion** | $ - | $ - | $ - | $ 110,907,988 | $ 114,235,228 | $ 117,662,285 | $ 121,192,153 | $ 124,827,918 | $ 128,572,755 | $ 132,429,938 | $ 136,402,836 | $ 140,494,921 |
| *- 50% Limit* | $ - | $ (364,946) | $ (2,621,317) | $ 165,325,307 | $ 170,689,733 | $ 176,202,448 | $ 181,869,392 | $ 187,696,607 | $ 193,690,243 | $ 199,856,571 | $ 206,201,994 | $ 212,733,056 |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 110,907,988 | $ 114,235,228 | $ 117,662,285 | $ 121,192,153 | $ 124,827,918 | $ 128,572,755 | $ 132,429,938 | $ 136,402,836 | $ 140,494,921 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (729,891) | $ (5,242,634) | $ 219,742,625 | $ 227,144,239 | $ 234,742,611 | $ 242,546,631 | $ 250,565,296 | $ 258,807,730 | $ 267,283,204 | $ 276,001,152 | $ 284,971,190 |
| *- Income Tax* | $ - | $ (189,772) | $ (1,363,085) | $ 57,133,083 | $ 59,057,502 | $ 61,033,079 | $ 63,062,124 | $ 65,146,977 | $ 67,290,010 | $ 69,493,633 | $ 71,760,299 | $ 74,092,510 |
| Net Profit/Loss | $ - | $ (540,119) | $ (3,879,549) | $ 162,609,543 | $ 168,086,737 | $ 173,709,532 | $ 179,484,507 | $ 185,418,319 | $ 191,517,720 | $ 197,789,571 | $ 204,240,852 | $ 210,878,681 |
| **Book Deductions** | $ - | $ 405,495 | $ 2,926,404 | $ 115,144,400 | $ 118,153,908 | $ 121,287,064 | $ 124,545,074 | $ 127,929,370 | $ 131,441,598 | $ 135,083,618 | $ 138,857,490 | $ 142,765,476 |
| *+ CAPEX #1 Depreciation* | $ - | $ 405,495 | $ 375,083 | $ 346,952 | $ 320,930 | $ 296,861 | $ 274,596 | $ 254,001 | $ 234,951 | $ 217,330 | $ 201,030 | $ 185,953 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 2,551,321 | $ 2,359,972 | $ 2,182,974 | $ 2,019,251 | $ 1,867,807 | $ 1,727,722 | $ 1,598,143 | $ 1,478,282 | $ 1,367,411 | $ 1,264,855 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,529,488 | $ 1,414,776 | $ 1,308,668 | $ 1,210,518 | $ 1,119,729 | $ 1,035,749 | $ 958,068 | $ 886,213 | $ 819,747 |
| *+ Depletion* | $ - | $ - | $ - | $ 110,907,988 | $ 114,235,228 | $ 117,662,285 | $ 121,192,153 | $ 124,827,918 | $ 128,572,755 | $ 132,429,938 | $ 136,402,836 | $ 140,494,921 |
| Cash Flow | $ - | $ (134,624) | $ (953,145) | $ 277,753,942 | $ 286,240,645 | $ 294,996,597 | $ 304,029,581 | $ 313,347,689 | $ 322,959,319 | $ 332,873,189 | $ 343,098,342 | $ 353,644,157 |
| **Captial Cost** | $ 5,406,600 | $ 34,017,614 | $ 20,393,173 | $ 4,484,356 | $ 4,753,417 | $ 5,038,622 | $ 5,340,940 | $ 5,661,396 | $ 6,001,080 | $ 6,361,144 | $ 6,742,813 | $ 6,412,798 |
| *- Fixed Capital Costs* | $ 10,813,200 | $ 67,214,852 | $ 34,817,294 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 5,406,600 | $ 33,607,426 | $ 17,408,647 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 410,188 | $ 434,799 | $ 460,887 | $ 488,540 | $ 517,853 | $ 548,924 | $ 581,859 | $ 616,771 | $ 653,777 | $ 693,004 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 2,549,727 | $ 2,702,710 | $ 2,864,873 | $ 3,036,765 | $ 3,218,971 | $ 3,412,110 | $ 3,616,836 | $ 3,833,846 | $ 4,063,877 | $ 4,307,710 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 1,320,758 | $ 1,400,004 | $ 1,484,004 | $ 1,573,044 | $ 1,667,427 | $ 1,767,473 | $ 1,873,521 | $ 1,985,932 | $ 2,105,088 |
| **Net Cash Flow** | $ (5,406,600) | $ (34,152,238) | $ (21,346,318) | $ 273,269,587 | $ 281,487,228 | $ 289,957,974 | $ 298,688,642 | $ 307,686,293 | $ 316,958,239 | $ 326,512,044 | $ 336,355,529 | $ 347,231,359 |
| Discounted Cash Flow | $ (5,406,600) | $ (31,047,489) | $ (17,641,585) | $ 205,311,485 | $ 192,259,564 | $ 180,041,089 | $ 168,601,951 | $ 157,891,719 | $ 147,863,358 | $ 138,472,980 | $ 129,679,617 | $ 121,702,473 |
| **Cumulative DCF** | $ (5,406,600) | $ (36,454,089) | $ (54,095,674) | $ 151,215,811 | $ 343,475,375 | $ 523,516,464 | $ 692,118,416 | $ 850,010,135 | $ 997,873,492 | $ 1,136,346,473 | $ 1,266,026,090 | $ 1,387,728,563 |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 108,132,002 | Payback Period | 3.5 |
| Constant Dollar OPEX | $ 415,594,391 | Internal Rate of Return | 177% |
| Constant Dollar REE Rev | $ 724,975,145 | Net Present Value | $ 2,326,320,433 |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ 1,569,468,650 |

# PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/13/2023
Case: SHXP-C3 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 1,033,641,206** | **$ 1,064,650,442** | **$ 1,096,589,955** | **$ 1,129,487,654** | **$ 1,163,372,283** | **$ 1,198,273,452** | **$ 1,234,221,655** | **$ 1,271,248,305** | **$ 1,309,385,754** | **$ 1,348,667,327** | **$ 1,389,127,347** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 1,033,641,206 | $ 1,064,650,442 | $ 1,096,589,955 | $ 1,129,487,654 | $ 1,163,372,283 | $ 1,198,273,452 | $ 1,234,221,655 | $ 1,271,248,305 | $ 1,309,385,754 | $ 1,348,667,327 | $ 1,389,127,347 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 592,672,112** | **$ 610,314,375** | **$ 628,623,806** | **$ 647,482,520** | **$ 666,906,996** | **$ 686,914,206** | **$ 707,521,632** | **$ 728,747,281** | **$ 750,609,699** | **$ 773,127,990** | **$ 796,321,830** |
| *- Operating Costs* | $ 592,538,228 | $ 610,314,375 | $ 628,623,806 | $ 647,482,520 | $ 666,906,996 | $ 686,914,206 | $ 707,521,632 | $ 728,747,281 | $ 750,609,699 | $ 773,127,990 | $ 796,321,830 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 133,884 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ 440,969,094 | $ 454,336,067 | $ 467,966,149 | $ 482,005,133 | $ 496,465,287 | $ 511,359,246 | $ 526,700,023 | $ 542,501,024 | $ 558,776,055 | $ 575,539,336 | $ 592,805,516 |
| **Book Deductions** | **$ 2,100,263** | **$ 1,942,744** | **$ 1,797,038** | **$ 1,662,260** | **$ 1,537,590** | **$ 1,422,271** | **$ 1,315,601** | **$ 1,216,931** | **$ 1,125,661** | **$ 955,962** | **$ 347,727** |
| *- CAPEX #1 Depreciation* | $ 172,006 | $ 159,106 | $ 147,173 | $ 136,135 | $ 125,925 | $ 116,481 | $ 107,744 | $ 99,664 | $ 92,189 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 1,169,991 | $ 1,082,241 | $ 1,001,073 | $ 925,993 | $ 856,543 | $ 792,303 | $ 732,880 | $ 677,914 | $ 627,070 | $ 580,040 | $ - |
| *- CAPEX #3 Depreciation* | $ 758,266 | $ 701,396 | $ 648,791 | $ 600,132 | $ 555,122 | $ 513,488 | $ 474,976 | $ 439,353 | $ 406,402 | $ 375,922 | $ 347,727 |
| Taxable Before Depletion | $ 438,868,831 | $ 452,393,323 | $ 466,169,111 | $ 480,342,873 | $ 494,927,697 | $ 509,936,975 | $ 525,384,422 | $ 541,284,093 | $ 557,650,394 | $ 574,583,375 | $ 592,457,789 |
| **Depletion** | $ 144,709,769 | $ 149,051,062 | $ 153,522,594 | $ 158,128,272 | $ 162,872,120 | $ 167,758,283 | $ 172,791,032 | $ 177,974,763 | $ 183,314,006 | $ 188,813,426 | $ 194,477,829 |
| *- 50% Limit* | $ 219,434,415 | $ 226,196,662 | $ 233,084,556 | $ 240,171,437 | $ 247,463,848 | $ 254,968,487 | $ 262,692,211 | $ 270,642,047 | $ 278,825,197 | $ 287,291,687 | $ 296,228,894 |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 144,709,769 | $ 149,051,062 | $ 153,522,594 | $ 158,128,272 | $ 162,872,120 | $ 167,758,283 | $ 172,791,032 | $ 177,974,763 | $ 183,314,006 | $ 188,813,426 | $ 194,477,829 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ 294,159,062** | **$ 303,342,261** | **$ 312,646,517** | **$ 322,214,602** | **$ 332,055,577** | **$ 342,178,692** | **$ 352,593,391** | **$ 363,309,331** | **$ 374,336,388** | **$ 385,769,949** | **$ 397,979,960** |
| *- Income Tax* | $ 76,481,356 | $ 78,868,988 | $ 81,288,095 | $ 83,775,796 | $ 86,334,450 | $ 88,966,460 | $ 91,674,282 | $ 94,460,426 | $ 97,327,461 | $ 100,300,187 | $ 103,474,790 |
| Net Profit/Loss | $ 217,677,706 | $ 224,473,273 | $ 231,358,423 | $ 238,438,805 | $ 245,721,127 | $ 253,212,232 | $ 260,919,109 | $ 268,848,905 | $ 277,008,927 | $ 285,469,762 | $ 294,505,171 |
| **Book Deductions** | **$ 146,810,032** | **$ 150,993,805** | **$ 155,319,631** | **$ 159,790,531** | **$ 164,409,710** | **$ 169,180,554** | **$ 174,106,633** | **$ 179,191,693** | **$ 184,439,667** | **$ 189,769,387** | **$ 194,825,556** |
| *+ CAPEX #1 Depreciation* | $ 172,006 | $ 159,106 | $ 147,173 | $ 136,135 | $ 125,925 | $ 116,481 | $ 107,744 | $ 99,664 | $ 92,189 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 1,169,991 | $ 1,082,241 | $ 1,001,073 | $ 925,993 | $ 856,543 | $ 792,303 | $ 732,880 | $ 677,914 | $ 627,070 | $ 580,040 | $ - |
| *+ CAPEX #3 Depreciation* | $ 758,266 | $ 701,396 | $ 648,791 | $ 600,132 | $ 555,122 | $ 513,488 | $ 474,976 | $ 439,353 | $ 406,402 | $ 375,922 | $ 347,727 |
| *+ Depletion* | $ 144,709,769 | $ 149,051,062 | $ 153,522,594 | $ 158,128,272 | $ 162,872,120 | $ 167,758,283 | $ 172,791,032 | $ 177,974,763 | $ 183,314,006 | $ 188,813,426 | $ 194,477,829 |
| Cash Flow | $ 364,487,738 | $ 375,467,079 | $ 386,678,054 | $ 398,229,337 | $ 410,130,837 | $ 422,392,786 | $ 435,025,742 | $ 448,040,598 | $ 461,448,594 | $ 475,239,150 | $ 489,330,727 |
| **Captial Cost** | **$ 2,231,394** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 2,231,394 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ 362,256,344 | $ 375,467,079 | $ 386,678,054 | $ 398,229,337 | $ 410,130,837 | $ 422,392,786 | $ 435,025,742 | $ 448,040,598 | $ 461,448,594 | $ 475,239,150 | $ 489,330,727 |
| Discounted Cash Flow | $ 115,426,035 | $ 108,759,439 | $ 101,824,417 | $ 95,332,937 | $ 89,256,420 | $ 83,568,161 | $ 78,243,203 | $ 73,258,218 | $ 68,591,393 | $ 64,219,338 | $ 60,112,310 |
| **Cumulative DCF** | $ 1,503,154,598 | $ 1,611,914,037 | $ 1,713,738,454 | $ 1,809,071,391 | $ 1,898,327,810 | $ 1,981,895,971 | $ 2,060,139,175 | $ 2,133,397,393 | $ 2,201,988,786 | $ 2,266,208,124 | $ 2,326,320,433 |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

## Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 108,132,002 | Payback Period | 0 | 3.5 |
| Constant Dollar OPEX | $ 415,594,391 | Internal Rate of Return | 0 | 177% |
| Constant Dollar REE Rev | $ 724,975,145 | Net Present Value | 0 | $ 2,326,320,433 |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ 1,569,468,650 |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/13/2023
By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 0.10 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 0.06 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 7 | Conveyor | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 0.17** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.03 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.03** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 611** | **$ 1,834** | **$ 616,370** | **$ 0.21** | **$ 0.15** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/13/2023
By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 2.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 500 | tph |
| Residue Rate | 339 | tph |
| Slurry Flow | 9250 | gpm |
| Leaching Time | 45 | min |
| Acid Consumption | 264.10 | kg/ton |
| Bioleaching Sol'n Consumption | 0 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 2 | -- |
| Thickener Capacity | 10.75 | ft^2/tph |
| Polymer Dose | 0.016 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 8 | 100% | 52,031 | gal | $ 181,404 | $ 1,451,235 | $ - | |
| 2 | Tank Mixer | 8 | 100% | 78 | HP | $ 187,044 | $ 1,496,351 | $ - | |
| 3 | Process Pump | 4 | 100% | 4,625 | gpm | $ 161,957 | $ 647,827 | $ - | |
| 4 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 5 | Belt Filter | 2 | 100% | 170 | TPH | $ 1,124,090 | $ 2,248,181 | $ - | |
| 6 | Solution Heater | 1 | 100% | 1 | MBTU/hr | $ 30,807 | $ 30,807 | $ - | |
| 7 | Thickener | 2 | 100% | 1,823 | ft^2 | $ 174,023 | $ 348,045 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 6,318,002** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 624.38 | Effiicency: | 0.85 | | $ 299.30 | $ 897.89 | $ 301,691 | $ 0.10 | $ 0.07 |
| 3 | Process Pump | 462.50 | | | | $ 221.70 | $ 665.10 | $ 223,475 | $ 0.07 | $ 0.06 |
| 4 | Conveyor | 31.00 | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 5 | Belt Filter | 82.21 | | | | $ 39.41 | $ 118.22 | $ 39,722 | $ 0.01 | $ 0.01 |
| 6 | Solution Heater | 78.00 | | | | $ 37.39 | $ 112.16 | $ 37,687 | $ 0.01 | $ 0.01 |
| 7 | Thickener | 2.32 | | | | $ 1.11 | $ 3.33 | $ 1,120 | $ 0.00 | $ 0.00 |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1280** | | | | **$ 613.76** | **$ 1,841.29** | **$ 618,673** | **$ 0.21** | **$ 0.15** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | - | L/hr | $ 0.008 | / L | $ - | $ - | $ - | $ - | $ - |
| 2 | H2SO4 | 132.05 | mt/hr | $ 220.000 | / mt | $ 232,412.27 | $ 697,236.82 | $ 234,271,572 | $ 78.35 | $ 58.10 |
| 3 | Natural Gas | 0.75 | MCF/hr | $ 2.860 | / MCF | $ 17.21 | $ 51.63 | $ 17,346 | $ 0.01 | $ 0.00 |
| 4 | Polymer | 33.61 | kg/hr | $ 2.250 | / kg | $ 605.06 | $ 1,815.18 | $ 609,901 | $ 0.20 | $ 0.15 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 233,034.54** | **$ 699,103.63** | **$ 234,898,819** | **$ 78.55** | **$ 58.26** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| | **Total Operating Cost:** | **$ 233,648** | **$ 700,945** | **$ 235,517,492** | **$ 78.76** | **$ 58.41** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/13/2023
By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 7.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8404 | gpm |
| Feed Rate | 31812 | lpm |
| Precip Mass Flow | 15.87 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 974.97 | ft^2/tpd |
| NaOH Dose | 18.31 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 7 | 100% | 54,024 | gal | $ 123,520 | $ 864,638 | $ - | |
| 2 | Tank Mixer | 7 | 100% | 27 | HP | $ 102,814 | $ 719,699 | $ - | |
| 3 | Process Pump | 21 | 100% | 1,201 | gpm | $ 48,414 | $ 1,016,688 | $ - | |
| 4 | Filter Press | 1 | 100% | 75 | ft^3 | $ 204,067 | $ 204,067 | $ - | |
| 5 | Conveyor | 1 | 100% | 16 | TPH | $ 14,370 | $ 14,370 | $ - | |
| 6 | Thickener | 7 | 100% | 2,210 | ft^2 | $ 203,112 | $ 1,421,787 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 4,241,248** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 189.08 | Effiicency: | 0.85 | | $ 90.64 | $ 271.91 | $ 91,363 | $ 0.03 | $ 0.02 |
| 3 | Process Pump | 630.28 | | | | $ 302.13 | $ 906.38 | $ 304,544 | $ 0.10 | $ 0.08 |
| 4 | Filter Press | 12.55 | | | | $ 6.02 | $ 18.05 | $ 6,065 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 0.98 | | | | $ 0.47 | $ 1.41 | $ 475 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 9.84 | | | | $ 4.72 | $ 14.15 | $ 4,754 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **843** | | | | **$ 403.97** | **$ 1,211.91** | **$ 407,201** | **$ 0.14** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 34.96 | mt/hr | $ 125.000 | / mt | $ 34,957.02 | $ 104,871.05 | $ 35,236,673 | $ 11.78 | $ 8.74 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 34,957.02** | **$ 104,871.05** | **$ 35,236,673** | **$ 11.78** | **$ 8.74** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 35,361** **$ 106,083** **$ 35,643,874** **$ 11.92** **$ 8.84**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/13/2023
By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 6.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8385 | gpm |
| Feed Rate | 31741 | lpm |
| Precip Mass Flow | 17.82 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 4942.15 | ft^2/tpd |
| NaOH Dose | 3.67 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 6 | 100% | 62,888 | gal | $ 134,538 | $ 807,228 | $ - | |
| 2 | Tank Mixer | 6 | 100% | 31 | HP | $ 112,013 | $ 672,076 | $ - | |
| 3 | Process Pump | 18 | 100% | 1,398 | gpm | $ 51,302 | $ 923,436 | $ - | |
| 4 | Filter Press | 1 | 100% | 84 | ft^3 | $ 221,954 | $ 221,954 | $ - | |
| 5 | Conveyor | 1 | 100% | 18 | TPH | $ 15,317 | $ 15,317 | $ - | |
| 6 | Thickener | 6 | 100% | 14,681 | ft^2 | $ 927,924 | $ 5,567,546 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 8,207,556** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 188.66 | Effiicency: | 0.85 | | | $ 90.44 | $ 271.31 | $ 91,160 | $ 0.03 | $ 0.02 |
| 3 | Process Pump | 628.88 | | | | | $ 301.46 | $ 904.37 | $ 303,868 | $ 0.10 | $ 0.08 |
| 4 | Filter Press | 14.10 | | | | | $ 6.76 | $ 20.28 | $ 6,815 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 1.11 | | | | | $ 0.53 | $ 1.59 | $ 534 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 56.02 | | | | | $ 26.85 | $ 80.56 | $ 27,069 | $ 0.01 | $ 0.01 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **889** | | | | | **$ 426.04** | **$ 1,278.12** | **$ 429,447** | **$ 0.14** | **$ 0.11** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 6.99 | mt/hr | $ 125.000 | / mt | $ 6,989.37 | $ 20,968.12 | $ 7,045,287 | $ 2.36 | $ 1.75 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 6,989.37** | **$ 20,968.12** | **$ 7,045,287** | **$ 2.36** | **$ 1.75** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 7,415** | **$ 22,246** | **$ 7,474,734** | **$ 2.50** | **$ 1.85** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3

Date: 3/13/2023
By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 17.82 | tph |
| Slurry Feed Rate | 888 | gpm |
| Leaching Time | 120 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 1122.31 | kg/ton |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 26,632 | gal | $ 33,004 | $ 132,015 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 40 | HP | $ 128,203 | $ 512,813 | $ - | |
| 3 | Process Pump | 1 | 100% | 888 | gpm | $ 86,298 | $ 86,298 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 731,126** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 159.79 | Effiicency: | 0.85 | | | $ 76.60 | $ 229.79 | $ 77,210 | $ 0.03 | $ 0.02 |
| 3 | Process Pump | 22.19 | | | | | $ 10.64 | $ 31.92 | $ 10,724 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **182** | | | | | **$ 87.24** | **$ 261.71** | **$ 87,933** | **$ 0.03** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 20.00 | mt/hr | $ 270.000 | / mt | $ 43,206.79 | $ 129,620.36 | $ 43,552,443 | $ 14.56 | $ 10.80 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 43,206.79** | **$ 129,620.36** | **$ 43,552,443** | **$ 14.56** | **$ 10.80** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 43,294** **$ 129,882** **$ 43,640,376** **$ 14.59** **$ 10.82**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3

Date: 3/13/2023
By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 888 | gpm |
| Feed Rate | 3360 | lpm |
| Precip Mass Flow | 0.15 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 1.86 | g/L of Feed |
| Oxalic Acid Dose | 7.58 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.09 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 4,439 | gal | $ 6,025 | $ 24,099 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 2 | HP | $ 25,115 | $ 100,458 | $ - | |
| 3 | Process Pump | 3 | 100% | 888 | gpm | $ 86,298 | $ 258,894 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 0.71 | ft^3 | $ 7,118 | $ 7,118 | $ - | |
| 6 | Roaster | 1 | 100% | 0.23 | MBTU/hr | $ 444,027 | $ 444,027 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 834,596** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 8.88 | Effiicency: | 0.85 | | | $ 4.26 | $ 12.77 | $ 4,289 | $ 0.00 | $ 0.00 |
| 3 | Process Pump | 66.58 | | | | | $ 31.92 | $ 95.75 | $ 32,171 | $ 0.01 | $ 0.01 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.12 | | | | | $ 0.06 | $ 0.17 | $ 57 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.03 | | | | | $ 0.02 | $ 0.05 | $ 17 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **76** | | | | | **$ 36.24** | **$ 108.73** | **$ 36,534** | **$ 0.01** | **$ 0.01** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 0.37 | mt/hr | $ 125.000 | / mt | $ 374.31 | $ 1,122.92 | $ 377,301 | $ 0.13 | $ 0.09 |
| 2 | Oxalic Acid | 1,529.27 | kg/hr | $ 0.600 | / kg | $ 7,340.48 | $ 22,021.45 | $ 7,399,208 | $ 2.47 | $ 1.84 |
| 3 | Natural Gas | 0.22 | MCF/hr | $ 2.860 | / MCF | $ 5.12 | $ 15.37 | $ 5,164 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 7,719.91** | **$ 23,159.74** | **$ 7,781,673** | **$ 2.60** | **$ 1.93** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 7,756** | **$ 23,268** | **$ 7,818,207** | **$ 2.61** | **$ 1.94** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/13/2023
By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8404 | gpm |
| Raffinate Rate: | 8404 | gpm |
| Extraction Feed (O+A) | 16807 | gpm |
| Scrub Feed (O+A) | 231 | gpm |
| Strip Feed (O+A) | 462 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 231 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 276313 | gal |
| Precip Mixing Intensity | 0.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 60.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,807 | gal | $ 96,082 | $ 192,164 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 42 | HP | $ 131,910 | $ 263,819 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 252,112 | gal | $ 440,633 | $ 881,266 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,404 | gpm | $ 203,385 | $ 406,770 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 231 | gal | $ 8,621 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,755 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,466 | gal | $ 39,537 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 462 | gal | $ 12,731 | $ 25,462 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 17,378 | $ 34,756 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,931 | gal | $ 58,385 | $ 116,770 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,688 | $ 19,688 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 60,143 | kg | $ 3,765,482 | $ - | $ 3,765,482 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 1,043,343 | kg | $ 1,129,780 | $ - | $ 1,129,780 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,765 | gal | $ 5,766 | $ 23,063 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 1.38 | HP | $ 19,229 | $ 76,916 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,688 | $ 39,376 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.02 | ft^3 | $ 465 | $ 465 | $ - | |
| 20 | Process Pump | 2 | 100% | 231 | gpm | $ 51,647 | $ 103,293 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,183,808** | **$ 4,895,262** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 84.04 | Effiicency: | 0.85 | | $ 40.28 | $ 120.85 | $ 40,606 | $ 0.01 | $ 0.01 |
| 3 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 420.19 | | | | $ 201.42 | $ 604.25 | $ 203,029 | $ 0.07 | $ 0.05 |
| 5 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.31 | | | | $ 1.11 | $ 3.32 | $ 1,116 | $ 0.00 | $ 0.00 |
| 11 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.46 | | | | $ 0.22 | $ 0.66 | $ 223 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 5.53 | | | | $ 2.65 | $ 7.95 | $ 2,672 | $ 0.00 | $ 0.00 |
| 17 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.92 | | | | $ 0.44 | $ 1.33 | $ 445 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | | | | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.55 | | | | $ 5.54 | $ 16.61 | $ 5,582 | $ 0.00 | $ 0.00 |
| | **TOTAL:** | **525** | | | | **$ 251.66** | **$ 754.98** | **$ 253,674** | **$ 0.08** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 108.80 | kg/hr | $ 0.940 | / kg | $ 818.14 | $ 2,454.43 | $ 824,688 | $ 0.28 | $ 0.20 |
| 2 | D2EHPA | 5.73 | kg/hr | $ 5.000 | / kg | $ 229.04 | $ 687.13 | $ 230,876 | $ 0.08 | $ 0.06 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.88 | mt/hr | $ 270.000 | / mt | $ 1,905.60 | $ 5,716.80 | $ 1,920,846 | $ 0.64 | $ 0.48 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,952.79** | **$ 8,858.36** | **$ 2,976,410** | **$ 1.00** | **$ 0.74** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,204** **$ 9,613** **$ 3,230,084** **$ 1.08** **$ 0.80**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3

Date: 3/13/2023
By: ks

## Module Information

Module ID: PM9
Module Name: CM Precipitation
Include? Yes
Number of Parallel PMs: 5.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8364 | gpm |
| Feed Rate | 31662 | lpm |
| Precip Mass Flow | 2.79 | tph |
| Precip Conditioning Time | 30 | min |
| Mixing Power Intensity | 0.00 | HP/1000 gal |
| Filter Press Cycle Time | 0.00 | min |
| Thickener Capacity | 2399.30 | ft^2/tpd |
| NaOH Dose | 2.96 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 5 | 100% | 50,185 | gal | $ 118,504 | $ 592,519 | $ - | |
| 2 | Tank Mixer | 5 | 100% | - | HP | $ - | $ - | $ - | |
| 3 | Process Pump | 15 | 100% | 1,673 | gpm | $ 54,944 | $ 824,159 | $ - | |
| 4 | Filter Press | 1 | 100% | - | ft^3 | $ - | $ - | $ - | |
| 5 | Conveyor | 1 | 100% | 3 | TPH | $ 5,528 | $ 5,528 | $ - | |
| 6 | Thickener | 5 | 100% | 1,337 | ft^2 | $ 135,678 | $ 678,390 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,100,595** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | - | Effiicency: | 0.85 | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Process Pump | 627.31 | | | | | $ 300.70 | $ 902.11 | $ 303,110 | $ 0.10 | $ 0.08 |
| 4 | Filter Press | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Conveyor | 0.17 | | | | | $ 0.08 | $ 0.25 | $ 83 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 4.25 | | | | | $ 2.04 | $ 6.11 | $ 2,054 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **632** | | | | | **$ 302.82** | **$ 908.47** | **$ 305,247** | **$ 0.10** | **$ 0.08** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 5.62 | mt/hr | $ 125.000 | / mt | $ 5,623.13 | $ 16,869.39 | $ 5,668,114 | $ 1.90 | $ 1.41 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 5,623.13** | **$ 16,869.39** | **$ 5,668,114** | **$ 1.90** | **$ 1.41** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 5,926** **$ 17,778** **$ 5,973,361** **$ 2.00** **$ 1.48**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3

Date: 3/13/2023
By: ks

## Module Information

Module ID: PM10
Module Name: CM Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 2.79 | tph |
| Slurry Feed Rate | 45 | gpm |
| Leaching Time | 60 | min |
| Mixing Power Intensity | 0.0 | HP/1000 gal |
| Acid Consumption | 411.78 | kg/ton |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 682 | gal | $ 1,018 | $ 4,070 | $ - | |
| 2 | Tank Mixer | 4 | 100% | - | HP | $ - | $ - | $ - | |
| 3 | Process Pump | 1 | 100% | 45 | gpm | $ 27,777 | $ 27,777 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 31,847** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | - | Effiicency: | 0.85 | $ - | $ - | $ - | $ - | $ - |
| 3 | Process Pump | 1.14 | | | $ 0.54 | $ 1.63 | $ 549 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1** | | | **$ 0.54** | **$ 1.63** | **$ 549** | **$ 0.00** | **$ 0.00** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 1.15 | mt/hr | $ 270.000 | / mt | $ 2,477.40 | $ 7,432.19 | $ 2,497,214 | $ 0.84 | $ 0.62 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,477.40** | **$ 7,432.19** | **$ 2,497,214** | **$ 0.84** | **$ 0.62** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 2,478** | **$ 7,434** | **$ 2,497,763** | **$ 0.84** | **$ 0.62** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3

Date: 3/13/2023
By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 933.00 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 3531.80 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 932.37 | gpm | HCL Dose | 7.58 | g/L of Feed |
| Mn Feed Rate | 3529.42 | lpm | | | |
| Co Precip Mass Flow | 0.53 | tph | | | |
| Mn Precip Mass Flow | 0.68 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 8.04 | g/L of Feed | | | |
| NaOH Dose | 12.59 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 13,995 | gal | $ 17,920 | $ 35,839 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 21 | HP | $ 89,187 | $ 178,373 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 933 | gpm | $ 87,951 | $ 87,951 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 2.52 | ft^3 | $ 17,777 | $ 17,777 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 13,986 | gal | $ 17,908 | $ 35,816 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 21 | HP | $ 89,153 | $ 178,306 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 932 | gpm | $ 87,928 | $ 87,928 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 3 | ft^3 | $ 21,114 | $ 21,114 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 643,104** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 41.99 | Effiicency: | 0.85 | | | $ 20.13 | $ 60.38 | $ 20,287 | $ 0.01 | $ 0.01 |
| 3 | Process Pump | 23.33 | | | | | $ 11.18 | $ 33.54 | $ 11,270 | $ 0.00 | $ 0.00 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.42 | | | | | $ 0.20 | $ 0.61 | $ 205 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 41.96 | | | | | $ 20.11 | $ 60.34 | $ 20,273 | $ 0.01 | $ 0.01 |
| 8 | Process Pump | 23.31 | | | | | $ 11.17 | $ 33.52 | $ 11,263 | $ 0.00 | $ 0.00 |
| 9 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 0.54 | | | | | $ 0.26 | $ 0.77 | $ 260 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **132** | | | | | **$ 63.05** | **$ 189.16** | **$ 63,557** | **$ 0.02** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 1.70 | mt/hr | $ 360.000 | / mt | $ 4,909.55 | $ 14,728.65 | $ 4,948,825 | $ 1.65 | $ 1.23 |
| 2 | NaOH | 2.67 | mt/hr | $ 125.000 | / mt | $ 2,665.81 | $ 7,997.44 | $ 2,687,139 | $ 0.90 | $ 0.67 |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 0.53 | $ 1.58 | $ 530 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 7,575.89** | **$ 22,727.66** | **$ 7,636,494** | **$ 2.55** | **$ 1.89** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 7,639** | **$ 22,917** | **$ 7,700,052** | **$ 2.58** | **$ 1.91** |

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | Shakespeare Lignite | Date: | 3/18/2023 |
| Case: | SHXP-C3 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | an |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | N/A |
| *(unit conversion)* | 12000 | TPD | Project NPV: | (898,811,078) |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 22.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C3 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 108,132,002 | | | | | |
| Constant Dollar Annual OPEX | $ | 413,129,924 | $ | 102.46 | /ton | $ 231.7 | /kg |
| Constant Dollar Annual REE Rev | $ | 296,372,095 | $ | 73.50 | /ton | $ 166.21 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | | |
|---|---|---|---|---|---|
| Payback Period | | 25.5 | years | *22.5* | *operating years* |
| Internal Rate of Return | | N/A | | | |
| Net Present Value | $ | (898,811,078) | | | |
| Total Severence Tax Paid | $ | - | | | |
| Total State and Federal Tax Paid | $ | - | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C3 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 1,250 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 2.52 | 31.54 | 14.31 | | | | | | | 14.31 |
| Yttrium | 17.93 | 224.14 | 101.67 | | | | | | | 101.67 |
| Lanthanum | 14.92 | 186.53 | 84.61 | | | | | | | 84.61 |
| Cerium | 29.73 | 371.57 | 168.54 | | | | | | | 168.54 |
| Praseodymium | 3.84 | 47.95 | 21.75 | | | | | | | 21.75 |
| Neodymium | 15.16 | 189.50 | 85.96 | | | | | | | 85.96 |
| Samarium | 3.24 | 40.54 | 18.39 | | | | | | | 18.39 |
| Europium | 0.73 | 9.12 | 4.14 | | | | | | | 4.14 |
| Gadolinium | 3.73 | 46.59 | 21.13 | | | | | | | 21.13 |
| Terbium | 0.49 | 6.08 | 2.76 | | | | | | | 2.76 |
| Dysprosium | 3.34 | 41.77 | 18.95 | | | | | | | 18.95 |
| Holmium | 0.62 | 7.76 | 3.52 | | | | | | | 3.52 |
| Erbium | 1.39 | 17.39 | 7.89 | | | | | | | 7.89 |
| Thulium | 0.37 | 4.63 | 2.10 | | | | | | | 2.10 |
| Ytterbium | 1.68 | 21.03 | 9.54 | | | | | | | 9.54 |
| Lutetium | 0.31 | 3.84 | 1.74 | | | | | | | 1.74 |
| Manganese | -- | 505.70 | 229.38 | | | 0.00 | -- | | | 229.38 |
| Cobalt | -- | 258.60 | 117.30 | | | 0.00 | -- | | | 117.30 |
| Nickel | -- | 718.70 | 326.00 | | | 0.00 | -- | | | 326.00 |
| Zinc | -- | 210.80 | 95.62 | | | 0.00 | -- | | | 95.62 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 1250.00 | 566.99 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 567.0 |
| Total non-REE CM | -- | 1693.80 | 768.30 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 768.3 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 14.31 | 0.00 | 4.70 | 0.00 | 0.00 | 0.00 | 4.71 | 9.60 | 32.9% | |
| Yttrium | 101.67 | 38.72 | 0.00 | 10.22 | 0.68 | 0.00 | 49.62 | 52.05 | 48.8% | |
| Lanthanum | 84.61 | 13.74 | 0.00 | 13.09 | 0.93 | 0.00 | 27.77 | 56.84 | 32.8% | |
| Cerium | 168.54 | 44.79 | 0.00 | 17.48 | 1.22 | 0.00 | 63.49 | 105.05 | 37.7% | |
| Praseodymium | 21.75 | 5.32 | 0.00 | 2.03 | 0.10 | 0.00 | 7.45 | 14.30 | 34.2% | |
| Neodymium | 85.96 | 25.19 | 0.00 | 5.00 | 0.38 | 0.00 | 30.56 | 55.39 | 35.6% | |
| Samarium | 18.39 | 6.77 | 0.00 | 0.39 | 0.06 | 0.00 | 7.22 | 11.17 | 39.3% | |
| Europium | 4.14 | 1.43 | 0.00 | 0.12 | 0.01 | 0.00 | 1.57 | 2.57 | 38.0% | |
| Gadolinium | 21.13 | 7.84 | 0.00 | 1.16 | 0.08 | 0.00 | 9.08 | 12.05 | 43.0% | |
| Terbium | 2.76 | 1.01 | 0.00 | 0.00 | 0.00 | 0.00 | 1.01 | 1.75 | 36.6% | |
| Dysprosium | 18.95 | 6.62 | 0.00 | 1.18 | 0.10 | 0.00 | 7.90 | 11.05 | 41.7% | |
| Holmium | 3.52 | 1.34 | 0.00 | 0.36 | 0.02 | 0.00 | 1.71 | 1.81 | 48.7% | |
| Erbium | 7.89 | 3.74 | 0.00 | 0.39 | 0.05 | 0.00 | 4.18 | 3.71 | 53.0% | |
| Thulium | 2.10 | 0.41 | 0.00 | 0.05 | 0.01 | 0.00 | 0.47 | 1.63 | 22.2% | |
| Ytterbium | 9.54 | 3.45 | 0.00 | 0.28 | 0.04 | 0.00 | 3.77 | 5.77 | 39.5% | |
| Lutetium | 1.74 | 0.48 | 0.00 | 0.13 | 0.01 | 0.00 | 0.62 | 1.12 | 35.5% | |
| Manganese | 229.38 | 0.02 | 0.00 | 32.77 | 2.19 | 0.00 | 34.98 | 194.40 | 15.2% | |
| Cobalt | 117.30 | 0.04 | 0.00 | 3.85 | 51.12 | 0.00 | 55.01 | 62.29 | 46.9% | |
| Nickel | 326.00 | 0.19 | 0.00 | 3.68 | 107.81 | 0.00 | 111.69 | 214.31 | 34.3% | |
| Zinc | 95.62 | 0.08 | 0.00 | 0.04 | 50.24 | 0.00 | 50.36 | 45.26 | 52.7% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 566.99 | 160.87 | 4.70 | 51.88 | 3.68 | 0.00 | 221.13 | 345.86 | 39.0% | <---- |
| Total non-REE CM | 768.30 | 0.33 | 0.00 | 40.35 | 211.37 | 0.00 | 252.04 | 516.25 | 32.8% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 339 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 16 | ton | $ | 1.50 |
| Dust/Volatiles | - | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | | |
|---|---|---|
| 3.86 | TPD REE | |
| 0.113 | TPD Sc | |
| 6.04 | TPD CM | |
| 0.28 | REE | |
| 32.9% | Sc | |
| 14.3% | Mn | |
| 38.8% | Metal-S | |
| 500 | Feed | |
| 1,250 | Grade | |
| 25.5 | PP | |
| N/A | IRR | |
| -898811078.1 | NPV | |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C3 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 0.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 2.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | Based on Design |
| Dissolved Solids in Leach | 32.2% | % | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Leaching Retention Time | 45 | min | Based on Design |
| Thickener UF % Solids | 46% | % | SHXP-C3 Samples |
| Residue Settling Rate | 15.0 | ft/hr | Not Available |
| **Reagent Use** | | | |
| Acid Consumption | 264.1 | kg/ton | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Bioleaching Sol'n Consumption | 0 | L/ton | |
| Polymer Dosage | 0.016 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 10.75 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 7.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | Based on Design |
| Precip Solid Yield | 7.54 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Precip Settling Rate | 2.24 | ft/hr | Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 1.5% | % | VT Override to 2% \| SHXP-C3 Samples |
| **Reagent Use** | | | |
| NaOH Dose | 18.31 | g/L of Feed | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 974.97 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 1250 | ppm | |
| Grade After Sorter | 1250 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 500 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 0 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 500 | tph | |
| Leaching Residue Rate | 339 | tph | |
| Slurry Flow (feed) | 9250 | gpm | |
| Product Flow | 8,404 | gpm | |
| Thickener Feed % Solids (eff) | 14% | % | |
| Liquid Density | 1.029 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 80.37 | g/L | |
| Leaching Vessel Size | 52031.25 | gal | |
| Leaching Vessel Size | 196.96 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 169.50 | tph | 70 to 200 tph |
| Thickener Area | 1822.72 | ft^2 | |
| Thickener Diameter | 48.19 | ft | ideal < 100' |
| Liquid Balance | 401.92 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,404 | gpm | |
| Slurry Feed Rate | 31,812 | lpm | |
| Precip Mass Flow | 15.87 | tph | |
| Thickener Feed % Solids (eff) | 0.73% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.023 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 204.50 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 74.75 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 18.64 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 2,210 | ft^2 | |
| Thickener Diameter | 53.06 | ft | ideal < 100' |

*PM5: REE Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |
| | | | |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45 | min | Based on Design |
| Precip Solid Yield | 8.49 | g/L | VT Override from 1.83 \|\| SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Precip Settling Rate | 0.044 | ft/hr | Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.88% | % | VT Override to 0.88% \| SHXP-C3 Samples |
| | | | |
| **Reagent Use** | | | |
| NaOH Dose | 3.67 | g/L of Feed | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| | | | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 4942.15 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 8385 | gpm | |
| Slurry Feed Rate | 31,741 | lpm | |
| Precip Mass Flow | 17.82 | tph | |
| Thickener Feed % Solids (eff) | 0.8% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.017 | g/cc | |
| | | | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 238.06 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 84.01 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 20.94 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 14681 | ft^2 | |
| Thickener Diameter | 136.75 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | 28.05 | m/m | target = 6.35 |

## PM6: RE-Redissolution

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 7.6% | % | VT Override from 10.7% \| Based on Design |
| Dissolved Solids in Re-Leach | 11% | % | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Leaching Retention Time | 120 | min | UK: Design |
| **Reagent Use** | | | |
| Acid Consumption | 1122.3 | kg/ton | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| | 0.0 | | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| Material Balance | | | |
|---|---|---|---|
| Solid Feed Rate | 17.82 | tph | |
| Slurry Flow (feed) | 888 | gpm | |
| Product Flow | 888 | gpm | |
| Liquid Density | 1.057 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 100.8 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 9.47 | v/v | target = 10.0, based on data |
| Liquid Balance | 20.94 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 80.2 | g/L | |

## PM7: REE Oxalate Precipitation

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | Based on Design |
| Precip Solid Yield | 0.7 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Thickener UF %Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| NaOH Dose | 1.86 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Oxalic Acid Dose | 7.58 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| Material Balance | | | |
|---|---|---|---|
| Slurry Feed Rate | 887.74 | gpm | |
| Slurry Feed Rate | 3360.44 | lpm | |
| Precip Mass Flow | 0.15 | tph | |
| Thickener Feed % Solids (eff) | 0.06% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.09 | tph | |
| Liquid Density | 1.056 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 16.80 | m^3 | |
| Filter Press Area | 0.71 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.18 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 5.72 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.72 | m/m | |

## PM8: Sc SX Separation

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | UK: Design |
| Settler Residence Time | 15.00 | min | UK: Design |
| Loading: Number of Stages | 2.00 | - | UK: Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Loading O:A Advance Ratio | 0.03 | v/v | UK: Design |
| Scrubbing: Number of Stages | | - | N/A |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | N/A |
| Scrubbing O:A Advance Ratio | | v/v | N/A |
| Stripping: Number of Stages | 2.00 | - | UK: Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | UK: Design |
| Sc2O3 Grade | 2% | % | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Precip Solid Yield | 0.4 | g/L | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| Material Balance (SX) | | |
|---|---|---|
| PLS Flow | 8403.7 | gpm |
| Raffinate Rate: | 8403.7 | gpm |
| Load Org Recycle | 8172.7 | gpm |
| Load Org Feed | 8403.7 | gpm |
| Loag Org Advance | 231.0 | gpm |
| Scrub Aq Product | 0.0 | gpm |
| Scrub Aq Recycle | 0.0 | gpm |
| Scrub Aq Feed | 0.0 | gpm |
| Scurb Org Product | 231.0 | gpm |
| Strip Aq Product | 18.4 | gpm |
| Strip Aq Recycle | 212.6 | gpm |
| Strip Aq Feed | 231.0 | gpm |
| Strip Org Product | 231.0 | gpm |
| Extraction Feed (O+A): | 16807.5 | gpm |
| Scrub Feed (O+A): | 231.0 | gpm |
| Strip Feed (O+A): | 462.1 | gpm |
| **Material Balance (Precip)** | | |
| Strip Solution Flow | 18.4 | gpm |
| Solid Density | 3.4 | g/cc |
| Solid Product Rate | 0.00 | tph |
| Liquid Density (after precip) | 1.13 | g/cc |
| Thickener Feed % Solids (eff) | 0.03% | % |
| **Total Volumes** | | |
| Total Extraction Volume | 537839 | gal |
| Total Scrubbing Volume | 0 | gal |
| Total Stripping Volume | 14787 | gal |
| Extraction Org Volume | 268919.28 | gal |
| Scrubbing Org Volume | 0.00 | gal |
| Stripping Org Volume | 7393.56 | gal |
| Total Org Volume | 276312.84 | gal |
| **Other Indicators** | | |
| Liquid Balance | 0.00 | gpm must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm should be close to zero |
| Conditioning Vessel Size | 10.47 | m^3 |
| Filter Press Area | 0.02 | ft^3 ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 |
| Thickener Diameter | 0.00 | ft ideal < 100' |

## PM9: CM Precipitation

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 30 | min | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Precip Solid Yield | 1.33 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Precip Settlng Rate | 10 | ft/hr | |
| Thickener UF % Solids | 25% | % | Not Available |
| **Reagent Use** | | | |
| NaOH Dose | 2.96 | g/L of Feed | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 0.00 | min | N/A |
| Thickener Capacity | 2399.30 | ft^2/tph | N/A |

| Material Balance | | | |
|---|---|---|---|
| Slurry Feed Rate | 8364 | gpm | |
| Slurry Feed Rate | 31,662 | lpm | |
| Precip Mass Flow | 2.79 | tph | |
| Thickener Feed % Solids (eff) | 0.1% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.016 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 189.97 | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | 3.27 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 1337 | ft^2 | |
| Thickener Diameter | 41.26 | ft | ideal < 100' |

## PM10: CM Redissolution

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 20.9% | % | VT Override from 20.9% \|\| SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Dissolved Solids in Re-Leach | 100% | % | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Leaching Retention Time | 60 | min | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| **Reagent Use** | | | |
| Acid Consumption | 411.78 | kg/ton | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 2.79 | tph | |
| Slurry Flow (feed) | 45 | gpm | |
| Product Flow | 45 | gpm | |
| Liquid Density | 1.173 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 2.6 | m^3 | |
| Releach Conc. Factor | 184.06 | v/v | target = 102, based on data |
| Liquid Balance | 3.27 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 244.8 | g/L | |

## PM11: CM Recovery

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 2.29 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | 0% | % | SHXP-C3 Samples |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 2.91 | g/L | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | 1% | % | SHXP-C3 Samples |
| **Reagent Use** | | | |
| Na2S Dose | 8.04 | g/L of Feed | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| NaOH Dose | 12.59 | g/L of Feed | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| HCL Dose | 7.58 | g/L of Feed | SHXP-C3/SHXP-C3 MASS BALANCE V3 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 933.00 | gpm | |
| Co Slurry Feed Rate | 3531.80 | lpm | |
| Co Precip Mass Flow | 0.53 | tph | |
| Co Thickener Feed % Solids (eff) | 0.22% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 932.37 | gpm | |
| Mn Slurry Feed Rate | 3529.42 | lpm | |
| Mn Precip Mass Flow | 0.68 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.27% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.06 | g/cc | |
| Mn Liquid Density | 1.06 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 53.0 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 2.52 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.63 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 52.9 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 3.2 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.80 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

## PM12: Biooxidation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Requirec | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C3 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 6,318,002 | $ - | $ 1,579,500 | $ 6,949,802 | $ 8,652,504 | $ 23,499,808 | IIIIIIIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 4,241,248 | $ - | $ 1,060,312 | $ 4,665,373 | $ 5,808,389 | $ 15,775,322 | IIIIIIIIIIIIIIII |
| PM5 | REE Precipitation | $ 8,207,556 | $ - | $ 2,051,889 | $ 9,028,311 | $ 11,240,248 | $ 30,528,004 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| PM6 | RE-Redissolution | $ 731,126 | $ - | $ 182,781 | $ 804,239 | $ 1,001,277 | $ 2,719,423 | III |
| PM7 | REE Oxalate Precipitation | $ 834,596 | $ - | $ 208,649 | $ 918,056 | $ 1,142,980 | $ 3,104,282 | III |
| PM8 | Sc SX Separation | $ 2,183,808 | $ 4,895,262 | $ 545,952 | $ 2,402,189 | $ 2,990,725 | $ 13,017,937 | IIIIIIIIIIIII |
| PM9 | CM Precipitation | $ 2,100,595 | $ - | $ 525,149 | $ 2,310,655 | $ 2,876,765 | $ 7,813,164 | IIIIIIII |
| PM10 | CM Redissolution | $ 31,847 | $ - | $ 7,962 | $ 35,032 | $ 43,615 | $ 118,456 | |
| PM11 | CM Recovery | $ 643,104 | $ - | $ 160,776 | $ 707,414 | $ 880,731 | $ 2,392,025 | II |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| | TOTAL | $ 27,755,542 | $ 4,895,262 | $ 6,938,886 | $ 30,531,096 | $ 38,011,215 | $ 108,132,002 | |

## Module Summary

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| Total Lang Factor (=1+sum % Base) | | 372% | |

Year of Construction: 2025

# OPERATING COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C3 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 1.72 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 0.66 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 108,132,002 | $ Total | $ 1,072.74 | $ 3,218.21 | $ 1,081,320 | $ 0.61 | $ 0.27 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 0.17 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 296,372,095 | $ Total | $ 1,470.10 | $ 4,410.30 | $ 1,481,860 | $ 0.83 | $ 0.37 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 0.26 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 108,132,002 | $ Total | $ 1,072.74 | $ 3,218.21 | $ 1,081,320 | $ 0.61 | $ 0.27 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 8,597.03** | **$ 25,791.08** | **$ 8,665,802** | **$ 4.86** | **$ 2.15** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 0.29 | $ 0.13 |
| PM2 | Roasting | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | PR | 1280 | | $ 613.76 | $ 1,841.29 | $ 618,673 | $ 0.35 | $ 0.15 |
| PM4 | Al Precipitation | PR | 843 | | $ 403.97 | $ 1,211.91 | $ 407,201 | $ 0.23 | $ 0.10 |
| PM5 | REE Precipitation | PR | 889 | | $ 426.04 | $ 1,278.12 | $ 429,447 | $ 0.24 | $ 0.11 |
| PM6 | RE-Redissolution | PR | 182 | | $ 87.24 | $ 261.71 | $ 87,933 | $ 0.05 | $ 0.02 |
| PM7 | REE Oxalate Precipitation | PR | 76 | | $ 36.24 | $ 108.73 | $ 36,534 | $ 0.02 | $ 0.01 |
| **Sub-Total** | | | **5624.3** | | **$ 2,696.01** | **$ 8,088.03** | **$ 2,717,579.20** | **$ 1.52** | **$ 0.67** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.06 | $ 0.03 |
| PM2 | Roasting | CR | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | CR | | $ 233,034.54 | $ 699,103.63 | $ 234,898,819 | $ 131.73 | $ 58.26 |
| PM4 | Al Precipitation | CR | | $ 34,957.02 | $ 104,871.05 | $ 35,236,673 | $ 19.76 | $ 8.74 |
| PM5 | REE Precipitation | CR | | $ 6,989.37 | $ 20,968.12 | $ 7,045,287 | $ 3.95 | $ 1.75 |
| PM6 | RE-Redissolution | CR | | $ 43,206.79 | $ 129,620.36 | $ 43,552,443 | $ 24.42 | $ 10.80 |
| PM7 | REE Oxalate Precipitation | CR | | $ 7,719.91 | $ 23,159.74 | $ 7,781,673 | $ 4.36 | $ 1.93 |
| **Sub-Total** | | | | **$ 344,637.63** | **$ 1,033,912.89** | **$ 347,394,732.39** | **$ 194.82** | **$ 86.16** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 8,136 | $ 0.10 | $ 271.20 | $ 813.60 | $ 273,370 | $ 0.15 | $ 0.07 |
| Non-Hazardous Precipitate | WD | Tons | 381 | $ 1.50 | $ 190.41 | $ 571.22 | $ 191,929 | $ 0.11 | $ 0.05 |
| Dust/Volatiles | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 461.61** | **$ 1,384.82** | **$ 465,298.38** | **$ 0.26** | **$ 0.12** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 347,795** | **$ 1,043,386** | **$ 350,577,610** | **$ 197** | **$ 87** |

## Overhead Costs

| Allocation | Code | Rate | Basis | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% | % of Direct Cost | $ 25,791 Daily Direct Cost | $ 1,289.55 | $ 3,868.66 | $ 1,299,870 | $ 0.73 | $ 0.32 |
| Variable Portion | OH | 15% | % of Direct Cost | $ 1,043,386 Daily Direct Cost | $ 52,169.29 | $ 156,507.86 | $ 52,586,641 | $ 29.49 | $ 13.04 |
| **TOTAL OVERHEAD COSTS** | | | | | **$ 53,458.84** | **$ 160,376.52** | **$ 53,886,511.81** | **$ 30.22** | **$ 13.36** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | SHXP-C3 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 8,597.03 | $ 25,791.08 | $ 8,665,802 | $ 5 | $ 2.15 | I |
| Power | $ 2,696.01 | $ 8,088.03 | $ 2,717,579 | $ 2 | $ 0.67 | |
| Consumables | $ 344,637.63 | $ 1,033,912.89 | $ 347,394,732 | $ 195 | $ 86.16 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| Leasing/Tolling | $ - | $ - | $ - | $ - | $ - | |
| Waste Disposal | $ 461.61 | $ 1,384.82 | $ 465,298 | $ 0 | $ 0.12 | |
| General Overhead | $ 53,458.84 | $ 160,376.52 | $ 53,886,512 | $ 30 | $ 13.36 | IIIIII |
| TOTAL | $ 409,851 | $ 1,229,553 | $ 413,129,924 | $ 232 | $ 102.46 | |

## Summary by Process Module

| | Process Module | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 611 | $ 1,834 | $ 616,370 | $ 0.35 | $ 0.15 | |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 233,648 | $ 700,945 | $ 235,517,492 | $ 132.08 | $ 58.41 | IIIIIIIIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 35,361 | $ 106,083 | $ 35,643,874 | $ 19.99 | $ 8.84 | IIIII |
| PM5 | REE Precipitation | $ 7,415 | $ 22,246 | $ 7,474,734 | $ 4.19 | $ 1.85 | |
| PM6 | RE-Redissolution | $ 43,294 | $ 129,882 | $ 43,640,376 | $ 24.47 | $ 10.82 | IIIII |
| PM7 | REE Oxalate Precipitation | $ 7,756 | $ 23,268 | $ 7,818,207 | $ 4.38 | $ 1.94 | |
| PM8 | Sc SX Separation | $ 3,204 | $ 9,613 | $ 3,230,084 | $ 1.81 | $ 0.80 | |
| PM9 | CM Precipitation | $ 5,926 | $ 17,778 | $ 5,973,361 | $ 3.35 | $ 1.48 | |
| PM10 | CM Redissolution | $ 2,478 | $ 7,434 | $ 2,497,763 | $ 1.40 | $ 0.62 | |
| PM11 | CM Recovery | $ 7,639 | $ 22,917 | $ 7,700,052 | $ 4.32 | $ 1.91 | |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| NA | Unallocated | $ 62,517 | $ 187,552 | $ 63,017,612 | $ 35 | $ 15.63 | IIIIIII |
| | TOTAL | $ 409,851 | $ 1,229,553 | $ 413,129,924 | $ 232 | $ 102.46 | |

## Summary by Cost Center

| Cost Center | Code | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| Operating Labor | OL | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 2 | $ 0.76 | |
| Technical Labor | TL | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 1 | $ 0.29 | |
| Power/Utilities | PR | $ 2,696.01 | $ 8,088.03 | $ 2,717,579 | $ 2 | $ 0.67 | |
| Lease Agreements | LS | $ - | $ - | $ - | $ - | $ - | |
| Consumables/Reagents | CR | $ 344,637.63 | $ 1,033,912.89 | $ 347,394,732 | $ 195 | $ 86.16 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| Waste Disposal | WD | $ 461.61 | $ 1,384.82 | $ 465,298 | $ 0 | $ 0.12 | |
| Plant Overhead | OH | $ 54,836.44 | $ 164,509.32 | $ 55,275,131 | $ 31 | $ 13.71 | IIIIII |
| General Sales, and Administration | GSA | $ 3,000.13 | $ 9,000.38 | $ 3,024,129 | $ 2 | $ 0.75 | |
| Reserved | | | | | | | |
| TOTAL | | $ 409,851 | $ 1,229,553.34 | $ 413,129,924 | $ 232 | $ 102.46 | |

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C3 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 296,372,095

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 1218 | 1250 |
| Contained Value | ($/ton) | $ 33.3 | $ 217.7 |
| Feed Basket Price | ($/kg) | $ 30.2 | $ 192.0 |
| Product Basket Price | ($/kg) | $ 30.0 | $ 166.5 |
| Production Rate | (kg/day) | 5194.07 | 5307.05 |
| | (t/day) | 5.19 | 5.31 |
| | (t/yr) | 1745.21 | 1783.17 |
| | (% of US Demand) | 17.5% | |

## Revenue Contribution Table

| | | Product Revenue ($/hr) | | | | | |
|---|---|---|---|---|---|---|---|
| **Element** | **REE/CM Price ($/kg) [1]** | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | **REE Revenue ($/day)** |
| Scandium | $ 4,200.00 | $ 23.4 | $ 30,289.8 | $ - | $ - | $ - | $ 727,519 |
| Yttrium | $ 6.00 | $ 221.3 | $ - | $ - | $ - | $ - | $ 5,311 |
| Lanthanum | $ 2.00 | $ 24.2 | $ - | $ - | $ - | $ - | $ 580 |
| Cerium | $ 2.00 | $ 82.5 | $ - | $ - | $ - | $ - | $ 1,981 |
| Praseodymium | $ 52.00 | $ 250.5 | $ - | $ - | $ - | $ - | $ 6,012 |
| Neodymium | $ 42.00 | $ 925.4 | $ - | $ - | $ - | $ - | $ 22,211 |
| Samarium | $ 2.11 | $ 12.4 | $ - | $ - | $ - | $ - | $ 298 |
| Europium | $ 34.00 | $ 42.3 | $ - | $ - | $ - | $ - | $ 1,016 |
| Gadolinium | $ 32.00 | $ 217.0 | $ - | $ - | $ - | $ - | $ 5,208 |
| Terbium | $ 400.00 | $ 355.9 | $ - | $ - | $ - | $ - | $ 8,541 |
| Dysprosium | $ 230.00 | $ 1,313.6 | $ - | $ - | $ - | $ - | $ 31,527 |
| Holmium | $ 52.96 | $ 61.0 | $ - | $ - | $ - | $ - | $ 1,464 |
| Erbium | $ 34.00 | $ 109.2 | $ - | $ - | $ - | $ - | $ 2,620 |
| Thulium | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Ytterbium | $ 28.72 | $ 84.5 | $ - | $ - | $ - | $ - | $ 2,028 |
| Lutetium | $ 796.64 | $ 326.2 | $ - | $ - | $ - | $ - | $ 7,828 |
| Manganese | $ 0.005 | $ - | $ - | $ 0.3 | $ - | $ - | $ 7 |
| Cobalt | $ 22.00 | $ - | $ - | $ - | $ 843.5 | $ - | $ 20,245 |
| Nickel | $ 18.00 | $ - | $ - | $ - | $ 1,455.5 | $ - | $ 34,931 |
| Zinc | $ 3.02 | $ - | $ - | $ - | $ 113.9 | $ - | $ 2,733 |
| Reserved | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Total: | | $ 4,049.5 | $ 30,289.8 | $ 0.3 | $ 2,412.9 | $ - | $ 882,060 |

[1] REEs listed as $/kg REO; Mn listed as $/kg 44% Mn Ore; Co listed as $/kg metal content

## Coal Revenue

List Coal Revenue Sources Here:

**Annual Coal Revenue:** $ -

## Other Revenue

List Other Revenue Sources Here:

**Annual Other Revenue:** $ -

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C3 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 323,853,790 | $ 333,569,404 | $ 343,576,486 | $ 353,883,780 | $ 364,500,294 | $ 375,435,303 | $ 386,698,362 | $ 398,299,313 | $ 410,248,292 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 323,853,790 | $ 333,569,404 | $ 343,576,486 | $ 353,883,780 | $ 364,500,294 | $ 375,435,303 | $ 386,698,362 | $ 398,299,313 | $ 410,248,292 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 324,396 | $ 2,316,230 | $ 454,619,900 | $ 467,893,985 | $ 481,558,221 | $ 495,623,828 | $ 510,102,334 | $ 525,005,581 | $ 540,345,735 | $ 556,135,292 | $ 572,387,085 |
| *- Operating Costs* | $ - | $ - | $ - | $ 451,438,222 | $ 464,981,369 | $ 478,930,810 | $ 493,298,734 | $ 508,097,696 | $ 523,340,627 | $ 539,040,846 | $ 555,212,072 | $ 571,868,434 |
| *- Loan #1 Interest* | $ - | $ 324,396 | $ 299,785 | $ 273,697 | $ 246,044 | $ 216,731 | $ 185,660 | $ 152,725 | $ 117,813 | $ 80,807 | $ 41,580 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 2,016,446 | $ 1,863,462 | $ 1,701,299 | $ 1,529,407 | $ 1,347,201 | $ 1,154,063 | $ 949,336 | $ 732,326 | $ 502,295 | $ 258,463 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 1,044,519 | $ 965,273 | $ 881,273 | $ 792,233 | $ 697,850 | $ 597,805 | $ 491,756 | $ 379,345 | $ 260,189 |
| Gross Profit | $ - | $ (324,396) | $ (2,316,230) | $ (130,766,110) | $ (134,324,581) | $ (137,981,735) | $ (141,740,048) | $ (145,602,040) | $ (149,570,278) | $ (153,647,373) | $ (157,835,979) | $ (162,138,793) |
| **Book Deductions** | $ - | $ 405,495 | $ 2,926,404 | $ 4,236,412 | $ 3,918,681 | $ 3,624,780 | $ 3,352,921 | $ 3,101,452 | $ 2,868,843 | $ 2,653,680 | $ 2,454,654 | $ 2,270,555 |
| *- CAPEX #1 Depreciation* | $ - | $ 405,495 | $ 375,083 | $ 346,952 | $ 320,930 | $ 296,861 | $ 274,596 | $ 254,001 | $ 234,951 | $ 217,330 | $ 201,030 | $ 185,953 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 2,551,321 | $ 2,359,972 | $ 2,182,974 | $ 2,019,251 | $ 1,867,807 | $ 1,727,722 | $ 1,598,143 | $ 1,478,282 | $ 1,367,411 | $ 1,264,855 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,529,488 | $ 1,414,776 | $ 1,308,668 | $ 1,210,518 | $ 1,119,729 | $ 1,035,749 | $ 958,068 | $ 886,213 | $ 819,747 |
| Taxable Before Depletion | $ - | $ (729,891) | $ (5,242,634) | $ (135,002,521) | $ (138,243,262) | $ (141,606,515) | $ (145,092,969) | $ (148,703,492) | $ (152,439,122) | $ (156,301,053) | $ (160,290,633) | $ (164,409,348) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ - | $ (364,946) | $ (2,621,317) | $ (67,501,261) | $ (69,121,631) | $ (70,803,258) | $ (72,546,484) | $ (74,351,746) | $ (76,219,561) | $ (78,150,527) | $ (80,145,317) | $ (82,204,674) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 45,339,531 | $ 46,699,717 | $ 48,100,708 | $ 49,543,729 | $ 51,030,041 | $ 52,560,942 | $ 54,137,771 | $ 55,761,904 | $ 57,434,761 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (729,891) | $ (5,242,634) | $ (135,002,521) | $ (138,243,262) | $ (141,606,515) | $ (145,092,969) | $ (148,703,492) | $ (152,439,122) | $ (156,301,053) | $ (160,290,633) | $ (164,409,348) |
| *- Income Tax* | $ - | $ (189,772) | $ (1,363,085) | $ (35,100,656) | $ (35,943,248) | $ (36,817,694) | $ (37,724,172) | $ (38,662,908) | $ (39,634,172) | $ (40,638,274) | $ (41,675,565) | $ (42,746,430) |
| Net Profit/Loss | $ - | $ (540,119) | $ (3,879,549) | $ (99,901,866) | $ (102,300,014) | $ (104,788,821) | $ (107,368,797) | $ (110,040,584) | $ (112,804,950) | $ (115,662,779) | $ (118,615,069) | $ (121,662,918) |
| **Book Deductions** | $ - | $ 405,495 | $ 2,926,404 | $ 4,236,412 | $ 3,918,681 | $ 3,624,780 | $ 3,352,921 | $ 3,101,452 | $ 2,868,843 | $ 2,653,680 | $ 2,454,654 | $ 2,270,555 |
| *+ CAPEX #1 Depreciation* | $ - | $ 405,495 | $ 375,083 | $ 346,952 | $ 320,930 | $ 296,861 | $ 274,596 | $ 254,001 | $ 234,951 | $ 217,330 | $ 201,030 | $ 185,953 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 2,551,321 | $ 2,359,972 | $ 2,182,974 | $ 2,019,251 | $ 1,867,807 | $ 1,727,722 | $ 1,598,143 | $ 1,478,282 | $ 1,367,411 | $ 1,264,855 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,529,488 | $ 1,414,776 | $ 1,308,668 | $ 1,210,518 | $ 1,119,729 | $ 1,035,749 | $ 958,068 | $ 886,213 | $ 819,747 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ - | $ (134,624) | $ (953,145) | $ (95,665,454) | $ (98,381,333) | $ (101,164,041) | $ (104,015,876) | $ (106,939,132) | $ (109,936,107) | $ (113,009,099) | $ (116,160,415) | $ (119,392,363) |
| **Captial Cost** | $ 5,406,600 | $ 34,017,614 | $ 20,393,173 | $ 4,484,356 | $ 4,753,417 | $ 5,038,622 | $ 5,340,940 | $ 5,661,396 | $ 6,001,080 | $ 6,361,144 | $ 6,742,813 | $ 6,412,798 |
| *- Fixed Capital Costs* | $ 10,813,200 | $ 67,214,852 | $ 34,817,294 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 5,406,600 | $ 33,607,426 | $ 17,408,647 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 410,188 | $ 434,799 | $ 460,887 | $ 488,540 | $ 517,853 | $ 548,924 | $ 581,859 | $ 616,771 | $ 653,777 | $ 693,004 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 2,549,727 | $ 2,702,710 | $ 2,864,873 | $ 3,036,765 | $ 3,218,971 | $ 3,412,110 | $ 3,616,836 | $ 3,833,846 | $ 4,063,877 | $ 4,307,710 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 1,320,758 | $ 1,400,004 | $ 1,484,004 | $ 1,573,044 | $ 1,667,427 | $ 1,767,473 | $ 1,873,521 | $ 1,985,932 | $ 2,105,088 |
| **Net Cash Flow** | $ (5,406,600) | $ (34,152,238) | $ (21,346,318) | $ (100,149,810) | $ (103,134,750) | $ (106,202,664) | $ (109,356,815) | $ (112,600,528) | $ (115,937,186) | $ (119,370,244) | $ (122,903,228) | $ (125,805,161) |
| Discounted Cash Flow | $ (5,406,600) | $ (31,047,489) | $ (17,641,585) | $ (75,244,035) | $ (70,442,422) | $ (65,943,498) | $ (61,729,071) | $ (57,781,875) | $ (54,085,553) | $ (50,624,636) | $ (47,384,515) | $ (44,093,941) |
| **Cumulative DCF** | $ (5,406,600) | $ (36,454,089) | $ (54,095,674) | $ (129,339,709) | $ (199,782,131) | $ (265,725,630) | $ (327,454,701) | $ (385,236,576) | $ (439,322,129) | $ (489,946,765) | $ (537,331,280) | $ (581,425,221) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 108,132,002 | Payback Period | 25.5 |
| Constant Dollar OPEX | $ 413,129,924 | Internal Rate of Return | N/A |
| Constant Dollar REE Rev | $ 296,372,095 | Net Present Value | $ (898,811,078) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ - |

## PRO FORMA CASH FLOW

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C3 | Years: | 13 through 23 | By: | ks |

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 422,555,741** | **$ 435,232,413** | **$ 448,289,385** | **$ 461,738,067** | **$ 475,590,209** | **$ 489,857,915** | **$ 504,553,653** | **$ 519,690,262** | **$ 535,280,970** | **$ 551,339,399** | **$ 567,879,581** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 422,555,741 | $ 435,232,413 | $ 448,289,385 | $ 461,738,067 | $ 475,590,209 | $ 489,857,915 | $ 504,553,653 | $ 519,690,262 | $ 535,280,970 | $ 551,339,399 | $ 567,879,581 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 589,158,370** | **$ 606,695,221** | **$ 624,896,078** | **$ 643,642,960** | **$ 662,952,249** | **$ 682,840,817** | **$ 703,326,041** | **$ 724,425,822** | **$ 746,158,597** | **$ 768,543,355** | **$ 791,599,655** |
| *- Operating Costs* | $ 589,024,487 | $ 606,695,221 | $ 624,896,078 | $ 643,642,960 | $ 662,952,249 | $ 682,840,817 | $ 703,326,041 | $ 724,425,822 | $ 746,158,597 | $ 768,543,355 | $ 791,599,655 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 133,884 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ (166,602,630) | $ (171,462,808) | $ (176,606,693) | $ (181,904,893) | $ (187,362,040) | $ (192,982,901) | $ (198,772,388) | $ (204,735,560) | $ (210,877,627) | $ (217,203,956) | $ (223,720,074) |
| **Book Deductions** | **$ 2,100,263** | **$ 1,942,744** | **$ 1,797,038** | **$ 1,662,260** | **$ 1,537,590** | **$ 1,422,271** | **$ 1,315,601** | **$ 1,216,931** | **$ 1,125,661** | **$ 955,962** | **$ 347,727** |
| *- CAPEX #1 Depreciation* | $ 172,006 | $ 159,106 | $ 147,173 | $ 136,135 | $ 125,925 | $ 116,481 | $ 107,744 | $ 99,664 | $ 92,189 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 1,169,991 | $ 1,082,241 | $ 1,001,073 | $ 925,993 | $ 856,543 | $ 792,303 | $ 732,880 | $ 677,914 | $ 627,070 | $ 580,040 | $ - |
| *- CAPEX #3 Depreciation* | $ 758,266 | $ 701,396 | $ 648,791 | $ 600,132 | $ 555,122 | $ 513,488 | $ 474,976 | $ 439,353 | $ 406,402 | $ 375,922 | $ 347,727 |
| Taxable Before Depletion | $ (168,702,893) | $ (173,405,552) | $ (178,403,730) | $ (183,567,153) | $ (188,899,631) | $ (194,405,172) | $ (200,087,989) | $ (205,952,491) | $ (212,003,288) | $ (218,159,917) | $ (224,067,802) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ (84,351,446) | $ (86,702,776) | $ (89,201,865) | $ (91,783,577) | $ (94,449,815) | $ (97,202,586) | $ (100,043,995) | $ (102,976,245) | $ (106,001,644) | $ (109,079,959) | $ (112,033,901) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 59,157,804 | $ 60,932,538 | $ 62,760,514 | $ 64,643,329 | $ 66,582,629 | $ 68,580,108 | $ 70,637,511 | $ 72,756,637 | $ 74,939,336 | $ 77,187,516 | $ 79,503,141 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ (168,702,893)** | **$ (173,405,552)** | **$ (178,403,730)** | **$ (183,567,153)** | **$ (188,899,631)** | **$ (194,405,172)** | **$ (200,087,989)** | **$ (205,952,491)** | **$ (212,003,288)** | **$ (218,159,917)** | **$ (224,067,802)** |
| *- Income Tax* | $ (43,862,752) | $ (45,085,443) | $ (46,384,970) | $ (47,727,460) | $ (49,113,904) | $ (50,545,345) | $ (52,022,877) | $ (53,547,648) | $ (55,120,855) | $ (56,721,578) | $ (58,257,628) |
| Net Profit/Loss | $ (124,840,141) | $ (128,320,108) | $ (132,018,760) | $ (135,839,693) | $ (139,785,727) | $ (143,859,828) | $ (148,065,112) | $ (152,404,843) | $ (156,882,433) | $ (161,438,339) | $ (165,810,173) |
| **Book Deductions** | **$ 2,100,263** | **$ 1,942,744** | **$ 1,797,038** | **$ 1,662,260** | **$ 1,537,590** | **$ 1,422,271** | **$ 1,315,601** | **$ 1,216,931** | **$ 1,125,661** | **$ 955,962** | **$ 347,727** |
| *+ CAPEX #1 Depreciation* | $ 172,006 | $ 159,106 | $ 147,173 | $ 136,135 | $ 125,925 | $ 116,481 | $ 107,744 | $ 99,664 | $ 92,189 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 1,169,991 | $ 1,082,241 | $ 1,001,073 | $ 925,993 | $ 856,543 | $ 792,303 | $ 732,880 | $ 677,914 | $ 627,070 | $ 580,040 | $ - |
| *+ CAPEX #3 Depreciation* | $ 758,266 | $ 701,396 | $ 648,791 | $ 600,132 | $ 555,122 | $ 513,488 | $ 474,976 | $ 439,353 | $ 406,402 | $ 375,922 | $ 347,727 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ (122,739,877) | $ (126,377,365) | $ (130,221,723) | $ (134,177,433) | $ (138,248,136) | $ (142,437,556) | $ (146,749,511) | $ (151,187,912) | $ (155,756,772) | $ (160,482,377) | $ (165,462,446) |
| **Captial Cost** | **$ 2,231,394** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 2,231,394 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ (124,971,271) | $ (126,377,365) | $ (130,221,723) | $ (134,177,433) | $ (138,248,136) | $ (142,437,556) | $ (146,749,511) | $ (151,187,912) | $ (155,756,772) | $ (160,482,377) | $ (165,462,446) |
| Discounted Cash Flow | $ (39,819,698) | $ (36,607,021) | $ (34,291,450) | $ (32,121,011) | $ (30,086,822) | $ (28,180,511) | $ (26,394,189) | $ (24,720,432) | $ (23,152,252) | $ (21,686,075) | $ (20,326,395) |
| **Cumulative DCF** | $ (621,244,920) | $ (657,851,941) | $ (692,143,390) | $ (724,264,401) | $ (754,351,223) | $ (782,531,735) | $ (808,925,924) | $ (833,646,356) | $ (856,798,608) | $ (878,484,683) | $ (898,811,078) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 108,132,002 | Payback Period | 0 | 25.5 |
| Constant Dollar OPEX | $ 413,129,924 | Internal Rate of Return | 0 | N/A |
| Constant Dollar REE Rev | $ 296,372,095 | Net Present Value | 0 | $ (898,811,078) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ - |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

Power Cost: $ 0.07 $/kWhr
Effiicency: 0.85

| ID | Unit Operation | Total HP | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | $ 287.61 | $ 862.84 | $ 289,913 | $ 0.16 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | $ 179.76 | $ 539.27 | $ 181,196 | $ 0.10 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.01 | $ 0.00 |
| 5 | Conveyor | 31.00 | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 6 | Conveyor | 31.00 | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 7 | Conveyor | - | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 0.29** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.06 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.06** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 611** | **$ 1,834** | **$ 616,370** | **$ 0.35** | **$ 0.15** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 2.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 500 | tph |
| Residue Rate | 339 | tph |
| Slurry Flow | 9250 | gpm |
| Leaching Time | 45 | min |
| Acid Consumption | 264.10 | kg/ton |
| Bioleaching Sol'n Consumption | 0 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 2 | -- |
| Thickener Capacity | 10.75 | ft^2/tph |
| Polymer Dose | 0.016 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 8 | 100% | 52,031 | gal | $ 181,404 | $ 1,451,235 | $ - | |
| 2 | Tank Mixer | 8 | 100% | 78 | HP | $ 187,044 | $ 1,496,351 | $ - | |
| 3 | Process Pump | 4 | 100% | 4,625 | gpm | $ 161,957 | $ 647,827 | $ - | |
| 4 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 5 | Belt Filter | 2 | 100% | 170 | TPH | $ 1,124,090 | $ 2,248,181 | $ - | |
| 6 | Solution Heater | 1 | 100% | 1 | MBTU/hr | $ 30,807 | $ 30,807 | $ - | |
| 7 | Thickener | 2 | 100% | 1,823 | ft^2 | $ 174,023 | $ 348,045 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 6,318,002** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 624.38 | Effiicency: | 0.85 | | $ 299.30 | $ 897.89 | $ 301,691 | $ 0.17 | $ 0.07 |
| 3 | Process Pump | 462.50 | | | | $ 221.70 | $ 665.10 | $ 223,475 | $ 0.13 | $ 0.06 |
| 4 | Conveyor | 31.00 | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 5 | Belt Filter | 82.21 | | | | $ 39.41 | $ 118.22 | $ 39,722 | $ 0.02 | $ 0.01 |
| 6 | Solution Heater | 78.00 | | | | $ 37.39 | $ 112.16 | $ 37,687 | $ 0.02 | $ 0.01 |
| 7 | Thickener | 2.32 | | | | $ 1.11 | $ 3.33 | $ 1,120 | $ 0.00 | $ 0.00 |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1280** | | | | **$ 613.76** | **$ 1,841.29** | **$ 618,673** | **$ 0.35** | **$ 0.15** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | - | L/hr | $ 0.008 | / L | $ - | $ - | $ - | $ - | $ - |
| 2 | H2SO4 | 132.05 | mt/hr | $ 220.000 | / mt | $ 232,412.27 | $ 697,236.82 | $ 234,271,572 | $ 131.38 | $ 58.10 |
| 3 | Natural Gas | 0.75 | MCF/hr | $ 2.860 | / MCF | $ 17.21 | $ 51.63 | $ 17,346 | $ 0.01 | $ 0.00 |
| 4 | Polymer | 33.61 | kg/hr | $ 2.250 | / kg | $ 605.06 | $ 1,815.18 | $ 609,901 | $ 0.34 | $ 0.15 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 233,034.54** | **$ 699,103.63** | **$ 234,898,819** | **$ 131.73** | **$ 58.26** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 233,648** | **$ 700,945** | **$ 235,517,492** | **$ 132.08** | **$ 58.41** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 7.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8404 | gpm |
| Feed Rate | 31812 | lpm |
| Precip Mass Flow | 15.87 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 974.97 | ft^2/tpd |
| NaOH Dose | 18.31 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 7 | 100% | 54,024 | gal | $ 123,520 | $ 864,638 | $ - | |
| 2 | Tank Mixer | 7 | 100% | 27 | HP | $ 102,814 | $ 719,699 | $ - | |
| 3 | Process Pump | 21 | 100% | 1,201 | gpm | $ 48,414 | $ 1,016,688 | $ - | |
| 4 | Filter Press | 1 | 100% | 75 | ft^3 | $ 204,067 | $ 204,067 | $ - | |
| 5 | Conveyor | 1 | 100% | 16 | TPH | $ 14,370 | $ 14,370 | $ - | |
| 6 | Thickener | 7 | 100% | 2,210 | ft^2 | $ 203,112 | $ 1,421,787 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 4,241,248** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 189.08 | Effiicency: | 0.85 | | $ 90.64 | $ 271.91 | $ 91,363 | $ 0.05 | $ 0.02 |
| 3 | Process Pump | 630.28 | | | | $ 302.13 | $ 906.38 | $ 304,544 | $ 0.17 | $ 0.08 |
| 4 | Filter Press | 12.55 | | | | $ 6.02 | $ 18.05 | $ 6,065 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 0.98 | | | | $ 0.47 | $ 1.41 | $ 475 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 9.84 | | | | $ 4.72 | $ 14.15 | $ 4,754 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **843** | | | | **$ 403.97** | **$ 1,211.91** | **$ 407,201** | **$ 0.23** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 34.96 | mt/hr | $ 125.000 | / mt | $ 34,957.02 | $ 104,871.05 | $ 35,236,673 | $ 19.76 | $ 8.74 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 34,957.02** | **$ 104,871.05** | **$ 35,236,673** | **$ 19.76** | **$ 8.74** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 35,361** **$ 106,083** **$ 35,643,874** **$ 19.99** **$ 8.84**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 6.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8385 | gpm |
| Feed Rate | 31741 | lpm |
| Precip Mass Flow | 17.82 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 4942.15 | ft^2/tpd |
| NaOH Dose | 3.67 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 6 | 100% | 62,888 | gal | $ 134,538 | $ 807,228 | $ - | |
| 2 | Tank Mixer | 6 | 100% | 31 | HP | $ 112,013 | $ 672,076 | $ - | |
| 3 | Process Pump | 18 | 100% | 1,398 | gpm | $ 51,302 | $ 923,436 | $ - | |
| 4 | Filter Press | 1 | 100% | 84 | ft^3 | $ 221,954 | $ 221,954 | $ - | |
| 5 | Conveyor | 1 | 100% | 18 | TPH | $ 15,317 | $ 15,317 | $ - | |
| 6 | Thickener | 6 | 100% | 14,681 | ft^2 | $ 927,924 | $ 5,567,546 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 8,207,556** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 188.66 | Effiicency: | 0.85 | | | $ 90.44 | $ 271.31 | $ 91,160 | $ 0.05 | $ 0.02 |
| 3 | Process Pump | 628.88 | | | | | $ 301.46 | $ 904.37 | $ 303,868 | $ 0.17 | $ 0.08 |
| 4 | Filter Press | 14.10 | | | | | $ 6.76 | $ 20.28 | $ 6,815 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 1.11 | | | | | $ 0.53 | $ 1.59 | $ 534 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 56.02 | | | | | $ 26.85 | $ 80.56 | $ 27,069 | $ 0.02 | $ 0.01 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **889** | | | | | **$ 426.04** | **$ 1,278.12** | **$ 429,447** | **$ 0.24** | **$ 0.11** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 6.99 | mt/hr | $ 125.000 | / mt | $ 6,989.37 | $ 20,968.12 | $ 7,045,287 | $ 3.95 | $ 1.75 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 6,989.37** | **$ 20,968.12** | **$ 7,045,287** | **$ 3.95** | **$ 1.75** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 7,415** **$ 22,246** **$ 7,474,734** **$ 4.19** **$ 1.85**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 17.82 | tph |
| Slurry Feed Rate | 888 | gpm |
| Leaching Time | 120 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 1122.31 | kg/ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 26,632 | gal | $ 33,004 | $ 132,015 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 40 | HP | $ 128,203 | $ 512,813 | $ - | |
| 3 | Process Pump | 1 | 100% | 888 | gpm | $ 86,298 | $ 86,298 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 731,126** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 159.79 | Effiicency: | 0.85 | | | $ 76.60 | $ 229.79 | $ 77,210 | $ 0.04 | $ 0.02 |
| 3 | Process Pump | 22.19 | | | | | $ 10.64 | $ 31.92 | $ 10,724 | $ 0.01 | $ 0.00 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **182** | | | | | **$ 87.24** | **$ 261.71** | **$ 87,933** | **$ 0.05** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 20.00 | mt/hr | $ 270.000 | / mt | $ 43,206.79 | $ 129,620.36 | $ 43,552,443 | $ 24.42 | $ 10.80 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 43,206.79** | **$ 129,620.36** | **$ 43,552,443** | **$ 24.42** | **$ 10.80** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 43,294** | **$ 129,882** | **$ 43,640,376** | **$ 24.47** | **$ 10.82** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 888 | gpm |
| Feed Rate | 3360 | lpm |
| Precip Mass Flow | 0.15 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 1.86 | g/L of Feed |
| Oxalic Acid Dose | 7.58 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.09 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 4,439 | gal | $ 6,025 | $ 24,099 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 2 | HP | $ 25,115 | $ 100,458 | $ - | |
| 3 | Process Pump | 3 | 100% | 888 | gpm | $ 86,298 | $ 258,894 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 0.71 | ft^3 | $ 7,118 | $ 7,118 | $ - | |
| 6 | Roaster | 1 | 100% | 0.23 | MBTU/hr | $ 444,027 | $ 444,027 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 834,596** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 8.88 | Effiicency: | 0.85 | | | $ 4.26 | $ 12.77 | $ 4,289 | $ 0.00 | $ 0.00 |
| 3 | Process Pump | 66.58 | | | | | $ 31.92 | $ 95.75 | $ 32,171 | $ 0.02 | $ 0.01 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.12 | | | | | $ 0.06 | $ 0.17 | $ 57 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.03 | | | | | $ 0.02 | $ 0.05 | $ 17 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **76** | | | | | **$ 36.24** | **$ 108.73** | **$ 36,534** | **$ 0.02** | **$ 0.01** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 0.37 | mt/hr | $ 125.000 | / mt | $ 374.31 | $ 1,122.92 | $ 377,301 | $ 0.21 | $ 0.09 |
| 2 | Oxalic Acid | 1,529.27 | kg/hr | $ 0.600 | / kg | $ 7,340.48 | $ 22,021.45 | $ 7,399,208 | $ 4.15 | $ 1.84 |
| 3 | Natural Gas | 0.22 | MCF/hr | $ 2.860 | / MCF | $ 5.12 | $ 15.37 | $ 5,164 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 7,719.91** | **$ 23,159.74** | **$ 7,781,673** | **$ 4.36** | **$ 1.93** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 7,756** **$ 23,268** **$ 7,818,207** **$ 4.38** **$ 1.94**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8404 | gpm |
| Raffinate Rate: | 8404 | gpm |
| Extraction Feed (O+A) | 16807 | gpm |
| Scrub Feed (O+A) | 231 | gpm |
| Strip Feed (O+A) | 462 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 231 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 276313 | gal |
| Precip Mixing Intensity | 0.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 60.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,807 | gal | $ 96,082 | $ 192,164 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 42 | HP | $ 131,910 | $ 263,819 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 252,112 | gal | $ 440,633 | $ 881,266 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,404 | gpm | $ 203,385 | $ 406,770 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 231 | gal | $ 8,621 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,755 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,466 | gal | $ 39,537 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 462 | gal | $ 12,731 | $ 25,462 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 17,378 | $ 34,756 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,931 | gal | $ 58,385 | $ 116,770 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,688 | $ 19,688 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 60,143 | kg | $ 3,765,482 | $ - | $ 3,765,482 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 1,043,343 | kg | $ 1,129,780 | $ - | $ 1,129,780 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,765 | gal | $ 5,766 | $ 23,063 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 1.38 | HP | $ 19,229 | $ 76,916 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,688 | $ 39,376 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.02 | ft^3 | $ 465 | $ 465 | $ - | |
| 20 | Process Pump | 2 | 100% | 231 | gpm | $ 51,647 | $ 103,293 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,183,808** | **$ 4,895,262** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 84.04 | Effiicency: | 0.85 | | $ 40.28 | $ 120.85 | $ 40,606 | $ 0.02 | $ 0.01 |
| 3 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 420.19 | | | | $ 201.42 | $ 604.25 | $ 203,029 | $ 0.11 | $ 0.05 |
| 5 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.31 | | | | $ 1.11 | $ 3.32 | $ 1,116 | $ 0.00 | $ 0.00 |
| 11 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.46 | | | | $ 0.22 | $ 0.66 | $ 223 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 5.53 | | | | $ 2.65 | $ 7.95 | $ 2,672 | $ 0.00 | $ 0.00 |
| 17 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.92 | | | | $ 0.44 | $ 1.33 | $ 445 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | | | | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.55 | | | | $ 5.54 | $ 16.61 | $ 5,582 | $ 0.00 | $ 0.00 |
| | **TOTAL:** | **525** | | | | **$ 251.66** | **$ 754.98** | **$ 253,674** | **$ 0.14** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 108.80 | kg/hr | $ 0.940 | / kg | $ 818.14 | $ 2,454.43 | $ 824,688 | $ 0.46 | $ 0.20 |
| 2 | D2EHPA | 5.73 | kg/hr | $ 5.000 | / kg | $ 229.04 | $ 687.13 | $ 230,876 | $ 0.13 | $ 0.06 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.88 | mt/hr | $ 270.000 | / mt | $ 1,905.60 | $ 5,716.80 | $ 1,920,846 | $ 1.08 | $ 0.48 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,952.79** | **$ 8,858.36** | **$ 2,976,410** | **$ 1.67** | **$ 0.74** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,204** **$ 9,613** **$ 3,230,084** **$ 1.81** **$ 0.80**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C3
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 933.00 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 3531.80 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 932.37 | gpm | HCL Dose | 7.58 | g/L of Feed |
| Mn Feed Rate | 3529.42 | lpm | | | |
| Co Precip Mass Flow | 0.53 | tph | | | |
| Mn Precip Mass Flow | 0.68 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 8.04 | g/L of Feed | | | |
| NaOH Dose | 12.59 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 13,995 | gal | $ 17,920 | $ 35,839 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 21 | HP | $ 89,187 | $ 178,373 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 933 | gpm | $ 87,951 | $ 87,951 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 2.52 | ft^3 | $ 17,777 | $ 17,777 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 13,986 | gal | $ 17,908 | $ 35,816 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 21 | HP | $ 89,153 | $ 178,306 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 932 | gpm | $ 87,928 | $ 87,928 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 3 | ft^3 | $ 21,114 | $ 21,114 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 643,104** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 41.99 | Effiicency: | 0.85 | | | $ 20.13 | $ 60.38 | $ 20,287 | $ 0.01 | $ 0.01 |
| 3 | Process Pump | 23.33 | | | | | $ 11.18 | $ 33.54 | $ 11,270 | $ 0.01 | $ 0.00 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.42 | | | | | $ 0.20 | $ 0.61 | 205 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 41.96 | | | | | $ 20.11 | $ 60.34 | $ 20,273 | $ 0.01 | $ 0.01 |
| 8 | Process Pump | 23.31 | | | | | $ 11.17 | $ 33.52 | $ 11,263 | $ 0.01 | $ 0.00 |
| 9 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 0.54 | | | | | $ 0.26 | $ 0.77 | 260 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **132** | | | | | **$ 63.05** | **$ 189.16** | **$ 63,557** | **$ 0.04** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 1.70 | mt/hr | $ 360.000 | / mt | $ 4,909.55 | $ 14,728.65 | $ 4,948,825 | $ 2.78 | $ 1.23 |
| 2 | NaOH | 2.67 | mt/hr | $ 125.000 | / mt | $ 2,665.81 | $ 7,997.44 | $ 2,687,139 | $ 1.51 | $ 0.67 |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 0.53 | $ 1.58 | 530 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 7,575.89** | **$ 22,727.66** | **$ 7,636,494** | **$ 4.28** | **$ 1.89** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 7,639** **$ 22,917** **$ 7,700,052** **$ 4.32** **$ 1.91**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | Shakespeare Lignite | Date: | 3/18/2023 |
| Case: | SHXP-C4 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | an |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | 180.2% |
| *(unit conversion)* | 12000 | TPD | Project NPV: | 2,129,214,160 |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 0.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C4 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 95,686,245 | | | | | |
| Constant Dollar Annual OPEX | $ | 505,190,449 | $ | 125.30 | /ton | $ 194.0 | /kg |
| Constant Dollar Annual REE Rev | $ | 782,160,698 | $ | 193.99 | /ton | $ 300.37 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | |
|---|---|---|---|---|
| Payback Period | 3.5 | years | *0.5* | *operating years* |
| Internal Rate of Return | 180% | | | |
| Net Present Value | $ 2,129,214,160 | | | |
| Total Severence Tax Paid | $ - | | | |
| Total State and Federal Tax Paid | $ 1,262,943,778 | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C4 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 1,250 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 2.78 | 34.80 | 15.78 | | | | | | | 15.78 |
| Yttrium | 18.88 | 236.02 | 107.06 | | | | | | | 107.06 |
| Lanthanum | 15.66 | 195.74 | 88.79 | | | | | | | 88.79 |
| Cerium | 30.20 | 377.56 | 171.26 | | | | | | | 171.26 |
| Praseodymium | 3.24 | 40.52 | 18.38 | | | | | | | 18.38 |
| Neodymium | 14.09 | 176.16 | 79.91 | | | | | | | 79.91 |
| Samarium | 2.97 | 37.15 | 16.85 | | | | | | | 16.85 |
| Europium | 0.68 | 8.51 | 3.86 | | | | | | | 3.86 |
| Gadolinium | 3.61 | 45.12 | 20.47 | | | | | | | 20.47 |
| Terbium | 0.49 | 6.15 | 2.79 | | | | | | | 2.79 |
| Dysprosium | 3.03 | 37.90 | 17.19 | | | | | | | 17.19 |
| Holmium | 0.59 | 7.37 | 3.34 | | | | | | | 3.34 |
| Erbium | 1.54 | 19.28 | 8.74 | | | | | | | 8.74 |
| Thulium | 0.34 | 4.23 | 1.92 | | | | | | | 1.92 |
| Ytterbium | 1.60 | 19.96 | 9.06 | | | | | | | 9.06 |
| Lutetium | 0.28 | 3.54 | 1.60 | | | | | | | 1.60 |
| Manganese | -- | 505.39 | 229.24 | | | 0.00 | -- | | | 229.24 |
| Cobalt | -- | 248.28 | 112.62 | | | 0.00 | -- | | | 112.62 |
| Nickel | -- | 759.69 | 344.59 | | | 0.00 | -- | | | 344.59 |
| Zinc | -- | 207.62 | 94.17 | | | 0.00 | -- | | | 94.17 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 1250.00 | 566.99 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 567.0 |
| Total non-REE CM | -- | 1720.98 | 780.62 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 780.6 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 15.78 | 0.00 | 13.67 | 0.01 | 0.00 | 0.00 | 13.69 | 2.09 | 86.7% | |
| Yttrium | 107.06 | 54.17 | 0.00 | 1.18 | 5.06 | 0.00 | 60.40 | 46.66 | 56.4% | |
| Lanthanum | 88.79 | 44.55 | 0.00 | 0.20 | 1.42 | 0.00 | 46.16 | 42.62 | 52.0% | |
| Cerium | 171.26 | 114.79 | 0.00 | 0.37 | 0.96 | 0.00 | 116.12 | 55.14 | 67.8% | |
| Praseodymium | 18.38 | 3.60 | 0.00 | 0.02 | 0.10 | 0.00 | 3.73 | 14.65 | 20.3% | |
| Neodymium | 79.91 | 39.77 | 0.00 | 0.01 | 0.52 | 0.00 | 40.31 | 39.60 | 50.4% | |
| Samarium | 16.85 | 9.20 | 0.00 | 0.03 | 0.13 | 0.00 | 9.36 | 7.49 | 55.5% | |
| Europium | 3.86 | 1.84 | 0.00 | 0.00 | 0.01 | 0.00 | 1.85 | 2.01 | 48.0% | |
| Gadolinium | 20.47 | 8.95 | 0.00 | 0.04 | 0.15 | 0.00 | 9.14 | 11.33 | 44.7% | |
| Terbium | 2.79 | 1.00 | 0.00 | 0.00 | 0.00 | 0.00 | 1.01 | 1.79 | 36.0% | |
| Dysprosium | 17.19 | 9.10 | 0.00 | 0.08 | 0.38 | 0.00 | 9.56 | 7.63 | 55.6% | |
| Holmium | 3.34 | 1.47 | 0.00 | 0.02 | 0.07 | 0.00 | 1.56 | 1.79 | 46.6% | |
| Erbium | 8.74 | 3.97 | 0.00 | 0.06 | 0.35 | 0.00 | 4.38 | 4.36 | 50.1% | |
| Thulium | 1.92 | 0.51 | 0.00 | 0.09 | 0.07 | 0.00 | 0.67 | 1.25 | 34.8% | |
| Ytterbium | 9.06 | 3.45 | 0.00 | 0.20 | 0.64 | 0.00 | 4.29 | 4.76 | 47.4% | |
| Lutetium | 1.60 | 0.56 | 0.00 | 0.02 | 0.10 | 0.00 | 0.68 | 0.92 | 42.6% | |
| Manganese | 229.24 | 0.93 | 0.00 | 29.65 | 117.61 | 0.00 | 148.19 | 81.05 | 64.6% | |
| Cobalt | 112.62 | 1.29 | 0.00 | 1.08 | 70.58 | 0.00 | 72.95 | 39.67 | 64.8% | |
| Nickel | 344.59 | 1.84 | 0.00 | 13.40 | 167.47 | 0.00 | 182.71 | 161.88 | 53.0% | |
| Zinc | 94.17 | 1.53 | 0.00 | 0.21 | 60.32 | 0.00 | 62.07 | 32.11 | 65.9% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 566.99 | 296.93 | 13.67 | 2.34 | 9.97 | 0.00 | 322.91 | 244.08 | 57.0% | <---- |
| Total non-REE CM | 780.62 | 5.60 | 0.00 | 44.34 | 415.98 | 0.00 | 465.92 | 314.71 | 59.7% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 402 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 8 | ton | $ | 1.50 |
| Dust/Volatiles | - | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | |
|---|---|
| 7.13 | TPD REE |
| 0.328 | TPD Sc |
| 11.05 | TPD CM |
| 0.52 | REE |
| 86.6% | Sc |
| 12.9% | Mn |
| 54.1% | Metal-S |
| 500 | Feed |
| 1,250 | Grade |
| 3.5 | PP |
| 180% | IRR |
| 2129214160 | NPV |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C4 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 0.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 10% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 2.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | UK: Design |
| Dissolved Solids in Leach | 19.6% | % | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Leaching Retention Time | 45 | min | UK: Design |
| Thickener UF % Solids | 47% | % | UK: SHXP-C4/SHXP-C4 Samples |
| Residue Settling Rate | 15.0 | ft/hr | Not Available, Assumed |
| **Reagent Use** | | | |
| Acid Consumption | 301.4 | kg/ton | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Bioleaching Sol'n Consumption | 0 | L/ton | N/A |
| Polymer Dosage | 0.016 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 3.00 | -- | VT: Design |
| Thickener Capacity | 8.56 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 7.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | UK: Design |
| Precip Solid Yield | 3.75 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 2.24 | ft/hr | UK: Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 0.5% | % | VT Override from 0.27% \| UK: SHXP-C4/SHXP-C4 Samples |
| **Reagent Use** | | | |
| NaOH Dose | 49.02 | g/L of Feed | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 884.99 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 1250 | ppm | |
| Grade After Sorter | 1250 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 500 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 0 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 500 | tph | |
| Leaching Residue Rate | 402 | tph | |
| Slurry Flow (feed) | 9250 | gpm | |
| Product Flow | 8,246 | gpm | |
| Thickener Feed % Solids (eff) | 16% | % | |
| Liquid Density | 1.018 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 48.92 | g/L | |
| Leaching Vessel Size | 52031.25 | gal | |
| Leaching Vessel Size | 196.96 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 134.00 | tph | 70 to 200 tph |
| Thickener Area | 1720.72 | ft^2 | |
| Thickener Diameter | 46.82 | ft | ideal < 100' |
| Liquid Balance | 244.65 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,246 | gpm | |
| Slurry Feed Rate | 31,216 | lpm | |
| Precip Mass Flow | 7.75 | tph | |
| Thickener Feed % Solids (eff) | 0.37% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.015 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 200.68 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 36.51 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 9.10 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 980 | ft^2 | |
| Thickener Diameter | 35.33 | ft | ideal < 100' |

*PM5: REE Precipitation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |

**Process Parameters**

| | | | |
|---|---|---|---|
| Precip Conditioning Time | 45 | min | UK: Based on Design |
| Precip Solid Yield | 4.23 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 0.044 | ft/hr | UK: Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.43% | % | VT Override from 0.23% \| UK: SHXP-C4/SHXP-C4 Samples |

**Reagent Use**

| | | | |
|---|---|---|---|
| NaOH Dose | 2.82 | g/L of Feed | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |

**Equipment Specifications**

| | | | |
|---|---|---|---|
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 5198.38 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Slurry Feed Rate | 8237 | gpm | |
| Slurry Feed Rate | 31,182 | lpm | |
| Precip Mass Flow | 8.73 | tph | |
| Thickener Feed % Solids (eff) | 0.4% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.012 | g/cc | |

**Other Indicators**

| | | | |
|---|---|---|---|
| Conditioning Vessel Size | 233.86 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 41.15 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 10.25 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 7564 | ft^2 | |
| Thickener Diameter | 98.16 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | 57.27 | m/m | target = 6.35 |

## PM6: RE-Redissolution

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 1.6% | % | VT override from 10.7% \|\| UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Dissolved Solids in Re-Leach | 25% | % | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Leaching Retention Time | 120 | min | UK: Design |
| **Reagent Use** | | | |
| Acid Consumption | 974.6 | kg/ton | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| Material Balance | | | |
|---|---|---|---|
| Solid Feed Rate | 8.73 | tph | |
| Slurry Flow (feed) | 2,158 | gpm | |
| Product Flow | 2,158 | gpm | |
| Liquid Density | 1.011 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 245.1 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 3.82 | v/v | target = 3.64, based on data |
| Liquid Balance | 10.26 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 16.2 | g/L | |

## PM7: REE Oxalate Precipitation

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | UK: Design |
| Precip Solid Yield | 0.3 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Thickener UF %Solids | | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 12.71 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Oxalic Acid Dose | 8.34 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| Material Balance | | | |
|---|---|---|---|
| Slurry Feed Rate | 2158.07 | gpm | |
| Slurry Feed Rate | 8169.18 | lpm | |
| Precip Mass Flow | 0.14 | tph | |
| Thickener Feed % Solids (eff) | 0.02% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.08 | tph | |
| Liquid Density | 1.011 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 40.85 | m^3 | |
| Filter Press Area | 1.28 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.16 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 6.28 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.70 | m/m | |

## PM8: Sc SX Separation

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | UK: Design |
| Settler Residence Time | 15.00 | min | UK: Design |
| Loading: Number of Stages | 2.00 | - | UK: Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Loading O:A Advance Ratio | 0.03 | v/v | UK: Design |
| Scrubbing: Number of Stages | | - | N/A |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | N/A |
| Scrubbing O:A Advance Ratio | | v/v | N/A |
| Stripping: Number of Stages | 2.00 | - | UK: Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | UK: Design |
| Sc2O3 Grade | 2% | % | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Precip Solid Yield | 0.4 | g/L | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| Material Balance (SX) | | |
|---|---|---|
| PLS Flow | 8246.5 | gpm |
| Raffinate Rate: | 8246.5 | gpm |
| Load Org Recycle | 8019.7 | gpm |
| Load Org Feed | 8246.5 | gpm |
| Loag Org Advance | 226.7 | gpm |
| Scrub Aq Product | 0.0 | gpm |
| Scrub Aq Recycle | 0.0 | gpm |
| Scrub Aq Feed | 0.0 | gpm |
| Scurb Org Product | 226.7 | gpm |
| Strip Aq Product | 18.1 | gpm |
| Strip Aq Recycle | 208.6 | gpm |
| Strip Aq Feed | 226.7 | gpm |
| Strip Org Product | 226.7 | gpm |
| Extraction Feed (O+A): | 16492.9 | gpm |
| Scrub Feed (O+A): | 226.7 | gpm |
| Strip Feed (O+A): | 453.4 | gpm |
| **Material Balance (Precip)** | | |
| Strip Solution Flow | 18.1 | gpm |
| Solid Density | 3.4 | g/cc |
| Solid Product Rate | 0.00 | tph |
| Liquid Density (after precip) | 1.13 | g/cc |
| Thickener Feed % Solids (eff) | 0.03% | % |
| **Total Volumes** | | |
| Total Extraction Volume | 527773 | gal |
| Total Scrubbing Volume | 0 | gal |
| Total Stripping Volume | 14510 | gal |
| Extraction Org Volume | 263886.57 | gal |
| Scrubbing Org Volume | 0.00 | gal |
| Stripping Org Volume | 7255.19 | gal |
| Total Org Volume | 271141.75 | gal |
| **Other Indicators** | | |
| Liquid Balance | 0.00 | gpm must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm should be close to zero |
| Conditioning Vessel Size | 10.27 | m^3 |
| Filter Press Area | 0.02 | ft^3 ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 |
| Thickener Diameter | 0.00 | ft ideal < 100' |

## PM9: CM Precipitation

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 0 | min | N/A |
| Precip Solid Yield | - | g/L | N/A |
| Precip Settlng Rate | 10 | ft/hr | N/A |
| Thickener UF % Solids | 10% | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 0.00 | g/L of Feed | N/A |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 0.00 | min | N/A |
| Thickener Capacity | 0.00 | ft^2/tph | N/A |

| Material Balance | | | |
|---|---|---|---|
| Slurry Feed Rate | 8227 | gpm | |
| Slurry Feed Rate | 31,143 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.012 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM10: CM Redissolution*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 10.0% | % | N/A |
| Dissolved Solids in Re-Leach | 10% | % | N/A |
| Leaching Retention Time | 10 | min | N/A |
| **Reagent Use** | | | |
| Acid Consumption | 0 | kg/ton | N/A |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 0.00 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 0 | gpm | |
| Liquid Density | 1.000 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | |
| Releach Conc. Factor | #DIV/0! | v/v | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | - | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

*PM11: CM Recovery*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 2.83 | g/L | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | | % | Not Applicable |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 1.56 | g/L | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Na2S Dose | 6.84 | g/L of Feed | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| NaOH Dose | 4.53 | g/L of Feed | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| HCL Dose | 19.38 | g/L of Feed | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 2157.91 | gpm | |
| Co Slurry Feed Rate | 8168.58 | lpm | |
| Co Precip Mass Flow | 1.53 | tph | |
| Co Thickener Feed % Solids (eff) | 0.28% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 2156.11 | gpm | |
| Mn Slurry Feed Rate | 8161.78 | lpm | |
| Mn Precip Mass Flow | 0.84 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.15% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.01 | g/cc | |
| Mn Liquid Density | 1.01 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 122.5 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 7.21 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 1.80 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 122.4 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 4.0 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.99 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

*PM12: Biooxidation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Requirec | 500 | gpm | |
| BioLeach Retention Time 1 | 20 | hr | 2.5 |
| BioLeach Retention Time 2 | 20 | hr | 60 |
| BioLeach Retention Time 3 | 20 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 3.5 | kJ/kg-K | |
| Temperature Drop | 10 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 500.0 | gpm | |
| Raw Leach Solution Req'd | 1892.7 | lpm | |
| Yearly Acid Production | 915,766,819 | liters solution/yr | |
| Yearly Acid Production | 1,098,920 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C4 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 7,234,587 | $ - | $ 1,808,647 | $ 7,958,046 | $ 9,907,767 | $ 26,909,047 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 3,448,822 | $ - | $ 862,206 | $ 3,793,705 | $ 4,723,162 | $ 12,827,895 | IIIIIIIIIIIII |
| PM5 | REE Precipitation | $ 5,795,457 | $ - | $ 1,448,864 | $ 6,375,003 | $ 7,936,879 | $ 21,556,204 | IIIIIIIIIIIIIIIIIIII |
| PM6 | RE-Redissolution | $ 1,274,195 | $ - | $ 318,549 | $ 1,401,615 | $ 1,745,010 | $ 4,739,369 | IIIII |
| PM7 | REE Oxalate Precipitation | $ 1,019,128 | $ - | $ 254,782 | $ 1,121,040 | $ 1,395,695 | $ 3,790,645 | IIII |
| PM8 | Sc SX Separation | $ 2,162,479 | $ 4,803,650 | $ 540,620 | $ 2,378,727 | $ 2,961,514 | $ 12,846,989 | IIIIIIIIIIIII |
| PM9 | CM Precipitation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM10 | CM Redissolution | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM11 | CM Recovery | $ 1,035,762 | $ - | $ 258,940 | $ 1,139,338 | $ 1,418,476 | $ 3,852,515 | IIII |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| | TOTAL | $ 24,434,089 | $ 4,803,650 | $ 6,108,522 | $ 26,877,498 | $ 33,462,486 | $ 95,686,245 | |

## Module Summary

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| Total Lang Factor (=1+sum % Base) | | 372% | |

Year of Construction: 2025

# OPERATING COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C4 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 1.18 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 0.45 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 95,686,245 | $ Total | $ 949.27 | $ 2,847.80 | $ 956,862 | $ 0.37 | $ 0.24 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 0.12 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 782,160,698 | $ Total | $ 3,879.77 | $ 11,639.30 | $ 3,910,803 | $ 1.50 | $ 0.97 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 0.18 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 95,686,245 | $ Total | $ 949.27 | $ 2,847.80 | $ 956,862 | $ 0.37 | $ 0.24 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 10,759.75** | **$ 32,279.26** | **$ 10,845,830** | **$ 4.17** | **$ 2.69** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 0.20 | $ 0.13 |
| PM2 | Roasting | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | PR | 1296 | | $ 621.02 | $ 1,863.07 | $ 625,992 | $ 0.24 | $ 0.16 |
| PM4 | Al Precipitation | PR | 815 | | $ 390.67 | $ 1,172.02 | $ 393,800 | $ 0.15 | $ 0.10 |
| PM5 | REE Precipitation | PR | 839 | | $ 402.40 | $ 1,207.19 | $ 405,616 | $ 0.16 | $ 0.10 |
| PM6 | RE-Redissolution | PR | 442 | | $ 212.07 | $ 636.20 | $ 213,765 | $ 0.08 | $ 0.05 |
| PM7 | REE Oxalate Precipitation | PR | 184 | | $ 88.05 | $ 264.15 | $ 88,753 | $ 0.03 | $ 0.02 |
| **Sub-Total** | | | **5460.5** | | **$ 2,617.50** | **$ 7,852.49** | **$ 2,638,436.62** | **$ 1.01** | **$ 0.65** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.04 | $ 0.03 |
| PM2 | Roasting | CR | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | CR | | $ 265,833.78 | $ 797,501.34 | $ 267,960,451 | $ 102.90 | $ 66.46 |
| PM4 | Al Precipitation | CR | | $ 91,803.85 | $ 275,411.55 | $ 92,538,281 | $ 35.54 | $ 22.95 |
| PM5 | REE Precipitation | CR | | $ 5,274.34 | $ 15,823.02 | $ 5,316,534 | $ 2.04 | $ 1.32 |
| PM6 | RE-Redissolution | CR | | $ 18,380.20 | $ 55,140.60 | $ 18,527,242 | $ 7.11 | $ 4.60 |
| PM7 | REE Oxalate Precipitation | CR | | $ 25,852.65 | $ 77,557.96 | $ 26,059,476 | $ 10.01 | $ 6.46 |
| **Sub-Total** | | | | **$ 422,017.71** | **$ 1,266,053.14** | **$ 425,393,855.85** | **$ 163.36** | **$ 105.50** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 9,648 | $ 0.10 | $ 321.60 | $ 964.80 | $ 324,173 | $ 0.12 | $ 0.08 |
| Non-Hazardous Precipitate | WD | Tons | 186 | $ 1.50 | $ 93.00 | $ 279.01 | $ 93,748 | $ 0.04 | $ 0.02 |
| Dust/Volatiles | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 414.60** | **$ 1,243.81** | **$ 417,920.54** | **$ 0.16** | **$ 0.10** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 425,050** | **$ 1,275,149** | **$ 428,450,213** | **$ 165** | **$ 106** |

## Overhead Costs

| Allocation | Code | Rate | Basis | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% | % of Direct Cost | $ 32,279 Daily Direct Cost | $ 1,613.96 | $ 4,841.89 | $ 1,626,874 | $ 0.62 | $ 0.40 |
| Variable Portion | OH | 15% | % of Direct Cost | $ 1,275,149 Daily Direct Cost | $ 63,757.47 | $ 191,272.42 | $ 64,267,532 | $ 24.68 | $ 15.94 |
| **TOTAL OVERHEAD COSTS** | | | | | **$ 65,371.43** | **$ 196,114.30** | **$ 65,894,406.45** | **$ 25.31** | **$ 16.34** |

## OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | SHXP-C4 | | | By: | ks |

### Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 10,759.75 | $ 32,279.26 | $ 10,845,830 | $ 4 | $ 2.69 | I |
| Power | $ 2,617.50 | $ 7,852.49 | $ 2,638,437 | $ 1 | $ 0.65 | |
| Consumables | $ 422,017.71 | $ 1,266,053.14 | $ 425,393,856 | $ 163 | $ 105.50 | IIII

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C4 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 782,160,698

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 1215 | 1250 |
| Contained Value | ($/ton) | $ 31.2 | $ 234.6 |
| Feed Basket Price | ($/kg) | $ 28.3 | $ 206.9 |
| Product Basket Price | ($/kg) | $ 24.3 | $ 296.4 |
| Production Rate | (kg/day) | 7421.36 | 7749.92 |
| | (t/day) | 7.42 | 7.75 |
| | (t/yr) | 2493.58 | 2603.97 |
| | (% of US Demand) | 24.9% | |

## Revenue Contribution Table

| | | Product Revenue ($/hr) | | | | | |
|---|---|---|---|---|---|---|---|
| **Element** | **REE/CM Price ($/kg) [1]** | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | **REE Revenue ($/day)** |
| Scandium | $ 4,200.00 | $ 4.8 | $ 88,070.1 | $ - | $ - | $ - | $ 2,113,798 |
| Yttrium | $ 6.00 | $ 309.5 | $ - | $ - | $ - | $ - | $ 7,429 |
| Lanthanum | $ 2.00 | $ 78.4 | $ - | $ - | $ - | $ - | $ 1,881 |
| Cerium | $ 2.00 | $ 211.5 | $ - | $ - | $ - | $ - | $ 5,076 |
| Praseodymium | $ 52.00 | $ 169.8 | $ - | $ - | $ - | $ - | $ 4,075 |
| Neodymium | $ 42.00 | $ 1,461.3 | $ - | $ - | $ - | $ - | $ 35,072 |
| Samarium | $ 2.11 | $ 16.9 | $ - | $ - | $ - | $ - | $ 405 |
| Europium | $ 34.00 | $ 54.3 | $ - | $ - | $ - | $ - | $ 1,303 |
| Gadolinium | $ 32.00 | $ 247.6 | $ - | $ - | $ - | $ - | $ 5,942 |
| Terbium | $ 400.00 | $ 352.7 | $ - | $ - | $ - | $ - | $ 8,464 |
| Dysprosium | $ 230.00 | $ 1,805.9 | $ - | $ - | $ - | $ - | $ 43,341 |
| Holmium | $ 52.96 | $ 66.8 | $ - | $ - | $ - | $ - | $ 1,604 |
| Erbium | $ 34.00 | $ 115.6 | $ - | $ - | $ - | $ - | $ 2,775 |
| Thulium | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Ytterbium | $ 28.72 | $ 84.7 | $ - | $ - | $ - | $ - | $ 2,033 |
| Lutetium | $ 796.64 | $ 381.8 | $ - | $ - | $ - | $ - | $ 9,164 |
| Manganese | $ 0.005 | $ - | $ - | $ 0.3 | $ - | $ - | $ 6 |
| Cobalt | $ 22.00 | $ - | $ - | $ - | $ 1,164.6 | $ - | $ 27,950 |
| Nickel | $ 18.00 | $ - | $ - | $ - | $ 2,260.8 | $ - | $ 54,260 |
| Zinc | $ 3.02 | $ - | $ - | $ - | $ 136.7 | $ - | $ 3,281 |
| Reserved | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Total: | | $ 5,361.6 | $ 88,070.1 | $ 0.3 | $ 3,562.1 | $ - | $ 2,327,859 |

[1] REEs listed as $/kg REO; Mn listed as $/kg 44% Mn Ore; Co listed as $/kg metal content

## Coal Revenue

List Coal Revenue Sources Here:

**Annual Coal Revenue:** $ -

## Other Revenue

List Other Revenue Sources Here:

**Annual Other Revenue:** $ -

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C4 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 854,688,113 | $ 880,328,756 | $ 906,738,619 | $ 933,940,777 | $ 961,959,001 | $ 990,817,771 | $ 1,020,542,304 | $ 1,051,158,573 | $ 1,082,693,330 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 854,688,113 | $ 880,328,756 | $ 906,738,619 | $ 933,940,777 | $ 961,959,001 | $ 990,817,771 | $ 1,020,542,304 | $ 1,051,158,573 | $ 1,082,693,330 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 287,059 | $ 2,049,637 | $ 554,850,718 | $ 571,173,682 | $ 587,979,193 | $ 605,281,297 | $ 623,094,439 | $ 641,433,468 | $ 660,313,650 | $ 679,750,679 | $ 699,760,687 |
| *- Operating Costs* | $ - | $ - | $ - | $ 552,035,244 | $ 568,596,302 | $ 585,654,191 | $ 603,223,816 | $ 621,320,531 | $ 639,960,147 | $ 659,158,951 | $ 678,933,720 | $ 699,301,731 |
| *- Loan #1 Interest* | $ - | $ 287,059 | $ 265,280 | $ 242,195 | $ 217,725 | $ 191,786 | $ 164,291 | $ 135,146 | $ 104,253 | $ 71,506 | $ 36,794 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 1,784,357 | $ 1,648,982 | $ 1,505,484 | $ 1,353,376 | $ 1,192,141 | $ 1,021,233 | $ 840,070 | $ 648,037 | $ 444,482 | $ 228,714 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 924,297 | $ 854,172 | $ 779,840 | $ 701,049 | $ 617,529 | $ 528,999 | $ 435,156 | $ 335,683 | $ 230,242 |
| Gross Profit | $ - | $ (287,059) | $ (2,049,637) | $ 299,837,395 | $ 309,155,074 | $ 318,759,426 | $ 328,659,480 | $ 338,864,562 | $ 349,384,303 | $ 360,228,653 | $ 371,407,893 | $ 382,932,643 |
| **Book Deductions** | $ - | $ 358,823 | $ 2,589,581 | $ 3,748,810 | $ 3,467,649 | $ 3,207,575 | $ 2,967,007 | $ 2,744,482 | $ 2,538,646 | $ 2,348,247 | $ 2,172,129 | $ 2,009,219 |
| *- CAPEX #1 Depreciation* | $ - | $ 358,823 | $ 331,912 | $ 307,018 | $ 283,992 | $ 262,693 | $ 242,991 | $ 224,766 | $ 207,909 | $ 192,316 | $ 177,892 | $ 164,550 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 2,257,670 | $ 2,088,344 | $ 1,931,719 | $ 1,786,840 | $ 1,652,827 | $ 1,528,865 | $ 1,414,200 | $ 1,308,135 | $ 1,210,025 | $ 1,119,273 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,353,447 | $ 1,251,939 | $ 1,158,043 | $ 1,071,190 | $ 990,851 | $ 916,537 | $ 847,797 | $ 784,212 | $ 725,396 |
| Taxable Before Depletion | $ - | $ (645,882) | $ (4,639,219) | $ 296,088,585 | $ 305,687,425 | $ 315,551,851 | $ 325,692,473 | $ 336,120,080 | $ 346,845,657 | $ 357,880,406 | $ 369,235,765 | $ 380,923,424 |
| **Depletion** | $ - | $ - | $ - | $ 119,656,336 | $ 123,246,026 | $ 126,943,407 | $ 130,751,709 | $ 134,674,260 | $ 138,714,488 | $ 142,875,923 | $ 147,162,200 | $ 151,577,066 |
| *- 50% Limit* | $ - | $ (322,941) | $ (2,319,609) | $ 148,044,293 | $ 152,843,712 | $ 157,775,925 | $ 162,846,236 | $ 168,060,040 | $ 173,422,828 | $ 178,940,203 | $ 184,617,882 | $ 190,461,712 |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 119,656,336 | $ 123,246,026 | $ 126,943,407 | $ 130,751,709 | $ 134,674,260 | $ 138,714,488 | $ 142,875,923 | $ 147,162,200 | $ 151,577,066 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (645,882) | $ (4,639,219) | $ 176,432,249 | $ 182,441,399 | $ 188,608,444 | $ 194,940,764 | $ 201,445,820 | $ 208,131,169 | $ 215,004,484 | $ 222,073,565 | $ 229,346,358 |
| *- Income Tax* | $ - | $ (167,929) | $ (1,206,197) | $ 45,872,385 | $ 47,434,764 | $ 49,038,195 | $ 50,684,599 | $ 52,375,913 | $ 54,114,104 | $ 55,901,166 | $ 57,739,127 | $ 59,630,053 |
| Net Profit/Loss | $ - | $ (477,953) | $ (3,433,022) | $ 130,559,864 | $ 135,006,635 | $ 139,570,249 | $ 144,256,165 | $ 149,069,907 | $ 154,017,065 | $ 159,103,318 | $ 164,334,438 | $ 169,716,305 |
| **Book Deductions** | $ - | $ 358,823 | $ 2,589,581 | $ 123,405,146 | $ 126,713,675 | $ 130,150,982 | $ 133,718,716 | $ 137,418,742 | $ 141,253,133 | $ 145,224,170 | $ 149,334,329 | $ 153,586,285 |
| *+ CAPEX #1 Depreciation* | $ - | $ 358,823 | $ 331,912 | $ 307,018 | $ 283,992 | $ 262,693 | $ 242,991 | $ 224,766 | $ 207,909 | $ 192,316 | $ 177,892 | $ 164,550 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 2,257,670 | $ 2,088,344 | $ 1,931,719 | $ 1,786,840 | $ 1,652,827 | $ 1,528,865 | $ 1,414,200 | $ 1,308,135 | $ 1,210,025 | $ 1,119,273 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,353,447 | $ 1,251,939 | $ 1,158,043 | $ 1,071,190 | $ 990,851 | $ 916,537 | $ 847,797 | $ 784,212 | $ 725,396 |
| *+ Depletion* | $ - | $ - | $ - | $ 119,656,336 | $ 123,246,026 | $ 126,943,407 | $ 130,751,709 | $ 134,674,260 | $ 138,714,488 | $ 142,875,923 | $ 147,162,200 | $ 151,577,066 |
| Cash Flow | $ - | $ (119,129) | $ (843,440) | $ 253,965,010 | $ 261,720,310 | $ 269,721,231 | $ 277,974,882 | $ 286,488,648 | $ 295,270,199 | $ 304,327,488 | $ 313,668,767 | $ 323,302,590 |
| **Captial Cost** | $ 4,784,312 | $ 30,102,261 | $ 18,045,963 | $ 3,968,216 | $ 4,206,309 | $ 4,458,688 | $ 4,726,209 | $ 5,009,782 | $ 5,310,369 | $ 5,628,991 | $ 5,966,730 | $ 5,674,699 |
| *- Fixed Capital Costs* | $ 9,568,625 | $ 59,478,570 | $ 30,809,899 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 4,784,312 | $ 29,739,285 | $ 15,404,950 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 362,976 | $ 384,755 | $ 407,840 | $ 432,310 | $ 458,249 | $ 485,744 | $ 514,888 | $ 545,782 | $ 578,529 | $ 613,240 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 2,256,259 | $ 2,391,634 | $ 2,535,132 | $ 2,687,240 | $ 2,848,475 | $ 3,019,383 | $ 3,200,546 | $ 3,392,579 | $ 3,596,134 | $ 3,811,902 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 1,168,742 | $ 1,238,867 | $ 1,313,199 | $ 1,391,991 | $ 1,475,510 | $ 1,564,041 | $ 1,657,883 | $ 1,757,356 | $ 1,862,797 |
| **Net Cash Flow** | $ (4,784,312) | $ (30,221,390) | $ (18,889,404) | $ 249,996,794 | $ 257,514,001 | $ 265,262,543 | $ 273,248,672 | $ 281,478,867 | $ 289,959,830 | $ 298,698,497 | $ 307,702,037 | $ 317,627,891 |
| Discounted Cash Flow | $ (4,784,312) | $ (27,473,991) | $ (15,611,077) | $ 187,826,291 | $ 175,885,528 | $ 164,707,169 | $ 154,241,752 | $ 144,443,166 | $ 135,268,401 | $ 126,677,321 | $ 118,632,455 | $ 111,326,638 |
| **Cumulative DCF** | $ (4,784,312) | $ (32,258,304) | $ (47,869,381) | $ 139,956,911 | $ 315,842,438 | $ 480,549,607 | $ 634,791,359 | $ 779,234,525 | $ 914,502,926 | $ 1,041,180,247 | $ 1,159,812,702 | $ 1,271,139,340 |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 95,686,245 | Payback Period | 3.5 |
| Constant Dollar OPEX | $ 505,190,449 | Internal Rate of Return | 180% |
| Constant Dollar REE Rev | $ 782,160,698 | Net Present Value | $ 2,129,214,160 |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ 1,262,943,778 |

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C4 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 1,115,174,130** | **$ 1,148,629,354** | **$ 1,183,088,234** | **$ 1,218,580,881** | **$ 1,255,138,308** | **$ 1,292,792,457** | **$ 1,331,576,231** | **$ 1,371,523,518** | **$ 1,412,669,223** | **$ 1,455,049,300** | **$ 1,498,700,779** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 1,115,174,130 | $ 1,148,629,354 | $ 1,183,088,234 | $ 1,218,580,881 | $ 1,255,138,308 | $ 1,292,792,457 | $ 1,331,576,231 | $ 1,371,523,518 | $ 1,412,669,223 | $ 1,455,049,300 | $ 1,498,700,779 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 720,399,257** | **$ 741,889,207** | **$ 764,145,883** | **$ 787,070,259** | **$ 810,682,367** | **$ 835,002,838** | **$ 860,052,923** | **$ 885,854,511** | **$ 912,430,146** | **$ 939,803,051** | **$ 967,997,142** |
| *- Operating Costs* | $ 720,280,783 | $ 741,889,207 | $ 764,145,883 | $ 787,070,259 | $ 810,682,367 | $ 835,002,838 | $ 860,052,923 | $ 885,854,511 | $ 912,430,146 | $ 939,803,051 | $ 967,997,142 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 118,474 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ 394,774,873 | $ 406,740,147 | $ 418,942,351 | $ 431,510,622 | $ 444,455,941 | $ 457,789,619 | $ 471,523,307 | $ 485,669,007 | $ 500,239,077 | $ 515,246,249 | $ 530,703,637 |
| **Book Deductions** | **$ 1,858,528** | **$ 1,719,138** | **$ 1,590,203** | **$ 1,470,937** | **$ 1,360,617** | **$ 1,258,571** | **$ 1,164,178** | **$ 1,076,865** | **$ 996,100** | **$ 845,933** | **$ 307,705** |
| *- CAPEX #1 Depreciation* | $ 152,209 | $ 140,793 | $ 130,234 | $ 120,466 | $ 111,431 | $ 103,074 | $ 95,343 | $ 88,193 | $ 81,578 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 1,035,327 | $ 957,678 | $ 885,852 | $ 819,413 | $ 757,957 | $ 701,110 | $ 648,527 | $ 599,888 | $ 554,896 | $ 513,279 | $ - |
| *- CAPEX #3 Depreciation* | $ 670,991 | $ 620,667 | $ 574,117 | $ 531,058 | $ 491,229 | $ 454,387 | $ 420,308 | $ 388,785 | $ 359,626 | $ 332,654 | $ 307,705 |
| Taxable Before Depletion | $ 392,916,345 | $ 405,021,009 | $ 417,352,149 | $ 430,039,685 | $ 443,095,324 | $ 456,531,048 | $ 470,359,129 | $ 484,592,142 | $ 499,242,977 | $ 514,400,317 | $ 530,395,932 |
| **Depletion** | $ 156,124,378 | $ 160,808,110 | $ 165,632,353 | $ 170,601,323 | $ 175,719,363 | $ 180,990,944 | $ 186,420,672 | $ 192,013,292 | $ 197,773,691 | $ 203,706,902 | $ 209,818,109 |
| *- 50% Limit* | $ 196,458,173 | $ 202,510,505 | $ 208,676,074 | $ 215,019,842 | $ 221,547,662 | $ 228,265,524 | $ 235,179,565 | $ 242,296,071 | $ 249,621,489 | $ 257,200,158 | $ 265,197,966 |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 156,124,378 | $ 160,808,110 | $ 165,632,353 | $ 170,601,323 | $ 175,719,363 | $ 180,990,944 | $ 186,420,672 | $ 192,013,292 | $ 197,773,691 | $ 203,706,902 | $ 209,818,109 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ 236,791,967** | **$ 244,212,900** | **$ 251,719,796** | **$ 259,438,361** | **$ 267,375,960** | **$ 275,540,104** | **$ 283,938,457** | **$ 292,578,850** | **$ 301,469,286** | **$ 310,693,415** | **$ 320,577,823** |
| *- Income Tax* | $ 61,565,911 | $ 63,495,354 | $ 65,447,147 | $ 67,453,974 | $ 69,517,750 | $ 71,640,427 | $ 73,823,999 | $ 76,070,501 | $ 78,382,014 | $ 80,780,288 | $ 83,350,234 |
| Net Profit/Loss | $ 175,226,056 | $ 180,717,546 | $ 186,272,649 | $ 191,984,387 | $ 197,858,211 | $ 203,899,677 | $ 210,114,458 | $ 216,508,349 | $ 223,087,271 | $ 229,913,127 | $ 237,227,589 |
| **Book Deductions** | **$ 157,982,906** | **$ 162,527,248** | **$ 167,222,555** | **$ 172,072,261** | **$ 177,079,980** | **$ 182,249,515** | **$ 187,584,850** | **$ 193,090,157** | **$ 198,769,791** | **$ 204,552,835** | **$ 210,125,814** |
| *+ CAPEX #1 Depreciation* | $ 152,209 | $ 140,793 | $ 130,234 | $ 120,466 | $ 111,431 | $ 103,074 | $ 95,343 | $ 88,193 | $ 81,578 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 1,035,327 | $ 957,678 | $ 885,852 | $ 819,413 | $ 757,957 | $ 701,110 | $ 648,527 | $ 599,888 | $ 554,896 | $ 513,279 | $ - |
| *+ CAPEX #3 Depreciation* | $ 670,991 | $ 620,667 | $ 574,117 | $ 531,058 | $ 491,229 | $ 454,387 | $ 420,308 | $ 388,785 | $ 359,626 | $ 332,654 | $ 307,705 |
| *+ Depletion* | $ 156,124,378 | $ 160,808,110 | $ 165,632,353 | $ 170,601,323 | $ 175,719,363 | $ 180,990,944 | $ 186,420,672 | $ 192,013,292 | $ 197,773,691 | $ 203,706,902 | $ 209,818,109 |
| Cash Flow | $ 333,208,961 | $ 343,244,793 | $ 353,495,205 | $ 364,056,648 | $ 374,938,191 | $ 386,149,192 | $ 397,699,309 | $ 409,598,506 | $ 421,857,063 | $ 434,465,961 | $ 447,353,403 |
| **Captial Cost** | **$ 1,974,565** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 1,974,565 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ 331,234,396 | $ 343,244,793 | $ 353,495,205 | $ 364,056,648 | $ 374,938,191 | $ 386,149,192 | $ 397,699,309 | $ 409,598,506 | $ 421,857,063 | $ 434,465,961 | $ 447,353,403 |
| Discounted Cash Flow | $ 105,541,487 | $ 99,425,790 | $ 93,086,336 | $ 87,152,267 | $ 81,597,474 | $ 76,397,559 | $ 71,529,716 | $ 66,972,629 | $ 62,706,364 | $ 58,709,633 | $ 54,955,564 |
| **Cumulative DCF** | $ 1,376,680,827 | $ 1,476,106,617 | $ 1,569,192,952 | $ 1,656,345,219 | $ 1,737,942,694 | $ 1,814,340,253 | $ 1,885,869,969 | $ 1,952,842,598 | $ 2,015,548,962 | $ 2,074,258,596 | $ 2,129,214,160 |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 95,686,245 | Payback Period | 0 | 3.5 |
| Constant Dollar OPEX | $ 505,190,449 | Internal Rate of Return | 0 | 180% |
| Constant Dollar REE Rev | $ 782,160,698 | Net Present Value | 0 | $ 2,129,214,160 |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ 1,262,943,778 |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 0.11 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 0.07 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.01 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 7 | Conveyor | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 0.20** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.04 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.04** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| | **Total Operating Cost:** | **$ 611** | **$ 1,834** | **$ 616,370** | **$ 0.24** | **$ 0.15** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 2.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 500 | tph |
| Residue Rate | 402 | tph |
| Slurry Flow | 9250 | gpm |
| Leaching Time | 45 | min |
| Acid Consumption | 301.38 | kg/ton |
| Bioleaching Sol'n Consumption | 0 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 3 | -- |
| Thickener Capacity | 8.56 | ft^2/tph |
| Polymer Dose | 0.016 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 8 | 100% | 52,031 | gal | $ 181,404 | $ 1,451,235 | $ - | |
| 2 | Tank Mixer | 8 | 100% | 78 | HP | $ 187,044 | $ 1,496,351 | $ - | |
| 3 | Process Pump | 4 | 100% | 4,625 | gpm | $ 161,957 | $ 647,827 | $ - | |
| 4 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 5 | Belt Filter | 3 | 100% | 134 | TPH | $ 1,060,160 | $ 3,180,481 | $ - | |
| 6 | Solution Heater | 1 | 100% | 1 | MBTU/hr | $ 30,807 | $ 30,807 | $ - | |
| 7 | Thickener | 2 | 100% | 1,721 | ft^2 | $ 166,165 | $ 332,330 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 7,234,587** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 624.38 | Effiicency: | 0.85 | | $ 299.30 | $ 897.89 | $ 301,691 | $ 0.12 | $ 0.07 |
| 3 | Process Pump | 462.50 | | | | $ 221.70 | $ 665.10 | $ 223,475 | $ 0.09 | $ 0.06 |
| 4 | Conveyor | 31.00 | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 5 | Belt Filter | 97.49 | | | | $ 46.73 | $ 140.19 | $ 47,104 | $ 0.02 | $ 0.01 |
| 6 | Solution Heater | 78.00 | | | | $ 37.39 | $ 112.16 | $ 37,687 | $ 0.01 | $ 0.01 |
| 7 | Thickener | 2.19 | | | | $ 1.05 | $ 3.15 | $ 1,058 | $ 0.00 | $ 0.00 |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1296** | | | | **$ 621.02** | **$ 1,863.07** | **$ 625,992** | **$ 0.24** | **$ 0.16** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | - | L/hr | $ 0.008 | / L | $ - | $ - | $ - | $ - | $ - |
| 2 | H2SO4 | 150.69 | mt/hr | $ 220.000 | / mt | $ 265,211.51 | $ 795,634.54 | $ 267,333,204 | $ 102.66 | $ 66.30 |
| 3 | Natural Gas | 0.75 | MCF/hr | $ 2.860 | / MCF | $ 17.21 | $ 51.63 | $ 17,346 | $ 0.01 | $ 0.00 |
| 4 | Polymer | 33.61 | kg/hr | $ 2.250 | / kg | $ 605.06 | $ 1,815.18 | $ 609,901 | $ 0.23 | $ 0.15 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 265,833.78** | **$ 797,501.34** | **$ 267,960,451** | **$ 102.90** | **$ 66.46** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 266,455** **$ 799,364** **$ 268,586,443** **$ 103.14** **$ 66.61**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 7.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8246 | gpm |
| Feed Rate | 31216 | lpm |
| Precip Mass Flow | 7.75 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 884.99 | ft^2/tpd |
| NaOH Dose | 49.02 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 7 | 100% | 53,013 | gal | $ 122,214 | $ 855,500 | $ - | |
| 2 | Tank Mixer | 7 | 100% | 27 | HP | $ 101,724 | $ 712,071 | $ - | |
| 3 | Process Pump | 21 | 100% | 1,178 | gpm | $ 48,066 | $ 1,009,389 | $ - | |
| 4 | Filter Press | 1 | 100% | 37 | ft^3 | $ 121,821 | $ 121,821 | $ - | |
| 5 | Conveyor | 1 | 100% | 8 | TPH | $ 9,696 | $ 9,696 | $ - | |
| 6 | Thickener | 7 | 100% | 980 | ft^2 | $ 105,764 | $ 740,345 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 3,448,822** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 185.55 | Effiicency: | 0.85 | | | $ 88.94 | $ 266.83 | $ 89,653 | $ 0.03 | $ 0.02 |
| 3 | Process Pump | 618.48 | | | | | $ 296.47 | $ 889.42 | $ 298,844 | $ 0.11 | $ 0.07 |
| 4 | Filter Press | 6.13 | | | | | $ 2.94 | $ 8.82 | $ 2,962 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 0.48 | | | | | $ 0.23 | $ 0.69 | $ 232 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 4.36 | | | | | $ 2.09 | $ 6.27 | $ 2,108 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **815** | | | | | **$ 390.67** | **$ 1,172.02** | **$ 393,800** | **$ 0.15** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 91.80 | mt/hr | $ 125.000 | / mt | $ 91,803.85 | $ 275,411.55 | $ 92,538,281 | $ 35.54 | $ 22.95 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 91,803.85** | **$ 275,411.55** | **$ 92,538,281** | **$ 35.54** | **$ 22.95** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 92,195** | **$ 276,584** | **$ 92,932,080** | **$ 35.69** | **$ 23.05** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 6.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8237 | gpm |
| Feed Rate | 31182 | lpm |
| Precip Mass Flow | 8.73 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 5198.38 | ft^2/tpd |
| NaOH Dose | 2.82 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 6 | 100% | 61,780 | gal | $ 133,200 | $ 799,198 | $ - | |
| 2 | Tank Mixer | 6 | 100% | 31 | HP | $ 110,895 | $ 665,371 | $ - | |
| 3 | Process Pump | 18 | 100% | 1,373 | gpm | $ 50,955 | $ 917,196 | $ - | |
| 4 | Filter Press | 1 | 100% | 41 | ft^3 | $ 132,775 | $ 132,775 | $ - | |
| 5 | Conveyor | 1 | 100% | 9 | TPH | $ 10,351 | $ 10,351 | $ - | |
| 6 | Thickener | 6 | 100% | 7,564 | ft^2 | $ 545,094 | $ 3,270,566 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 5,795,457** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 185.34 | Effiicency: | 0.85 | | $ 88.84 | $ 266.53 | $ 89,554 | $ 0.03 | $ 0.02 |
| 3 | Process Pump | 617.80 | | | | $ 296.15 | $ 888.44 | $ 298,514 | $ 0.11 | $ 0.07 |
| 4 | Filter Press | 6.91 | | | | $ 3.31 | $ 9.94 | $ 3,339 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 0.54 | | | | $ 0.26 | $ 0.78 | $ 262 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 28.87 | | | | $ 13.84 | $ 41.51 | $ 13,948 | $ 0.01 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **839** | | | | **$ 402.40** | **$ 1,207.19** | **$ 405,616** | **$ 0.16** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 5.27 | mt/hr | $ 125.000 | / mt | $ 5,274.34 | $ 15,823.02 | $ 5,316,534 | $ 2.04 | $ 1.32 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 5,274.34** | **$ 15,823.02** | **$ 5,316,534** | **$ 2.04** | **$ 1.32** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 5,677** | **$ 17,030** | **$ 5,722,151** | **$ 2.20** | **$ 1.42** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 8.73 | tph |
| Slurry Feed Rate | 2158 | gpm |
| Leaching Time | 120 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 974.63 | kg/ton |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 64,742 | gal | $ 76,692 | $ 306,767 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 97 | HP | $ 211,583 | $ 846,330 | $ - | |
| 3 | Process Pump | 1 | 100% | 2,158 | gpm | $ 121,098 | $ 121,098 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,274,195** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 388.45 | Effiicency: | 0.85 | | $ 186.21 | $ 558.62 | $ 187,696 | $ 0.07 | $ 0.05 |
| 3 | Process Pump | 53.95 | | | | $ 25.86 | $ 77.59 | $ 26,069 | $ 0.01 | $ 0.01 |
| 4 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **442** | | | | **$ 212.07** | **$ 636.20** | **$ 213,765** | **$ 0.08** | **$ 0.05** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 8.51 | mt/hr | $ 270.000 | / mt | $ 18,380.20 | $ 55,140.60 | $ 18,527,242 | $ 7.11 | $ 4.60 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 18,380.20** | **$ 55,140.60** | **$ 18,527,242** | **$ 7.11** | **$ 4.60** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 18,592** | **$ 55,777** | **$ 18,741,007** | **$ 7.20** | **$ 4.65** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 2158 | gpm |
| Feed Rate | 8169 | lpm |
| Precip Mass Flow | 0.14 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 60.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 12.71 | g/L of Feed |
| Oxalic Acid Dose | 8.34 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.08 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 10,790 | gal | $ 14,000 | $ 56,000 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 5 | HP | $ 41,448 | $ 165,793 | $ - | |
| 3 | Process Pump | 3 | 100% | 2,158 | gpm | $ 121,098 | $ 363,295 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 1.28 | ft^3 | $ 10,907 | $ 10,907 | $ - | |
| 6 | Roaster | 1 | 100% | 0.21 | MBTU/hr | $ 423,133 | $ 423,133 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,019,128** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 21.58 | Effiicency: | 0.85 | | | $ 10.34 | $ 31.03 | $ 10,428 | $ 0.00 | $ 0.00 |
| 3 | Process Pump | 161.86 | | | | | $ 77.59 | $ 232.76 | $ 78,207 | $ 0.03 | $ 0.02 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.21 | | | | | $ 0.10 | $ 0.31 | $ 104 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.03 | | | | | $ 0.02 | $ 0.05 | $ 15 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **184** | | | | | **$ 88.05** | **$ 264.15** | **$ 88,753** | **$ 0.03** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 6.23 | mt/hr | $ 125.000 | / mt | $ 6,228.71 | $ 18,686.12 | $ 6,278,537 | $ 2.41 | $ 1.56 |
| 2 | Oxalic Acid | 4,087.36 | kg/hr | $ 0.600 | / kg | $ 19,619.31 | $ 58,857.94 | $ 19,776,269 | $ 7.59 | $ 4.90 |
| 3 | Natural Gas | 0.20 | MCF/hr | $ 2.860 | / MCF | $ 4.63 | $ 13.90 | $ 4,670 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 25,852.65** | **$ 77,557.96** | **$ 26,059,476** | **$ 10.01** | **$ 6.46** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 25,941** | **$ 77,822** | **$ 26,148,229** | **$ 10.04** | **$ 6.49** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8246 | gpm |
| Raffinate Rate: | 8246 | gpm |
| Extraction Feed (O+A) | 16493 | gpm |
| Scrub Feed (O+A) | 227 | gpm |
| Strip Feed (O+A) | 453 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 227 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 271142 | gal |
| Precip Mixing Intensity | 0.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 60.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,493 | gal | $ 95,067 | $ 190,134 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 41 | HP | $ 130,512 | $ 261,023 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 247,394 | gal | $ 435,976 | $ 871,952 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,246 | gpm | $ 201,925 | $ 403,849 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 227 | gal | $ 8,530 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,630 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,401 | gal | $ 39,119 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 453 | gal | $ 12,597 | $ 25,193 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 17,194 | $ 34,388 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,802 | gal | $ 57,768 | $ 115,536 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,547 | $ 19,547 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 59,017 | kg | $ 3,695,013 | $ - | $ 3,695,013 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 1,023,817 | kg | $ 1,108,637 | $ - | $ 1,108,637 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,713 | gal | $ 5,663 | $ 22,653 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 1.36 | HP | $ 19,025 | $ 76,100 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,547 | $ 39,093 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.02 | ft^3 | $ 459 | $ 459 | $ - | |
| 20 | Process Pump | 2 | 100% | 227 | gpm | $ 51,276 | $ 102,552 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,162,479** | **$ 4,803,650** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 82.46 | Effiicency: | 0.85 | | $ 39.53 | $ 118.59 | $ 39,846 | $ 0.02 | $ 0.01 |
| 3 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 412.32 | | | | $ 197.65 | $ 592.95 | $ 199,230 | $ 0.08 | $ 0.05 |
| 5 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.27 | | | | $ 1.09 | $ 3.26 | $ 1,096 | $ 0.00 | $ 0.00 |
| 11 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.45 | | | | $ 0.22 | $ 0.65 | $ 218 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 5.43 | | | | $ 2.60 | $ 7.80 | $ 2,622 | $ 0.00 | $ 0.00 |
| 17 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.90 | | | | $ 0.43 | $ 1.30 | $ 437 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | | | | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.34 | | | | $ 5.43 | $ 16.30 | $ 5,478 | $ 0.00 | $ 0.00 |
| | **TOTAL:** | **515** | | | | **$ 246.95** | **$ 740.85** | **$ 248,927** | **$ 0.10** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 106.76 | kg/hr | $ 0.940 | / kg | $ 802.83 | $ 2,408.49 | $ 809,254 | $ 0.31 | $ 0.20 |
| 2 | D2EHPA | 5.62 | kg/hr | $ 5.000 | / kg | $ 224.76 | $ 674.27 | $ 226,555 | $ 0.09 | $ 0.06 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.87 | mt/hr | $ 270.000 | / mt | $ 1,869.94 | $ 5,609.82 | $ 1,884,898 | $ 0.72 | $ 0.47 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,897.53** | **$ 8,692.58** | **$ 2,920,707** | **$ 1.12** | **$ 0.72** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,144** **$ 9,433** **$ 3,169,634** **$ 1.22** **$ 0.79**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 2157.91 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 8168.58 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 2156.11 | gpm | HCL Dose | 19.38 | g/L of Feed |
| Mn Feed Rate | 8161.78 | lpm | | | |
| Co Precip Mass Flow | 1.53 | tph | | | |
| Mn Precip Mass Flow | 0.84 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 6.84 | g/L of Feed | | | |
| NaOH Dose | 4.53 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 32,369 | gal | $ 39,717 | $ 79,435 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 49 | HP | $ 143,114 | $ 286,227 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 2,158 | gpm | $ 121,095 | $ 121,095 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 7.21 | ft^3 | $ 37,885 | $ 37,885 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 32,342 | gal | $ 39,686 | $ 79,372 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 49 | HP | $ 143,046 | $ 286,093 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 2,156 | gpm | $ 121,056 | $ 121,056 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 4 | ft^3 | $ 24,598 | $ 24,598 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,035,762** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 97.11 | Effiicency: | 0.85 | | $ 46.55 | $ 139.64 | $ 46,920 | $ 0.02 | $ 0.01 |
| 3 | Process Pump | 53.95 | | | | $ 25.86 | $ 77.58 | $ 26,067 | $ 0.01 | $ 0.01 |
| 4 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 1.21 | | | | $ 0.58 | $ 1.74 | $ 585 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 97.03 | | | | $ 46.51 | $ 139.53 | $ 46,881 | $ 0.02 | $ 0.01 |
| 8 | Process Pump | 53.90 | | | | $ 25.84 | $ 77.52 | $ 26,045 | $ 0.01 | $ 0.01 |
| 9 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 0.66 | | | | $ 0.32 | $ 0.96 | $ 321 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **304** | | | | **$ 145.65** | **$ 436.96** | **$ 146,820** | **$ 0.06** | **$ 0.04** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 3.35 | mt/hr | $ 360.000 | / mt | $ 9,654.51 | $ 28,963.53 | $ 9,731,746 | $ 3.74 | $ 2.41 |
| 2 | NaOH | 2.22 | mt/hr | $ 125.000 | / mt | $ 2,216.21 | $ 6,648.63 | $ 2,233,939 | $ 0.86 | $ 0.55 |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 3.84 | $ 11.52 | $ 3,872 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 11,874.56** | **$ 35,623.68** | **$ 11,969,557** | **$ 4.60** | **$ 2.97** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 12,020** **$ 36,061** **$ 12,116,378** **$ 4.65** **$ 3.01**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | Shakespeare Lignite | Date: | 3/18/2023 |
| Case: | SHXP-C4 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | an |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | N/A |
| *(unit conversion)* | 12000 | TPD | Project NPV: | (1,121,631,006) |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 22.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C4 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 95,686,245 | | | | | |
| Constant Dollar Annual OPEX | $ | 502,720,956 | $ | 124.68 | /ton | $ 324.0 | /kg |
| Constant Dollar Annual REE Rev | $ | 352,683,651 | $ | 87.47 | /ton | $ 227.27 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | | |
|---|---|---|---|---|---|
| Payback Period | 25.5 | years | *22.5* | *operating years* | |
| Internal Rate of Return | N/A | | | | |
| Net Present Value | $ | (1,121,631,006) | | | |
| Total Severence Tax Paid | $ | - | | | |
| Total State and Federal Tax Paid | $ | - | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C4 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 1,250 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 2.78 | 34.80 | 15.78 | | | | | | | 15.78 |
| Yttrium | 18.88 | 236.02 | 107.06 | | | | | | | 107.06 |
| Lanthanum | 15.66 | 195.74 | 88.79 | | | | | | | 88.79 |
| Cerium | 30.20 | 377.56 | 171.26 | | | | | | | 171.26 |
| Praseodymium | 3.24 | 40.52 | 18.38 | | | | | | | 18.38 |
| Neodymium | 14.09 | 176.16 | 79.91 | | | | | | | 79.91 |
| Samarium | 2.97 | 37.15 | 16.85 | | | | | | | 16.85 |
| Europium | 0.68 | 8.51 | 3.86 | | | | | | | 3.86 |
| Gadolinium | 3.61 | 45.12 | 20.47 | | | | | | | 20.47 |
| Terbium | 0.49 | 6.15 | 2.79 | | | | | | | 2.79 |
| Dysprosium | 3.03 | 37.90 | 17.19 | | | | | | | 17.19 |
| Holmium | 0.59 | 7.37 | 3.34 | | | | | | | 3.34 |
| Erbium | 1.54 | 19.28 | 8.74 | | | | | | | 8.74 |
| Thulium | 0.34 | 4.23 | 1.92 | | | | | | | 1.92 |
| Ytterbium | 1.60 | 19.96 | 9.06 | | | | | | | 9.06 |
| Lutetium | 0.28 | 3.54 | 1.60 | | | | | | | 1.60 |
| Manganese | -- | 505.39 | 229.24 | | | 0.00 | -- | | | 229.24 |
| Cobalt | -- | 248.28 | 112.62 | | | 0.00 | -- | | | 112.62 |
| Nickel | -- | 759.69 | 344.59 | | | 0.00 | -- | | | 344.59 |
| Zinc | -- | 207.62 | 94.17 | | | 0.00 | -- | | | 94.17 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 1250.00 | 566.99 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 567.0 |
| Total non-REE CM | -- | 1720.98 | 780.62 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 780.6 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 15.78 | 0.00 | 5.89 | 0.01 | 0.00 | 0.00 | 5.90 | 9.88 | 37.4% | |
| Yttrium | 107.06 | 38.92 | 0.00 | 0.84 | 3.63 | 0.00 | 43.40 | 63.66 | 40.5% | |
| Lanthanum | 88.79 | 29.28 | 0.00 | 0.13 | 0.93 | 0.00 | 30.34 | 58.45 | 34.2% | |
| Cerium | 171.26 | 60.61 | 0.00 | 0.19 | 0.51 | 0.00 | 61.31 | 109.94 | 35.8% | |
| Praseodymium | 18.38 | 2.43 | 0.00 | 0.02 | 0.07 | 0.00 | 2.52 | 15.86 | 13.7% | |
| Neodymium | 79.91 | 22.08 | 0.00 | 0.01 | 0.29 | 0.00 | 22.38 | 57.52 | 28.0% | |
| Samarium | 16.85 | 5.30 | 0.00 | 0.02 | 0.08 | 0.00 | 5.40 | 11.46 | 32.0% | |
| Europium | 3.86 | 1.12 | 0.00 | 0.00 | 0.01 | 0.00 | 1.13 | 2.73 | 29.3% | |
| Gadolinium | 20.47 | 5.66 | 0.00 | 0.03 | 0.09 | 0.00 | 5.78 | 14.69 | 28.2% | |
| Terbium | 2.79 | 0.81 | 0.00 | 0.00 | 0.00 | 0.00 | 0.81 | 1.98 | 29.1% | |
| Dysprosium | 17.19 | 5.16 | 0.00 | 0.05 | 0.21 | 0.00 | 5.41 | 11.77 | 31.5% | |
| Holmium | 3.34 | 1.01 | 0.00 | 0.01 | 0.05 | 0.00 | 1.07 | 2.28 | 31.9% | |
| Erbium | 8.74 | 2.82 | 0.00 | 0.05 | 0.25 | 0.00 | 3.12 | 5.62 | 35.7% | |
| Thulium | 1.92 | 0.38 | 0.00 | 0.07 | 0.05 | 0.00 | 0.50 | 1.42 | 26.0% | |
| Ytterbium | 9.06 | 2.34 | 0.00 | 0.13 | 0.44 | 0.00 | 2.91 | 6.15 | 32.1% | |
| Lutetium | 1.60 | 0.37 | 0.00 | 0.02 | 0.07 | 0.00 | 0.45 | 1.15 | 28.3% | |
| Manganese | 229.24 | 0.66 | 0.00 | 21.22 | 84.16 | 0.00 | 106.05 | 123.20 | 46.3% | |
| Cobalt | 112.62 | 0.93 | 0.00 | 0.78 | 50.85 | 0.00 | 52.56 | 60.06 | 46.7% | |
| Nickel | 344.59 | 1.28 | 0.00 | 9.32 | 116.45 | 0.00 | 127.05 | 217.54 | 36.9% | |
| Zinc | 94.17 | 1.09 | 0.00 | 0.15 | 42.91 | 0.00 | 44.15 | 50.02 | 46.9% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 566.99 | 178.30 | 5.89 | 1.57 | 6.68 | 0.00 | 192.43 | 374.56 | 33.9% | <---- |
| Total non-REE CM | 780.62 | 3.97 | 0.00 | 31.46 | 294.38 | 0.00 | 329.81 | 450.82 | 42.2% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 402 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 8 | ton | $ | 1.50 |
| Dust/Volatiles | - | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | | |
|---|---|---|
| 4.28 | TPD REE | |
| 0.141 | TPD Sc | |
| 7.82 | TPD CM | |
| 0.31 | REE | |
| 37.3% | Sc | |
| 9.3% | Mn | |
| 38.1% | Metal-S | |
| 500 | Feed | |
| 1,250 | Grade | |
| 25.5 | PP | |
| N/A | IRR | |
| -1121631006 | NPV | |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C4 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 0.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 10% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 2.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | UK: Design |
| Dissolved Solids in Leach | 19.6% | % | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Leaching Retention Time | 45 | min | UK: Design |
| Thickener UF % Solids | 47% | % | UK: SHXP-C4/SHXP-C4 Samples |
| Residue Settling Rate | 15.0 | ft/hr | Not Available, Assumed |
| **Reagent Use** | | | |
| Acid Consumption | 301.4 | kg/ton | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Bioleaching Sol'n Consumption | 0 | L/ton | N/A |
| Polymer Dosage | 0.016 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 3.00 | -- | VT: Design |
| Thickener Capacity | 8.56 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 7.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | UK: Design |
| Precip Solid Yield | 3.75 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 2.24 | ft/hr | UK: Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 0.5% | % | VT Override from 0.27% \| UK: SHXP-C4/SHXP-C4 Samples |
| **Reagent Use** | | | |
| NaOH Dose | 49.02 | g/L of Feed | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 884.99 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 1250 | ppm | |
| Grade After Sorter | 1250 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 500 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 0 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 500 | tph | |
| Leaching Residue Rate | 402 | tph | |
| Slurry Flow (feed) | 9250 | gpm | |
| Product Flow | 8,246 | gpm | |
| Thickener Feed % Solids (eff) | 16% | % | |
| Liquid Density | 1.018 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 48.92 | g/L | |
| Leaching Vessel Size | 52031.25 | gal | |
| Leaching Vessel Size | 196.96 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 134.00 | tph | 70 to 200 tph |
| Thickener Area | 1720.72 | ft^2 | |
| Thickener Diameter | 46.82 | ft | ideal < 100' |
| Liquid Balance | 244.65 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,246 | gpm | |
| Slurry Feed Rate | 31,216 | lpm | |
| Precip Mass Flow | 7.75 | tph | |
| Thickener Feed % Solids (eff) | 0.37% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.015 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 200.68 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 36.51 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 9.10 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 980 | ft^2 | |
| Thickener Diameter | 35.33 | ft | ideal < 100' |

*PM5: REE Precipitation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |

**Process Parameters**

| | | | |
|---|---|---|---|
| Precip Conditioning Time | 45 | min | UK: Based on Design |
| Precip Solid Yield | 4.23 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 0.044 | ft/hr | UK: Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.43% | % | VT Override from 0.23% \| UK: SHXP-C4/SHXP-C4 Samples |

**Reagent Use**

| | | | |
|---|---|---|---|
| NaOH Dose | 2.82 | g/L of Feed | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |

**Equipment Specifications**

| | | | |
|---|---|---|---|
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 5198.38 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Slurry Feed Rate | 8237 | gpm | |
| Slurry Feed Rate | 31,182 | lpm | |
| Precip Mass Flow | 8.73 | tph | |
| Thickener Feed % Solids (eff) | 0.4% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.012 | g/cc | |

**Other Indicators**

| | | | |
|---|---|---|---|
| Conditioning Vessel Size | 233.86 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 41.15 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 10.25 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 7564 | ft^2 | |
| Thickener Diameter | 98.16 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | 57.27 | m/m | target = 6.35 |

## PM6: RE-Redissolution

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 1.6% | % | VT override from 10.7% \|\| UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Dissolved Solids in Re-Leach | 25% | % | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Leaching Retention Time | 120 | min | UK: Design |
| **Reagent Use** | | | |
| Acid Consumption | 974.6 | kg/ton | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 8.73 | tph | |
| Slurry Flow (feed) | 2,158 | gpm | |
| Product Flow | 2,158 | gpm | |
| Liquid Density | 1.011 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 245.1 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 3.82 | v/v | target = 3.64, based on data |
| Liquid Balance | 10.26 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 16.2 | g/L | |

## PM7: REE Oxalate Precipitation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | UK: Design |
| Precip Solid Yield | 0.3 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Thickener UF %Solids | | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 12.71 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Oxalic Acid Dose | 8.34 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

**Material Balance**

| | | | |
|---|---|---|---|
| Slurry Feed Rate | 2158.07 | gpm | |
| Slurry Feed Rate | 8169.18 | lpm | |
| Precip Mass Flow | 0.14 | tph | |
| Thickener Feed % Solids (eff) | 0.02% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.08 | tph | |
| Liquid Density | 1.011 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 40.85 | m^3 | |
| Filter Press Area | 1.28 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.16 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 6.32 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.71 | m/m | |

## PM8: Sc SX Separation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | UK: Design |
| Settler Residence Time | 15.00 | min | UK: Design |
| Loading: Number of Stages | 2.00 | - | UK: Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Loading O:A Advance Ratio | 0.03 | v/v | UK: Design |
| Scrubbing: Number of Stages | | - | N/A |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | N/A |
| Scrubbing O:A Advance Ratio | | v/v | N/A |
| Stripping: Number of Stages | 2.00 | - | UK: Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | UK: Design |
| Sc2O3 Grade | 2% | % | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Precip Solid Yield | 0.4 | g/L | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance (SX)**

| | | |
|---|---|---|
| PLS Flow | 8246.5 | gpm |
| Raffinate Rate: | 8246.5 | gpm |
| Load Org Recycle | 8019.7 | gpm |
| Load Org Feed | 8246.5 | gpm |
| Loag Org Advance | 226.7 | gpm |
| Scrub Aq Product | 0.0 | gpm |
| Scrub Aq Recycle | 0.0 | gpm |
| Scrub Aq Feed | 0.0 | gpm |
| Scurb Org Product | 226.7 | gpm |
| Strip Aq Product | 18.1 | gpm |
| Strip Aq Recycle | 208.6 | gpm |
| Strip Aq Feed | 226.7 | gpm |
| Strip Org Product | 226.7 | gpm |
| Extraction Feed (O+A): | 16492.9 | gpm |
| Scrub Feed (O+A): | 226.7 | gpm |
| Strip Feed (O+A): | 453.4 | gpm |

**Material Balance (Precip)**

| | | |
|---|---|---|
| Strip Solution Flow | 18.1 | gpm |
| Solid Density | 3.4 | g/cc |
| Solid Product Rate | 0.00 | tph |
| Liquid Density (after precip) | 1.13 | g/cc |
| Thickener Feed % Solids (eff) | 0.03% | % |

**Total Volumes**

| | | |
|---|---|---|
| Total Extraction Volume | 527773 | gal |
| Total Scrubbing Volume | 0 | gal |
| Total Stripping Volume | 14510 | gal |
| Extraction Org Volume | 263886.57 | gal |
| Scrubbing Org Volume | 0.00 | gal |
| Stripping Org Volume | 7255.19 | gal |
| Total Org Volume | 271141.75 | gal |

**Other Indicators**

| | | | |
|---|---|---|---|
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 10.27 | m^3 | |
| Filter Press Area | 0.02 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

## PM9: CM Precipitation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 0 | min | N/A |
| Precip Solid Yield | - | g/L | N/A |
| Precip Settlng Rate | 10 | ft/hr | N/A |
| Thickener UF % Solids | 10% | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 0.00 | g/L of Feed | N/A |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 0.00 | min | N/A |
| Thickener Capacity | 0.00 | ft^2/tph | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Slurry Feed Rate | 8227 | gpm | |
| Slurry Feed Rate | 31,143 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.012 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

## PM10: CM Redissolution

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 10.0% | % | N/A |
| Dissolved Solids in Re-Leach | 10% | % | N/A |
| Leaching Retention Time | 10 | min | N/A |
| **Reagent Use** | | | |
| Acid Consumption | 0 | kg/ton | N/A |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 0.00 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 0 | gpm | |
| Liquid Density | 1.000 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | |
| Releach Conc. Factor | #DIV/0! | v/v | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | - | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

## PM11: CM Recovery

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 2.83 | g/L | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | | % | Not Applicable |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 1.56 | g/L | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Na2S Dose | 6.84 | g/L of Feed | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| NaOH Dose | 4.53 | g/L of Feed | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| HCL Dose | 19.38 | g/L of Feed | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 2157.91 | gpm | |
| Co Slurry Feed Rate | 8168.58 | lpm | |
| Co Precip Mass Flow | 1.53 | tph | |
| Co Thickener Feed % Solids (eff) | 0.28% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 2156.11 | gpm | |
| Mn Slurry Feed Rate | 8161.78 | lpm | |
| Mn Precip Mass Flow | 0.84 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.15% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.01 | g/cc | |
| Mn Liquid Density | 1.01 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 122.5 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 7.21 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 1.80 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 122.4 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 4.0 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.99 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

## PM12: Biooxidation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Require | 500 | gpm | |
| BioLeach Retention Time 1 | 20 | hr | 2.5 |
| BioLeach Retention Time 2 | 20 | hr | 60 |
| BioLeach Retention Time 3 | 20 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 3.5 | kJ/kg-K | |
| Temperature Drop | 10 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 500.0 | gpm | |
| Raw Leach Solution Req'd | 1892.7 | lpm | |
| Yearly Acid Production | 915,766,819 | liters solution/yr | |
| Yearly Acid Production | 1,098,920 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C4 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 7,234,587 | $ - | $ 1,808,647 | $ 7,958,046 | $ 9,907,767 | $ 26,909,047 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 3,448,822 | $ - | $ 862,206 | $ 3,793,705 | $ 4,723,162 | $ 12,827,895 | IIIIIIIIIIIII |
| PM5 | REE Precipitation | $ 5,795,457 | $ - | $ 1,448,864 | $ 6,375,003 | $ 7,936,879 | $ 21,556,204 | IIIIIIIIIIIIIIIIIIII |
| PM6 | RE-Redissolution | $ 1,274,195 | $ - | $ 318,549 | $ 1,401,615 | $ 1,745,010 | $ 4,739,369 | IIIII |
| PM7 | REE Oxalate Precipitation | $ 1,019,128 | $ - | $ 254,782 | $ 1,121,040 | $ 1,395,695 | $ 3,790,645 | IIII |
| PM8 | Sc SX Separation | $ 2,162,479 | $ 4,803,650 | $ 540,620 | $ 2,378,727 | $ 2,961,514 | $ 12,846,989 | IIIIIIIIIIIII |
| PM9 | CM Precipitation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM10 | CM Redissolution | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM11 | CM Recovery | $ 1,035,762 | $ - | $ 258,940 | $ 1,139,338 | $ 1,418,476 | $ 3,852,515 | IIII |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| | TOTAL | $ 24,434,089 | $ 4,803,650 | $ 6,108,522 | $ 26,877,498 | $ 33,462,486 | $ 95,686,245 | |

## Module Summary

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| Total Lang Factor (=1+sum % Base) | | 372% | |

Year of Construction: 2025

# OPERATING COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C4 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 1.98 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 0.76 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 95,686,245 | $ Total | $ 949.27 | $ 2,847.80 | $ 956,862 | $ 0.62 | $ 0.24 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 0.20 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 352,683,651 | $ Total | $ 1,749.42 | $ 5,248.27 | $ 1,763,418 | $ 1.14 | $ 0.44 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 0.30 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 95,686,245 | $ Total | $ 949.27 | $ 2,847.80 | $ 956,862 | $ 0.62 | $ 0.24 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 8,629.41** | **$ 25,888.23** | **$ 8,698,445** | **$ 5.61** | **$ 2.16** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 0.33 | $ 0.13 |
| PM2 | Roasting | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | PR | 1296 | | $ 621.02 | $ 1,863.07 | $ 625,992 | $ 0.40 | $ 0.16 |
| PM4 | Al Precipitation | PR | 815 | | $ 390.67 | $ 1,172.02 | $ 393,800 | $ 0.25 | $ 0.10 |
| PM5 | REE Precipitation | PR | 839 | | $ 402.40 | $ 1,207.19 | $ 405,616 | $ 0.26 | $ 0.10 |
| PM6 | RE-Redissolution | PR | 442 | | $ 212.07 | $ 636.20 | $ 213,765 | $ 0.14 | $ 0.05 |
| PM7 | REE Oxalate Precipitation | PR | 184 | | $ 88.05 | $ 264.15 | $ 88,753 | $ 0.06 | $ 0.02 |
| **Sub-Total** | | | **5460.5** | | **$ 2,617.50** | **$ 7,852.49** | **$ 2,638,436.62** | **$ 1.70** | **$ 0.65** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.07 | $ 0.03 |
| PM2 | Roasting | CR | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | CR | | $ 265,833.78 | $ 797,501.34 | $ 267,960,451 | $ 172.68 | $ 66.46 |
| PM4 | Al Precipitation | CR | | $ 91,803.85 | $ 275,411.55 | $ 92,538,281 | $ 59.63 | $ 22.95 |
| PM5 | REE Precipitation | CR | | $ 5,274.34 | $ 15,823.02 | $ 5,316,534 | $ 3.43 | $ 1.32 |
| PM6 | RE-Redissolution | CR | | $ 18,380.20 | $ 55,140.60 | $ 18,527,242 | $ 11.94 | $ 4.60 |
| PM7 | REE Oxalate Precipitation | CR | | $ 25,852.65 | $ 77,557.96 | $ 26,059,476 | $ 16.79 | $ 6.46 |
| **Sub-Total** | | | | **$ 422,017.71** | **$ 1,266,053.14** | **$ 425,393,855.85** | **$ 274.13** | **$ 105.50** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 9,648 | $ 0.10 | $ 321.60 | $ 964.80 | $ 324,173 | $ 0.21 | $ 0.08 |
| Non-Hazardous Precipitate | WD | Tons | 186 | $ 1.50 | $ 93.00 | $ 279.01 | $ 93,748 | $ 0.06 | $ 0.02 |
| Dust/Volatiles | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 414.60** | **$ 1,243.81** | **$ 417,920.54** | **$ 0.27** | **$ 0.10** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 425,050** | **$ 1,275,149** | **$ 428,450,213** | **$ 276** | **$ 106** |

## Overhead Costs

| Allocation | Code | Rate | Basis | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% | % of Direct Cost | $ 25,888 Daily Direct Cost | $ 1,294.41 | $ 3,883.23 | $ 1,304,767 | $ 0.84 | $ 0.32 |
| Variable Portion | OH | 15% | % of Direct Cost | $ 1,275,149 Daily Direct Cost | $ 63,757.47 | $ 191,272.42 | $ 64,267,532 | $ 41.42 | $ 15.94 |
| **TOTAL OVERHEAD COSTS** | | | | | **$ 65,051.88** | **$ 195,155.65** | **$ 65,572,298.67** | **$ 42.26** | **$ 16.26** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | SHXP-C4 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 8,629.41 | $ 25,888.23 | $ 8,698,445 | $ 6 | $ 2.16 | I |
| Power | $ 2,617.50 | $ 7,852.49 | $ 2,638,437 | $ 2 | $ 0.65 | |
| Consumables | $ 422,017.71 | $ 1,266,053.14 | $ 425,393,856 | $ 274 | $ 105.50 | IIII

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C4 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 352,683,651

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 1215 | 1250 |
| Contained Value | ($/ton) | $ 31.2 | $ 234.6 |
| Feed Basket Price | ($/kg) | $ 28.3 | $ 206.9 |
| Product Basket Price | ($/kg) | $ 24.6 | $ 221.4 |
| Production Rate | (kg/day) | 4476.79 | 4618.43 |
| | (t/day) | 4.48 | 4.62 |
| | (t/yr) | 1504.20 | 1551.79 |
| | (% of US Demand) | 15.0% | |

## Revenue Contribution Table

| | | Product Revenue ($/hr) | | | | | | |
|---|---|---|---|---|---|---|---|---|
| **Element** | **REE/CM Price ($/kg) [1]** | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | **REE Revenue ($/day)** | |
| Scandium | $ 4,200.00 | $ 2.1 | $ 37,966.8 | $ - | $ - | $ - | $ 911,253 | IIII

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C4 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 385,386,948 | $ 396,948,556 | $ 408,857,013 | $ 421,122,723 | $ 433,756,405 | $ 446,769,097 | $ 460,172,170 | $ 473,977,335 | $ 488,196,655 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 385,386,948 | $ 396,948,556 | $ 408,857,013 | $ 421,122,723 | $ 433,756,405 | $ 446,769,097 | $ 460,172,170 | $ 473,977,335 | $ 488,196,655 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 287,059 | $ 2,049,637 | $ 552,152,236 | $ 568,394,246 | $ 585,116,373 | $ 602,332,593 | $ 620,057,274 | $ 638,305,188 | $ 657,091,522 | $ 676,431,887 | $ 696,342,331 |
| *- Operating Costs* | $ - | $ - | $ - | $ 549,336,763 | $ 565,816,865 | $ 582,791,371 | $ 600,275,113 | $ 618,283,366 | $ 636,831,867 | $ 655,936,823 | $ 675,614,928 | $ 695,883,375 |
| *- Loan #1 Interest* | $ - | $ 287,059 | $ 265,280 | $ 242,195 | $ 217,725 | $ 191,786 | $ 164,291 | $ 135,146 | $ 104,253 | $ 71,506 | $ 36,794 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 1,784,357 | $ 1,648,982 | $ 1,505,484 | $ 1,353,376 | $ 1,192,141 | $ 1,021,233 | $ 840,070 | $ 648,037 | $ 444,482 | $ 228,714 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 924,297 | $ 854,172 | $ 779,840 | $ 701,049 | $ 617,529 | $ 528,999 | $ 435,156 | $ 335,683 | $ 230,242 |
| Gross Profit | $ - | $ (287,059) | $ (2,049,637) | $ (166,765,288) | $ (171,445,690) | $ (176,259,360) | $ (181,209,870) | $ (186,300,869) | $ (191,536,091) | $ (196,919,352) | $ (202,454,552) | $ (208,145,676) |
| **Book Deductions** | $ - | $ 358,823 | $ 2,589,581 | $ 3,748,810 | $ 3,467,649 | $ 3,207,575 | $ 2,967,007 | $ 2,744,482 | $ 2,538,646 | $ 2,348,247 | $ 2,172,129 | $ 2,009,219 |
| *- CAPEX #1 Depreciation* | $ - | $ 358,823 | $ 331,912 | $ 307,018 | $ 283,992 | $ 262,693 | $ 242,991 | $ 224,766 | $ 207,909 | $ 192,316 | $ 177,892 | $ 164,550 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 2,257,670 | $ 2,088,344 | $ 1,931,719 | $ 1,786,840 | $ 1,652,827 | $ 1,528,865 | $ 1,414,200 | $ 1,308,135 | $ 1,210,025 | $ 1,119,273 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,353,447 | $ 1,251,939 | $ 1,158,043 | $ 1,071,190 | $ 990,851 | $ 916,537 | $ 847,797 | $ 784,212 | $ 725,396 |
| Taxable Before Depletion | $ - | $ (645,882) | $ (4,639,219) | $ (170,514,098) | $ (174,913,339) | $ (179,466,936) | $ (184,176,877) | $ (189,045,351) | $ (194,074,736) | $ (199,267,599) | $ (204,626,681) | $ (210,154,895) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ - | $ (322,941) | $ (2,319,609) | $ (85,257,049) | $ (87,456,669) | $ (89,733,468) | $ (92,088,439) | $ (94,522,675) | $ (97,037,368) | $ (99,633,799) | $ (102,313,340) | $ (105,077,447) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 53,954,173 | $ 55,572,798 | $ 57,239,982 | $ 58,957,181 | $ 60,725,897 | $ 62,547,674 | $ 64,424,104 | $ 66,356,827 | $ 68,347,532 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (645,882) | $ (4,639,219) | $ (170,514,098) | $ (174,913,339) | $ (179,466,936) | $ (184,176,877) | $ (189,045,351) | $ (194,074,736) | $ (199,267,599) | $ (204,626,681) | $ (210,154,895) |
| *- Income Tax* | $ - | $ (167,929) | $ (1,206,197) | $ (44,333,665) | $ (45,477,468) | $ (46,661,403) | $ (47,885,988) | $ (49,151,791) | $ (50,459,431) | $ (51,809,576) | $ (53,202,937) | $ (54,640,273) |
| Net Profit/Loss | $ - | $ (477,953) | $ (3,433,022) | $ (126,180,432) | $ (129,435,871) | $ (132,805,532) | $ (136,290,889) | $ (139,893,559) | $ (143,615,305) | $ (147,458,023) | $ (151,423,744) | $ (155,514,622) |
| **Book Deductions** | $ - | $ 358,823 | $ 2,589,581 | $ 3,748,810 | $ 3,467,649 | $ 3,207,575 | $ 2,967,007 | $ 2,744,482 | $ 2,538,646 | $ 2,348,247 | $ 2,172,129 | $ 2,009,219 |
| *+ CAPEX #1 Depreciation* | $ - | $ 358,823 | $ 331,912 | $ 307,018 | $ 283,992 | $ 262,693 | $ 242,991 | $ 224,766 | $ 207,909 | $ 192,316 | $ 177,892 | $ 164,550 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 2,257,670 | $ 2,088,344 | $ 1,931,719 | $ 1,786,840 | $ 1,652,827 | $ 1,528,865 | $ 1,414,200 | $ 1,308,135 | $ 1,210,025 | $ 1,119,273 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,353,447 | $ 1,251,939 | $ 1,158,043 | $ 1,071,190 | $ 990,851 | $ 916,537 | $ 847,797 | $ 784,212 | $ 725,396 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ - | $ (119,129) | $ (843,440) | $ (122,431,623) | $ (125,968,221) | $ (129,597,957) | $ (133,323,882) | $ (137,149,078) | $ (141,076,659) | $ (145,109,776) | $ (149,251,615) | $ (153,505,403) |
| **Captial Cost** | $ 4,784,312 | $ 30,102,261 | $ 18,045,963 | $ 3,968,216 | $ 4,206,309 | $ 4,458,688 | $ 4,726,209 | $ 5,009,782 | $ 5,310,369 | $ 5,628,991 | $ 5,966,730 | $ 5,674,699 |
| *- Fixed Capital Costs* | $ 9,568,625 | $ 59,478,570 | $ 30,809,899 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 4,784,312 | $ 29,739,285 | $ 15,404,950 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 362,976 | $ 384,755 | $ 407,840 | $ 432,310 | $ 458,249 | $ 485,744 | $ 514,888 | $ 545,782 | $ 578,529 | $ 613,240 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 2,256,259 | $ 2,391,634 | $ 2,535,132 | $ 2,687,240 | $ 2,848,475 | $ 3,019,383 | $ 3,200,546 | $ 3,392,579 | $ 3,596,134 | $ 3,811,902 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 1,168,742 | $ 1,238,867 | $ 1,313,199 | $ 1,391,991 | $ 1,475,510 | $ 1,564,041 | $ 1,657,883 | $ 1,757,356 | $ 1,862,797 |
| **Net Cash Flow** | $ (4,784,312) | $ (30,221,390) | $ (18,889,404) | $ (126,399,839) | $ (130,174,531) | $ (134,056,645) | $ (138,050,091) | $ (142,158,859) | $ (146,387,028) | $ (150,738,767) | $ (155,218,345) | $ (159,180,102) |
| Discounted Cash Flow | $ (4,784,312) | $ (27,473,991) | $ (15,611,077) | $ (94,966,070) | $ (88,910,956) | $ (83,238,629) | $ (77,925,677) | $ (72,949,973) | $ (68,290,629) | $ (63,927,952) | $ (59,843,391) | $ (55,791,655) |
| **Cumulative DCF** | $ (4,784,312) | $ (32,258,304) | $ (47,869,381) | $ (142,835,451) | $ (231,746,407) | $ (314,985,036) | $ (392,910,713) | $ (465,860,686) | $ (534,151,315) | $ (598,079,267) | $ (657,922,658) | $ (713,714,313) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 95,686,245 | Payback Period | 25.5 |
| Constant Dollar OPEX | $ 502,720,956 | Internal Rate of Return | N/A |
| Constant Dollar REE Rev | $ 352,683,651 | Net Present Value | $ (1,121,631,006) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ - |

## PRO FORMA CASH FLOW

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C4 | Years: | 13 through 23 | By: | ks |

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 502,842,555** | **$ 517,927,832** | **$ 533,465,667** | **$ 549,469,637** | **$ 565,953,726** | **$ 582,932,338** | **$ 600,420,308** | **$ 618,432,917** | **$ 636,985,904** | **$ 656,095,482** | **$ 675,778,346** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 502,842,555 | $ 517,927,832 | $ 533,465,667 | $ 549,469,637 | $ 565,953,726 | $ 582,932,338 | $ 600,420,308 | $ 618,432,917 | $ 636,985,904 | $ 656,095,482 | $ 675,778,346 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 716,878,351** | **$ 738,262,673** | **$ 760,410,553** | **$ 783,222,870** | **$ 806,719,556** | **$ 830,921,143** | **$ 855,848,777** | **$ 881,524,240** | **$ 907,969,967** | **$ 935,209,066** | **$ 963,265,338** |
| *- Operating Costs* | $ 716,759,877 | $ 738,262,673 | $ 760,410,553 | $ 783,222,870 | $ 806,719,556 | $ 830,921,143 | $ 855,848,777 | $ 881,524,240 | $ 907,969,967 | $ 935,209,066 | $ 963,265,338 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 118,474 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ (214,035,795) | $ (220,334,841) | $ (226,944,886) | $ (233,753,233) | $ (240,765,830) | $ (247,988,805) | $ (255,428,469) | $ (263,091,323) | $ (270,984,063) | $ (279,113,585) | $ (287,486,992) |
| **Book Deductions** | **$ 1,858,528** | **$ 1,719,138** | **$ 1,590,203** | **$ 1,470,937** | **$ 1,360,617** | **$ 1,258,571** | **$ 1,164,178** | **$ 1,076,865** | **$ 996,100** | **$ 845,933** | **$ 307,705** |
| *- CAPEX #1 Depreciation* | $ 152,209 | $ 140,793 | $ 130,234 | $ 120,466 | $ 111,431 | $ 103,074 | $ 95,343 | $ 88,193 | $ 81,578 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 1,035,327 | $ 957,678 | $ 885,852 | $ 819,413 | $ 757,957 | $ 701,110 | $ 648,527 | $ 599,888 | $ 554,896 | $ 513,279 | $ - |
| *- CAPEX #3 Depreciation* | $ 670,991 | $ 620,667 | $ 574,117 | $ 531,058 | $ 491,229 | $ 454,387 | $ 420,308 | $ 388,785 | $ 359,626 | $ 332,654 | $ 307,705 |
| Taxable Before Depletion | $ (215,894,323) | $ (222,053,979) | $ (228,535,089) | $ (235,224,171) | $ (242,126,447) | $ (249,247,376) | $ (256,592,647) | $ (264,168,188) | $ (271,980,163) | $ (279,959,517) | $ (287,794,697) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ (107,947,162) | $ (111,026,990) | $ (114,267,545) | $ (117,612,085) | $ (121,063,224) | $ (124,623,688) | $ (128,296,324) | $ (132,084,094) | $ (135,990,081) | $ (139,979,759) | $ (143,897,349) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 70,397,958 | $ 72,509,896 | $ 74,685,193 | $ 76,925,749 | $ 79,233,522 | $ 81,610,527 | $ 84,058,843 | $ 86,580,608 | $ 89,178,027 | $ 91,853,367 | $ 94,608,968 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ (215,894,323)** | **$ (222,053,979)** | **$ (228,535,089)** | **$ (235,224,171)** | **$ (242,126,447)** | **$ (249,247,376)** | **$ (256,592,647)** | **$ (264,168,188)** | **$ (271,980,163)** | **$ (279,959,517)** | **$ (287,794,697)** |
| *- Income Tax* | $ (56,132,524) | $ (57,734,035) | $ (59,419,123) | $ (61,158,284) | $ (62,952,876) | $ (64,804,318) | $ (66,714,088) | $ (68,683,729) | $ (70,714,842) | $ (72,789,475) | $ (74,826,621) |
| Net Profit/Loss | $ (159,761,799) | $ (164,319,945) | $ (169,115,966) | $ (174,065,886) | $ (179,173,571) | $ (184,443,058) | $ (189,878,559) | $ (195,484,459) | $ (201,265,320) | $ (207,170,043) | $ (212,968,076) |
| **Book Deductions** | **$ 1,858,528** | **$ 1,719,138** | **$ 1,590,203** | **$ 1,470,937** | **$ 1,360,617** | **$ 1,258,571** | **$ 1,164,178** | **$ 1,076,865** | **$ 996,100** | **$ 845,933** | **$ 307,705** |
| *+ CAPEX #1 Depreciation* | $ 152,209 | $ 140,793 | $ 130,234 | $ 120,466 | $ 111,431 | $ 103,074 | $ 95,343 | $ 88,193 | $ 81,578 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 1,035,327 | $ 957,678 | $ 885,852 | $ 819,413 | $ 757,957 | $ 701,110 | $ 648,527 | $ 599,888 | $ 554,896 | $ 513,279 | $ - |
| *+ CAPEX #3 Depreciation* | $ 670,991 | $ 620,667 | $ 574,117 | $ 531,058 | $ 491,229 | $ 454,387 | $ 420,308 | $ 388,785 | $ 359,626 | $ 332,654 | $ 307,705 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ (157,903,271) | $ (162,600,807) | $ (167,525,763) | $ (172,594,949) | $ (177,812,954) | $ (183,184,487) | $ (188,714,381) | $ (194,407,594) | $ (200,269,221) | $ (206,324,110) | $ (212,660,371) |
| **Captial Cost** | **$ 1,974,565** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 1,974,565 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ (159,877,837) | $ (162,600,807) | $ (167,525,763) | $ (172,594,949) | $ (177,812,954) | $ (183,184,487) | $ (188,714,381) | $ (194,407,594) | $ (200,269,221) | $ (206,324,110) | $ (212,660,371) |
| Discounted Cash Flow | $ (50,942,006) | $ (47,099,662) | $ (44,114,769) | $ (41,317,858) | $ (38,697,279) | $ (36,242,074) | $ (33,941,940) | $ (31,787,195) | $ (29,768,744) | $ (27,880,695) | $ (26,124,470) |
| **Cumulative DCF** | $ (764,656,319) | $ (811,755,980) | $ (855,870,750) | $ (897,188,608) | $ (935,885,888) | $ (972,127,962) | $ (1,006,069,902) | $ (1,037,857,097) | $ (1,067,625,841) | $ (1,095,506,536) | $ (1,121,631,006) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 95,686,245 | Payback Period | 0 | 25.5 |
| Constant Dollar OPEX | $ 502,720,956 | Internal Rate of Return | 0 | N/A |
| Constant Dollar REE Rev | $ 352,683,651 | Net Present Value | 0 | $ (1,121,631,006) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ - |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | $ 287.61 | $ 862.84 | $ 289,913 | $ 0.19 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 0.12 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.01 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 7 | Conveyor | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 0.33** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.07 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.07** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 611** **$ 1,834** **$ 616,370** **$ 0.40** **$ 0.15**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 2.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 500 | tph |
| Residue Rate | 402 | tph |
| Slurry Flow | 9250 | gpm |
| Leaching Time | 45 | min |
| Acid Consumption | 301.38 | kg/ton |
| Bioleaching Sol'n Consumption | 0 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 3 | -- |
| Thickener Capacity | 8.56 | ft^2/tph |
| Polymer Dose | 0.016 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 8 | 100% | 52,031 | gal | $ 181,404 | $ 1,451,235 | $ - | |
| 2 | Tank Mixer | 8 | 100% | 78 | HP | $ 187,044 | $ 1,496,351 | $ - | |
| 3 | Process Pump | 4 | 100% | 4,625 | gpm | $ 161,957 | $ 647,827 | $ - | |
| 4 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 5 | Belt Filter | 3 | 100% | 134 | TPH | $ 1,060,160 | $ 3,180,481 | $ - | |
| 6 | Solution Heater | 1 | 100% | 1 | MBTU/hr | $ 30,807 | $ 30,807 | $ - | |
| 7 | Thickener | 2 | 100% | 1,721 | ft^2 | $ 166,165 | $ 332,330 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 7,234,587** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 624.38 | Effiicency: | 0.85 | | $ 299.30 | $ 897.89 | $ 301,691 | $ 0.19 | $ 0.07 |
| 3 | Process Pump | 462.50 | | | | $ 221.70 | $ 665.10 | $ 223,475 | $ 0.14 | $ 0.06 |
| 4 | Conveyor | 31.00 | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 5 | Belt Filter | 97.49 | | | | $ 46.73 | $ 140.19 | $ 47,104 | $ 0.03 | $ 0.01 |
| 6 | Solution Heater | 78.00 | | | | $ 37.39 | $ 112.16 | $ 37,687 | $ 0.02 | $ 0.01 |
| 7 | Thickener | 2.19 | | | | $ 1.05 | $ 3.15 | $ 1,058 | $ 0.00 | $ 0.00 |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1296** | | | | **$ 621.02** | **$ 1,863.07** | **$ 625,992** | **$ 0.40** | **$ 0.16** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | - | L/hr | $ 0.008 | / L | $ - | $ - | $ - | $ - | $ - |
| 2 | H2SO4 | 150.69 | mt/hr | $ 220.000 | / mt | $ 265,211.51 | $ 795,634.54 | $ 267,333,204 | $ 172.27 | $ 66.30 |
| 3 | Natural Gas | 0.75 | MCF/hr | $ 2.860 | / MCF | $ 17.21 | $ 51.63 | $ 17,346 | $ 0.01 | $ 0.00 |
| 4 | Polymer | 33.61 | kg/hr | $ 2.250 | / kg | $ 605.06 | $ 1,815.18 | $ 609,901 | $ 0.39 | $ 0.15 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 265,833.78** | **$ 797,501.34** | **$ 267,960,451** | **$ 172.68** | **$ 66.46** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 266,455** | **$ 799,364** | **$ 268,586,443** | **$ 173.08** | **$ 66.61** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 7.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8246 | gpm |
| Feed Rate | 31216 | lpm |
| Precip Mass Flow | 7.75 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 884.99 | ft^2/tpd |
| NaOH Dose | 49.02 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 7 | 100% | 53,013 | gal | $ 122,214 | $ 855,500 | $ - | |
| 2 | Tank Mixer | 7 | 100% | 27 | HP | $ 101,724 | $ 712,071 | $ - | |
| 3 | Process Pump | 21 | 100% | 1,178 | gpm | $ 48,066 | $ 1,009,389 | $ - | |
| 4 | Filter Press | 1 | 100% | 37 | ft^3 | $ 121,821 | $ 121,821 | $ - | |
| 5 | Conveyor | 1 | 100% | 8 | TPH | $ 9,696 | $ 9,696 | $ - | |
| 6 | Thickener | 7 | 100% | 980 | ft^2 | $ 105,764 | $ 740,345 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 3,448,822** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 185.55 | Effiicency: | 0.85 | | | $ 88.94 | $ 266.83 | $ 89,653 | $ 0.06 | $ 0.02 |
| 3 | Process Pump | 618.48 | | | | | $ 296.47 | $ 889.42 | $ 298,844 | $ 0.19 | $ 0.07 |
| 4 | Filter Press | 6.13 | | | | | $ 2.94 | $ 8.82 | $ 2,962 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 0.48 | | | | | $ 0.23 | $ 0.69 | $ 232 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 4.36 | | | | | $ 2.09 | $ 6.27 | $ 2,108 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **815** | | | | | **$ 390.67** | **$ 1,172.02** | **$ 393,800** | **$ 0.25** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 91.80 | mt/hr | $ 125.000 | / mt | $ 91,803.85 | $ 275,411.55 | $ 92,538,281 | $ 59.63 | $ 22.95 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 91,803.85** | **$ 275,411.55** | **$ 92,538,281** | **$ 59.63** | **$ 22.95** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 92,195** | **$ 276,584** | **$ 92,932,080** | **$ 59.89** | **$ 23.05** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 6.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8237 | gpm |
| Feed Rate | 31182 | lpm |
| Precip Mass Flow | 8.73 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 5198.38 | ft^2/tpd |
| NaOH Dose | 2.82 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 6 | 100% | 61,780 | gal | $ 133,200 | $ 799,198 | $ - | |
| 2 | Tank Mixer | 6 | 100% | 31 | HP | $ 110,895 | $ 665,371 | $ - | |
| 3 | Process Pump | 18 | 100% | 1,373 | gpm | $ 50,955 | $ 917,196 | $ - | |
| 4 | Filter Press | 1 | 100% | 41 | ft^3 | $ 132,775 | $ 132,775 | $ - | |
| 5 | Conveyor | 1 | 100% | 9 | TPH | $ 10,351 | $ 10,351 | $ - | |
| 6 | Thickener | 6 | 100% | 7,564 | ft^2 | $ 545,094 | $ 3,270,566 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 5,795,457** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 185.34 | Effiicency: | 0.85 | | | $ 88.84 | $ 266.53 | $ 89,554 | $ 0.06 | $ 0.02 |
| 3 | Process Pump | 617.80 | | | | | $ 296.15 | $ 888.44 | $ 298,514 | $ 0.19 | $ 0.07 |
| 4 | Filter Press | 6.91 | | | | | $ 3.31 | $ 9.94 | $ 3,339 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 0.54 | | | | | $ 0.26 | $ 0.78 | $ 262 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 28.87 | | | | | $ 13.84 | $ 41.51 | $ 13,948 | $ 0.01 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **839** | | | | | **$ 402.40** | **$ 1,207.19** | **$ 405,616** | **$ 0.26** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 5.27 | mt/hr | $ 125.000 | / mt | $ 5,274.34 | $ 15,823.02 | $ 5,316,534 | $ 3.43 | $ 1.32 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 5,274.34** | **$ 15,823.02** | **$ 5,316,534** | **$ 3.43** | **$ 1.32** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 5,677** | **$ 17,030** | **$ 5,722,151** | **$ 3.69** | **$ 1.42** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4

Date: 3/18/2023
By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 8.73 | tph |
| Slurry Feed Rate | 2158 | gpm |
| Leaching Time | 120 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 974.63 | kg/ton |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 64,742 | gal | $ 76,692 | $ 306,767 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 97 | HP | $ 211,583 | $ 846,330 | $ - | |
| 3 | Process Pump | 1 | 100% | 2,158 | gpm | $ 121,098 | $ 121,098 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,274,195** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 388.45 | Effiicency: | 0.85 | | | $ 186.21 | $ 558.62 | $ 187,696 | $ 0.12 | $ 0.05 |
| 3 | Process Pump | 53.95 | | | | | $ 25.86 | $ 77.59 | $ 26,069 | $ 0.02 | $ 0.01 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **442** | | | | | **$ 212.07** | **$ 636.20** | **$ 213,765** | **$ 0.14** | **$ 0.05** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 8.51 | mt/hr | $ 270.000 | / mt | $ 18,380.20 | $ 55,140.60 | $ 18,527,242 | $ 11.94 | $ 4.60 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 18,380.20** | **$ 55,140.60** | **$ 18,527,242** | **$ 11.94** | **$ 4.60** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 18,592** | **$ 55,777** | **$ 18,741,007** | **$ 12.08** | **$ 4.65** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 2158 | gpm |
| Feed Rate | 8169 | lpm |
| Precip Mass Flow | 0.14 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 60.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 12.71 | g/L of Feed |
| Oxalic Acid Dose | 8.34 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.08 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 10,790 | gal | $ 14,000 | $ 56,000 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 5 | HP | $ 41,448 | $ 165,793 | $ - | |
| 3 | Process Pump | 3 | 100% | 2,158 | gpm | $ 121,098 | $ 363,295 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 1.28 | ft^3 | $ 10,907 | $ 10,907 | $ - | |
| 6 | Roaster | 1 | 100% | 0.21 | MBTU/hr | $ 423,133 | $ 423,133 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,019,128** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 21.58 | Effiicency: | 0.85 | | | $ 10.34 | $ 31.03 | $ 10,428 | $ 0.01 | $ 0.00 |
| 3 | Process Pump | 161.86 | | | | | $ 77.59 | $ 232.76 | $ 78,207 | $ 0.05 | $ 0.02 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.21 | | | | | $ 0.10 | $ 0.31 | $ 104 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.03 | | | | | $ 0.02 | $ 0.05 | $ 15 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **184** | | | | | **$ 88.05** | **$ 264.15** | **$ 88,753** | **$ 0.06** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 6.23 | mt/hr | $ 125.000 | / mt | $ 6,228.71 | $ 18,686.12 | $ 6,278,537 | $ 4.05 | $ 1.56 |
| 2 | Oxalic Acid | 4,087.36 | kg/hr | $ 0.600 | / kg | $ 19,619.31 | $ 58,857.94 | $ 19,776,269 | $ 12.74 | $ 4.90 |
| 3 | Natural Gas | 0.20 | MCF/hr | $ 2.860 | / MCF | $ 4.63 | $ 13.90 | $ 4,670 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 25,852.65** | **$ 77,557.96** | **$ 26,059,476** | **$ 16.79** | **$ 6.46** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 25,941** | **$ 77,822** | **$ 26,148,229** | **$ 16.85** | **$ 6.49** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8246 | gpm |
| Raffinate Rate: | 8246 | gpm |
| Extraction Feed (O+A) | 16493 | gpm |
| Scrub Feed (O+A) | 227 | gpm |
| Strip Feed (O+A) | 453 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 227 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 271142 | gal |
| Precip Mixing Intensity | 0.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 60.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,493 | gal | $ 95,067 | $ 190,134 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 41 | HP | $ 130,512 | $ 261,023 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 247,394 | gal | $ 435,976 | $ 871,952 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,246 | gpm | $ 201,925 | $ 403,849 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 227 | gal | $ 8,530 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,630 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,401 | gal | $ 39,119 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 453 | gal | $ 12,597 | $ 25,193 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 17,194 | $ 34,388 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,802 | gal | $ 57,768 | $ 115,536 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,547 | $ 19,547 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 59,017 | kg | $ 3,695,013 | $ - | $ 3,695,013 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 1,023,817 | kg | $ 1,108,637 | $ - | $ 1,108,637 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,713 | gal | $ 5,663 | $ 22,653 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 1.36 | HP | $ 19,025 | $ 76,100 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,547 | $ 39,093 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.02 | ft^3 | $ 459 | $ 459 | $ - | |
| 20 | Process Pump | 2 | 100% | 227 | gpm | $ 51,276 | $ 102,552 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,162,479** | **$ 4,803,650** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 82.46 | Effiicency: | 0.85 | | $ 39.53 | $ 118.59 | $ 39,846 | $ 0.03 | $ 0.01 |
| 3 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 412.32 | | | | $ 197.65 | $ 592.95 | $ 199,230 | $ 0.13 | $ 0.05 |
| 5 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.27 | | | | $ 1.09 | $ 3.26 | $ 1,096 | $ 0.00 | $ 0.00 |
| 11 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.45 | | | | $ 0.22 | $ 0.65 | $ 218 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 5.43 | | | | $ 2.60 | $ 7.80 | $ 2,622 | $ 0.00 | $ 0.00 |
| 17 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.90 | | | | $ 0.43 | $ 1.30 | $ 437 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | | | | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.34 | | | | $ 5.43 | $ 16.30 | $ 5,478 | $ 0.00 | $ 0.00 |
| | **TOTAL:** | **515** | | | | **$ 246.95** | **$ 740.85** | **$ 248,927** | **$ 0.16** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 106.76 | kg/hr | $ 0.940 | / kg | $ 802.83 | $ 2,408.49 | $ 809,254 | $ 0.52 | $ 0.20 |
| 2 | D2EHPA | 5.62 | kg/hr | $ 5.000 | / kg | $ 224.76 | $ 674.27 | $ 226,555 | $ 0.15 | $ 0.06 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.87 | mt/hr | $ 270.000 | / mt | $ 1,869.94 | $ 5,609.82 | $ 1,884,898 | $ 1.21 | $ 0.47 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,897.53** | **$ 8,692.58** | **$ 2,920,707** | **$ 1.88** | **$ 0.72** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,144** **$ 9,433** **$ 3,169,634** **$ 2.04** **$ 0.79**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C4
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 2157.91 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 8168.58 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 2156.11 | gpm | HCL Dose | 19.38 | g/L of Feed |
| Mn Feed Rate | 8161.78 | lpm | | | |
| Co Precip Mass Flow | 1.53 | tph | | | |
| Mn Precip Mass Flow | 0.84 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 6.84 | g/L of Feed | | | |
| NaOH Dose | 4.53 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 32,369 | gal | $ 39,717 | $ 79,435 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 49 | HP | $ 143,114 | $ 286,227 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 2,158 | gpm | $ 121,095 | $ 121,095 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 7.21 | ft^3 | $ 37,885 | $ 37,885 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 32,342 | gal | $ 39,686 | $ 79,372 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 49 | HP | $ 143,046 | $ 286,093 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 2,156 | gpm | $ 121,056 | $ 121,056 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 4 | ft^3 | $ 24,598 | $ 24,598 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,035,762** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 97.11 | Effiicency: | 0.85 | | | $ 46.55 | $ 139.64 | $ 46,920 | $ 0.03 | $ 0.01 |
| 3 | Process Pump | 53.95 | | | | | $ 25.86 | $ 77.58 | $ 26,067 | $ 0.02 | $ 0.01 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 1.21 | | | | | $ 0.58 | $ 1.74 | $ 585 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 97.03 | | | | | $ 46.51 | $ 139.53 | $ 46,881 | $ 0.03 | $ 0.01 |
| 8 | Process Pump | 53.90 | | | | | $ 25.84 | $ 77.52 | $ 26,045 | $ 0.02 | $ 0.01 |
| 9 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 0.66 | | | | | $ 0.32 | $ 0.96 | $ 321 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **304** | | | | | **$ 145.65** | **$ 436.96** | **$ 146,820** | **$ 0.09** | **$ 0.04** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 3.35 | mt/hr | $ 360.000 | / mt | $ 9,654.51 | $ 28,963.53 | $ 9,731,746 | $ 6.27 | $ 2.41 |
| 2 | NaOH | 2.22 | mt/hr | $ 125.000 | / mt | $ 2,216.21 | $ 6,648.63 | $ 2,233,939 | $ 1.44 | $ 0.55 |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 3.84 | $ 11.52 | $ 3,872 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 11,874.56** | **$ 35,623.68** | **$ 11,969,557** | **$ 7.71** | **$ 2.97** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 12,020** **$ 36,061** **$ 12,116,378** **$ 7.81** **$ 3.01**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | Shakespeare Lignite | Date: | 3/18/2023 |
| Case: | SHXP-C5 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | an |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | N/A |
| *(unit conversion)* | 12000 | TPD | Project NPV: | (1,418,227,391) |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 22.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C5 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 101,481,430 | | | | | |
| Constant Dollar Annual OPEX | $ | 462,339,819 | $ | 114.67 | /ton | $ 628.0 | /kg |
| Constant Dollar Annual REE Rev | $ | 270,473,485 | $ | 67.08 | /ton | $ 367.40 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | | |
|---|---|---|---|---|---|
| Payback Period | | 25.5 | years | *22.5* | *operating years* |
| Internal Rate of Return | | N/A | | | |
| Net Present Value | $ | (1,418,227,391) | | | |
| Total Severence Tax Paid | $ | - | | | |
| Total State and Federal Tax Paid | $ | - | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-CS | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 1,250 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 2.98 | 37.24 | 16.89 | | | | | | | 16.89 |
| Yttrium | 17.25 | 215.68 | 97.83 | | | | | | | 97.83 |
| Lanthanum | 15.45 | 193.09 | 87.59 | | | | | | | 87.59 |
| Cerium | 31.48 | 393.53 | 178.50 | | | | | | | 178.50 |
| Praseodymium | 3.73 | 46.64 | 21.15 | | | | | | | 21.15 |
| Neodymium | 14.95 | 186.84 | 84.75 | | | | | | | 84.75 |
| Samarium | 3.70 | 46.29 | 21.00 | | | | | | | 21.00 |
| Europium | 0.72 | 9.02 | 4.09 | | | | | | | 4.09 |
| Gadolinium | 3.52 | 43.96 | 19.94 | | | | | | | 19.94 |
| Terbium | 0.43 | 5.42 | 2.46 | | | | | | | 2.46 |
| Dysprosium | 2.20 | 27.55 | 12.50 | | | | | | | 12.50 |
| Holmium | 0.47 | 5.89 | 2.67 | | | | | | | 2.67 |
| Erbium | 0.74 | 9.28 | 4.21 | | | | | | | 4.21 |
| Thulium | 0.21 | 2.61 | 1.18 | | | | | | | 1.18 |
| Ytterbium | 1.69 | 21.14 | 9.59 | | | | | | | 9.59 |
| Lutetium | 0.47 | 5.82 | 2.64 | | | | | | | 2.64 |
| Manganese | -- | 465.70 | 211.24 | | | 0.00 | -- | | | 211.24 |
| Cobalt | -- | 162.20 | 73.57 | | | 0.00 | -- | | | 73.57 |
| Nickel | -- | 486.70 | 220.76 | | | 0.00 | -- | | | 220.76 |
| Zinc | -- | 133.60 | 60.60 | | | 0.00 | -- | | | 60.60 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 1250.00 | 566.99 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 567.0 |
| Total non-REE CM | -- | 1248.20 | 566.17 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 566.2 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 16.89 | 0.00 | 4.90 | 0.00 | 0.00 | 0.00 | 4.91 | 11.99 | 29.0% | |
| Yttrium | 97.83 | 6.12 | 0.00 | 16.28 | 5.15 | 0.00 | 27.55 | 70.28 | 28.2% | |
| Lanthanum | 87.59 | 4.52 | 0.00 | 8.97 | 4.84 | 0.00 | 18.32 | 69.26 | 20.9% | |
| Cerium | 178.50 | 11.90 | 0.00 | 0.14 | 2.89 | 0.00 | 14.92 | 163.58 | 8.4% | |
| Praseodymium | 21.15 | 1.53 | 0.00 | 0.14 | 0.18 | 0.00 | 1.85 | 19.30 | 8.8% | |
| Neodymium | 84.75 | 5.36 | 0.00 | 1.66 | 3.59 | 0.00 | 10.62 | 74.13 | 12.5% | |
| Samarium | 21.00 | 1.89 | 0.00 | 0.49 | 1.01 | 0.00 | 3.39 | 17.61 | 16.1% | |
| Europium | 4.09 | 0.39 | 0.00 | 0.00 | 0.12 | 0.00 | 0.52 | 3.57 | 12.6% | |
| Gadolinium | 19.94 | 1.75 | 0.00 | 0.54 | 0.92 | 0.00 | 3.21 | 16.73 | 16.1% | |
| Terbium | 2.46 | 0.23 | 0.00 | 0.01 | 0.02 | 0.00 | 0.26 | 2.20 | 10.4% | |
| Dysprosium | 12.50 | 0.92 | 0.00 | 0.59 | 0.61 | 0.00 | 2.12 | 10.38 | 16.9% | |
| Holmium | 2.67 | 0.23 | 0.00 | 0.25 | 0.10 | 0.00 | 0.58 | 2.09 | 21.7% | |
| Erbium | 4.21 | 0.34 | 0.00 | 0.62 | 0.24 | 0.00 | 1.21 | 3.00 | 28.7% | |
| Thulium | 1.18 | 0.06 | 0.00 | 0.05 | 0.05 | 0.00 | 0.16 | 1.02 | 13.7% | |
| Ytterbium | 9.59 | 0.37 | 0.00 | 0.68 | 0.26 | 0.00 | 1.30 | 8.28 | 13.6% | |
| Lutetium | 2.64 | 0.06 | 0.00 | 0.24 | 0.08 | 0.00 | 0.38 | 2.26 | 14.4% | |
| Manganese | 211.24 | 0.01 | 0.00 | 69.14 | 6.58 | 0.00 | 75.74 | 135.50 | 35.9% | |
| Cobalt | 73.57 | 0.03 | 0.00 | 2.90 | 25.17 | 0.00 | 28.09 | 45.48 | 38.2% | |
| Nickel | 220.76 | 0.08 | 0.00 | 4.10 | 56.23 | 0.00 | 60.40 | 160.36 | 27.4% | |
| Zinc | 60.60 | 0.04 | 0.00 | 0.15 | 22.42 | 0.00 | 22.61 | 37.99 | 37.3% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 566.99 | 35.67 | 4.90 | 30.66 | 20.06 | 0.00 | 91.29 | 475.70 | 16.1% | <---- |
| Total non-REE CM | 566.17 | 0.16 | 0.00 | 76.29 | 110.39 | 0.00 | 186.84 | 379.33 | 33.0% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 422 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 29 | ton | $ | 1.50 |
| Dust/Volatiles | - | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | | |
|---|---|---|
| 0.86 | TPD REE | |
| 0.118 | TPD Sc | |
| 4.48 | TPD CM | |
| 0.06 | REE | |
| 29.0% | Sc | |
| 32.7% | Mn | |
| 29.2% | Metal-S | |
| 500 | Feed | |
| 1,250 | Grade | |
| 25.5 | PP | |
| N/A | IRR | |
| -1418227391 | NPV | |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C5 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 0.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 2.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | Based on Design |
| Dissolved Solids in Leach | 15.7% | % | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Leaching Retention Time | 45 | min | Based on Design |
| Thickener UF % Solids | 45% | % | VT Override from 7.9% \| SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Residue Settling Rate | 15.0 | ft/hr | Not Available |
| **Reagent Use** | | | |
| Acid Consumption | 58.1 | kg/ton | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Bioleaching Sol'n Consumption | 5487 | L/ton | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Polymer Dosage | 0.05 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 7.81 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | Based on Design |
| Precip Solid Yield | 14.03 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Precip Settling Rate | 2.24 | ft/hr | Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 1.9% | % | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| **Reagent Use** | | | |
| NaOH Dose | 49.76 | g/L of Feed | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 287.91 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 1250 | ppm | |
| Grade After Sorter | 1250 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 500 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 0 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 500 | tph | |
| Leaching Residue Rate | 421.5 | tph | |
| Slurry Flow (feed) | 9250 | gpm | |
| Product Flow | 8,198 | gpm | |
| Thickener Feed % Solids (eff) | 17% | % | |
| Liquid Density | 1.014 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 39.18 | g/L | |
| Leaching Vessel Size | 52031.25 | gal | |
| Leaching Vessel Size | 196.96 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 210.75 | tph | 70 to 200 tph |
| Thickener Area | 1646.69 | ft^2 | |
| Thickener Diameter | 45.80 | ft | ideal < 100' |
| Liquid Balance | 195.97 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,198 | gpm | |
| Slurry Feed Rate | 31,032 | lpm | |
| Precip Mass Flow | 28.80 | tph | |
| Thickener Feed % Solids (eff) | 1.38% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.004 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 349.11 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 135.67 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 33.83 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 2,073 | ft^2 | |
| Thickener Diameter | 51.38 | ft | ideal < 100' |

*PM5: REE Precipitation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |

**Process Parameters**

| | | | |
|---|---|---|---|
| Precip Conditioning Time | 45 | min | UK: Based on Design |
| Precip Solid Yield | 8.82 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 0.044 | ft/hr | UK: Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.94% | % | VT Override from 1.51% \|\| UK: SHXP-C4/SHXP-C4 Samples |

**Reagent Use**

| | | | |
|---|---|---|---|
| NaOH Dose | 5.77 | g/L of Feed | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |

**Equipment Specifications**

| | | | |
|---|---|---|---|
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 5431.06 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Slurry Feed Rate | 8164 | gpm | |
| Slurry Feed Rate | 30,904 | lpm | |
| Precip Mass Flow | 18.04 | tph | |
| Thickener Feed % Solids (eff) | 0.9% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 0.998 | g/cc | |

**Other Indicators**

| | | | |
|---|---|---|---|
| Conditioning Vessel Size | 231.78 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 85.01 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 21.19 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 16326 | ft^2 | |
| Thickener Diameter | 144.21 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | 27.72 | m/m | target = 6.35 |

*PM6: RE-Redissolution*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 3.0% | % | VT Override from 10.7% \|\| SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Dissolved Solids in Re-Leach | 60% | % | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Leaching Retention Time | 120 | min | Based on Design |
| **Reagent Use** | | | |
| Acid Consumption | 585.0 | kg/ton | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| **Material Balance** | | | |
|---|---|---|---|
| Solid Feed Rate | 18.04 | tph | |
| Slurry Flow (feed) | 2,354 | gpm | |
| Product Flow | 2,354 | gpm | |
| Liquid Density | 1.022 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 267.3 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 3.48 | v/v | target = 3.63 based on data |
| Liquid Balance | 21.19 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 30.6 | g/L | |

*PM7: REE Oxalate Precipitation*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | Based on Design |
| Precip Solid Yield | 0.0 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Thickener UF %Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| NaOH Dose | 0.00 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Oxalic Acid Dose | 8.73 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| **Material Balance** | | | |
|---|---|---|---|
| Slurry Feed Rate | 2353.88 | gpm | |
| Slurry Feed Rate | 8910.41 | lpm | |
| Precip Mass Flow | 0.02 | tph | |
| Thickener Feed % Solids (eff) | 0.00% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.01 | tph | |
| Liquid Density | 1.022 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 44.55 | m^3 | |
| Filter Press Area | 0.10 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.02 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 39.70 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.70 | m/m | |

*PM8: Sc SX Separation*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | UK: Design |
| Settler Residence Time | 15.00 | min | UK: Design |
| Loading: Number of Stages | 2.00 | - | UK: Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Loading O:A Advance Ratio | 0.03 | v/v | UK: Design |
| Scrubbing: Number of Stages | | - | N/A |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | N/A |
| Scrubbing O:A Advance Ratio | | v/v | N/A |
| Stripping: Number of Stages | 2.00 | - | UK: Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | UK: Design |
| Sc2O3 Grade | 2% | % | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Precip Solid Yield | 0.4 | g/L | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| **Material Balance (SX)** | | | |
|---|---|---|---|
| PLS Flow | 8197.8 | gpm | |
| Raffinate Rate: | 8197.8 | gpm | |
| Load Org Recycle | 7972.4 | gpm | |
| Load Org Feed | 8197.8 | gpm | |
| Loag Org Advance | 225.4 | gpm | |
| Scrub Aq Product | 0.0 | gpm | |
| Scrub Aq Recycle | 0.0 | gpm | |
| Scrub Aq Feed | 0.0 | gpm | |
| Scurb Org Product | 225.4 | gpm | |
| Strip Aq Product | 18.0 | gpm | |
| Strip Aq Recycle | 207.4 | gpm | |
| Strip Aq Feed | 225.4 | gpm | |
| Strip Org Product | 225.4 | gpm | |
| Extraction Feed (O+A): | 16395.6 | gpm | |
| Scrub Feed (O+A): | 225.4 | gpm | |
| Strip Feed (O+A): | 450.8 | gpm | |
| **Material Balance (Precip)** | | | |
| Strip Solution Flow | 18.0 | gpm | |
| Solid Density | 3.4 | g/cc | |
| Solid Product Rate | 0.00 | tph | |
| Liquid Density (after precip) | 1.13 | g/cc | |
| Thickener Feed % Solids (eff) | 0.03% | % | |
| **Total Volumes** | | | |
| Total Extraction Volume | 524658 | gal | |
| Total Scrubbing Volume | 0 | gal | |
| Total Stripping Volume | 14425 | gal | |
| Extraction Org Volume | 262328.82 | gal | |
| Scrubbing Org Volume | 0.00 | gal | |
| Stripping Org Volume | 7212.36 | gal | |
| Total Org Volume | 269541.18 | gal | |
| **Other Indicators** | | | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 10.21 | m^3 | |
| Filter Press Area | 0.02 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM9: CM Precipitation*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 0 | min | N/A |
| Precip Solid Yield | - | g/L | N/A |
| Precip Settlng Rate | 10 | ft/hr | N/A |
| Thickener UF % Solids | 25% | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 7.24 | g/L of Feed | N/A |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 0.00 | min | N/A |
| Thickener Capacity | 0.00 | ft^2/tph | N/A |

| **Material Balance** | | | |
|---|---|---|---|
| Slurry Feed Rate | 8143 | gpm | |
| Slurry Feed Rate | 30,824 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 0.998 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

## PM10: CM Redissolution

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 2.0% | % | N/A |
| Dissolved Solids in Re-Leach | 100% | % | N/A |
| Leaching Retention Time | 45 | min | N/A |
| **Reagent Use** | | | |
| Acid Consumption | 441 | kg/ton | N/A |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 0.00 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 0 | gpm | |
| Liquid Density | 1.000 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | |
| Releach Conc. Factor | #DIV/0! | v/v | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | - | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

## PM11: CM Recovery

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 4.25 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | | % | Not Applicable |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 2.64 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Na2S Dose | 8.67 | g/L of Feed | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| NaOH Dose | 4.51 | g/L of Feed | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| HCL Dose | 0.00 | g/L of Feed | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 2353.86 | gpm | |
| Co Slurry Feed Rate | 8910.32 | lpm | |
| Co Precip Mass Flow | 2.50 | tph | |
| Co Thickener Feed % Solids (eff) | 0.42% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 2350.92 | gpm | |
| Mn Slurry Feed Rate | 8899.20 | lpm | |
| Mn Precip Mass Flow | 1.55 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.26% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.02 | g/cc | |
| Mn Liquid Density | 1.02 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 133.7 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 11.79 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 2.94 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 133.5 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 7.3 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 1.82 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

## PM12: Biooxidation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Requirec | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C5 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 6,416,147 | $ - | $ 1,604,037 | $ 7,057,761 | $ 8,786,913 | $ 23,864,858 | IIIIIIIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 3,040,947 | $ - | $ 760,237 | $ 3,345,041 | $ 4,164,577 | $ 11,310,802 | IIIIIIIIIII |
| PM5 | REE Precipitation | $ 8,673,320 | $ - | $ 2,168,330 | $ 9,540,651 | $ 11,878,111 | $ 32,260,412 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| PM6 | RE-Redissolution | $ 1,347,124 | $ - | $ 336,781 | $ 1,481,837 | $ 1,844,887 | $ 5,010,629 | IIIIII |
| PM7 | REE Oxalate Precipitation | $ 786,753 | $ - | $ 196,688 | $ 865,428 | $ 1,077,458 | $ 2,926,327 | III |
| PM8 | Sc SX Separation | $ 2,155,838 | $ 4,775,293 | $ 538,960 | $ 2,371,422 | $ 2,952,421 | $ 12,793,934 | IIIIIIIIIIII |
| PM9 | CM Precipitation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM10 | CM Redissolution | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM11 | CM Recovery | $ 1,115,980 | $ - | $ 278,995 | $ 1,227,578 | $ 1,528,335 | $ 4,150,888 | IIIII |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| | TOTAL | $ 25,999,768 | $ 4,775,293 | $ 6,499,942 | $ 28,599,745 | $ 35,606,682 | $ 101,481,430 | |

## Module Summary

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| Total Lang Factor (=1+sum % Base) | | 372% | |

Year of Construction: 2025

# OPERATING COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C5 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 4.17 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 1.60 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 101,481,430 | $ Total | $ 1,006.76 | $ 3,020.28 | $ 1,014,814 | $ 1.38 | $ 0.25 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 0.42 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 270,473,485 | $ Total | $ 1,341.63 | $ 4,024.90 | $ 1,352,367 | $ 1.84 | $ 0.34 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 0.63 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 101,481,430 | $ Total | $ 1,006.76 | $ 3,020.28 | $ 1,014,814 | $ 1.38 | $ 0.25 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 8,336.60** | **$ 25,009.81** | **$ 8,403,298** | **$ 11.41** | **$ 2.08** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 0.70 | $ 0.13 |
| PM2 | Roasting | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | PR | 1300 | | $ 623.25 | $ 1,869.74 | $ 628,231 | $ 0.85 | $ 0.16 |
| PM4 | Al Precipitation | PR | 829 | | $ 397.44 | $ 1,192.32 | $ 400,621 | $ 0.54 | $ 0.10 |
| PM5 | REE Precipitation | PR | 874 | | $ 418.80 | $ 1,256.40 | $ 422,150 | $ 0.57 | $ 0.10 |
| PM6 | RE-Redissolution | PR | 483 | | $ 231.31 | $ 693.93 | $ 233,160 | $ 0.32 | $ 0.06 |
| PM7 | REE Oxalate Precipitation | PR | 200 | | $ 95.92 | $ 287.76 | $ 96,687 | $ 0.13 | $ 0.02 |
| **Sub-Total** | | | **5595.6** | | **$ 2,682.29** | **$ 8,046.88** | **$ 2,703,751.09** | **$ 3.67** | **$ 0.67** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.14 | $ 0.03 |
| PM2 | Roasting | CR | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | CR | | $ 219,847.57 | $ 659,542.70 | $ 221,606,349 | $ 301.02 | $ 54.96 |
| PM4 | Al Precipitation | CR | | $ 92,650.53 | $ 277,951.59 | $ 93,391,734 | $ 126.86 | $ 23.16 |
| PM5 | REE Precipitation | CR | | $ 10,700.25 | $ 32,100.76 | $ 10,785,856 | $ 14.65 | $ 2.68 |
| PM6 | RE-Redissolution | CR | | $ 22,790.39 | $ 68,371.18 | $ 22,972,717 | $ 31.20 | $ 5.70 |
| PM7 | REE Oxalate Precipitation | CR | | $ 22,410.78 | $ 67,232.34 | $ 22,590,067 | $ 30.69 | $ 5.60 |
| **Sub-Total** | | | | **$ 387,142.22** | **$ 1,161,426.67** | **$ 390,239,362.46** | **$ 530.08** | **$ 96.79** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 10,116 | $ 0.10 | $ 337.20 | $ 1,011.60 | $ 339,898 | $ 0.46 | $ 0.08 |
| Non-Hazardous Precipitate | WD | Tons | 691 | $ 1.50 | $ 345.55 | $ 1,036.65 | $ 348,316 | $ 0.47 | $ 0.09 |
| Dust/Volatiles | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 682.75** | **$ 2,048.25** | **$ 688,213.62** | **$ 0.93** | **$ 0.17** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 390,507** | **$ 1,171,522** | **$ 393,631,327** | **$ 535** | **$ 98** |

## Overhead Costs

| Allocation | Code | Rate | Basis | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% | % of Direct Cost | $ 25,010 Daily Direct Cost | $ 1,250.49 | $ 3,751.47 | $ 1,260,495 | $ 1.71 | $ 0.31 |
| Variable Portion | OH | 15% | % of Direct Cost | $ 1,171,522 Daily Direct Cost | $ 58,576.09 | $ 175,728.27 | $ 59,044,699 | $ 80.20 | $ 14.64 |
| **TOTAL OVERHEAD COSTS** | | | | | **$ 59,826.58** | **$ 179,479.74** | **$ 60,305,193.72** | **$ 81.92** | **$ 14.96** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | SHXP-C5 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 8,336.60 | $ 25,009.81 | $ 8,403,298 | $ 11 | $ 2.08 | I |
| Power | $ 2,682.29 | $ 8,046.88 | $ 2,703,751 | $ 4 | $ 0.67 | |
| Consumables | $ 387,142.22 | $ 1,161,426.67 | $ 390,239,362 | $ 530 | $ 96.79 | IIII

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C5 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 270,473,485

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 1213 | 1250 |
| Contained Value | ($/ton) | $ 30.6 | $ 248.3 |
| Feed Basket Price | ($/kg) | $ 27.8 | $ 218.9 |
| Product Basket Price | ($/kg) | $ 25.7 | $ 370.6 |
| Production Rate | (kg/day) | 2073.27 | 2191.03 |
| | (t/day) | 2.07 | 2.19 |
| | (t/yr) | 696.62 | 736.19 |
| | (% of US Demand) | 7.0% | |

## Revenue Contribution Table

| Element | REE/CM Price ($/kg) [1] | Product Revenue ($/hr) MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | REE Revenue ($/day) | |
|---|---|---|---|---|---|---|---|---|
| Scandium | $ 4,200.00 | $ 0.7 | $ 31,581.6 | $ - | $ - | $ - | $ 757,977 | IIII

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C5 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 295,553,680 | $ 304,420,290 | $ 313,552,899 | $ 322,959,486 | $ 332,648,271 | $ 342,627,719 | $ 352,906,550 | $ 363,493,747 | $ 374,398,559 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 295,553,680 | $ 304,420,290 | $ 313,552,899 | $ 322,959,486 | $ 332,648,271 | $ 342,627,719 | $ 352,906,550 | $ 363,493,747 | $ 374,398,559 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 304,444 | $ 2,173,772 | $ 508,197,194 | $ 523,101,017 | $ 538,444,380 | $ 554,240,013 | $ 570,501,004 | $ 587,240,802 | $ 604,473,230 | $ 622,212,493 | $ 640,473,189 |
| *- Operating Costs* | $ - | $ - | $ - | $ 505,211,203 | $ 520,367,539 | $ 535,978,565 | $ 552,057,922 | $ 568,619,660 | $ 585,678,250 | $ 603,248,597 | $ 621,346,055 | $ 639,986,437 |
| *- Loan #1 Interest* | $ - | $ 304,444 | $ 281,347 | $ 256,863 | $ 230,911 | $ 203,401 | $ 174,241 | $ 143,331 | $ 110,567 | $ 75,837 | $ 39,023 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 1,892,426 | $ 1,748,851 | $ 1,596,662 | $ 1,435,342 | $ 1,264,343 | $ 1,083,083 | $ 890,948 | $ 687,285 | $ 471,402 | $ 242,566 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 980,277 | $ 905,905 | $ 827,071 | $ 743,507 | $ 654,929 | $ 561,037 | $ 461,511 | $ 356,014 | $ 244,186 |
| Gross Profit | $ - | $ (304,444) | $ (2,173,772) | $ (212,643,514) | $ (218,680,727) | $ (224,891,480) | $ (231,280,527) | $ (237,852,733) | $ (244,613,083) | $ (251,566,679) | $ (258,718,746) | $ (266,074,629) |
| **Book Deductions** | $ - | $ 380,555 | $ 2,746,418 | $ 3,975,854 | $ 3,677,665 | $ 3,401,841 | $ 3,146,702 | $ 2,910,700 | $ 2,692,397 | $ 2,490,468 | $ 2,303,682 | $ 2,130,906 |
| *- CAPEX #1 Depreciation* | $ - | $ 380,555 | $ 352,014 | $ 325,613 | $ 301,192 | $ 278,602 | $ 257,707 | $ 238,379 | $ 220,501 | $ 203,963 | $ 188,666 | $ 174,516 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 2,394,404 | $ 2,214,824 | $ 2,048,712 | $ 1,895,059 | $ 1,752,929 | $ 1,621,460 | $ 1,499,850 | $ 1,387,361 | $ 1,283,309 | $ 1,187,061 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,435,418 | $ 1,327,762 | $ 1,228,180 | $ 1,136,066 | $ 1,050,861 | $ 972,047 | $ 899,143 | $ 831,707 | $ 769,329 |
| Taxable Before Depletion | $ - | $ (685,000) | $ (4,920,190) | $ (216,619,368) | $ (222,358,392) | $ (228,293,321) | $ (234,427,229) | $ (240,763,433) | $ (247,305,480) | $ (254,057,147) | $ (261,022,429) | $ (268,205,536) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ - | $ (342,500) | $ (2,460,095) | $ (108,309,684) | $ (111,179,196) | $ (114,146,660) | $ (117,213,615) | $ (120,381,716) | $ (123,652,740) | $ (127,028,573) | $ (130,511,214) | $ (134,102,768) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 41,377,515 | $ 42,618,841 | $ 43,897,406 | $ 45,214,328 | $ 46,570,758 | $ 47,967,881 | $ 49,406,917 | $ 50,889,125 | $ 52,415,798 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (685,000) | $ (4,920,190) | $ (216,619,368) | $ (222,358,392) | $ (228,293,321) | $ (234,427,229) | $ (240,763,433) | $ (247,305,480) | $ (254,057,147) | $ (261,022,429) | $ (268,205,536) |
| *- Income Tax* | $ - | $ (178,100) | $ (1,279,249) | $ (56,321,036) | $ (57,813,182) | $ (59,356,263) | $ (60,951,080) | $ (62,598,492) | $ (64,299,425) | $ (66,054,858) | $ (67,865,831) | $ (69,733,439) |
| Net Profit/Loss | $ - | $ (506,900) | $ (3,640,941) | $ (160,298,333) | $ (164,545,210) | $ (168,937,057) | $ (173,476,150) | $ (178,164,940) | $ (183,006,055) | $ (188,002,289) | $ (193,156,597) | $ (198,472,096) |
| **Book Deductions** | $ - | $ 380,555 | $ 2,746,418 | $ 3,975,854 | $ 3,677,665 | $ 3,401,841 | $ 3,146,702 | $ 2,910,700 | $ 2,692,397 | $ 2,490,468 | $ 2,303,682 | $ 2,130,906 |
| *+ CAPEX #1 Depreciation* | $ - | $ 380,555 | $ 352,014 | $ 325,613 | $ 301,192 | $ 278,602 | $ 257,707 | $ 238,379 | $ 220,501 | $ 203,963 | $ 188,666 | $ 174,516 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 2,394,404 | $ 2,214,824 | $ 2,048,712 | $ 1,895,059 | $ 1,752,929 | $ 1,621,460 | $ 1,499,850 | $ 1,387,361 | $ 1,283,309 | $ 1,187,061 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,435,418 | $ 1,327,762 | $ 1,228,180 | $ 1,136,066 | $ 1,050,861 | $ 972,047 | $ 899,143 | $ 831,707 | $ 769,329 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ - | $ (126,344) | $ (894,523) | $ (156,322,478) | $ (160,867,545) | $ (165,535,217) | $ (170,329,447) | $ (175,254,240) | $ (180,313,658) | $ (185,511,821) | $ (190,852,915) | $ (196,341,190) |
| **Captial Cost** | $ 5,074,072 | $ 31,925,388 | $ 19,138,907 | $ 4,208,549 | $ 4,461,062 | $ 4,728,726 | $ 5,012,449 | $ 5,313,196 | $ 5,631,988 | $ 5,969,907 | $ 6,328,102 | $ 6,018,384 |
| *- Fixed Capital Costs* | $ 10,148,143 | $ 63,080,857 | $ 32,675,884 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 5,074,072 | $ 31,540,429 | $ 16,337,942 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 384,959 | $ 408,057 | $ 432,540 | $ 458,493 | $ 486,002 | $ 515,163 | $ 546,072 | $ 578,837 | $ 613,567 | $ 650,381 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 2,392,908 | $ 2,536,482 | $ 2,688,671 | $ 2,849,992 | $ 3,020,991 | $ 3,202,251 | $ 3,394,386 | $ 3,598,049 | $ 3,813,932 | $ 4,042,768 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 1,239,526 | $ 1,313,898 | $ 1,392,732 | $ 1,476,296 | $ 1,564,873 | $ 1,658,766 | $ 1,758,292 | $ 1,863,789 | $ 1,975,617 |
| **Net Cash Flow** | $ (5,074,072) | $ (32,051,732) | $ (20,033,430) | $ (160,531,027) | $ (165,328,607) | $ (170,263,943) | $ (175,341,897) | $ (180,567,437) | $ (185,945,646) | $ (191,481,729) | $ (197,181,017) | $ (202,359,574) |
| Discounted Cash Flow | $ (5,074,072) | $ (29,137,939) | $ (16,556,554) | $ (120,609,337) | $ (112,921,663) | $ (105,720,513) | $ (98,975,929) | $ (92,659,646) | $ (86,745,016) | $ (81,206,945) | $ (76,021,818) | $ (70,925,796) |
| **Cumulative DCF** | $ (5,074,072) | $ (34,212,010) | $ (50,768,564) | $ (171,377,900) | $ (284,299,563) | $ (390,020,076) | $ (488,996,005) | $ (581,655,651) | $ (668,400,668) | $ (749,607,613) | $ (825,629,430) | $ (896,555,227) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 101,481,430 | Payback Period | 25.5 |
| Constant Dollar OPEX | $ 462,339,819 | Internal Rate of Return | N/A |
| Constant Dollar REE Rev | $ 270,473,485 | Net Present Value | $ (1,418,227,391) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ - |

## PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C5 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 385,630,516** | **$ 397,199,432** | **$ 409,115,415** | **$ 421,388,877** | **$ 434,030,543** | **$ 447,051,460** | **$ 460,463,003** | **$ 474,276,894** | **$ 488,505,200** | **$ 503,160,356** | **$ 518,255,167** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 385,630,516 | $ 397,199,432 | $ 409,115,415 | $ 421,388,877 | $ 434,030,543 | $ 447,051,460 | $ 460,463,003 | $ 474,276,894 | $ 488,505,200 | $ 503,160,356 | $ 518,255,167 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 659,311,679** | **$ 678,961,611** | **$ 699,330,459** | **$ 720,310,373** | **$ 741,919,684** | **$ 764,177,274** | **$ 787,102,593** | **$ 810,715,670** | **$ 835,037,140** | **$ 860,088,255** | **$ 885,890,902** |
| *- Operating Costs* | $ 659,186,030 | $ 678,961,611 | $ 699,330,459 | $ 720,310,373 | $ 741,919,684 | $ 764,177,274 | $ 787,102,593 | $ 810,715,670 | $ 835,037,140 | $ 860,088,255 | $ 885,890,902 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 125,649 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ (273,681,163) | $ (281,762,179) | $ (290,215,044) | $ (298,921,496) | $ (307,889,140) | $ (317,125,815) | $ (326,639,589) | $ (336,438,777) | $ (346,531,940) | $ (356,927,898) | $ (367,635,735) |
| **Book Deductions** | **$ 1,971,088** | **$ 1,823,257** | **$ 1,686,512** | **$ 1,560,024** | **$ 1,443,022** | **$ 1,334,796** | **$ 1,234,686** | **$ 1,142,084** | **$ 1,056,428** | **$ 897,166** | **$ 326,341** |
| *- CAPEX #1 Depreciation* | $ 161,427 | $ 149,320 | $ 138,121 | $ 127,762 | $ 118,180 | $ 109,316 | $ 101,118 | $ 93,534 | $ 86,519 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 1,098,031 | $ 1,015,679 | $ 939,503 | $ 869,040 | $ 803,862 | $ 743,573 | $ 687,805 | $ 636,219 | $ 588,503 | $ 544,365 | $ - |
| *- CAPEX #3 Depreciation* | $ 711,630 | $ 658,257 | $ 608,888 | $ 563,221 | $ 520,980 | $ 481,906 | $ 445,763 | $ 412,331 | $ 381,406 | $ 352,801 | $ 326,341 |
| Taxable Before Depletion | $ (275,652,251) | $ (283,585,436) | $ (291,901,557) | $ (300,481,520) | $ (309,332,163) | $ (318,460,610) | $ (327,874,275) | $ (337,580,861) | $ (347,588,368) | $ (357,825,064) | $ (367,962,076) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ (137,826,126) | $ (141,792,718) | $ (145,950,778) | $ (150,240,760) | $ (154,666,081) | $ (159,230,305) | $ (163,937,138) | $ (168,790,431) | $ (173,794,184) | $ (178,912,532) | $ (183,981,038) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 53,988,272 | $ 55,607,920 | $ 57,276,158 | $ 58,994,443 | $ 60,764,276 | $ 62,587,204 | $ 64,464,820 | $ 66,398,765 | $ 68,390,728 | $ 70,442,450 | $ 72,555,723 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ (275,652,251)** | **$ (283,585,436)** | **$ (291,901,557)** | **$ (300,481,520)** | **$ (309,332,163)** | **$ (318,460,610)** | **$ (327,874,275)** | **$ (337,580,861)** | **$ (347,588,368)** | **$ (357,825,064)** | **$ (367,962,076)** |
| *- Income Tax* | $ (71,669,585) | $ (73,732,213) | $ (75,894,405) | $ (78,125,195) | $ (80,426,362) | $ (82,799,759) | $ (85,247,312) | $ (87,771,024) | $ (90,372,976) | $ (93,034,517) | $ (95,670,140) |
| Net Profit/Loss | $ (203,982,666) | $ (209,853,222) | $ (216,007,152) | $ (222,356,325) | $ (228,905,800) | $ (235,660,852) | $ (242,626,964) | $ (249,809,837) | $ (257,215,392) | $ (264,790,548) | $ (272,291,936) |
| **Book Deductions** | **$ 1,971,088** | **$ 1,823,257** | **$ 1,686,512** | **$ 1,560,024** | **$ 1,443,022** | **$ 1,334,796** | **$ 1,234,686** | **$ 1,142,084** | **$ 1,056,428** | **$ 897,166** | **$ 326,341** |
| *+ CAPEX #1 Depreciation* | $ 161,427 | $ 149,320 | $ 138,121 | $ 127,762 | $ 118,180 | $ 109,316 | $ 101,118 | $ 93,534 | $ 86,519 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 1,098,031 | $ 1,015,679 | $ 939,503 | $ 869,040 | $ 803,862 | $ 743,573 | $ 687,805 | $ 636,219 | $ 588,503 | $ 544,365 | $ - |
| *+ CAPEX #3 Depreciation* | $ 711,630 | $ 658,257 | $ 608,888 | $ 563,221 | $ 520,980 | $ 481,906 | $ 445,763 | $ 412,331 | $ 381,406 | $ 352,801 | $ 326,341 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ (202,011,577) | $ (208,029,966) | $ (214,320,640) | $ (220,796,301) | $ (227,462,778) | $ (234,326,056) | $ (241,392,278) | $ (248,667,753) | $ (256,158,964) | $ (263,893,382) | $ (271,965,596) |
| **Captial Cost** | **$ 2,094,154** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 2,094,154 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ (204,105,731) | $ (208,029,966) | $ (214,320,640) | $ (220,796,301) | $ (227,462,778) | $ (234,326,056) | $ (241,392,278) | $ (248,667,753) | $ (256,158,964) | $ (263,893,382) | $ (271,965,596) |
| Discounted Cash Flow | $ (65,034,376) | $ (60,258,871) | $ (56,437,323) | $ (52,856,879) | $ (49,502,528) | $ (46,360,161) | $ (43,416,523) | $ (40,659,165) | $ (38,076,398) | $ (35,660,063) | $ (33,409,878) |
| **Cumulative DCF** | $ (961,589,603) | $ (1,021,848,474) | $ (1,078,285,797) | $ (1,131,142,675) | $ (1,180,645,203) | $ (1,227,005,364) | $ (1,270,421,887) | $ (1,311,081,052) | $ (1,349,157,450) | $ (1,384,817,513) | $ (1,418,227,391) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 101,481,430 | Payback Period | 0 | 25.5 |
| Constant Dollar OPEX | $ 462,339,819 | Internal Rate of Return | 0 | N/A |
| Constant Dollar REE Rev | $ 270,473,485 | Net Present Value | 0 | $ (1,418,227,391) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ - |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5

Date: 3/18/2023
By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 0.39 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 0.25 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.02 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.02 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.02 | $ 0.00 |
| 7 | Conveyor | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 0.70** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.14 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.14** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 611** | **$ 1,834** | **$ 616,370** | **$ 0.84** | **$ 0.15** | |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 2.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 500 | tph |
| Residue Rate | 422 | tph |
| Slurry Flow | 9250 | gpm |
| Leaching Time | 45 | min |
| Acid Consumption | 58.11 | kg/ton |
| Bioleaching Sol'n Consumption | 5487 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 2 | -- |
| Thickener Capacity | 7.81 | ft^2/tph |
| Polymer Dose | 0.050 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 8 | 100% | 52,031 | gal | $ 181,404 | $ 1,451,235 | $ - | |
| 2 | Tank Mixer | 8 | 100% | 78 | HP | $ 187,044 | $ 1,496,351 | $ - | |
| 3 | Process Pump | 4 | 100% | 4,625 | gpm | $ 161,957 | $ 647,827 | $ - | |
| 4 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 5 | Belt Filter | 2 | 100% | 211 | TPH | $ 1,186,781 | $ 2,373,561 | $ - | |
| 6 | Solution Heater | 1 | 100% | 1 | MBTU/hr | $ 30,807 | $ 30,807 | $ - | |
| 7 | Thickener | 2 | 100% | 1,647 | ft^2 | $ 160,405 | $ 320,809 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 6,416,147** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 624.38 | Effiicency: | 0.85 | | $ 299.30 | $ 897.89 | $ 301,691 | $ 0.41 | $ 0.07 |
| 3 | Process Pump | 462.50 | | | | $ 221.70 | $ 665.10 | $ 223,475 | $ 0.30 | $ 0.06 |
| 4 | Conveyor | 31.00 | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.02 | $ 0.00 |
| 5 | Belt Filter | 102.21 | | | | $ 49.00 | $ 146.99 | $ 49,388 | $ 0.07 | $ 0.01 |
| 6 | Solution Heater | 78.00 | | | | $ 37.39 | $ 112.16 | $ 37,687 | $ 0.05 | $ 0.01 |
| 7 | Thickener | 2.09 | | | | $ 1.00 | $ 3.01 | $ 1,012 | $ 0.00 | $ 0.00 |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1300** | | | | **$ 623.25** | **$ 1,869.74** | **$ 628,231** | **$ 0.85** | **$ 0.16** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | 2,743,500.00 | L/hr | $ 0.008 | / L | $ 166,804.80 | $ 500,414.40 | $ 168,139,238 | $ 228.39 | $ 41.70 |
| 2 | H2SO4 | 29.05 | mt/hr | $ 220.000 | / mt | $ 51,134.75 | $ 153,404.24 | $ 51,543,824 | $ 70.01 | $ 12.78 |
| 3 | Natural Gas | 0.75 | MCF/hr | $ 2.860 | / MCF | $ 17.21 | $ 51.63 | $ 17,346 | $ 0.02 | $ 0.00 |
| 4 | Polymer | 105.05 | kg/hr | $ 2.250 | / kg | $ 1,890.81 | $ 5,672.44 | $ 1,905,940 | $ 2.59 | $ 0.47 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 219,847.57** | **$ 659,542.70** | **$ 221,606,349** | **$ 301.02** | **$ 54.96** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 220,471** **$ 661,412** **$ 222,234,580** **$ 301.87** **$ 55.12**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5

Date: 3/18/2023
By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 4.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8198 | gpm |
| Feed Rate | 31032 | lpm |
| Precip Mass Flow | 28.80 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 287.91 | ft^2/tpd |
| NaOH Dose | 49.76 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 4 | 100% | 92,225 | gal | $ 166,863 | $ 667,452 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 46 | HP | $ 139,011 | $ 556,043 | $ - | |
| 3 | Process Pump | 12 | 100% | 2,049 | gpm | $ 59,368 | $ 712,417 | $ - | |
| 4 | Filter Press | 1 | 100% | 136 | ft^3 | $ 313,424 | $ 313,424 | $ - | |
| 5 | Conveyor | 1 | 100% | 29 | TPH | $ 19,933 | $ 19,933 | $ - | |
| 6 | Thickener | 4 | 100% | 2,073 | ft^2 | $ 192,919 | $ 771,676 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 3,040,947** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 184.45 | Effiicency: | 0.85 | | | $ 88.42 | $ 265.25 | $ 89,124 | $ 0.12 | $ 0.02 |
| 3 | Process Pump | 614.83 | | | | | $ 294.72 | $ 884.17 | $ 297,080 | $ 0.40 | $ 0.07 |
| 4 | Filter Press | 22.78 | | | | | $ 10.92 | $ 32.76 | $ 11,006 | $ 0.01 | $ 0.00 |
| 5 | Conveyor | 1.79 | | | | | $ 0.86 | $ 2.57 | $ 863 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 5.27 | | | | | $ 2.53 | $ 7.58 | $ 2,548 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **829** | | | | | **$ 397.44** | **$ 1,192.32** | **$ 400,621** | **$ 0.54** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 92.65 | mt/hr | $ 125.000 | / mt | $ 92,650.53 | $ 277,951.59 | $ 93,391,734 | $ 126.86 | $ 23.16 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 92,650.53** | **$ 277,951.59** | **$ 93,391,734** | **$ 126.86** | **$ 23.16** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 93,048** **$ 279,144** **$ 93,792,355** **$ 127.40** **$ 23.26**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5

Date: 3/18/2023
By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 6.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8164 | gpm |
| Feed Rate | 30904 | lpm |
| Precip Mass Flow | 18.04 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 5431.06 | ft^2/tpd |
| NaOH Dose | 5.77 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 6 | 100% | 61,230 | gal | $ 132,531 | $ 795,185 | $ - | |
| 2 | Tank Mixer | 6 | 100% | 31 | HP | $ 110,337 | $ 662,021 | $ - | |
| 3 | Process Pump | 18 | 100% | 1,361 | gpm | $ 50,782 | $ 914,070 | $ - | |
| 4 | Filter Press | 1 | 100% | 85 | ft^3 | $ 223,860 | $ 223,860 | $ - | |
| 5 | Conveyor | 1 | 100% | 18 | TPH | $ 15,417 | $ 15,417 | $ - | |
| 6 | Thickener | 6 | 100% | 16,326 | ft^2 | $ 1,010,461 | $ 6,062,767 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 8,673,320** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 183.69 | Effiicency: | 0.85 | | | $ 88.05 | $ 264.16 | $ 88,756 | $ 0.12 | $ 0.02 |
| 3 | Process Pump | 612.30 | | | | | $ 293.51 | $ 880.52 | $ 295,854 | $ 0.40 | $ 0.07 |
| 4 | Filter Press | 14.27 | | | | | $ 6.84 | $ 20.53 | $ 6,897 | $ 0.01 | $ 0.00 |
| 5 | Conveyor | 1.12 | | | | | $ 0.54 | $ 1.61 | $ 540 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 62.30 | | | | | $ 29.86 | $ 89.59 | $ 30,102 | $ 0.04 | $ 0.01 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **874** | | | | | **$ 418.80** | **$ 1,256.40** | **$ 422,150** | **$ 0.57** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 10.70 | mt/hr | $ 125.000 | / mt | $ 10,700.25 | $ 32,100.76 | $ 10,785,856 | $ 14.65 | $ 2.68 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 10,700.25** | **$ 32,100.76** | **$ 10,785,856** | **$ 14.65** | **$ 2.68** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 11,119** **$ 33,357** **$ 11,208,006** **$ 15.22** **$ 2.78**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 18.04 | tph |
| Slurry Feed Rate | 2354 | gpm |
| Leaching Time | 120 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 585.00 | kg/ton |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 70,616 | gal | $ 83,282 | $ 333,128 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 106 | HP | $ 222,205 | $ 888,819 | $ - | |
| 3 | Process Pump | 1 | 100% | 2,354 | gpm | $ 125,177 | $ 125,177 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,347,124** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 423.70 | Effiicency: | 0.85 | | | $ 203.10 | $ 609.30 | $ 204,726 | $ 0.28 | $ 0.05 |
| 3 | Process Pump | 58.85 | | | | | $ 28.21 | $ 84.63 | $ 28,434 | $ 0.04 | $ 0.01 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **483** | | | | | **$ 231.31** | **$ 693.93** | **$ 233,160** | **$ 0.32** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 10.55 | mt/hr | $ 270.000 | / mt | $ 22,790.39 | $ 68,371.18 | $ 22,972,717 | $ 31.20 | $ 5.70 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 22,790.39** | **$ 68,371.18** | **$ 22,972,717** | **$ 31.20** | **$ 5.70** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 23,022** | **$ 69,065** | **$ 23,205,877** | **$ 31.52** | **$ 5.76** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5

Date: 3/18/2023
By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 2354 | gpm |
| Feed Rate | 8910 | lpm |
| Precip Mass Flow | 0.02 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 0.00 | g/L of Feed |
| Oxalic Acid Dose | 8.73 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.01 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 11,769 | gal | $ 15,203 | $ 60,812 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 6 | HP | $ 43,529 | $ 174,116 | $ - | |
| 3 | Process Pump | 3 | 100% | 2,354 | gpm | $ 125,177 | $ 375,530 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 0.10 | ft^3 | $ 1,754 | $ 1,754 | $ - | |
| 6 | Roaster | 1 | 100% | 0.03 | MBTU/hr | $ 174,539 | $ 174,539 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 786,753** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 23.54 | Effiicency: | 0.85 | | $ 11.28 | $ 33.85 | $ 11,374 | $ 0.02 | $ 0.00 |
| 3 | Process Pump | 176.54 | | | | $ 84.63 | $ 253.88 | $ 85,303 | $ 0.12 | $ 0.02 |
| 4 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.02 | | | | $ 0.01 | $ 0.02 | $ 8 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.00 | | | | $ 0.00 | $ 0.01 | $ 2 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **200** | | | | **$ 95.92** | **$ 287.76** | **$ 96,687** | **$ 0.13** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 2 | Oxalic Acid | 4,668.76 | kg/hr | $ 0.600 | / kg | $ 22,410.05 | $ 67,230.14 | $ 22,589,329 | $ 30.68 | $ 5.60 |
| 3 | Natural Gas | 0.03 | MCF/hr | $ 2.860 | / MCF | $ 0.73 | $ 2.20 | $ 738 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 22,410.78** | **$ 67,232.34** | **$ 22,590,067** | **$ 30.69** | **$ 5.60** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 22,507** **$ 67,520** **$ 22,686,754** **$ 30.82** **$ 5.63**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8198 | gpm |
| Raffinate Rate: | 8198 | gpm |
| Extraction Feed (O+A) | 16396 | gpm |
| Scrub Feed (O+A) | 225 | gpm |
| Strip Feed (O+A) | 451 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 225 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 269541 | gal |
| Precip Mixing Intensity | 0.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 60.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,396 | gal | $ 94,751 | $ 189,502 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 41 | HP | $ 130,076 | $ 260,153 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 245,933 | gal | $ 434,527 | $ 869,053 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,198 | gpm | $ 201,469 | $ 402,939 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 225 | gal | $ 8,502 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,592 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,381 | gal | $ 38,989 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 451 | gal | $ 12,555 | $ 25,109 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 17,137 | $ 34,273 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,762 | gal | $ 57,576 | $ 115,152 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,503 | $ 19,503 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 58,669 | kg | $ 3,673,201 | $ - | $ 3,673,201 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 1,017,774 | kg | $ 1,102,092 | $ - | $ 1,102,092 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,697 | gal | $ 5,632 | $ 22,526 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 1.35 | HP | $ 18,962 | $ 75,847 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,503 | $ 39,005 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.02 | ft^3 | $ 457 | $ 457 | $ - | |
| 20 | Process Pump | 2 | 100% | 225 | gpm | $ 51,160 | $ 102,320 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,155,838** | **$ 4,775,293** | |

## Power/Utilities

Power Cost: $ 0.07 $/kWhr
Effiicency: 0.85

| ID | Unit Operation | Total HP | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 81.98 | $ 39.30 | $ 117.89 | $ 39,611 | $ 0.05 | $ 0.01 |
| 3 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 409.89 | $ 196.48 | $ 589.44 | $ 198,053 | $ 0.27 | $ 0.05 |
| 5 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.25 | $ 1.08 | $ 3.24 | $ 1,089 | $ 0.00 | $ 0.00 |
| 11 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.45 | $ 0.22 | $ 0.65 | $ 217 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 5.39 | $ 2.59 | $ 7.76 | $ 2,606 | $ 0.00 | $ 0.00 |
| 17 | Thickener | - | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.90 | $ 0.43 | $ 1.29 | $ 434 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.27 | $ 5.40 | $ 16.21 | $ 5,445 | $ 0.01 | $ 0.00 |
| | **TOTAL:** | **512** | **$ 245.49** | **$ 736.48** | **$ 247,457** | **$ 0.34** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 106.13 | kg/hr | $ 0.940 | / kg | $ 798.09 | $ 2,394.28 | $ 804,477 | $ 1.09 | $ 0.20 |
| 2 | D2EHPA | 5.59 | kg/hr | $ 5.000 | / kg | $ 223.43 | $ 670.29 | $ 225,218 | $ 0.31 | $ 0.06 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.86 | mt/hr | $ 270.000 | / mt | $ 1,858.90 | $ 5,576.70 | $ 1,873,772 | $ 2.55 | $ 0.46 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,880.42** | **$ 8,641.27** | **$ 2,903,466** | **$ 3.94** | **$ 0.72** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,126** **$ 9,378** **$ 3,150,923** **$ 4.28** **$ 0.78**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 2353.86 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 8910.32 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 2350.92 | gpm | HCL Dose | 0.00 | g/L of Feed |
| Mn Feed Rate | 8899.20 | lpm | | | |
| Co Precip Mass Flow | 2.50 | tph | | | |
| Mn Precip Mass Flow | 1.55 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 8.67 | g/L of Feed | | | |
| NaOH Dose | 4.51 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 35,308 | gal | $ 43,133 | $ 86,266 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 53 | HP | $ 150,304 | $ 300,608 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 2,354 | gpm | $ 125,176 | $ 125,176 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 11.79 | ft^3 | $ 53,988 | $ 53,988 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 35,264 | gal | $ 43,082 | $ 86,164 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 53 | HP | $ 150,198 | $ 300,396 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 2,351 | gpm | $ 125,117 | $ 125,117 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 7 | ft^3 | $ 38,266 | $ 38,266 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,115,980** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 105.92 | Effiicency: | 0.85 | | $ 50.77 | $ 152.32 | $ 51,181 | $ 0.07 | $ 0.01 |
| 3 | Process Pump | 58.85 | | | | $ 28.21 | $ 84.62 | $ 28,434 | $ 0.04 | $ 0.01 |
| 4 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 1.98 | | | | $ 0.95 | $ 2.85 | $ 957 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 105.79 | | | | $ 50.71 | $ 152.13 | $ 51,117 | $ 0.07 | $ 0.01 |
| 8 | Process Pump | 58.77 | | | | $ 28.17 | $ 84.52 | $ 28,398 | $ 0.04 | $ 0.01 |
| 9 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 1.23 | | | | $ 0.59 | $ 1.77 | $ 593 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **333** | | | | **$ 159.41** | **$ 478.22** | **$ 160,680** | **$ 0.22** | **$ 0.04** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 4.64 | mt/hr | $ 360.000 | / mt | $ 13,353.00 | $ 40,059.00 | $ 13,459,824 | $ 18.28 | $ 3.34 |
| 2 | NaOH | 2.41 | mt/hr | $ 125.000 | / mt | $ 2,408.48 | $ 7,225.43 | $ 2,427,744 | $ 3.30 | $ 0.60 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 15,761.48** | **$ 47,284.43** | **$ 15,887,568** | **$ 21.58** | **$ 3.94** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 15,921** **$ 47,763** **$ 16,048,248** **$ 21.80** **$ 3.98**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | Shakespeare Lignite | Date: | 3/18/2023 |
| Case: | SHXP-C5 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | an |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | 188.1% |
| *(unit conversion)* | 12000 | TPD | Project NPV: | 2,438,692,001 |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 0.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C5 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 101,481,430 | | | | | |
| Constant Dollar Annual OPEX | $ | 465,309,821 | $ | 115.40 | /ton | $ 311.1 | /kg |
| Constant Dollar Annual REE Rev | $ | 786,995,656 | $ | 195.19 | /ton | $ 526.16 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | |
|---|---|---|---|---|
| Payback Period | 3.5 | years | *0.5* | *operating years* |
| Internal Rate of Return | 188% | | | |
| Net Present Value | $ 2,438,692,001 | | | |
| Total Severence Tax Paid | $ - | | | |
| Total State and Federal Tax Paid | $ 1,598,200,025 | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C5 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 1,250 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 2.98 | 37.24 | 16.89 | | | | | | | 16.89 |
| Yttrium | 17.25 | 215.68 | 97.83 | | | | | | | 97.83 |
| Lanthanum | 15.45 | 193.09 | 87.59 | | | | | | | 87.59 |
| Cerium | 31.48 | 393.53 | 178.50 | | | | | | | 178.50 |
| Praseodymium | 3.73 | 46.64 | 21.15 | | | | | | | 21.15 |
| Neodymium | 14.95 | 186.84 | 84.75 | | | | | | | 84.75 |
| Samarium | 3.70 | 46.29 | 21.00 | | | | | | | 21.00 |
| Europium | 0.72 | 9.02 | 4.09 | | | | | | | 4.09 |
| Gadolinium | 3.52 | 43.96 | 19.94 | | | | | | | 19.94 |
| Terbium | 0.43 | 5.42 | 2.46 | | | | | | | 2.46 |
| Dysprosium | 2.20 | 27.55 | 12.50 | | | | | | | 12.50 |
| Holmium | 0.47 | 5.89 | 2.67 | | | | | | | 2.67 |
| Erbium | 0.74 | 9.28 | 4.21 | | | | | | | 4.21 |
| Thulium | 0.21 | 2.61 | 1.18 | | | | | | | 1.18 |
| Ytterbium | 1.69 | 21.14 | 9.59 | | | | | | | 9.59 |
| Lutetium | 0.47 | 5.82 | 2.64 | | | | | | | 2.64 |
| Manganese | -- | 465.70 | 211.24 | | | 0.00 | -- | | | 211.24 |
| Cobalt | -- | 162.20 | 73.57 | | | 0.00 | -- | | | 73.57 |
| Nickel | -- | 486.70 | 220.76 | | | 0.00 | -- | | | 220.76 |
| Zinc | -- | 133.60 | 60.60 | | | 0.00 | -- | | | 60.60 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 1250.00 | 566.99 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 567.0 |
| Total non-REE CM | -- | 1248.20 | 566.17 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 566.2 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 16.89 | 0.00 | 14.63 | 0.01 | 0.00 | 0.00 | 14.65 | 2.25 | 86.7% | |
| Yttrium | 97.83 | 10.10 | 0.00 | 26.84 | 8.49 | 0.00 | 45.43 | 52.40 | 46.4% | |
| Lanthanum | 87.59 | 9.16 | 0.00 | 18.18 | 9.82 | 0.00 | 37.15 | 50.43 | 42.4% | |
| Cerium | 178.50 | 26.63 | 0.00 | 0.31 | 6.46 | 0.00 | 33.39 | 145.11 | 18.7% | |
| Praseodymium | 21.15 | 3.04 | 0.00 | 0.28 | 0.35 | 0.00 | 3.67 | 17.49 | 17.3% | |
| Neodymium | 84.75 | 13.49 | 0.00 | 4.18 | 9.04 | 0.00 | 26.72 | 58.03 | 31.5% | |
| Samarium | 21.00 | 3.15 | 0.00 | 0.81 | 1.68 | 0.00 | 5.65 | 15.35 | 26.9% | |
| Europium | 4.09 | 0.68 | 0.00 | 0.01 | 0.22 | 0.00 | 0.90 | 3.19 | 22.0% | |
| Gadolinium | 19.94 | 3.29 | 0.00 | 1.01 | 1.74 | 0.00 | 6.04 | 13.90 | 30.3% | |
| Terbium | 2.46 | 0.34 | 0.00 | 0.02 | 0.03 | 0.00 | 0.38 | 2.08 | 15.5% | |
| Dysprosium | 12.50 | 2.01 | 0.00 | 1.29 | 1.33 | 0.00 | 4.62 | 7.87 | 37.0% | |
| Holmium | 2.67 | 0.46 | 0.00 | 0.51 | 0.20 | 0.00 | 1.17 | 1.50 | 43.9% | |
| Erbium | 4.21 | 0.81 | 0.00 | 1.50 | 0.58 | 0.00 | 2.90 | 1.31 | 68.8% | |
| Thulium | 1.18 | 0.10 | 0.00 | 0.08 | 0.08 | 0.00 | 0.26 | 0.92 | 21.9% | |
| Ytterbium | 9.59 | 0.56 | 0.00 | 1.05 | 0.40 | 0.00 | 2.01 | 7.58 | 21.0% | |
| Lutetium | 2.64 | 0.09 | 0.00 | 0.34 | 0.11 | 0.00 | 0.54 | 2.10 | 20.4% | |
| Manganese | 211.24 | 0.02 | 0.00 | 93.91 | 8.93 | 0.00 | 102.86 | 108.37 | 48.7% | |
| Cobalt | 73.57 | 0.04 | 0.00 | 4.19 | 36.36 | 0.00 | 40.59 | 32.98 | 55.2% | |
| Nickel | 220.76 | 0.11 | 0.00 | 6.20 | 85.03 | 0.00 | 91.34 | 129.42 | 41.4% | |
| Zinc | 60.60 | 0.05 | 0.00 | 0.18 | 27.29 | 0.00 | 27.52 | 33.08 | 45.4% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 566.99 | 73.92 | 14.63 | 56.41 | 40.53 | 0.00 | 185.48 | 381.51 | 32.7% | <---- |
| Total non-REE CM | 566.17 | 0.23 | 0.00 | 104.47 | 157.62 | 0.00 | 262.31 | 303.86 | 46.3% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 422 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 29 | ton | $ | 1.50 |
| Dust/Volatiles | - | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | |
|---|---|
| 1.77 | TPD REE |
| 0.351 | TPD Sc |
| 6.29 | TPD CM |
| 0.13 | REE |
| 86.6% | Sc |
| 44.5% | Mn |
| 41.9% | Metal-S |
| 500 | Feed |
| 1,250 | Grade |
| 3.5 | PP |
| 188% | IRR |
| 2438692001 | NPV |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C5 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 0.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 2.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | Based on Design |
| Dissolved Solids in Leach | 15.7% | % | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Leaching Retention Time | 45 | min | Based on Design |
| Thickener UF % Solids | 45% | % | VT Override from 7.9% \| SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Residue Settling Rate | 15.0 | ft/hr | Not Available |
| **Reagent Use** | | | |
| Acid Consumption | 58.1 | kg/ton | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Bioleaching Sol'n Consumption | 5487 | L/ton | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Polymer Dosage | 0.05 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 7.81 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | Based on Design |
| Precip Solid Yield | 14.03 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Precip Settling Rate | 2.24 | ft/hr | Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 1.9% | % | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| **Reagent Use** | | | |
| NaOH Dose | 49.76 | g/L of Feed | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 287.91 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 1250 | ppm | |
| Grade After Sorter | 1250 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 500 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 0 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 500 | tph | |
| Leaching Residue Rate | 421.5 | tph | |
| Slurry Flow (feed) | 9250 | gpm | |
| Product Flow | 8,198 | gpm | |
| Thickener Feed % Solids (eff) | 17% | % | |
| Liquid Density | 1.014 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 39.18 | g/L | |
| Leaching Vessel Size | 52031.25 | gal | |
| Leaching Vessel Size | 196.96 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 210.75 | tph | 70 to 200 tph |
| Thickener Area | 1646.69 | ft^2 | |
| Thickener Diameter | 45.80 | ft | ideal < 100' |
| Liquid Balance | 195.97 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,198 | gpm | |
| Slurry Feed Rate | 31,032 | lpm | |
| Precip Mass Flow | 28.80 | tph | |
| Thickener Feed % Solids (eff) | 1.38% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.004 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 349.11 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 135.67 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 33.83 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 2,073 | ft^2 | |
| Thickener Diameter | 51.38 | ft | ideal < 100' |

*PM5: REE Precipitation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 6.0 | -- | VT: Process Design |

**Process Parameters**

| | | | |
|---|---|---|---|
| Precip Conditioning Time | 45 | min | UK: Based on Design |
| Precip Solid Yield | 8.82 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 0.044 | ft/hr | UK: Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.94% | % | VT Override from 1.51% \|\| UK: SHXP-C4/SHXP-C4 Samples |

**Reagent Use**

| | | | |
|---|---|---|---|
| NaOH Dose | 5.77 | g/L of Feed | SHXP-C4/SHXP-C4 Mass Balance-F2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |

**Equipment Specifications**

| | | | |
|---|---|---|---|
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 5431.06 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Slurry Feed Rate | 8164 | gpm | |
| Slurry Feed Rate | 30,904 | lpm | |
| Precip Mass Flow | 18.04 | tph | |
| Thickener Feed % Solids (eff) | 0.9% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 0.998 | g/cc | |

**Other Indicators**

| | | | |
|---|---|---|---|
| Conditioning Vessel Size | 231.78 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 85.01 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 21.19 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 16326 | ft^2 | |
| Thickener Diameter | 144.21 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | 27.72 | m/m | target = 6.35 |

*PM6: RE-Redissolution*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 3.0% | % | VT Override from 10.7% \|\| SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Dissolved Solids in Re-Leach | 60% | % | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Leaching Retention Time | 120 | min | Based on Design |
| **Reagent Use** | | | |
| Acid Consumption | 585.0 | kg/ton | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Solid Feed Rate | 18.04 | tph | |
| Slurry Flow (feed) | 2,354 | gpm | |
| Product Flow | 2,354 | gpm | |
| Liquid Density | 1.022 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 267.3 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 3.48 | v/v | target = 3.63 based on data |
| Liquid Balance | 21.19 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 30.6 | g/L | |

*PM7: REE Oxalate Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | Based on Design |
| Precip Solid Yield | 0.0 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Thickener UF %Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| NaOH Dose | 0.00 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Oxalic Acid Dose | 8.73 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 2353.88 | gpm | |
| Slurry Feed Rate | 8910.41 | lpm | |
| Precip Mass Flow | 0.02 | tph | |
| Thickener Feed % Solids (eff) | 0.00% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.01 | tph | |
| Liquid Density | 1.022 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 44.55 | m^3 | |
| Filter Press Area | 0.10 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.02 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 39.51 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.69 | m/m | |

*PM8: Sc SX Separation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | UK: Design |
| Settler Residence Time | 15.00 | min | UK: Design |
| Loading: Number of Stages | 2.00 | - | UK: Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Loading O:A Advance Ratio | 0.03 | v/v | UK: Design |
| Scrubbing: Number of Stages | | - | N/A |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | N/A |
| Scrubbing O:A Advance Ratio | | v/v | N/A |
| Stripping: Number of Stages | 2.00 | - | UK: Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | UK: Design |
| Sc2O3 Grade | 2% | % | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Precip Solid Yield | 0.4 | g/L | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance (SX)** | | | |
| PLS Flow | 8197.8 | gpm | |
| Raffinate Rate: | 8197.8 | gpm | |
| Load Org Recycle | 7972.4 | gpm | |
| Load Org Feed | 8197.8 | gpm | |
| Loag Org Advance | 225.4 | gpm | |
| Scrub Aq Product | 0.0 | gpm | |
| Scrub Aq Recycle | 0.0 | gpm | |
| Scrub Aq Feed | 0.0 | gpm | |
| Scurb Org Product | 225.4 | gpm | |
| Strip Aq Product | 18.0 | gpm | |
| Strip Aq Recycle | 207.4 | gpm | |
| Strip Aq Feed | 225.4 | gpm | |
| Strip Org Product | 225.4 | gpm | |
| Extraction Feed (O+A): | 16395.6 | gpm | |
| Scrub Feed (O+A): | 225.4 | gpm | |
| Strip Feed (O+A): | 450.8 | gpm | |
| **Material Balance (Precip)** | | | |
| Strip Solution Flow | 18.0 | gpm | |
| Solid Density | 3.4 | g/cc | |
| Solid Product Rate | 0.00 | tph | |
| Liquid Density (after precip) | 1.13 | g/cc | |
| Thickener Feed % Solids (eff) | 0.03% | % | |
| **Total Volumes** | | | |
| Total Extraction Volume | 524658 | gal | |
| Total Scrubbing Volume | 0 | gal | |
| Total Stripping Volume | 14425 | gal | |
| Extraction Org Volume | 262328.82 | gal | |
| Scrubbing Org Volume | 0.00 | gal | |
| Stripping Org Volume | 7212.36 | gal | |
| Total Org Volume | 269541.18 | gal | |
| **Other Indicators** | | | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 10.21 | m^3 | |
| Filter Press Area | 0.02 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM9: CM Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 0 | min | N/A |
| Precip Solid Yield | - | g/L | N/A |
| Precip Settlng Rate | 10 | ft/hr | N/A |
| Thickener UF % Solids | 25% | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 7.24 | g/L of Feed | N/A |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 0.00 | min | N/A |
| Thickener Capacity | 0.00 | ft^2/tph | N/A |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 8143 | gpm | |
| Slurry Feed Rate | 30,824 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 0.998 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM10: CM Redissolution*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 2.0% | % | N/A |
| Dissolved Solids in Re-Leach | 100% | % | N/A |
| Leaching Retention Time | 45 | min | N/A |
| **Reagent Use** | | | |
| Acid Consumption | 441 | kg/ton | N/A |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 0.00 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 0 | gpm | |
| Liquid Density | 1.000 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | |
| Releach Conc. Factor | #DIV/0! | v/v | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | - | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

*PM11: CM Recovery*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 4.25 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | | % | Not Applicable |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 2.64 | g/L | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Na2S Dose | 8.67 | g/L of Feed | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| NaOH Dose | 4.51 | g/L of Feed | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| HCL Dose | 0.00 | g/L of Feed | SHXP-C5/SHXP-C5 Mass Balance-V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 2353.86 | gpm | |
| Co Slurry Feed Rate | 8910.32 | lpm | |
| Co Precip Mass Flow | 2.50 | tph | |
| Co Thickener Feed % Solids (eff) | 0.42% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 2350.92 | gpm | |
| Mn Slurry Feed Rate | 8899.20 | lpm | |
| Mn Precip Mass Flow | 1.55 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.26% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.02 | g/cc | |
| Mn Liquid Density | 1.02 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 133.7 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 11.79 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 2.94 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 133.5 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 7.3 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 1.82 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

*PM12: Biooxidation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Requirec | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C5 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM3 | Leaching | $ 6,416,147 | $ - | $ 1,604,037 | $ 7,057,761 | $ 8,786,913 | $ 23,864,858 | IIIIIIIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 3,040,947 | $ - | $ 760,237 | $ 3,345,041 | $ 4,164,577 | $ 11,310,802 | IIIIIIIIIII |
| PM5 | REE Precipitation | $ 8,673,320 | $ - | $ 2,168,330 | $ 9,540,651 | $ 11,878,111 | $ 32,260,412 | IIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIIII |
| PM6 | RE-Redissolution | $ 1,347,124 | $ - | $ 336,781 | $ 1,481,837 | $ 1,844,887 | $ 5,010,629 | IIIIII |
| PM7 | REE Oxalate Precipitation | $ 786,753 | $ - | $ 196,688 | $ 865,428 | $ 1,077,458 | $ 2,926,327 | III |
| PM8 | Sc SX Separation | $ 2,155,838 | $ 4,775,293 | $ 538,960 | $ 2,371,422 | $ 2,952,421 | $ 12,793,934 | IIIIIIIIIIII |
| PM9 | CM Precipitation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM10 | CM Redissolution | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM11 | CM Recovery | $ 1,115,980 | $ - | $ 278,995 | $ 1,227,578 | $ 1,528,335 | $ 4,150,888 | IIIII |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | $ - | |
| | TOTAL | $ 25,999,768 | $ 4,775,293 | $ 6,499,942 | $ 28,599,745 | $ 35,606,682 | $ 101,481,430 | |

## Module Summary

| Module | Type | % of Base | Typical |
|---|---|---|---|
| Piping, Materials and Labor | Other Direct | 30% | 7 to 25% |
| Electrical, Materials and Labor | Other Direct | 10% | 10% to 25% |
| Instrumentation | Other Direct | 10% | 10% to 20% |
| Plant Services | Other Direct | 30% | 20 to 50% |
| Process Buildings | Other Direct | 20% | 20 to 30% |
| Site Improvements | Other Direct | 10% | 3 to 18% |
| Equipment Installation | Installation | 25% | 17 to 25% |
| Contractor's Fee | Indirect | 24% | 23 to 47% |
| Engineering | Indirect | 52% | 28 to 52% |
| Project Management and Construction | Indirect | 12% | 12 to 24% |
| Contingency | Indirect | 50% | 20 to 50% |
| Total Lang Factor (=1+sum % Base) | | 372% | |

Year of Construction: 2025

# OPERATING COST ESTIMATE

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C5 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 2.05 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 0.79 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 101,481,430 | $ Total | $ 1,006.76 | $ 3,020.28 | $ 1,014,814 | $ 0.68 | $ 0.25 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 0.21 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 786,995,656 | $ Total | $ 3,903.75 | $ 11,711.24 | $ 3,934,978 | $ 2.63 | $ 0.98 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 0.31 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 101,481,430 | $ Total | $ 1,006.76 | $ 3,020.28 | $ 1,014,814 | $ 0.68 | $ 0.25 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 10,898.72** | **$ 32,696.16** | **$ 10,985,908** | **$ 7.34** | **$ 2.72** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 0.34 | $ 0.13 |
| PM2 | Roasting | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | PR | 1300 | | $ 623.25 | $ 1,869.74 | $ 628,231 | $ 0.42 | $ 0.16 |
| PM4 | Al Precipitation | PR | 829 | | $ 397.44 | $ 1,192.32 | $ 400,621 | $ 0.27 | $ 0.10 |
| PM5 | REE Precipitation | PR | 874 | | $ 418.80 | $ 1,256.40 | $ 422,150 | $ 0.28 | $ 0.10 |
| PM6 | RE-Redissolution | PR | 483 | | $ 231.31 | $ 693.93 | $ 233,160 | $ 0.16 | $ 0.06 |
| PM7 | REE Oxalate Precipitation | PR | 200 | | $ 95.92 | $ 287.76 | $ 96,687 | $ 0.06 | $ 0.02 |
| **Sub-Total** | | | **5595.6** | | **$ 2,682.29** | **$ 8,046.88** | **$ 2,703,751.09** | **$ 1.81** | **$ 0.67** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.07 | $ 0.03 |
| PM2 | Roasting | CR | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | CR | | $ 219,847.57 | $ 659,542.70 | $ 221,606,349 | $ 148.16 | $ 54.96 |
| PM4 | Al Precipitation | CR | | $ 92,650.53 | $ 277,951.59 | $ 93,391,734 | $ 62.44 | $ 23.16 |
| PM5 | REE Precipitation | CR | | $ 10,700.25 | $ 32,100.76 | $ 10,785,856 | $ 7.21 | $ 2.68 |
| PM6 | RE-Redissolution | CR | | $ 22,790.39 | $ 68,371.18 | $ 22,972,717 | $ 15.36 | $ 5.70 |
| PM7 | REE Oxalate Precipitation | CR | | $ 22,410.78 | $ 67,232.34 | $ 22,590,067 | $ 15.10 | $ 5.60 |
| **Sub-Total** | | | | **$ 387,142.22** | **$ 1,161,426.67** | **$ 390,239,362.46** | **$ 260.90** | **$ 96.79** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 10,116 | $ 0.10 | $ 337.20 | $ 1,011.60 | $ 339,898 | $ 0.23 | $ 0.08 |
| Non-Hazardous Precipitate | WD | Tons | 691 | $ 1.50 | $ 345.55 | $ 1,036.65 | $ 348,316 | $ 0.23 | $ 0.09 |
| Dust/Volatiles | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 682.75** | **$ 2,048.25** | **$ 688,213.62** | **$ 0.46** | **$ 0.17** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 390,507** | **$ 1,171,522** | **$ 393,631,327** | **$ 263** | **$ 98** |

## Overhead Costs

| Allocation | Code | Rate | Basis | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% | % of Direct Cost | $ 32,696 Daily Direct Cost | $ 1,634.81 | $ 4,904.42 | $ 1,647,886 | $ 1.10 | $ 0.41 |
| Variable Portion | OH | 15% | % of Direct Cost | $ 1,171,522 Daily Direct Cost | $ 58,576.09 | $ 175,728.27 | $ 59,044,699 | $ 39.48 | $ 14.64 |
| **TOTAL OVERHEAD COSTS** | | | | | **$ 60,210.90** | **$ 180,632.69** | **$ 60,692,585.35** | **$ 40.58** | **$ 15.05** |

## OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | SHXP-C5 | | | By: | ks |

### Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 10,898.72 | $ 32,696.16 | $ 10,985,908 | $ 7 | $ 2.72 | I |
| Power | $ 2,682.29 | $ 8,046.88 | $ 2,703,751 | $ 2 | $ 0.67 | |
| Consumables | $ 387,142.22 | $ 1,161,426.67 | $ 390,239,362 | $ 261 | $ 96.79 | IIII

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | Shakespeare Lignite | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | SHXP-C5 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 786,995,656

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 1213 | 1250 |
| Contained Value | ($/ton) | $ 30.6 | $ 248.3 |
| Feed Basket Price | ($/kg) | $ 27.8 | $ 218.9 |
| Product Basket Price | ($/kg) | $ 26.1 | $ 532.8 |
| Production Rate | (kg/day) | 4100.06 | 4451.58 |
| | (t/day) | 4.10 | 4.45 |
| | (t/yr) | 1377.62 | 1495.73 |
| | (% of US Demand) | 13.8% | |

## Revenue Contribution Table

| | | Product Revenue ($/hr) | | | | | | |
|---|---|---|---|---|---|---|---|---|
| **Element** | **REE/CM Price ($/kg) [1]** | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | **REE Revenue ($/day)** | |
| Scandium | $ 4,200.00 | $ 2.2 | $ 94,264.4 | $ - | $ - | $ - | $ 2,262,398 | IIII

# PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C5 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 859,971,402 | $ 885,770,544 | $ 912,343,660 | $ 939,713,970 | $ 967,905,389 | $ 996,942,551 | $ 1,026,850,827 | $ 1,057,656,352 | $ 1,089,386,043 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 859,971,402 | $ 885,770,544 | $ 912,343,660 | $ 939,713,970 | $ 967,905,389 | $ 996,942,551 | $ 1,026,850,827 | $ 1,057,656,352 | $ 1,089,386,043 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 304,444 | $ 2,173,772 | $ 511,442,596 | $ 526,443,781 | $ 541,887,426 | $ 557,786,351 | $ 574,153,732 | $ 591,003,112 | $ 608,348,409 | $ 626,203,928 | $ 644,584,367 |
| *- Operating Costs* | $ - | $ - | $ - | $ 508,456,605 | $ 523,710,303 | $ 539,421,612 | $ 555,604,260 | $ 572,272,388 | $ 589,440,560 | $ 607,123,777 | $ 625,337,490 | $ 644,097,615 |
| *- Loan #1 Interest* | $ - | $ 304,444 | $ 281,347 | $ 256,863 | $ 230,911 | $ 203,401 | $ 174,241 | $ 143,331 | $ 110,567 | $ 75,837 | $ 39,023 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 1,892,426 | $ 1,748,851 | $ 1,596,662 | $ 1,435,342 | $ 1,264,343 | $ 1,083,083 | $ 890,948 | $ 687,285 | $ 471,402 | $ 242,566 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 980,277 | $ 905,905 | $ 827,071 | $ 743,507 | $ 654,929 | $ 561,037 | $ 461,511 | $ 356,014 | $ 244,186 |
| Gross Profit | $ - | $ (304,444) | $ (2,173,772) | $ 348,528,806 | $ 359,326,763 | $ 370,456,234 | $ 381,927,619 | $ 393,751,657 | $ 405,939,439 | $ 418,502,418 | $ 431,452,424 | $ 444,801,676 |
| **Book Deductions** | $ - | $ 380,555 | $ 2,746,418 | $ 3,975,854 | $ 3,677,665 | $ 3,401,841 | $ 3,146,702 | $ 2,910,700 | $ 2,692,397 | $ 2,490,468 | $ 2,303,682 | $ 2,130,906 |
| *- CAPEX #1 Depreciation* | $ - | $ 380,555 | $ 352,014 | $ 325,613 | $ 301,192 | $ 278,602 | $ 257,707 | $ 238,379 | $ 220,501 | $ 203,963 | $ 188,666 | $ 174,516 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 2,394,404 | $ 2,214,824 | $ 2,048,712 | $ 1,895,059 | $ 1,752,929 | $ 1,621,460 | $ 1,499,850 | $ 1,387,361 | $ 1,283,309 | $ 1,187,061 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,435,418 | $ 1,327,762 | $ 1,228,180 | $ 1,136,066 | $ 1,050,861 | $ 972,047 | $ 899,143 | $ 831,707 | $ 769,329 |
| Taxable Before Depletion | $ - | $ (685,000) | $ (4,920,190) | $ 344,552,951 | $ 355,649,097 | $ 367,054,393 | $ 378,780,916 | $ 390,840,957 | $ 403,247,041 | $ 416,011,950 | $ 429,148,741 | $ 442,670,769 |
| **Depletion** | $ - | $ - | $ - | $ 120,395,996 | $ 124,007,876 | $ 127,728,112 | $ 131,559,956 | $ 135,506,754 | $ 139,571,957 | $ 143,759,116 | $ 148,071,889 | $ 152,514,046 |
| *- 50% Limit* | $ - | $ (342,500) | $ (2,460,095) | $ 172,276,476 | $ 177,824,549 | $ 183,527,197 | $ 189,390,458 | $ 195,420,479 | $ 201,623,521 | $ 208,005,975 | $ 214,574,371 | $ 221,335,385 |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 120,395,996 | $ 124,007,876 | $ 127,728,112 | $ 131,559,956 | $ 135,506,754 | $ 139,571,957 | $ 143,759,116 | $ 148,071,889 | $ 152,514,046 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (685,000) | $ (4,920,190) | $ 224,156,955 | $ 231,641,221 | $ 239,326,281 | $ 247,220,960 | $ 255,334,203 | $ 263,675,084 | $ 272,252,834 | $ 281,076,852 | $ 290,156,723 |
| *- Income Tax* | $ - | $ (178,100) | $ (1,279,249) | $ 58,280,808 | $ 60,226,718 | $ 62,224,833 | $ 64,277,450 | $ 66,386,893 | $ 68,555,522 | $ 70,785,737 | $ 73,079,982 | $ 75,440,748 |
| Net Profit/Loss | $ - | $ (506,900) | $ (3,640,941) | $ 165,876,147 | $ 171,414,504 | $ 177,101,448 | $ 182,943,511 | $ 188,947,310 | $ 195,119,562 | $ 201,467,098 | $ 207,996,870 | $ 214,715,975 |
| **Book Deductions** | $ - | $ 380,555 | $ 2,746,418 | $ 124,371,851 | $ 127,685,542 | $ 131,129,953 | $ 134,706,658 | $ 138,417,454 | $ 142,264,354 | $ 146,249,583 | $ 150,375,572 | $ 154,644,952 |
| *+ CAPEX #1 Depreciation* | $ - | $ 380,555 | $ 352,014 | $ 325,613 | $ 301,192 | $ 278,602 | $ 257,707 | $ 238,379 | $ 220,501 | $ 203,963 | $ 188,666 | $ 174,516 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 2,394,404 | $ 2,214,824 | $ 2,048,712 | $ 1,895,059 | $ 1,752,929 | $ 1,621,460 | $ 1,499,850 | $ 1,387,361 | $ 1,283,309 | $ 1,187,061 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 1,435,418 | $ 1,327,762 | $ 1,228,180 | $ 1,136,066 | $ 1,050,861 | $ 972,047 | $ 899,143 | $ 831,707 | $ 769,329 |
| *+ Depletion* | $ - | $ - | $ - | $ 120,395,996 | $ 124,007,876 | $ 127,728,112 | $ 131,559,956 | $ 135,506,754 | $ 139,571,957 | $ 143,759,116 | $ 148,071,889 | $ 152,514,046 |
| Cash Flow | $ - | $ (126,344) | $ (894,523) | $ 290,247,998 | $ 299,100,045 | $ 308,231,401 | $ 317,650,169 | $ 327,364,764 | $ 337,383,917 | $ 347,716,681 | $ 358,372,442 | $ 369,360,928 |
| **Captial Cost** | $ 5,074,072 | $ 31,925,388 | $ 19,138,907 | $ 4,208,549 | $ 4,461,062 | $ 4,728,726 | $ 5,012,449 | $ 5,313,196 | $ 5,631,988 | $ 5,969,907 | $ 6,328,102 | $ 6,018,384 |
| *- Fixed Capital Costs* | $ 10,148,143 | $ 63,080,857 | $ 32,675,884 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 5,074,072 | $ 31,540,429 | $ 16,337,942 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 384,959 | $ 408,057 | $ 432,540 | $ 458,493 | $ 486,002 | $ 515,163 | $ 546,072 | $ 578,837 | $ 613,567 | $ 650,381 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 2,392,908 | $ 2,536,482 | $ 2,688,671 | $ 2,849,992 | $ 3,020,991 | $ 3,202,251 | $ 3,394,386 | $ 3,598,049 | $ 3,813,932 | $ 4,042,768 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 1,239,526 | $ 1,313,898 | $ 1,392,732 | $ 1,476,296 | $ 1,564,873 | $ 1,658,766 | $ 1,758,292 | $ 1,863,789 | $ 1,975,617 |
| **Net Cash Flow** | $ (5,074,072) | $ (32,051,732) | $ (20,033,430) | $ 286,039,448 | $ 294,638,983 | $ 303,502,675 | $ 312,637,720 | $ 322,051,568 | $ 331,751,929 | $ 341,746,773 | $ 352,044,340 | $ 363,342,543 |
| Discounted Cash Flow | $ (5,074,072) | $ (29,137,939) | $ (16,556,554) | $ 214,905,671 | $ 201,242,390 | $ 188,451,282 | $ 176,475,842 | $ 165,263,377 | $ 154,764,723 | $ 144,933,993 | $ 135,728,333 | $ 127,349,345 |
| **Cumulative DCF** | $ (5,074,072) | $ (34,212,010) | $ (50,768,564) | $ 164,137,108 | $ 365,379,498 | $ 553,830,780 | $ 730,306,622 | $ 895,569,999 | $ 1,050,334,722 | $ 1,195,268,715 | $ 1,330,997,048 | $ 1,458,346,393 |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

## Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 101,481,430 | Payback Period | 3.5 |
| Constant Dollar OPEX | $ 465,309,821 | Internal Rate of Return | 188% |
| Constant Dollar REE Rev | $ 786,995,656 | Net Present Value | $ 2,438,692,001 |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ 1,598,200,025 |

# PRO FORMA CASH FLOW

Feedstock: Shakespeare Lignite | Plant Size: 500 TPH | Date: 3/18/2023
Case: SHXP-C5 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 1,122,067,624** | **$ 1,155,729,653** | **$ 1,190,401,542** | **$ 1,226,113,588** | **$ 1,262,896,996** | **$ 1,300,783,906** | **$ 1,339,807,423** | **$ 1,380,001,646** | **$ 1,421,401,695** | **$ 1,464,043,746** | **$ 1,507,965,058** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 1,122,067,624 | $ 1,155,729,653 | $ 1,190,401,542 | $ 1,226,113,588 | $ 1,262,896,996 | $ 1,300,783,906 | $ 1,339,807,423 | $ 1,380,001,646 | $ 1,421,401,695 | $ 1,464,043,746 | $ 1,507,965,058 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 663,546,192** | **$ 683,323,159** | **$ 703,822,854** | **$ 724,937,540** | **$ 746,685,666** | **$ 769,086,236** | **$ 792,158,823** | **$ 815,923,588** | **$ 840,401,295** | **$ 865,613,334** | **$ 891,581,734** |
| *- Operating Costs* | $ 663,420,543 | $ 683,323,159 | $ 703,822,854 | $ 724,937,540 | $ 746,685,666 | $ 769,086,236 | $ 792,158,823 | $ 815,923,588 | $ 840,401,295 | $ 865,613,334 | $ 891,581,734 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 125,649 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ 458,521,432 | $ 472,406,493 | $ 486,578,688 | $ 501,176,049 | $ 516,211,330 | $ 531,697,670 | $ 547,648,600 | $ 564,078,058 | $ 581,000,400 | $ 598,430,412 | $ 616,383,324 |
| **Book Deductions** | **$ 1,971,088** | **$ 1,823,257** | **$ 1,686,512** | **$ 1,560,024** | **$ 1,443,022** | **$ 1,334,796** | **$ 1,234,686** | **$ 1,142,084** | **$ 1,056,428** | **$ 897,166** | **$ 326,341** |
| *- CAPEX #1 Depreciation* | $ 161,427 | $ 149,320 | $ 138,121 | $ 127,762 | $ 118,180 | $ 109,316 | $ 101,118 | $ 93,534 | $ 86,519 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 1,098,031 | $ 1,015,679 | $ 939,503 | $ 869,040 | $ 803,862 | $ 743,573 | $ 687,805 | $ 636,219 | $ 588,503 | $ 544,365 | $ - |
| *- CAPEX #3 Depreciation* | $ 711,630 | $ 658,257 | $ 608,888 | $ 563,221 | $ 520,980 | $ 481,906 | $ 445,763 | $ 412,331 | $ 381,406 | $ 352,801 | $ 326,341 |
| Taxable Before Depletion | $ 456,550,343 | $ 470,583,237 | $ 484,892,176 | $ 499,616,025 | $ 514,768,308 | $ 530,362,875 | $ 546,413,914 | $ 562,935,974 | $ 579,943,972 | $ 597,533,246 | $ 616,056,984 |
| **Depletion** | $ 157,089,467 | $ 161,802,151 | $ 166,656,216 | $ 171,655,902 | $ 176,805,579 | $ 182,109,747 | $ 187,573,039 | $ 193,200,230 | $ 198,996,237 | $ 204,966,124 | $ 211,115,108 |
| *- 50% Limit* | $ 228,275,172 | $ 235,291,618 | $ 242,446,088 | $ 249,808,012 | $ 257,384,154 | $ 265,181,437 | $ 273,206,957 | $ 281,467,987 | $ 289,971,986 | $ 298,766,623 | $ 308,028,492 |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 157,089,467 | $ 161,802,151 | $ 166,656,216 | $ 171,655,902 | $ 176,805,579 | $ 182,109,747 | $ 187,573,039 | $ 193,200,230 | $ 198,996,237 | $ 204,966,124 | $ 211,115,108 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ 299,460,876** | **$ 308,781,085** | **$ 318,235,960** | **$ 327,960,122** | **$ 337,962,729** | **$ 348,253,128** | **$ 358,840,875** | **$ 369,735,743** | **$ 380,947,734** | **$ 392,567,121** | **$ 404,941,875** |
| *- Income Tax* | $ 77,859,828 | $ 80,283,082 | $ 82,741,350 | $ 85,269,632 | $ 87,870,309 | $ 90,545,813 | $ 93,298,628 | $ 96,131,293 | $ 99,046,411 | $ 102,067,452 | $ 105,284,888 |
| Net Profit/Loss | $ 221,601,048 | $ 228,498,003 | $ 235,494,610 | $ 242,690,491 | $ 250,092,419 | $ 257,707,314 | $ 265,542,248 | $ 273,604,450 | $ 281,901,324 | $ 290,499,670 | $ 299,656,988 |
| **Book Deductions** | **$ 159,060,556** | **$ 163,625,408** | **$ 168,342,728** | **$ 173,215,926** | **$ 178,248,602** | **$ 183,444,542** | **$ 188,807,725** | **$ 194,342,315** | **$ 200,052,665** | **$ 205,863,290** | **$ 211,441,449** |
| *+ CAPEX #1 Depreciation* | $ 161,427 | $ 149,320 | $ 138,121 | $ 127,762 | $ 118,180 | $ 109,316 | $ 101,118 | $ 93,534 | $ 86,519 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 1,098,031 | $ 1,015,679 | $ 939,503 | $ 869,040 | $ 803,862 | $ 743,573 | $ 687,805 | $ 636,219 | $ 588,503 | $ 544,365 | $ - |
| *+ CAPEX #3 Depreciation* | $ 711,630 | $ 658,257 | $ 608,888 | $ 563,221 | $ 520,980 | $ 481,906 | $ 445,763 | $ 412,331 | $ 381,406 | $ 352,801 | $ 326,341 |
| *+ Depletion* | $ 157,089,467 | $ 161,802,151 | $ 166,656,216 | $ 171,655,902 | $ 176,805,579 | $ 182,109,747 | $ 187,573,039 | $ 193,200,230 | $ 198,996,237 | $ 204,966,124 | $ 211,115,108 |
| Cash Flow | $ 380,661,604 | $ 392,123,411 | $ 403,837,339 | $ 415,906,417 | $ 428,341,021 | $ 441,151,857 | $ 454,349,973 | $ 467,946,765 | $ 481,953,989 | $ 496,362,960 | $ 511,098,437 |
| **Captial Cost** | **$ 2,094,154** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 2,094,154 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ 378,567,450 | $ 392,123,411 | $ 403,837,339 | $ 415,906,417 | $ 428,341,021 | $ 441,151,857 | $ 454,349,973 | $ 467,946,765 | $ 481,953,989 | $ 496,362,960 | $ 511,098,437 |
| Discounted Cash Flow | $ 120,623,256 | $ 113,584,185 | $ 106,342,993 | $ 99,564,689 | $ 93,219,486 | $ 87,279,543 | $ 81,718,836 | $ 76,513,035 | $ 71,639,389 | $ 67,073,810 | $ 62,786,385 |
| **Cumulative DCF** | $ 1,578,969,649 | $ 1,692,553,834 | $ 1,798,896,826 | $ 1,898,461,516 | $ 1,991,681,002 | $ 2,078,960,545 | $ 2,160,679,381 | $ 2,237,192,417 | $ 2,308,831,806 | $ 2,375,905,616 | $ 2,438,692,001 |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

## Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 101,481,430 | Payback Period | 0 | 3.5 |
| Constant Dollar OPEX | $ 465,309,821 | Internal Rate of Return | 0 | 188% |
| Constant Dollar REE Rev | $ 786,995,656 | Net Present Value | 0 | $ 2,438,692,001 |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ 1,598,200,025 |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 0.19 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 0.12 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.01 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 7 | Conveyor | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 0.34** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.07 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.07** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 611** **$ 1,834** **$ 616,370** **$ 0.41** **$ 0.15**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 2.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 500 | tph |
| Residue Rate | 422 | tph |
| Slurry Flow | 9250 | gpm |
| Leaching Time | 45 | min |
| Acid Consumption | 58.11 | kg/ton |
| Bioleaching Sol'n Consumption | 5487 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 2 | -- |
| Thickener Capacity | 7.81 | ft^2/tph |
| Polymer Dose | 0.050 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 8 | 100% | 52,031 | gal | $ 181,404 | $ 1,451,235 | $ - | |
| 2 | Tank Mixer | 8 | 100% | 78 | HP | $ 187,044 | $ 1,496,351 | $ - | |
| 3 | Process Pump | 4 | 100% | 4,625 | gpm | $ 161,957 | $ 647,827 | $ - | |
| 4 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 5 | Belt Filter | 2 | 100% | 211 | TPH | $ 1,186,781 | $ 2,373,561 | $ - | |
| 6 | Solution Heater | 1 | 100% | 1 | MBTU/hr | $ 30,807 | $ 30,807 | $ - | |
| 7 | Thickener | 2 | 100% | 1,647 | ft^2 | $ 160,405 | $ 320,809 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 6,416,147** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 624.38 | Effiicency: | 0.85 | | $ 299.30 | $ 897.89 | $ 301,691 | $ 0.20 | $ 0.07 |
| 3 | Process Pump | 462.50 | | | | $ 221.70 | $ 665.10 | $ 223,475 | $ 0.15 | $ 0.06 |
| 4 | Conveyor | 31.00 | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.01 | $ 0.00 |
| 5 | Belt Filter | 102.21 | | | | $ 49.00 | $ 146.99 | $ 49,388 | $ 0.03 | $ 0.01 |
| 6 | Solution Heater | 78.00 | | | | $ 37.39 | $ 112.16 | $ 37,687 | $ 0.03 | $ 0.01 |
| 7 | Thickener | 2.09 | | | | $ 1.00 | $ 3.01 | $ 1,012 | $ 0.00 | $ 0.00 |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1300** | | | | **$ 623.25** | **$ 1,869.74** | **$ 628,231** | **$ 0.42** | **$ 0.16** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | 2,743,500.00 | L/hr | $ 0.008 | / L | $ 166,804.80 | $ 500,414.40 | $ 168,139,238 | $ 112.41 | $ 41.70 |
| 2 | H2SO4 | 29.05 | mt/hr | $ 220.000 | / mt | $ 51,134.75 | $ 153,404.24 | $ 51,543,824 | $ 34.46 | $ 12.78 |
| 3 | Natural Gas | 0.75 | MCF/hr | $ 2.860 | / MCF | $ 17.21 | $ 51.63 | $ 17,346 | $ 0.01 | $ 0.00 |
| 4 | Polymer | 105.05 | kg/hr | $ 2.250 | / kg | $ 1,890.81 | $ 5,672.44 | $ 1,905,940 | $ 1.27 | $ 0.47 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 219,847.57** | **$ 659,542.70** | **$ 221,606,349** | **$ 148.16** | **$ 54.96** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 220,471** **$ 661,412** **$ 222,234,580** **$ 148.58** **$ 55.12**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 4.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8198 | gpm |
| Feed Rate | 31032 | lpm |
| Precip Mass Flow | 28.80 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 287.91 | ft^2/tpd |
| NaOH Dose | 49.76 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 4 | 100% | 92,225 | gal | $ 166,863 | $ 667,452 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 46 | HP | $ 139,011 | $ 556,043 | $ - | |
| 3 | Process Pump | 12 | 100% | 2,049 | gpm | $ 59,368 | $ 712,417 | $ - | |
| 4 | Filter Press | 1 | 100% | 136 | ft^3 | $ 313,424 | $ 313,424 | $ - | |
| 5 | Conveyor | 1 | 100% | 29 | TPH | $ 19,933 | $ 19,933 | $ - | |
| 6 | Thickener | 4 | 100% | 2,073 | ft^2 | $ 192,919 | $ 771,676 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 3,040,947** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 184.45 | Effiicency: | 0.85 | | $ 88.42 | $ 265.25 | $ 89,124 | $ 0.06 | $ 0.02 |
| 3 | Process Pump | 614.83 | | | | $ 294.72 | $ 884.17 | $ 297,080 | $ 0.20 | $ 0.07 |
| 4 | Filter Press | 22.78 | | | | $ 10.92 | $ 32.76 | $ 11,006 | $ 0.01 | $ 0.00 |
| 5 | Conveyor | 1.79 | | | | $ 0.86 | $ 2.57 | $ 863 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 5.27 | | | | $ 2.53 | $ 7.58 | $ 2,548 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **829** | | | | **$ 397.44** | **$ 1,192.32** | **$ 400,621** | **$ 0.27** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 92.65 | mt/hr | $ 125.000 | / mt | $ 92,650.53 | $ 277,951.59 | $ 93,391,734 | $ 62.44 | $ 23.16 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 92,650.53** | **$ 277,951.59** | **$ 93,391,734** | **$ 62.44** | **$ 23.16** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 93,048** **$ 279,144** **$ 93,792,355** **$ 62.71** **$ 23.26**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 6.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8164 | gpm |
| Feed Rate | 30904 | lpm |
| Precip Mass Flow | 18.04 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 5431.06 | ft^2/tpd |
| NaOH Dose | 5.77 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 6 | 100% | 61,230 | gal | $ 132,531 | $ 795,185 | $ - | |
| 2 | Tank Mixer | 6 | 100% | 31 | HP | $ 110,337 | $ 662,021 | $ - | |
| 3 | Process Pump | 18 | 100% | 1,361 | gpm | $ 50,782 | $ 914,070 | $ - | |
| 4 | Filter Press | 1 | 100% | 85 | ft^3 | $ 223,860 | $ 223,860 | $ - | |
| 5 | Conveyor | 1 | 100% | 18 | TPH | $ 15,417 | $ 15,417 | $ - | |
| 6 | Thickener | 6 | 100% | 16,326 | ft^2 | $ 1,010,461 | $ 6,062,767 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 8,673,320** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 183.69 | Effiicency: | 0.85 | | | $ 88.05 | $ 264.16 | $ 88,756 | $ 0.06 | $ 0.02 |
| 3 | Process Pump | 612.30 | | | | | $ 293.51 | $ 880.52 | $ 295,854 | $ 0.20 | $ 0.07 |
| 4 | Filter Press | 14.27 | | | | | $ 6.84 | $ 20.53 | $ 6,897 | $ 0.00 | $ 0.00 |
| 5 | Conveyor | 1.12 | | | | | $ 0.54 | $ 1.61 | $ 540 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 62.30 | | | | | $ 29.86 | $ 89.59 | $ 30,102 | $ 0.02 | $ 0.01 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **874** | | | | | **$ 418.80** | **$ 1,256.40** | **$ 422,150** | **$ 0.28** | **$ 0.10** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 10.70 | mt/hr | $ 125.000 | / mt | $ 10,700.25 | $ 32,100.76 | $ 10,785,856 | $ 7.21 | $ 2.68 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 10,700.25** | **$ 32,100.76** | **$ 10,785,856** | **$ 7.21** | **$ 2.68** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 11,119** **$ 33,357** **$ 11,208,006** **$ 7.49** **$ 2.78**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5

Date: 3/18/2023
By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 18.04 | tph |
| Slurry Feed Rate | 2354 | gpm |
| Leaching Time | 120 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 585.00 | kg/ton |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 70,616 | gal | $ 83,282 | $ 333,128 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 106 | HP | $ 222,205 | $ 888,819 | $ - | |
| 3 | Process Pump | 1 | 100% | 2,354 | gpm | $ 125,177 | $ 125,177 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,347,124** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 423.70 | Effiicency: | 0.85 | | | $ 203.10 | $ 609.30 | $ 204,726 | $ 0.14 | $ 0.05 |
| 3 | Process Pump | 58.85 | | | | | $ 28.21 | $ 84.63 | $ 28,434 | $ 0.02 | $ 0.01 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **483** | | | | | **$ 231.31** | **$ 693.93** | **$ 233,160** | **$ 0.16** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 10.55 | mt/hr | $ 270.000 | / mt | $ 22,790.39 | $ 68,371.18 | $ 22,972,717 | $ 15.36 | $ 5.70 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 22,790.39** | **$ 68,371.18** | **$ 22,972,717** | **$ 15.36** | **$ 5.70** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| | **Total Operating Cost:** | **$ 23,022** | **$ 69,065** | **$ 23,205,877** | **$ 15.51** | **$ 5.76** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 2354 | gpm |
| Feed Rate | 8910 | lpm |
| Precip Mass Flow | 0.02 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 0.00 | g/L of Feed |
| Oxalic Acid Dose | 8.73 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.01 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 11,769 | gal | $ 15,203 | $ 60,812 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 6 | HP | $ 43,529 | $ 174,116 | $ - | |
| 3 | Process Pump | 3 | 100% | 2,354 | gpm | $ 125,177 | $ 375,530 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 0.10 | ft^3 | $ 1,754 | $ 1,754 | $ - | |
| 6 | Roaster | 1 | 100% | 0.03 | MBTU/hr | $ 174,539 | $ 174,539 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 786,753** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 23.54 | Effiicency: | 0.85 | | $ 11.28 | $ 33.85 | $ 11,374 | $ 0.01 | $ 0.00 |
| 3 | Process Pump | 176.54 | | | | $ 84.63 | $ 253.88 | $ 85,303 | $ 0.06 | $ 0.02 |
| 4 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.02 | | | | $ 0.01 | $ 0.02 | $ 8 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.00 | | | | $ 0.00 | $ 0.01 | $ 2 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **200** | | | | **$ 95.92** | **$ 287.76** | **$ 96,687** | **$ 0.06** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 2 | Oxalic Acid | 4,668.76 | kg/hr | $ 0.600 | / kg | $ 22,410.05 | $ 67,230.14 | $ 22,589,329 | $ 15.10 | $ 5.60 |
| 3 | Natural Gas | 0.03 | MCF/hr | $ 2.860 | / MCF | $ 0.73 | $ 2.20 | $ 738 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 22,410.78** | **$ 67,232.34** | **$ 22,590,067** | **$ 15.10** | **$ 5.60** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 22,507** **$ 67,520** **$ 22,686,754** **$ 15.17** **$ 5.63**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8198 | gpm |
| Raffinate Rate: | 8198 | gpm |
| Extraction Feed (O+A) | 16396 | gpm |
| Scrub Feed (O+A) | 225 | gpm |
| Strip Feed (O+A) | 451 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 225 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 269541 | gal |
| Precip Mixing Intensity | 0.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 60.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,396 | gal | $ 94,751 | $ 189,502 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 41 | HP | $ 130,076 | $ 260,153 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 245,933 | gal | $ 434,527 | $ 869,053 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,198 | gpm | $ 201,469 | $ 402,939 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 225 | gal | $ 8,502 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,592 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,381 | gal | $ 38,989 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 451 | gal | $ 12,555 | $ 25,109 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 17,137 | $ 34,273 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,762 | gal | $ 57,576 | $ 115,152 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,503 | $ 19,503 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 58,669 | kg | $ 3,673,201 | $ - | $ 3,673,201 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 1,017,774 | kg | $ 1,102,092 | $ - | $ 1,102,092 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,697 | gal | $ 5,632 | $ 22,526 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 1.35 | HP | $ 18,962 | $ 75,847 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,503 | $ 39,005 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.02 | ft^3 | $ 457 | $ 457 | $ - | |
| 20 | Process Pump | 2 | 100% | 225 | gpm | $ 51,160 | $ 102,320 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,155,838** | **$ 4,775,293** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 81.98 | Effiicency: | 0.85 | | $ 39.30 | $ 117.89 | $ 39,611 | $ 0.03 | $ 0.01 |
| 3 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 409.89 | | | | $ 196.48 | $ 589.44 | $ 198,053 | $ 0.13 | $ 0.05 |
| 5 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.25 | | | | $ 1.08 | $ 3.24 | $ 1,089 | $ 0.00 | $ 0.00 |
| 11 | Steel Tanks | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.45 | | | | $ 0.22 | $ 0.65 | $ 217 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 5.39 | | | | $ 2.59 | $ 7.76 | $ 2,606 | $ 0.00 | $ 0.00 |
| 17 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.90 | | | | $ 0.43 | $ 1.29 | $ 434 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | | | | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.27 | | | | $ 5.40 | $ 16.21 | $ 5,445 | $ 0.00 | $ 0.00 |
| | **TOTAL:** | **512** | | | | **$ 245.49** | **$ 736.48** | **$ 247,457** | **$ 0.17** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 106.13 | kg/hr | $ 0.940 | / kg | $ 798.09 | $ 2,394.28 | $ 804,477 | $ 0.54 | $ 0.20 |
| 2 | D2EHPA | 5.59 | kg/hr | $ 5.000 | / kg | $ 223.43 | $ 670.29 | $ 225,218 | $ 0.15 | $ 0.06 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.86 | mt/hr | $ 270.000 | / mt | $ 1,858.90 | $ 5,576.70 | $ 1,873,772 | $ 1.25 | $ 0.46 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,880.42** | **$ 8,641.27** | **$ 2,903,466** | **$ 1.94** | **$ 0.72** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,126** **$ 9,378** **$ 3,150,923** **$ 2.11** **$ 0.78**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: Shakespeare Lignite
Case: SHXP-C5
Date: 3/18/2023
By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 2353.86 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 8910.32 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 2350.92 | gpm | HCL Dose | 0.00 | g/L of Feed |
| Mn Feed Rate | 8899.20 | lpm | | | |
| Co Precip Mass Flow | 2.50 | tph | | | |
| Mn Precip Mass Flow | 1.55 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 8.67 | g/L of Feed | | | |
| NaOH Dose | 4.51 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 35,308 | gal | $ 43,133 | $ 86,266 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 53 | HP | $ 150,304 | $ 300,608 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 2,354 | gpm | $ 125,176 | $ 125,176 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 11.79 | ft^3 | $ 53,988 | $ 53,988 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 35,264 | gal | $ 43,082 | $ 86,164 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 53 | HP | $ 150,198 | $ 300,396 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 2,351 | gpm | $ 125,117 | $ 125,117 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 7 | ft^3 | $ 38,266 | $ 38,266 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,115,980** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 105.92 | Effiicency: | 0.85 | | $ 50.77 | $ 152.32 | $ 51,181 | $ 0.03 | $ 0.01 |
| 3 | Process Pump | 58.85 | | | | $ 28.21 | $ 84.62 | $ 28,434 | $ 0.02 | $ 0.01 |
| 4 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 1.98 | | | | $ 0.95 | $ 2.85 | $ 957 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 105.79 | | | | $ 50.71 | $ 152.13 | $ 51,117 | $ 0.03 | $ 0.01 |
| 8 | Process Pump | 58.77 | | | | $ 28.17 | $ 84.52 | $ 28,398 | $ 0.02 | $ 0.01 |
| 9 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 1.23 | | | | $ 0.59 | $ 1.77 | $ 593 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **333** | | | | **$ 159.41** | **$ 478.22** | **$ 160,680** | **$ 0.11** | **$ 0.04** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 4.64 | mt/hr | $ 360.000 | / mt | $ 13,353.00 | $ 40,059.00 | $ 13,459,824 | $ 9.00 | $ 3.34 |
| 2 | NaOH | 2.41 | mt/hr | $ 125.000 | / mt | $ 2,408.48 | $ 7,225.43 | $ 2,427,744 | $ 1.62 | $ 0.60 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 15,761.48** | **$ 47,284.43** | **$ 15,887,568** | **$ 10.62** | **$ 3.94** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 15,921** **$ 47,763** **$ 16,048,248** **$ 10.73** **$ 3.98**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | WK No. 13 | Date: | 3/18/2023 |
| Case: | WK-1 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | an |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | N/A |
| *(unit conversion)* | 12000 | TPD | Project NPV: | (1,701,387,802) |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 22.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | WK No. 13 | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | WK-1 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 154,031,629 | | | | | |
| Constant Dollar Annual OPEX | $ | 269,835,330 | $ | 66.92 | /ton | $ 2,021.4 | /kg |
| Constant Dollar Annual REE Rev | $ | 43,226,968 | $ | 10.72 | /ton | $ 323.82 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | | |
|---|---|---|---|---|---|
| Payback Period | | 25.5 | years | *22.5* | *operating years* |
| Internal Rate of Return | | N/A | | | |
| Net Present Value | $ | (1,701,387,802) | | | |
| Total Severence Tax Paid | $ | - | | | |
| Total State and Federal Tax Paid | $ | - | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-1 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 274 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 6.87 | 18.83 | 8.54 | | | | | | | 8.54 |
| Yttrium | 10.09 | 27.66 | 12.55 | | | | | | | 12.55 |
| Lanthanum | 18.70 | 51.25 | 23.25 | | | | | | | 23.25 |
| Cerium | 37.75 | 103.46 | 46.93 | | | | | | | 46.93 |
| Praseodymium | 3.60 | 9.87 | 4.48 | | | | | | | 4.48 |
| Neodymium | 9.03 | 24.75 | 11.23 | | | | | | | 11.23 |
| Samarium | 3.51 | 9.63 | 4.37 | | | | | | | 4.37 |
| Europium | 0.61 | 1.68 | 0.76 | | | | | | | 0.76 |
| Gadolinium | 3.02 | 8.29 | 3.76 | | | | | | | 3.76 |
| Terbium | 0.50 | 1.37 | 0.62 | | | | | | | 0.62 |
| Dysprosium | 1.77 | 4.85 | 2.20 | | | | | | | 2.20 |
| Holmium | 0.55 | 1.52 | 0.69 | | | | | | | 0.69 |
| Erbium | 1.17 | 3.21 | 1.46 | | | | | | | 1.46 |
| Thulium | 0.22 | 0.60 | 0.27 | | | | | | | 0.27 |
| Ytterbium | 1.91 | 5.25 | 2.38 | | | | | | | 2.38 |
| Lutetium | 0.69 | 1.89 | 0.86 | | | | | | | 0.86 |
| Manganese | -- | 220.90 | 100.20 | | | 0.00 | -- | | | 100.20 |
| Cobalt | -- | 22.30 | 10.12 | | | 0.00 | -- | | | 10.12 |
| Nickel | -- | 48.00 | 21.77 | | | 0.00 | -- | | | 21.77 |
| Zinc | -- | 110.50 | 50.12 | | | 0.00 | -- | | | 50.12 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 274.10 | 124.33 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 124.3 |
| Total non-REE CM | -- | 401.70 | 182.21 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 182.2 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 8.54 | 0.00 | 0.80 | 0.00 | 0.00 | 0.00 | 0.80 | 7.74 | 9.3% | |
| Yttrium | 12.55 | 0.69 | 0.00 | 0.06 | 0.01 | 0.00 | 0.75 | 11.79 | 6.0% | |
| Lanthanum | 23.25 | 3.44 | 0.00 | 0.03 | 0.01 | 0.00 | 3.47 | 19.78 | 14.9% | |
| Cerium | 46.93 | 6.63 | 0.00 | 0.01 | 0.01 | 0.00 | 6.65 | 40.28 | 14.2% | |
| Praseodymium | 4.48 | 0.11 | 0.00 | 0.00 | 0.00 | 0.00 | 0.11 | 4.36 | 2.5% | |
| Neodymium | 11.23 | 3.10 | 0.00 | 0.01 | 0.01 | 0.00 | 3.12 | 8.11 | 27.8% | |
| Samarium | 4.37 | 0.76 | 0.00 | 0.01 | 0.00 | 0.00 | 0.76 | 3.60 | 17.5% | |
| Europium | 0.76 | 0.11 | 0.00 | 0.00 | 0.00 | 0.00 | 0.11 | 0.65 | 14.9% | |
| Gadolinium | 3.76 | 0.47 | 0.00 | 0.01 | 0.00 | 0.00 | 0.48 | 3.27 | 12.9% | |
| Terbium | 0.62 | 0.01 | 0.00 | 0.00 | 0.00 | 0.00 | 0.01 | 0.61 | 2.2% | |
| Dysprosium | 2.20 | 0.15 | 0.00 | 0.00 | 0.00 | 0.00 | 0.15 | 2.05 | 6.9% | |
| Holmium | 0.69 | 0.03 | 0.00 | 0.00 | 0.00 | 0.00 | 0.03 | 0.65 | 5.1% | |
| Erbium | 1.46 | 0.06 | 0.00 | 0.00 | 0.00 | 0.00 | 0.06 | 1.40 | 3.9% | |
| Thulium | 0.27 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.27 | 1.8% | |
| Ytterbium | 2.38 | 0.02 | 0.00 | 0.00 | 0.00 | 0.00 | 0.03 | 2.35 | 1.1% | |
| Lutetium | 0.86 | 0.01 | 0.00 | 0.00 | 0.00 | 0.00 | 0.01 | 0.85 | 0.7% | |
| Manganese | 100.20 | 0.02 | 0.00 | 19.94 | 1.44 | 0.00 | 21.40 | 78.79 | 21.4% | |
| Cobalt | 10.12 | 0.03 | 0.00 | 0.78 | 0.18 | 0.00 | 0.98 | 9.13 | 9.7% | |
| Nickel | 21.77 | 0.08 | 0.00 | 2.14 | 0.27 | 0.00 | 2.49 | 19.28 | 11.4% | |
| Zinc | 50.12 | 0.14 | 0.00 | 0.21 | 5.74 | 0.00 | 6.08 | 44.04 | 12.1% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 124.33 | 15.59 | 0.80 | 0.12 | 0.05 | 0.00 | 16.55 | 107.78 | 13.3% | <---- |
| Total non-REE CM | 182.21 | 0.27 | 0.00 | 23.07 | 7.62 | 0.00 | 30.96 | 151.25 | 17.0% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 388 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 11 | ton | $ | 1.50 |
| Dust/Volatiles | 20 | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | |
|---|---|
| 0.37 | TPD REE |
| 0.019 | TPD Sc |
| 0.74 | TPD CM |
| 0.13 | REE |
| 9.3% | Sc |
| 19.9% | Mn |
| 7.5% | Metal-S |
| 500 | Feed |
| 274 | Grade |
| 25.5 | PP |
| N/A | IRR |
| -1701387802 | NPV |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-1 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 4.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 2.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | Based on Design |
| Dissolved Solids in Leach | 19.2% | % | WK1/WK1 Mass Balance-V2 |
| Leaching Retention Time | 45 | min | Based on Design |
| Thickener UF % Solids | 20% | % | VT Override from 9.3% \| WK1/WK1 Samples |
| Residue Settling Rate | 15.0 | ft/hr | Not Available |
| **Reagent Use** | | | |
| Acid Consumption | 228.5 | kg/ton | WK1/WK1 Mass Balance-V2 |
| Bioleaching Sol'n Consumption | 0 | L/ton | |
| Polymer Dosage | 0.016 | g/L | WK1/WK1 Samples |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Process Design |
| Thickener Capacity | 2.50 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 7.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45.0 | min | Based on Design |
| Precip Solid Yield | 5.58 | g/L | WK1/WK1 Mass Balance-V2 |
| Precip Settling Rate | 2.24 | ft/hr | Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 0.9% | % | VT Override from 0.49% \| WK1/WK1 Samples |
| **Reagent Use** | | | |
| NaOH Dose | 2.38 | g/L of Feed | WK1/WK1 Mass Balance-V2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | |
| Thickener Capacity | 915.35 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 274 | ppm | |
| Grade After Sorter | 274 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 480 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 581 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 480 | tph | |
| Leaching Residue Rate | 387.84 | tph | |
| Slurry Flow (feed) | 8880 | gpm | |
| Product Flow | 7,912 | gpm | |
| Thickener Feed % Solids (eff) | 16% | % | |
| Liquid Density | 1.017 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 47.92 | g/L | |
| Leaching Vessel Size | 49950.00 | gal | |
| Leaching Vessel Size | 189.08 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 193.92 | tph | 70 to 200 tph |
| Thickener Area | 483.89 | ft^2 | |
| Thickener Diameter | 24.83 | ft | ideal < 100' |
| Liquid Balance | 230.07 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 7,912 | gpm | |
| Slurry Feed Rate | 29,949 | lpm | |
| Precip Mass Flow | 11.04 | tph | |
| Thickener Feed % Solids (eff) | 0.55% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.013 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 192.53 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 52.03 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 12.97 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 1,444 | ft^2 | |
| Thickener Diameter | 42.89 | ft | ideal < 100' |

*PM5: REE Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 8.0 | -- | VT: Process Design |
| | | | |
| **Process Parameters** | | | |
| Precip Conditioning Time | 45 | min | Based on Design |
| Precip Solid Yield | 1.28 | g/L | WK1/WK1 Mass Balance-V2 |
| Precip Settling Rate | 0.044 | ft/hr | Lily Banks/Settling Tests |
| Thickener UF % Solids | 2.35% | % | VT Override from 0.14% \| WK1/WK1 Samples |
| | | | |
| **Reagent Use** | | | |
| NaOH Dose | 0.66 | g/L of Feed | WK1/WK1 Mass Balance-V2 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| | | | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 5357.12 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 7899 | gpm | |
| Slurry Feed Rate | 29,900 | lpm | |
| Precip Mass Flow | 2.54 | tph | |
| Thickener Feed % Solids (eff) | 2.00% | % | VT Override from 0.13% |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.013 | g/cc | |
| | | | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 168.19 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 11.97 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 2.98 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 1701 | ft^2 | |
| Thickener Diameter | 46.55 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | 188.99 | m/m | target = 6.35 |

*PM6: RE-Redissolution*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 1.0% | % | VT Override from 10.7% \| WK1/WK1 Mass Balance-V2 |
| Dissolved Solids in Re-Leach | 13% | % | WK1/WK1 Mass Balance-V2 |
| Leaching Retention Time | 120 | min | Based on Design |
| **Reagent Use** | | | |
| Acid Consumption | 1002.2 | kg/ton | WK1/WK1 Mass Balance-V2 |
| | 119.8 | | WK1/WK1 Mass Balance-V2 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| **Material Balance** | | | |
|---|---|---|---|
| Solid Feed Rate | 2.54 | tph | |
| Slurry Flow (feed) | 1,062 | gpm | |
| Product Flow | 1,062 | gpm | |
| Liquid Density | 1.007 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 120.6 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 7.45 | v/v | target = 8.2, based on test data |
| Liquid Balance | 2.98 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 9.6 | g/L | |

*PM7: REE Oxalate Precipitation*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | Based on Design |
| Precip Solid Yield | 0.1 | g/L | WK1/WK1 Mass Balance-V2 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Thickener UF %Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| NaOH Dose | 1.95 | g/L | WK1/WK1 Mass Balance-V2 |
| Oxalic Acid Dose | 6.30 | g/L | WK1/WK1 Mass Balance-V2 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | |
| Filter Press Cycle Time | 30.00 | min | |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| **Material Balance** | | | |
|---|---|---|---|
| Slurry Feed Rate | 1062.25 | gpm | |
| Slurry Feed Rate | 4021.06 | lpm | |
| Precip Mass Flow | 0.04 | tph | |
| Thickener Feed % Solids (eff) | 0.01% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.02 | tph | |
| Liquid Density | 1.007 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 20.11 | m^3 | |
| Filter Press Area | 0.18 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.04 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 21.36 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.68 | m/m | |

*PM8: Sc SX Separation*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | Based on Design |
| Settler Residence Time | 15.00 | min | Based on Design |
| Loading: Number of Stages | 2.00 | - | Based on Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | Based on Design |
| Loading O:A Advance Ratio | 0.03 | v/v | Based on Design |
| Scrubbing: Number of Stages | 0.00 | - | Not Applicable |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | Not Applicable |
| Scrubbing O:A Advance Ratio | 0.00 | v/v | Not Applicable |
| Stripping: Number of Stages | 2.00 | - | Based on Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | Based on Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | Based on Design |
| Sc2O3 Grade | 2% | % | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | Not Availabe |
| Precip Solid Yield | 0.4 | g/L | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | |
| Thickener UF % Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | Data not available |
| Diluent Density | 0.84 | g/ml | |
| Extractant Density | 0.92 | g/ml | |
| Extractant Concentration | 5% | % Volume | |
| Scrubbing Acid Molarity | | mol/L | Not Applicable |
| Srubbing Acid Molar Mass | | g/mol | Not Applicable |
| Scrub Acid Stock Conc | 37% | % | Not Applicable |
| Stripping Solution Molarity | 2.00 | Mol/L | |
| Stripping Solution Acid Molar M | 98.000 | g/mol | |
| Strip Acid Stock Conc | 93% | % | |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | Not Applicable |
| Neutralization Molar Mass | | g/mol | Not Applicable |
| Neutralization Stock Conc | 50% | % | Not Applicable |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| **Material Balance (SX)** | | |
|---|---|---|
| PLS Flow | 7911.8 | gpm |
| Raffinate Rate: | 7911.8 | gpm |
| Load Org Recycle | 7694.3 | gpm |
| Load Org Feed | 7911.8 | gpm |
| Loag Org Advance | 217.5 | gpm |
| Scrub Aq Product | 0.0 | gpm |
| Scrub Aq Recycle | 0.0 | gpm |
| Scrub Aq Feed | 0.0 | gpm |
| Scurb Org Product | 217.5 | gpm |
| Strip Aq Product | 17.4 | gpm |
| Strip Aq Recycle | 200.2 | gpm |
| Strip Aq Feed | 217.5 | gpm |
| Strip Org Product | 217.5 | gpm |
| Extraction Feed (O+A): | 15823.6 | gpm |
| Scrub Feed (O+A): | 217.5 | gpm |
| Strip Feed (O+A): | 435.0 | gpm |
| **Material Balance (Precip)** | | |
| Strip Solution Flow | 17.4 | gpm |
| Solid Density | 3.4 | g/cc |
| Solid Product Rate | 0.00 | tph |
| Liquid Density (after precip) | 1.13 | g/cc |
| Thickener Feed % Solids (eff) | 0.03% | % |
| **Total Volumes** | | |
| Total Extraction Volume | 506355 | gal |
| Total Scrubbing Volume | 0 | gal |
| Total Stripping Volume | 13922 | gal |
| Extraction Org Volume | 253177.72 | gal |
| Scrubbing Org Volume | 0.00 | gal |
| Stripping Org Volume | 6960.76 | gal |
| Total Org Volume | 260138.49 | gal |

| **Other Indicators** | | | |
|---|---|---|---|
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 9.85 | m^3 | |
| Filter Press Area | 0.01 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM9: CM Precipitation*

| **Flowsheet Design** | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 0 | min | N/A |
| Precip Solid Yield | - | g/L | N/A |
| Precip Settlng Rate | 10 | ft/hr | N/A |
| Thickener UF % Solids | 25% | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 7.24 | g/L of Feed | N/A |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 0.00 | min | N/A |
| Thickener Capacity | 0.00 | ft^2/tph | N/A |

| **Material Balance** | | | |
|---|---|---|---|
| Slurry Feed Rate | 7896 | gpm | |
| Slurry Feed Rate | 29,889 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.013 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

## PM10: CM Redissolution

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 2.0% | % | N/A |
| Dissolved Solids in Re-Leach | 100% | % | N/A |
| Leaching Retention Time | 45 | min | N/A |
| **Reagent Use** | | | |
| Acid Consumption | 441 | kg/ton | N/A |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 0.00 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 0 | gpm | |
| Liquid Density | 1.000 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | |
| Releach Conc. Factor | #DIV/0! | v/v | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | - | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

## PM11: CM Recovery

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 0.19 | g/L | WK1/WK1 Mass Balance-V2 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | | % | Not Applicable |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 2.49 | g/L | WK1/WK1 Mass Balance-V2 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Na2S Dose | 6.15 | g/L of Feed | WK1/WK1 Mass Balance-V2 |
| NaOH Dose | 3.58 | g/L of Feed | WK1/WK1 Mass Balance-V2 |
| HCL Dose | 5.45 | g/L of Feed | WK1/WK1 Mass Balance-V2 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 1062.21 | gpm | |
| Co Slurry Feed Rate | 4020.89 | lpm | |
| Co Precip Mass Flow | 0.05 | tph | |
| Co Thickener Feed % Solids (eff) | 0.02% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 1062.15 | gpm | |
| Mn Slurry Feed Rate | 4020.67 | lpm | |
| Mn Precip Mass Flow | 0.66 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.25% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.01 | g/cc | |
| Mn Liquid Density | 1.00 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 60.3 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 0.24 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.06 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 60.3 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 3.1 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.78 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

## PM12: Biooxidation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Requirec | 500 | gpm | |
| BioLeach Retention Time 1 | 20 | hr | 2.5 |
| BioLeach Retention Time 2 | 20 | hr | 60 |
| BioLeach Retention Time 3 | 20 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 3.5 | kJ/kg-K | |
| Temperature Drop | 10 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 500.0 | gpm | |
| Raw Leach Solution Req'd | 1892.7 | lpm | |
| Yearly Acid Production | 915,766,819 | liters solution/yr | |
| Yearly Acid Production | 1,098,920 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-1 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ 19,585,772 | $ - | $ 4,896,443 | $ 21,544,350 | $ 26,822,715 | $ 72,849,280 | IIIIIIII

# OPERATING COST ESTIMATE

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-1 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 23.02 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 8.84 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 154,031,629 | $ Total | $ 1,528.09 | $ 4,584.27 | $ 1,540,316 | $ 11.54 | $ 0.38 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 2.30 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 43,226,968 | $ Total | $ 214.42 | $ 643.26 | $ 216,135 | $ 1.62 | $ 0.05 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 3.45 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 154,031,629 | $ Total | $ 1,528.09 | $ 4,584.27 | $ 1,540,316 | $ 11.54 | $ 0.38 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 8,252.05** | **$ 24,756.16** | **$ 8,318,069** | **$ 62.31** | **$ 2.06** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 3.86 | $ 0.13 |
| PM2 | Roasting | PR | 198 | | $ 94.99 | $ 284.96 | $ 95,747 | $ 0.72 | $ 0.02 |
| PM3 | Leaching | PR | 1244 | | $ 596.19 | $ 1,788.56 | $ 600,955 | $ 4.50 | $ 0.15 |
| PM4 | Al Precipitation | PR | 787 | | $ 377.37 | $ 1,132.11 | $ 380,390 | $ 2.85 | $ 0.09 |
| PM5 | REE Precipitation | PR | 781 | | $ 374.35 | $ 1,123.06 | $ 377,349 | $ 2.83 | $ 0.09 |
| PM6 | RE-Redissolution | PR | 218 | | $ 104.38 | $ 313.15 | $ 105,220 | $ 0.79 | $ 0.03 |
| PM7 | REE Oxalate Precipitation | PR | 90 | | $ 43.30 | $ 129.90 | $ 43,646 | $ 0.33 | $ 0.01 |
| PM8 | Sc SX Separation | PR | 494 | | $ 236.93 | $ 710.79 | $ 238,824 | $ 1.79 | $ 0.06 |
| PM9 | CM Precipitation | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM10 | CM Redissolution | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM11 | CM Recovery | PR | 149 | | $ 71.55 | $ 214.66 | $ 72,125 | $ 0.54 | $ 0.02 |
| PM12 | Biooxidation | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM13 | Reserved | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM14 | Reserved | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM15 | Reserved | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | **5027.1** | | **$ 2,409.74** | **$ 7,229.23** | **$ 2,429,021.09** | **$ 18.20** | **$ 0.60** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.76 | $ 0.03 |
| PM2 | Roasting | CR | | $ 1,281.99 | $ 3,845.96 | $ 1,292,244 | $ 9.68 | $ 0.32 |
| PM3 | Leaching | CR | | $ 193,641.57 | $ 580,924.72 | $ 195,190,707 | $ 1,462.19 | $ 48.41 |
| PM4 | Al Precipitation | CR | | $ 4,267.83 | $ 12,803.50 | $ 4,301,974 | $ 32.23 | $ 1.07 |
| PM5 | REE Precipitation | CR | | $ 1,188.70 | $ 3,566.10 | $ 1,198,211 | $ 8.98 | $ 0.30 |
| PM6 | RE-Redissolution | CR | | $ 5,498.33 | $ 16,494.99 | $ 5,542,316 | $ 41.52 | $ 1.37 |
| PM7 | REE Oxalate Precipitation | CR | | $ 7,772.22 | $ 23,316.65 | $ 7,834,393 | $ 58.69 | $ 1.94 |
| PM8 | Sc SX Separation | CR | | $ 2,779.94 | $ 8,339.83 | $ 2,802,181 | $ 20.99 | $ 0.69 |
| PM9 | CM Precipitation | CR | | $ - | $ - | $ - | $ - | $ - |
| PM10 | CM Redissolution | CR | | $ - | $ - | $ - | $ - | $ - |
| PM11 | CM Recovery | CR | | $ 5,141.22 | $ 15,423.67 | $ 5,182,354 | $ 38.82 | $ 1.29 |
| PM12 | Biooxidation | CR | | $ - | $ - | $ - | $ - | $ - |
| PM13 | Reserved | CR | | $ - | $ - | $ - | $ - | $ - |
| PM14 | Reserved | CR | | $ - | $ - | $ - | $ - | $ - |
| PM15 | Reserved | CR | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ 221,672.61** | **$ 665,017.82** | **$ 223,445,987.56** | **$ 1,673.85** | **$ 55.42** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM8 | Sc SX Separation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM9 | CM Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM10 | CM Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM11 | CM Recovery | LS | | $ - | $ - | $ - | $ - | $ - |
| PM12 | Biooxidation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM13 | Reserved | LS | | $ - | $ - | $ - | $ - | $ - |
| PM14 | Reserved | LS | | $ - | $ - | $ - | $ - | $ - |
| PM15 | Reserved | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 9,308 | $ 0.10 | $ 310.27 | $ 930.82 | $ 312,754 | $ 2.34 | $ 0.08 |
| Non-Hazardous Precipitate | WD | Tons | 265 | $ 1.50 | $ 132.52 | $ 397.57 | $ 133,585 | $ 1.00 | $ 0.03 |
| Dust/Volatiles | WD | Tons | 28,800 | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 442.80** | **$ 1,328.39** | **$ 446,339.29** | **$ 3.34** | **$ 0.11** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 224,525** | **$ 673,575** | **$ 226,321,348** | **$ 1,695** | **$ 56** |

## Overhead Costs

| Allocation | Code | Rate | Basis | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% % of Direct Cost | $ 24,756 Daily Direct Cost | $ 1,237.81 | $ 3,713.42 | $ 1,247,710 | $ 9.35 | $ 0.31 |
| Variable Portion | OH | 15% % of Direct Cost | $ 673,575 Daily Direct Cost | $ 33,678.77 | $ 101,036.32 | $ 33,948,202 | $ 254.31 | $ 8.42 |
| **TOTAL OVERHEAD COSTS** | | | | **$ 34,916.58** | **$ 104,749.74** | **$ 35,195,912.54** | **$ 263.66** | **$ 8.73** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | WK No. 13 | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | WK-1 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 8,252.05 | $ 24,756.16 | $ 8,318,069 | $ 62 | $ 2.06 | I |
| Power | $ 2,409.74 | $ 7,229.23 | $ 2,429,021 | $ 18 | $ 0.60 | |
| Consumables | $ 221,672.61 | $ 665,017.82 | $ 223,445,988 | $ 1,674 | $ 55.42 | IIIIIIIIIIIIIIIIIIIIIIII |
| Leasing/Tolling | $ - | $ - | $ - | $ - | $ - | |
| Waste Disposal | $ 442.80 | $ 1,328.39 | $ 446,339 | $ 3 | $ 0.11 | |
| General Overhead | $ 34,916.58 | $ 104,749.74 | $ 35,195,913 | $ 264 | $ 8.73 | IIII |
| TOTAL | $ 267,694 | $ 803,081 | $ 269,835,330 | $ 2,021 | $ 66.92 | |

## Summary by Process Module

| | Process Module | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 611 | $ 1,834 | $ 616,370 | $ 4.62 | $ 0.15 | |
| PM2 | Roasting | $ 1,377 | $ 4,131 | $ 1,387,991 | $ 10.40 | $ 0.34 | |
| PM3 | Leaching | $ 194,238 | $ 582,713 | $ 195,791,662 | $ 1,466.69 | $ 48.56 | IIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 4,645 | $ 13,936 | $ 4,682,365 | $ 35.08 | $ 1.16 | |
| PM5 | REE Precipitation | $ 1,563 | $ 4,689 | $ 1,575,560 | $ 11.80 | $ 0.39 | |
| PM6 | RE-Redissolution | $ 5,603 | $ 16,808 | $ 5,647,536 | $ 42.31 | $ 1.40 | |
| PM7 | REE Oxalate Precipitation | $ 7,816 | $ 23,447 | $ 7,878,039 | $ 59.02 | $ 1.95 | |
| PM8 | Sc SX Separation | $ 3,017 | $ 9,051 | $ 3,041,006 | $ 22.78 | $ 0.75 | |
| PM9 | CM Precipitation | $ - | $ - | $ - | $ - | $ - | |
| PM10 | CM Redissolution | $ - | $ - | $ - | $ - | $ - | |
| PM11 | CM Recovery | $ 5,213 | $ 15,638 | $ 5,254,479 | $ 39.36 | $ 1.30 | |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| NA | Unallocated | $ 43,611 | $ 130,834 | $ 43,960,321 | $ 329 | $ 10.90 | IIIII |
| | TOTAL | $ 267,694 | $ 803,081 | $ 269,835,330 | $ 2,021 | $ 66.92 | |

## Summary by Cost Center

| Cost Center | Code | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| Operating Labor | OL | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 23 | $ 0.76 | |
| Technical Labor | TL | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 9 | $ 0.29 | |
| Power/Utilities | PR | $ 2,409.74 | $ 7,229.23 | $ 2,429,021 | $ 18 | $ 0.60 | |
| Lease Agreements | LS | $ - | $ - | $ - | $ - | $ - | |
| Consumables/Reagents | CR | $ 221,672.61 | $ 665,017.82 | $ 223,445,988 | $ 1,674 | $ 55.42 | IIIIIIIIIIIIIIIIIIIIIIII |
| Waste Disposal | WD | $ 442.80 | $ 1,328.39 | $ 446,339 | $ 3 | $ 0.11 | |
| Plant Overhead | OH | $ 36,749.53 | $ 110,248.59 | $ 37,043,528 | $ 277 | $ 9.19 | IIII |
| General Sales, and Administration | GSA | $ 2,199.80 | $ 6,599.40 | $ 2,217,399 | $ 17 | $ 0.55 | |
| Reserved | | | | | | | |
| TOTAL | | $ 267,694 | $ 803,081.34 | $ 269,835,330 | $ 2,021 | $ 66.92 | |

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | WK No. 13 | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | WK-1 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 43,226,968

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 255 | 274 |
| Contained Value | ($/ton) | $ 6.2 | $ 116.2 |
| Feed Basket Price | ($/kg) | $ 26.8 | $ 467.5 |
| Product Basket Price | ($/kg) | $ 17.2 | $ 326.9 |
| Production Rate | (kg/day) | 378.15 | 397.30 |
| | (t/day) | 0.38 | 0.40 |
| | (t/yr) | 127.06 | 133.49 |
| | (% of US Demand) | 1.3% | |

## Revenue Contribution Table

| Element | REE/CM Price ($/kg) [1] | Product Revenue ($/hr) MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | REE Revenue ($/day) |
|---|---|---|---|---|---|---|---|
| Scandium | $ 4,200.00 | $ 0.0 | $ 5,140.5 | $ - | $ - | $ - | $ 123,371 |
| Yttrium | $ 6.00 | $ 3.9 | $ - | $ - | $ - | $ - | $ 95 |
| Lanthanum | $ 2.00 | $ 6.0 | $ - | $ - | $ - | $ - | $ 145 |
| Cerium | $ 2.00 | $ 12.2 | $ - | $ - | $ - | $ - | $ 293 |
| Praseodymium | $ 52.00 | $ 5.1 | $ - | $ - | $ - | $ - | $ 123 |
| Neodymium | $ 42.00 | $ 114.0 | $ - | $ - | $ - | $ - | $ 2,735 |
| Samarium | $ 2.11 | $ 1.4 | $ - | $ - | $ - | $ - | $ 33 |
| Europium | $ 34.00 | $ 3.3 | $ - | $ - | $ - | $ - | $ 80 |
| Gadolinium | $ 32.00 | $ 13.0 | $ - | $ - | $ - | $ - | $ 313 |
| Terbium | $ 400.00 | $ 4.7 | $ - | $ - | $ - | $ - | $ 114 |
| Dysprosium | $ 230.00 | $ 29.2 | $ - | $ - | $ - | $ - | $ 701 |
| Holmium | $ 52.96 | $ 1.5 | $ - | $ - | $ - | $ - | $ 37 |
| Erbium | $ 34.00 | $ 1.6 | $ - | $ - | $ - | $ - | $ 39 |
| Thulium | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Ytterbium | $ 28.72 | $ 0.6 | $ - | $ - | $ - | $ - | $ 15 |
| Lutetium | $ 796.64 | $ 3.6 | $ - | $ - | $ - | $ - | $ 87 |
| Manganese | $ 0.005 | $ - | $ - | $ 0.2 | $ - | $ - | $ 4 |
| Cobalt | $ 22.00 | $ - | $ - | $ - | $ 2.9 | $ - | $ 69 |
| Nickel | $ 18.00 | $ - | $ - | $ - | $ 3.6 | $ - | $ 86 |
| Zinc | $ 3.02 | $ - | $ - | $ - | $ 13.0 | $ - | $ 312 |
| Reserved | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Total: | | $ 200.4 | $ 5,140.5 | $ 0.2 | $ 19.5 | $ - | $ 128,652 |

[1] REEs listed as $/kg REO; Mn listed as $/kg 44% Mn Ore; Co listed as $/kg metal content

## Coal Revenue

List Coal Revenue Sources Here:

**Annual Coal Revenue:** $ -

## Other Revenue

List Other Revenue Sources Here:

**Annual Other Revenue:** $ -

## PRO FORMA CASH FLOW

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-1 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 47,235,276 | $ 48,652,334 | $ 50,111,904 | $ 51,615,261 | $ 53,163,719 | $ 54,758,630 | $ 56,401,389 | $ 58,093,431 | $ 59,836,234 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 47,235,276 | $ 48,652,334 | $ 50,111,904 | $ 51,615,261 | $ 53,163,719 | $ 54,758,630 | $ 56,401,389 | $ 58,093,431 | $ 59,836,234 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 462,095 | $ 3,299,419 | $ 299,388,579 | $ 307,850,998 | $ 316,555,791 | $ 325,509,539 | $ 334,718,981 | $ 344,191,012 | $ 353,932,687 | $ 363,951,226 | $ 374,254,010 |
| *- Operating Costs* | $ - | $ - | $ - | $ 294,856,350 | $ 303,702,041 | $ 312,813,102 | $ 322,197,495 | $ 331,863,420 | $ 341,819,322 | $ 352,073,902 | $ 362,636,119 | $ 373,515,203 |
| *- Loan #1 Interest* | $ - | $ 462,095 | $ 427,037 | $ 389,875 | $ 350,484 | $ 308,729 | $ 264,469 | $ 217,553 | $ 167,822 | $ 115,108 | $ 59,230 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 2,872,382 | $ 2,654,460 | $ 2,423,463 | $ 2,178,606 | $ 1,919,058 | $ 1,643,937 | $ 1,352,308 | $ 1,043,182 | $ 715,508 | $ 368,174 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 1,487,894 | $ 1,375,010 | $ 1,255,354 | $ 1,128,518 | $ 994,072 | $ 851,559 | $ 700,496 | $ 540,368 | $ 370,633 |
| Gross Profit | $ - | $ (462,095) | $ (3,299,419) | $ (252,153,303) | $ (259,198,664) | $ (266,443,887) | $ (273,894,278) | $ (281,555,262) | $ (289,432,382) | $ (297,531,298) | $ (305,857,795) | $ (314,417,776) |
| **Book Deductions** | $ - | $ 577,619 | $ 4,168,597 | $ 6,034,674 | $ 5,582,073 | $ 5,163,418 | $ 4,776,162 | $ 4,417,949 | $ 4,086,603 | $ 3,780,108 | $ 3,496,600 | $ 3,234,355 |
| *- CAPEX #1 Depreciation* | $ - | $ 577,619 | $ 534,297 | $ 494,225 | $ 457,158 | $ 422,871 | $ 391,156 | $ 361,819 | $ 334,683 | $ 309,582 | $ 286,363 | $ 264,886 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 3,634,300 | $ 3,361,728 | $ 3,109,598 | $ 2,876,378 | $ 2,660,650 | $ 2,461,101 | $ 2,276,518 | $ 2,105,780 | $ 1,947,846 | $ 1,801,758 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 2,178,722 | $ 2,015,317 | $ 1,864,169 | $ 1,724,356 | $ 1,595,029 | $ 1,475,402 | $ 1,364,747 | $ 1,262,391 | $ 1,167,712 |
| Taxable Before Depletion | $ - | $ (1,039,713) | $ (7,468,016) | $ (258,187,977) | $ (264,780,737) | $ (271,607,305) | $ (278,670,440) | $ (285,973,212) | $ (293,518,985) | $ (301,311,406) | $ (309,354,395) | $ (317,652,131) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ - | $ (519,857) | $ (3,734,008) | $ (129,093,989) | $ (132,390,369) | $ (135,803,652) | $ (139,335,220) | $ (142,986,606) | $ (146,759,492) | $ (150,655,703) | $ (154,677,197) | $ (158,826,066) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 6,612,939 | $ 6,811,327 | $ 7,015,667 | $ 7,226,137 | $ 7,442,921 | $ 7,666,208 | $ 7,896,194 | $ 8,133,080 | $ 8,377,073 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (1,039,713) | $ (7,468,016) | $ (258,187,977) | $ (264,780,737) | $ (271,607,305) | $ (278,670,440) | $ (285,973,212) | $ (293,518,985) | $ (301,311,406) | $ (309,354,395) | $ (317,652,131) |
| *- Income Tax* | $ - | $ (270,326) | $ (1,941,684) | $ (67,128,874) | $ (68,842,992) | $ (70,617,899) | $ (72,454,314) | $ (74,353,035) | $ (76,314,936) | $ (78,340,966) | $ (80,432,143) | $ (82,589,554) |
| Net Profit/Loss | $ - | $ (769,388) | $ (5,526,332) | $ (191,059,103) | $ (195,937,746) | $ (200,989,405) | $ (206,216,126) | $ (211,620,177) | $ (217,204,049) | $ (222,970,440) | $ (228,922,252) | $ (235,062,577) |
| **Book Deductions** | $ - | $ 577,619 | $ 4,168,597 | $ 6,034,674 | $ 5,582,073 | $ 5,163,418 | $ 4,776,162 | $ 4,417,949 | $ 4,086,603 | $ 3,780,108 | $ 3,496,600 | $ 3,234,355 |
| *+ CAPEX #1 Depreciation* | $ - | $ 577,619 | $ 534,297 | $ 494,225 | $ 457,158 | $ 422,871 | $ 391,156 | $ 361,819 | $ 334,683 | $ 309,582 | $ 286,363 | $ 264,886 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 3,634,300 | $ 3,361,728 | $ 3,109,598 | $ 2,876,378 | $ 2,660,650 | $ 2,461,101 | $ 2,276,518 | $ 2,105,780 | $ 1,947,846 | $ 1,801,758 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 2,178,722 | $ 2,015,317 | $ 1,864,169 | $ 1,724,356 | $ 1,595,029 | $ 1,475,402 | $ 1,364,747 | $ 1,262,391 | $ 1,167,712 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ - | $ (191,769) | $ (1,357,734) | $ (185,024,429) | $ (190,355,672) | $ (195,825,988) | $ (201,439,964) | $ (207,202,227) | $ (213,117,446) | $ (219,190,332) | $ (225,425,652) | $ (231,828,222) |
| **Captial Cost** | $ 7,701,581 | $ 48,457,333 | $ 29,049,620 | $ 6,387,865 | $ 6,771,137 | $ 7,177,405 | $ 7,608,049 | $ 8,064,532 | $ 8,548,404 | $ 9,061,309 | $ 9,604,987 | $ 9,134,888 |
| *- Fixed Capital Costs* | $ 15,403,163 | $ 95,746,060 | $ 49,596,459 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 7,701,581 | $ 47,873,030 | $ 24,798,230 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 584,303 | $ 619,361 | $ 656,523 | $ 695,915 | $ 737,669 | $ 781,930 | $ 828,845 | $ 878,576 | $ 931,291 | $ 987,168 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 3,632,029 | $ 3,849,951 | $ 4,080,948 | $ 4,325,805 | $ 4,585,353 | $ 4,860,474 | $ 5,152,103 | $ 5,461,229 | $ 5,788,903 | $ 6,136,237 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 1,881,391 | $ 1,994,275 | $ 2,113,931 | $ 2,240,767 | $ 2,375,213 | $ 2,517,726 | $ 2,668,789 | $ 2,828,917 | $ 2,998,652 |
| **Net Cash Flow** | $ (7,701,581) | $ (48,649,103) | $ (30,407,355) | $ (191,412,294) | $ (197,126,809) | $ (203,003,393) | $ (209,048,013) | $ (215,266,760) | $ (221,665,850) | $ (228,251,641) | $ (235,030,639) | $ (240,963,110) |
| Discounted Cash Flow | $ (7,701,581) | $ (44,226,457) | $ (25,130,045) | $ (143,810,890) | $ (134,640,263) | $ (126,049,135) | $ (118,002,154) | $ (110,465,885) | $ (103,408,755) | $ (96,800,977) | $ (90,614,486) | $ (84,456,100) |
| **Cumulative DCF** | $ (7,701,581) | $ (51,928,039) | $ (77,058,084) | $ (220,868,973) | $ (355,509,236) | $ (481,558,372) | $ (599,560,525) | $ (710,026,411) | $ (813,435,166) | $ (910,236,143) | $ (1,000,850,629) | $ (1,085,306,729) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 154,031,629 | Payback Period | 25.5 |
| Constant Dollar OPEX | $ 269,835,330 | Internal Rate of Return | N/A |
| Constant Dollar REE Rev | $ 43,226,968 | Net Present Value | $ (1,701,387,802) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ - |

## PRO FORMA CASH FLOW

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-1 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 61,631,321** | **$ 63,480,260** | **$ 65,384,668** | **$ 67,346,208** | **$ 69,366,595** | **$ 71,447,592** | **$ 73,591,020** | **$ 75,798,751** | **$ 78,072,713** | **$ 80,414,895** | **$ 82,827,342** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 61,631,321 | $ 63,480,260 | $ 65,384,668 | $ 67,346,208 | $ 69,366,595 | $ 71,447,592 | $ 73,591,020 | $ 75,798,751 | $ 78,072,713 | $ 80,414,895 | $ 82,827,342 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 384,911,373** | **$ 396,262,278** | **$ 408,150,147** | **$ 420,394,651** | **$ 433,006,491** | **$ 445,996,685** | **$ 459,376,586** | **$ 473,157,884** | **$ 487,352,620** | **$ 501,973,199** | **$ 517,032,395** |
| *- Operating Costs* | $ 384,720,659 | $ 396,262,278 | $ 408,150,147 | $ 420,394,651 | $ 433,006,491 | $ 445,996,685 | $ 459,376,586 | $ 473,157,884 | $ 487,352,620 | $ 501,973,199 | $ 517,032,395 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 190,714 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ (323,280,052) | $ (332,782,018) | $ (342,765,479) | $ (353,048,443) | $ (363,639,896) | $ (374,549,093) | $ (385,785,566) | $ (397,359,133) | $ (409,279,907) | $ (421,558,304) | $ (434,205,053) |
| **Book Deductions** | **$ 2,991,778** | **$ 2,767,395** | **$ 2,559,840** | **$ 2,367,852** | **$ 2,190,263** | **$ 2,025,994** | **$ 1,874,044** | **$ 1,733,491** | **$ 1,603,479** | **$ 1,361,746** | **$ 495,330** |
| *- CAPEX #1 Depreciation* | $ 245,019 | $ 226,643 | $ 209,645 | $ 193,921 | $ 179,377 | $ 165,924 | $ 153,480 | $ 141,969 | $ 131,321 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 1,666,626 | $ 1,541,629 | $ 1,426,007 | $ 1,319,056 | $ 1,220,127 | $ 1,128,617 | $ 1,043,971 | $ 965,673 | $ 893,248 | $ 826,254 | $ - |
| *- CAPEX #3 Depreciation* | $ 1,080,133 | $ 999,123 | $ 924,189 | $ 854,875 | $ 790,759 | $ 731,452 | $ 676,593 | $ 625,849 | $ 578,910 | $ 535,492 | $ 495,330 |
| Taxable Before Depletion | $ (326,271,830) | $ (335,549,413) | $ (345,325,319) | $ (355,416,295) | $ (365,830,160) | $ (376,575,087) | $ (387,659,610) | $ (399,092,624) | $ (410,883,386) | $ (422,920,050) | $ (434,700,383) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ (163,135,915) | $ (167,774,706) | $ (172,662,659) | $ (177,708,148) | $ (182,915,080) | $ (188,287,543) | $ (193,829,805) | $ (199,546,312) | $ (205,441,693) | $ (211,460,025) | $ (217,350,192) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 8,628,385 | $ 8,887,236 | $ 9,153,854 | $ 9,428,469 | $ 9,711,323 | $ 10,002,663 | $ 10,302,743 | $ 10,611,825 | $ 10,930,180 | $ 11,258,085 | $ 11,595,828 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ (326,271,830)** | **$ (335,549,413)** | **$ (345,325,319)** | **$ (355,416,295)** | **$ (365,830,160)** | **$ (376,575,087)** | **$ (387,659,610)** | **$ (399,092,624)** | **$ (410,883,386)** | **$ (422,920,050)** | **$ (434,700,383)** |
| *- Income Tax* | $ (84,830,676) | $ (87,242,847) | $ (89,784,583) | $ (92,408,237) | $ (95,115,841) | $ (97,909,523) | $ (100,791,499) | $ (103,764,082) | $ (106,829,680) | $ (109,959,213) | $ (113,022,100) |
| Net Profit/Loss | $ (241,441,154) | $ (248,306,566) | $ (255,540,736) | $ (263,008,058) | $ (270,714,318) | $ (278,665,564) | $ (286,868,111) | $ (295,328,541) | $ (304,053,705) | $ (312,960,837) | $ (321,678,284) |
| **Book Deductions** | **$ 2,991,778** | **$ 2,767,395** | **$ 2,559,840** | **$ 2,367,852** | **$ 2,190,263** | **$ 2,025,994** | **$ 1,874,044** | **$ 1,733,491** | **$ 1,603,479** | **$ 1,361,746** | **$ 495,330** |
| *+ CAPEX #1 Depreciation* | $ 245,019 | $ 226,643 | $ 209,645 | $ 193,921 | $ 179,377 | $ 165,924 | $ 153,480 | $ 141,969 | $ 131,321 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 1,666,626 | $ 1,541,629 | $ 1,426,007 | $ 1,319,056 | $ 1,220,127 | $ 1,128,617 | $ 1,043,971 | $ 965,673 | $ 893,248 | $ 826,254 | $ - |
| *+ CAPEX #3 Depreciation* | $ 1,080,133 | $ 999,123 | $ 924,189 | $ 854,875 | $ 790,759 | $ 731,452 | $ 676,593 | $ 625,849 | $ 578,910 | $ 535,492 | $ 495,330 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ (238,449,376) | $ (245,539,171) | $ (252,980,896) | $ (260,640,206) | $ (268,524,055) | $ (276,639,571) | $ (284,994,067) | $ (293,595,051) | $ (302,450,227) | $ (311,599,091) | $ (321,182,954) |
| **Captial Cost** | **$ 3,178,571** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 3,178,571 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ (241,627,947) | $ (245,539,171) | $ (252,980,896) | $ (260,640,206) | $ (268,524,055) | $ (276,639,571) | $ (284,994,067) | $ (293,595,051) | $ (302,450,227) | $ (311,599,091) | $ (321,182,954) |
| Discounted Cash Flow | $ (76,990,110) | $ (71,123,952) | $ (66,617,777) | $ (62,395,193) | $ (58,438,658) | $ (54,731,664) | $ (51,258,688) | $ (48,005,137) | $ (44,957,299) | $ (42,106,563) | $ (39,456,033) |
| **Cumulative DCF** | $ (1,162,296,839) | $ (1,233,420,791) | $ (1,300,038,567) | $ (1,362,433,760) | $ (1,420,872,418) | $ (1,475,604,083) | $ (1,526,862,771) | $ (1,574,867,908) | $ (1,619,825,207) | $ (1,661,931,770) | $ (1,701,387,802) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 154,031,629 | Payback Period | 0 | 25.5 |
| Constant Dollar OPEX | $ 269,835,330 | Internal Rate of Return | 0 | N/A |
| Constant Dollar REE Rev | $ 43,226,968 | Net Present Value | 0 | $ (1,701,387,802) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ - |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-1 | By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 2.17 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 1.36 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.10 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.11 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.11 | $ 0.00 |
| 7 | Conveyor | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 3.86** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.76 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.76** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 611** | **$ 1,834** | **$ 616,370** | **$ 4.62** | **$ 0.15** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-1 | By: ks

## Module Information

Module ID: PM2
Module Name: Roasting
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 500 | tph |
| External Heat | 10.0% | % |
| Product Rate | 480 | tph |
| Roasting Temperature | 575 | deg C |
| Bin Storage Capacity | 20% | % of hourly prod. |
| Material Heat Capacity | 0.80 | kg/kg-K |
| Reaction Enthalpy | 1783.00 | kJ/kg |
| % of Feed Reacted | 50% | % |
| Scurbber Airflow | 20.00 | cfm/tph |
| Quencher Aiflow | 20.00 | cfm/tph |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Storage Bins | 2 | 100% | 100 | Tons | $ 115,387 | $ 230,773 | $ - | |
| 2 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 3 | Roaster | 1 | 100% | 581 | MBTU/hr | $ 18,994,975 | $ 18,994,975 | $ - | |
| 4 | Gas Scrubber | 1 | 100% | 10,000 | cfm | $ 57,958 | $ 57,958 | $ - | |
| 5 | Spray Chamber Quencher | 1 | 100% | 10,000 | cfm | $ 206,510 | $ 206,510 | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 19,585,772** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Storage Bins | 10.00 | Power Cost: | $ 0.07 | $/kWhr | | $ 4.79 | $ 14.38 | $ 4,832 | $ 0.04 | $ 0.00 |
| 2 | Conveyor | 31.00 | Effiicency: | 0.85 | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.11 | $ 0.00 |
| 3 | Roaster | 87.16 | | | | | $ 41.78 | $ 125.34 | $ 42,113 | $ 0.32 | $ 0.01 |
| 4 | Gas Scrubber | 50.00 | | | | | $ 23.97 | $ 71.90 | $ 24,159 | $ 0.18 | $ 0.01 |
| 5 | Spray Chamber Quencher | 20.00 | | | | | $ 9.59 | $ 28.76 | $ 9,664 | $ 0.07 | $ 0.00 |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **198** | | | | | **$ 94.99** | **$ 284.96** | **$ 95,747** | **$ 0.72** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Natural Gas | 56.03 | MCF/hr | $ 2.860 | / MCF | $ 1,281.99 | $ 3,845.96 | $ 1,292,244 | $ 9.68 | $ 0.32 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 1,281.99** | **$ 3,845.96** | **$ 1,292,244** | **$ 9.68** | **$ 0.32** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 1,377** **$ 4,131** **$ 1,387,991** **$ 10.40** **$ 0.34**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-1 | By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 2.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 480 | tph |
| Residue Rate | 388 | tph |
| Slurry Flow | 8880 | gpm |
| Leaching Time | 45 | min |
| Acid Consumption | 228.51 | kg/ton |
| Bioleaching Sol'n Consumption | 0 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 2 | -- |
| Thickener Capacity | 2.50 | ft^2/tph |
| Polymer Dose | 0.016 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 8 | 100% | 49,950 | gal | $ 177,287 | $ 1,418,297 | $ - | |
| 2 | Tank Mixer | 8 | 100% | 75 | HP | $ 182,787 | $ 1,462,293 | $ - | |
| 3 | Process Pump | 4 | 100% | 4,440 | gpm | $ 159,455 | $ 637,818 | $ - | |
| 4 | Conveyor | 1 | 100% | 480 | TPH | $ 93,438 | $ 93,438 | $ - | |
| 5 | Belt Filter | 2 | 100% | 194 | TPH | $ 1,162,425 | $ 2,324,850 | $ - | |
| 6 | Solution Heater | 1 | 100% | 1 | MBTU/hr | $ 30,018 | $ 30,018 | $ - | |
| 7 | Thickener | 2 | 100% | 484 | ft^2 | $ 60,048 | $ 120,097 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 6,086,811** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 599.40 | Effiicency: | 0.85 | | $ 287.32 | $ 861.97 | $ 289,623 | $ 2.17 | $ 0.07 |
| 3 | Process Pump | 444.00 | | | | $ 212.83 | $ 638.50 | $ 214,536 | $ 1.61 | $ 0.05 |
| 4 | Conveyor | 29.76 | | | | $ 14.27 | $ 42.80 | $ 14,380 | $ 0.11 | $ 0.00 |
| 5 | Belt Filter | 94.05 | | | | $ 45.08 | $ 135.25 | $ 45,444 | $ 0.34 | $ 0.01 |
| 6 | Solution Heater | 75.90 | | | | $ 36.38 | $ 109.15 | $ 36,675 | $ 0.27 | $ 0.01 |
| 7 | Thickener | 0.62 | | | | $ 0.30 | $ 0.89 | $ 297 | $ 0.00 | $ 0.00 |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1244** | | | | **$ 596.19** | **$ 1,788.56** | **$ 600,955** | **$ 4.50** | **$ 0.15** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | - | L/hr | $ 0.008 | / L | $ - | $ - | $ - | $ - | $ - |
| 2 | H2SO4 | 109.68 | mt/hr | $ 220.000 | / mt | $ 193,043.97 | $ 579,131.91 | $ 194,588,321 | $ 1,457.68 | $ 48.26 |
| 3 | Natural Gas | 0.73 | MCF/hr | $ 2.860 | / MCF | $ 16.75 | $ 50.24 | $ 16,881 | $ 0.13 | $ 0.00 |
| 4 | Polymer | 32.27 | kg/hr | $ 2.250 | / kg | $ 580.86 | $ 1,742.57 | $ 585,505 | $ 4.39 | $ 0.15 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 193,641.57** | **$ 580,924.72** | **$ 195,190,707** | **$ 1,462.19** | **$ 48.41** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 194,238** | **$ 582,713** | **$ 195,791,662** | **$ 1,466.69** | **$ 48.56** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-1 | By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 7.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 7912 | gpm |
| Feed Rate | 29949 | lpm |
| Precip Mass Flow | 11.04 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 915.35 | ft^2/tpd |
| NaOH Dose | 2.38 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 7 | 100% | 50,862 | gal | $ 119,400 | $ 835,798 | $ - | |
| 2 | Tank Mixer | 7 | 100% | 25 | HP | $ 99,375 | $ 695,627 | $ - | |
| 3 | Process Pump | 21 | 100% | 1,130 | gpm | $ 47,313 | $ 993,565 | $ - | |
| 4 | Filter Press | 1 | 100% | 52 | ft^3 | $ 157,202 | $ 157,202 | $ - | |
| 5 | Conveyor | 1 | 100% | 11 | TPH | $ 11,777 | $ 11,777 | $ - | |
| 6 | Thickener | 7 | 100% | 1,444 | ft^2 | $ 144,372 | $ 1,010,601 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 3,704,569** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 178.02 | Effiicency: | 0.85 | | $ 85.33 | $ 256.00 | $ 86,015 | $ 0.64 | $ 0.02 |
| 3 | Process Pump | 593.39 | | | | $ 284.44 | $ 853.32 | $ 286,717 | $ 2.15 | $ 0.07 |
| 4 | Filter Press | 8.74 | | | | $ 4.19 | $ 12.56 | $ 4,221 | $ 0.03 | $ 0.00 |
| 5 | Conveyor | 0.68 | | | | $ 0.33 | $ 0.98 | $ 331 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 6.43 | | | | $ 3.08 | $ 9.25 | $ 3,107 | $ 0.02 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **787** | | | | **$ 377.37** | **$ 1,132.11** | **$ 380,390** | **$ 2.85** | **$ 0.09** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 4.27 | mt/hr | $ 125.000 | / mt | $ 4,267.83 | $ 12,803.50 | $ 4,301,974 | $ 32.23 | $ 1.07 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 4,267.83** | **$ 12,803.50** | **$ 4,301,974** | **$ 32.23** | **$ 1.07** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 4,645** **$ 13,936** **$ 4,682,365** **$ 35.08** **$ 1.16**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-1 | By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 8.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 7899 | gpm |
| Feed Rate | 29900 | lpm |
| Precip Mass Flow | 2.54 | tph |
| Precip Conditioning Time | 45 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 5357.12 | ft^2/tpd |
| NaOH Dose | 0.66 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 8 | 100% | 44,431 | gal | $ 110,659 | $ 885,274 | $ - | |
| 2 | Tank Mixer | 8 | 100% | 22 | HP | $ 92,081 | $ 736,646 | $ - | |
| 3 | Process Pump | 24 | 100% | 987 | gpm | $ 44,935 | $ 1,078,446 | $ - | |
| 4 | Filter Press | 1 | 100% | 12 | ft^3 | $ 54,578 | $ 54,578 | $ - | |
| 5 | Conveyor | 1 | 100% | 3 | TPH | $ 5,255 | $ 5,255 | $ - | |
| 6 | Thickener | 8 | 100% | 1,701 | ft^2 | $ 164,620 | $ 1,316,964 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 4,077,163** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 177.72 | Effiicency: | 0.85 | | $ 85.19 | $ 255.58 | $ 85,874 | $ 0.64 | $ 0.02 |
| 3 | Process Pump | 592.41 | | | | $ 283.97 | $ 851.92 | $ 286,247 | $ 2.14 | $ 0.07 |
| 4 | Filter Press | 2.01 | | | | $ 0.96 | $ 2.89 | $ 971 | $ 0.01 | $ 0.00 |
| 5 | Conveyor | 0.16 | | | | $ 0.08 | $ 0.23 | $ 76 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 8.65 | | | | $ 4.15 | $ 12.44 | $ 4,181 | $ 0.03 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **781** | | | | **$ 374.35** | **$ 1,123.06** | **$ 377,349** | **$ 2.83** | **$ 0.09** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 1.19 | mt/hr | $ 125.000 | / mt | $ 1,188.70 | $ 3,566.10 | $ 1,198,211 | $ 8.98 | $ 0.30 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 1,188.70** | **$ 3,566.10** | **$ 1,198,211** | **$ 8.98** | **$ 0.30** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 1,563** | **$ 4,689** | **$ 1,575,560** | **$ 11.80** | **$ 0.39** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-1 | By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 2.54 | tph |
| Slurry Feed Rate | 1062 | gpm |
| Leaching Time | 120 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 1002.22 | kg/ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 31,868 | gal | $ 39,133 | $ 156,534 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 48 | HP | $ 141,860 | $ 567,439 | $ - | |
| 3 | Process Pump | 1 | 100% | 1,062 | gpm | $ 92,412 | $ 92,412 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 816,384** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 191.21 | Effiicency: | 0.85 | | | $ 91.65 | $ 274.96 | $ 92,388 | $ 0.69 | $ 0.02 |
| 3 | Process Pump | 26.56 | | | | | $ 12.73 | $ 38.19 | $ 12,832 | $ 0.10 | $ 0.00 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **218** | | | | | **$ 104.38** | **$ 313.15** | **$ 105,220** | **$ 0.79** | **$ 0.03** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 2.55 | mt/hr | $ 270.000 | / mt | $ 5,498.33 | $ 16,494.99 | $ 5,542,316 | $ 41.52 | $ 1.37 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 5,498.33** | **$ 16,494.99** | **$ 5,542,316** | **$ 41.52** | **$ 1.37** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 5,603** **$ 16,808** **$ 5,647,536** **$ 42.31** **$ 1.40**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-1 | By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 1062 | gpm |
| Feed Rate | 4021 | lpm |
| Precip Mass Flow | 0.04 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 1.95 | g/L of Feed |
| Oxalic Acid Dose | 6.30 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.02 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 5,311 | gal | $ 7,144 | $ 28,575 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 3 | HP | $ 27,790 | $ 111,159 | $ - | |
| 3 | Process Pump | 3 | 100% | 1,062 | gpm | $ 92,412 | $ 277,236 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 0.18 | ft^3 | $ 2,635 | $ 2,635 | $ - | |
| 6 | Roaster | 1 | 100% | 0.06 | MBTU/hr | $ 228,907 | $ 228,907 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 648,512** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 10.62 | Effiicency: | 0.85 | | | $ 5.09 | $ 15.28 | $ 5,133 | $ 0.04 | $ 0.00 |
| 3 | Process Pump | 79.67 | | | | | $ 38.19 | $ 114.57 | $ 38,495 | $ 0.29 | $ 0.01 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.03 | | | | | $ 0.01 | $ 0.04 | $ 14 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.01 | | | | | $ 0.00 | $ 0.01 | $ 4 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **90** | | | | | **$ 43.30** | **$ 129.90** | **$ 43,646** | **$ 0.33** | **$ 0.01** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 0.47 | mt/hr | $ 125.000 | / mt | $ 469.71 | $ 1,409.13 | $ 473,469 | $ 3.55 | $ 0.12 |
| 2 | Oxalic Acid | 1,521.09 | kg/hr | $ 0.600 | / kg | $ 7,301.22 | $ 21,903.65 | $ 7,359,625 | $ 55.13 | $ 1.83 |
| 3 | Natural Gas | 0.06 | MCF/hr | $ 2.860 | / MCF | $ 1.29 | $ 3.86 | $ 1,299 | $ 0.01 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 7,772.22** | **$ 23,316.65** | **$ 7,834,393** | **$ 58.69** | **$ 1.94** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 7,816** **$ 23,447** **$ 7,878,039** **$ 59.02** **$ 1.95**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-1 | By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 7912 | gpm |
| Raffinate Rate: | 7912 | gpm |
| Extraction Feed (O+A) | 15824 | gpm |
| Scrub Feed (O+A) | 218 | gpm |
| Strip Feed (O+A) | 435 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 17 | gpm |
| Organic Advance | 218 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 260138 | gal |
| Precip Mixing Intensity | 0.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 30.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 15,824 | gal | $ 92,877 | $ 185,755 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 40 | HP | $ 127,497 | $ 254,995 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 237,354 | gal | $ 425,936 | $ 851,871 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 7,912 | gpm | $ 198,759 | $ 397,519 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 218 | gal | $ 8,334 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,362 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,263 | gal | $ 38,218 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 435 | gal | $ 12,306 | $ 24,613 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 16,797 | $ 33,594 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,526 | gal | $ 56,437 | $ 112,875 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 17 | gpm | $ 19,240 | $ 19,240 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 56,622 | kg | $ 3,545,065 | $ - | $ 3,545,065 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 982,269 | kg | $ 1,063,647 | $ - | $ 1,063,647 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,603 | gal | $ 5,445 | $ 21,780 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 1.30 | HP | $ 18,586 | $ 74,343 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 17 | gpm | $ 19,240 | $ 38,481 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.01 | ft^3 | $ 270 | $ 270 | $ - | |
| 20 | Process Pump | 2 | 100% | 218 | gpm | $ 50,472 | $ 100,944 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,116,279** | **$ 4,608,712** | |

## Power/Utilities

Power Cost: $ 0.07 $/kWhr
Effiicency: 0.85

| ID | Unit Operation | Total HP | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 79.12 | $ 37.93 | $ 113.78 | $ 38,229 | $ 0.29 | $ 0.01 |
| 3 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 395.59 | $ 189.63 | $ 568.88 | $ 191,145 | $ 1.43 | $ 0.05 |
| 5 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.18 | $ 1.04 | $ 3.13 | $ 1,051 | $ 0.01 | $ 0.00 |
| 11 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.43 | $ 0.21 | $ 0.62 | $ 210 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 5.21 | $ 2.50 | $ 7.49 | $ 2,515 | $ 0.02 | $ 0.00 |
| 17 | Thickener | - | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.87 | $ 0.42 | $ 1.25 | $ 419 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 10.88 | $ 5.21 | $ 15.64 | $ 5,255 | $ 0.04 | $ 0.00 |
| | **TOTAL:** | **494** | **$ 236.93** | **$ 710.79** | **$ 238,824** | **$ 1.79** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 102.43 | kg/hr | $ 0.940 | / kg | $ 770.25 | $ 2,310.75 | $ 776,414 | $ 5.82 | $ 0.19 |
| 2 | D2EHPA | 5.39 | kg/hr | $ 5.000 | / kg | $ 215.64 | $ 646.91 | $ 217,361 | $ 1.63 | $ 0.05 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.83 | mt/hr | $ 270.000 | / mt | $ 1,794.05 | $ 5,382.16 | $ 1,808,407 | $ 13.55 | $ 0.45 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,779.94** | **$ 8,339.83** | **$ 2,802,181** | **$ 20.99** | **$ 0.69** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,017** **$ 9,051** **$ 3,041,006** **$ 22.78** **$ 0.75**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-1 | By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 1062.21 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 4020.89 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 1062.15 | gpm | HCL Dose | 5.45 | g/L of Feed |
| Mn Feed Rate | 4020.67 | lpm | | | |
| Co Precip Mass Flow | 0.05 | tph | | | |
| Mn Precip Mass Flow | 0.66 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 6.15 | g/L of Feed | | | |
| NaOH Dose | 3.58 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 15,933 | gal | $ 20,267 | $ 40,534 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 24 | HP | $ 95,955 | $ 191,910 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 1,062 | gpm | $ 92,410 | $ 92,410 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 0.24 | ft^3 | $ 3,242 | $ 3,242 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 15,932 | gal | $ 20,266 | $ 40,532 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 24 | HP | $ 95,952 | $ 191,904 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 1,062 | gpm | $ 92,409 | $ 92,409 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 3 | ft^3 | $ 20,759 | $ 20,759 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 673,701** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 47.80 | Effiicency: | 0.85 | | $ 22.91 | $ 68.74 | $ 23,096 | $ 0.17 | $ 0.01 |
| 3 | Process Pump | 26.56 | | | | $ 12.73 | $ 38.19 | $ 12,831 | $ 0.10 | $ 0.00 |
| 4 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.04 | | | | $ 0.02 | $ 0.06 | 19 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 47.80 | | | | $ 22.91 | $ 68.73 | $ 23,095 | $ 0.17 | $ 0.01 |
| 8 | Process Pump | 26.55 | | | | $ 12.73 | $ 38.19 | $ 12,830 | $ 0.10 | $ 0.00 |
| 9 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 0.52 | | | | $ 0.25 | $ 0.75 | $ 254 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **149** | | | | **$ 71.55** | **$ 214.66** | **$ 72,125** | **$ 0.54** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 1.48 | mt/hr | $ 360.000 | / mt | $ 4,276.51 | $ 12,829.53 | $ 4,310,722 | $ 32.29 | $ 1.07 |
| 2 | NaOH | 0.86 | mt/hr | $ 125.000 | / mt | $ 864.68 | $ 2,594.04 | $ 871,596 | $ 6.53 | $ 0.22 |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 0.04 | $ 0.11 | 36 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 5,141.22** | **$ 15,423.67** | **$ 5,182,354** | **$ 38.82** | **$ 1.29** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 5,213** **$ 15,638** **$ 5,254,479** **$ 39.36** **$ 1.30**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | WK No. 13 | Date: | 3/18/2023 |
| Case: | WK-2 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | an |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | N/A |
| *(unit conversion)* | 12000 | TPD | Project NPV: | (1,928,199,815) |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 22.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | WK No. 13 | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | WK-2 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 233,537,701 | | | | | |
| Constant Dollar Annual OPEX | $ | 282,811,518 | $ | 70.14 | /ton | $ 1,910.7 | /kg |
| Constant Dollar Annual REE Rev | $ | 32,506,257 | $ | 8.06 | /ton | $ 219.62 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | |
|---|---|---|---|---|
| Payback Period | 25.5 | years | *22.5* | *operating years* |
| Internal Rate of Return | N/A | | | |
| Net Present Value | $ (1,928,199,815) | | | |
| Total Severence Tax Paid | $ - | | | |
| Total State and Federal Tax Paid | $ - | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-2 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 283 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 6.36 | 18.01 | 8.17 | | | | | | | 8.17 |
| Yttrium | 7.19 | 20.36 | 9.23 | | | | | | | 9.23 |
| Lanthanum | 16.54 | 46.84 | 21.25 | | | | | | | 21.25 |
| Cerium | 37.25 | 105.48 | 47.85 | | | | | | | 47.85 |
| Praseodymium | 3.83 | 10.85 | 4.92 | | | | | | | 4.92 |
| Neodymium | 17.35 | 49.14 | 22.29 | | | | | | | 22.29 |
| Samarium | 3.47 | 9.82 | 4.46 | | | | | | | 4.46 |
| Europium | 0.58 | 1.63 | 0.74 | | | | | | | 0.74 |
| Gadolinium | 2.59 | 7.34 | 3.33 | | | | | | | 3.33 |
| Terbium | 0.44 | 1.25 | 0.57 | | | | | | | 0.57 |
| Dysprosium | 1.53 | 4.32 | 1.96 | | | | | | | 1.96 |
| Holmium | 0.47 | 1.34 | 0.61 | | | | | | | 0.61 |
| Erbium | 1.04 | 2.94 | 1.33 | | | | | | | 1.33 |
| Thulium | 0.19 | 0.53 | 0.24 | | | | | | | 0.24 |
| Ytterbium | 0.96 | 2.71 | 1.23 | | | | | | | 1.23 |
| Lutetium | 0.22 | 0.63 | 0.28 | | | | | | | 0.28 |
| Manganese | -- | 230.70 | 104.64 | | | 0.00 | -- | | | 104.64 |
| Cobalt | -- | 24.20 | 10.98 | | | 0.00 | -- | | | 10.98 |
| Nickel | -- | 38.50 | 17.46 | | | 0.00 | -- | | | 17.46 |
| Zinc | -- | 122.40 | 55.52 | | | 0.00 | -- | | | 55.52 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 283.20 | 128.46 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 128.5 |
| Total non-REE CM | -- | 415.80 | 188.60 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 188.6 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 8.17 | 0.00 | 0.58 | 0.00 | 0.00 | 0.00 | 0.58 | 7.59 | 7.1% | |
| Yttrium | 9.23 | 0.60 | 0.00 | 0.00 | 0.03 | 0.00 | 0.64 | 8.60 | 6.9% | |
| Lanthanum | 21.25 | 3.39 | 0.00 | 0.00 | 0.08 | 0.00 | 3.47 | 17.78 | 16.3% | |
| Cerium | 47.85 | 7.42 | 0.00 | 0.01 | 0.13 | 0.00 | 7.56 | 40.29 | 15.8% | |
| Praseodymium | 4.92 | 0.85 | 0.00 | 0.00 | 0.01 | 0.00 | 0.87 | 4.05 | 17.7% | |
| Neodymium | 22.29 | 3.56 | 0.00 | 0.01 | 0.06 | 0.00 | 3.62 | 18.67 | 16.3% | |
| Samarium | 4.46 | 0.71 | 0.00 | 0.00 | 0.01 | 0.00 | 0.73 | 3.73 | 16.3% | |
| Europium | 0.74 | 0.12 | 0.00 | 0.00 | 0.00 | 0.00 | 0.12 | 0.62 | 16.7% | |
| Gadolinium | 3.33 | 0.42 | 0.00 | 0.00 | 0.01 | 0.00 | 0.43 | 2.90 | 12.9% | |
| Terbium | 0.57 | 0.03 | 0.00 | 0.00 | 0.00 | 0.00 | 0.03 | 0.54 | 4.8% | |
| Dysprosium | 1.96 | 0.16 | 0.00 | 0.00 | 0.01 | 0.00 | 0.17 | 1.79 | 8.6% | |
| Holmium | 0.61 | 0.03 | 0.00 | 0.00 | 0.00 | 0.00 | 0.03 | 0.58 | 4.5% | |
| Erbium | 1.33 | 0.06 | 0.00 | 0.00 | 0.00 | 0.00 | 0.06 | 1.27 | 4.6% | |
| Thulium | 0.24 | 0.01 | 0.00 | 0.00 | 0.00 | 0.00 | 0.01 | 0.23 | 3.2% | |
| Ytterbium | 1.23 | 0.04 | 0.00 | 0.00 | 0.00 | 0.00 | 0.04 | 1.19 | 3.2% | |
| Lutetium | 0.28 | 0.01 | 0.00 | 0.00 | 0.00 | 0.00 | 0.01 | 0.28 | 2.2% | |
| Manganese | 104.64 | 0.10 | 0.00 | 34.36 | 4.17 | 0.00 | 38.63 | 66.01 | 36.9% | |
| Cobalt | 10.98 | 0.01 | 0.00 | 0.63 | 0.27 | 0.00 | 0.91 | 10.07 | 8.3% | |
| Nickel | 17.46 | 0.02 | 0.00 | 1.03 | 0.43 | 0.00 | 1.48 | 15.98 | 8.5% | |
| Zinc | 55.52 | 0.05 | 0.00 | 2.41 | 11.16 | 0.00 | 13.61 | 41.91 | 24.5% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 128.46 | 17.40 | 0.58 | 0.03 | 0.35 | 0.00 | 18.35 | 110.10 | 14.3% | <---- |
| Total non-REE CM | 188.60 | 0.17 | 0.00 | 38.43 | 16.03 | 0.00 | 54.63 | 133.97 | 29.0% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 224 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 12 | ton | $ | 1.50 |
| Dust/Volatiles | 20 | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | | |
|---|---|---|
| 0.42 | TPD REE | |
| 0.014 | TPD Sc | |
| 1.31 | TPD CM | |
| 0.14 | REE | |
| 7.1% | Sc | |
| 32.8% | Mn | |
| 14.1% | Metal-S | |
| 500 | Feed | |
| 283 | Grade | |
| 25.5 | PP | |
| N/A | IRR | |
| -1928199815 | NPV | |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-2 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 4.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 13.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | Based on Design |
| Dissolved Solids in Leach | 53.4% | % | WK2/WK2 Mass Balance-V1 |
| Leaching Retention Time | 300 | min | Based on Design |
| Thickener UF % Solids | 20% | % | VT Override from 6.53% \| WK2/WK2 Mass Balance-V1 |
| Residue Settling Rate | 15.0 | ft/hr | Not Available |
| **Reagent Use** | | | |
| Acid Consumption | 224.1 | kg/ton | WK2/WK2 Mass Balance-V1 |
| Bioleaching Sol'n Consumption | 0 | L/ton | |
| Polymer Dosage | 0.016 | g/L | WK1/WK1 Samples |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 11.72 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 12.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 72.0 | min | UK: Design |
| Precip Solid Yield | 5.72 | g/L | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| Precip Settling Rate | 2.24 | ft/hr | UK: Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 0.9% | % | VT Override from 0.52% \| UK: SHXP-C4/SHXP-C4 Samples |
| **Reagent Use** | | | |
| NaOH Dose | 2.95 | g/L of Feed | UK: SHXP-C4/SHXP-C4 Mass Balance-F2 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 897.48 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 283 | ppm | |
| Grade After Sorter | 283 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 480 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 581 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 480 | tph | |
| Leaching Residue Rate | 223.536 | tph | |
| Slurry Flow (feed) | 8880 | gpm | |
| Product Flow | 8,322 | gpm | |
| Thickener Feed % Solids (eff) | 9% | % | |
| Liquid Density | 1.046 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 133.35 | g/L | |
| Leaching Vessel Size | 51230.77 | gal | |
| Leaching Vessel Size | 193.93 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 111.77 | tph | 70 to 200 tph |
| Thickener Area | 201.48 | ft^2 | |
| Thickener Diameter | 16.02 | ft | ideal < 100' |
| Liquid Balance | 640.23 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 8,322 | gpm | |
| Slurry Feed Rate | 31,502 | lpm | |
| Precip Mass Flow | 11.91 | tph | |
| Thickener Feed % Solids (eff) | 0.55% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.042 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 189.01 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 56.11 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 13.99 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 891 | ft^2 | |
| Thickener Diameter | 33.68 | ft | ideal < 100' |

*PM5: REE Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| | | | |
| **Process Parameters** | | | |
| Precip Conditioning Time | 72 | min | Based on Design |
| Precip Solid Yield | 2.34 | g/L | WK2/WK2 Mass Balance-V1 |
| Precip Settling Rate | 0.044 | ft/hr | Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.23% | % | VT Override from 0.13% \| WK2/WK2 Mass Balance-V1 |
| | | | |
| **Reagent Use** | | | |
| NaOH Dose | 1.74 | g/L of Feed | WK2/WK2 Mass Balance-V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| | | | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 7892.66 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 8308 | gpm | |
| Slurry Feed Rate | 31,449 | lpm | |
| Precip Mass Flow | 4.86 | tph | |
| Thickener Feed % Solids (eff) | 0.2% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.041 | g/cc | |
| | | | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 452.87 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 22.91 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 5.71 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 7671 | ft^2 | |
| Thickener Diameter | 98.85 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | 98.77 | m/m | target = 6.35 |

*PM6: RE-Redissolution*

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 1.2% | % | VT Override from 10.73% \|\| WK2/WK2 Mass Balance-V1 |
| Dissolved Solids in Re-Leach | 2% | % | WK2/WK2 Mass Balance-V1 |
| Leaching Retention Time | 120 | min | Based on Design |
| **Reagent Use** | | | |
| Acid Consumption | 873.3 | kg/ton | WK2/WK2 Mass Balance-V1 |
| | 0.0 | | WK2/WK2 Mass Balance-V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| Material Balance | | | |
|---|---|---|---|
| Solid Feed Rate | 4.86 | tph | |
| Slurry Flow (feed) | 1,677 | gpm | |
| Product Flow | 1,677 | gpm | |
| Liquid Density | 1.008 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 190.4 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 4.96 | v/v | target = 4.8, based on test data |
| Liquid Balance | 5.71 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 11.6 | g/L | |

*PM7: REE Oxalate Precipitation*

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | Based on Design |
| Precip Solid Yield | 0.1 | g/L | WK2/WK2 Mass Balance-V1 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Thickener UF %Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| NaOH Dose | 2.11 | g/L | WK2/WK2 Mass Balance-V1 |
| Oxalic Acid Dose | 6.27 | g/L | WK2/WK2 Mass Balance-V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| Material Balance | | | |
|---|---|---|---|
| Slurry Feed Rate | 1676.59 | gpm | |
| Slurry Feed Rate | 6346.58 | lpm | |
| Precip Mass Flow | 0.04 | tph | |
| Thickener Feed % Solids (eff) | 0.01% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.03 | tph | |
| Liquid Density | 1.008 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 31.73 | m^3 | |
| Filter Press Area | 0.21 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.05 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 17.91 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.67 | m/m | |

*PM8: Sc SX Separation*

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | Based on Design |
| Settler Residence Time | 15.00 | min | Based on Design |
| Loading: Number of Stages | 2.00 | - | Based on Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | Based on Design |
| Loading O:A Advance Ratio | 0.03 | v/v | Based on Design |
| Scrubbing: Number of Stages | | - | Not Applicable |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | Not Applicable |
| Scrubbing O:A Advance Ratio | | v/v | Not Applicable |
| Stripping: Number of Stages | 2.00 | - | Based on Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | Based on Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | Based on Design |
| Sc2O3 Grade | 2% | % | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Precip Solid Yield | 0.4 | g/L | Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | |
| Thickener UF % Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | Data not available |
| Diluent Density | 0.84 | g/ml | |
| Extractant Density | 0.92 | g/ml | |
| Extractant Concentration | 5% | % Volume | |
| Scrubbing Acid Molarity | | mol/L | Not Applicable |
| Srubbing Acid Molar Mass | | g/mol | Not Applicable |
| Scrub Acid Stock Conc | 37% | % | Not Applicable |
| Stripping Solution Molarity | 2.00 | Mol/L | |
| Stripping Solution Acid Molar M | 98.000 | g/mol | |
| Strip Acid Stock Conc | 93% | % | |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | Not Applicable |
| Neutralization Molar Mass | | g/mol | Not Applicable |
| Neutralization Stock Conc | 50% | % | Not Applicable |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| Material Balance (SX) | | |
|---|---|---|
| PLS Flow | 8322.0 | gpm |
| Raffinate Rate: | 8322.0 | gpm |
| Load Org Recycle | 8093.2 | gpm |
| Load Org Feed | 8322.0 | gpm |
| Loag Org Advance | 228.8 | gpm |
| Scrub Aq Product | 0.0 | gpm |
| Scrub Aq Recycle | 0.0 | gpm |
| Scrub Aq Feed | 0.0 | gpm |
| Scurb Org Product | 228.8 | gpm |
| Strip Aq Product | 18.3 | gpm |
| Strip Aq Recycle | 210.5 | gpm |
| Strip Aq Feed | 228.8 | gpm |
| Strip Org Product | 228.8 | gpm |
| Extraction Feed (O+A): | 16643.9 | gpm |
| Scrub Feed (O+A): | 228.8 | gpm |
| Strip Feed (O+A): | 457.6 | gpm |
| **Material Balance (Precip)** | | |
| Strip Solution Flow | 18.3 | gpm |
| Solid Density | 3.4 | g/cc |
| Solid Product Rate | 0.00 | tph |
| Liquid Density (after precip) | 1.13 | g/cc |
| Thickener Feed % Solids (eff) | 0.03% | % |
| **Total Volumes** | | |
| Total Extraction Volume | 532606 | gal |
| Total Scrubbing Volume | 0 | gal |
| Total Stripping Volume | 14643 | gal |
| Extraction Org Volume | 266303.04 | gal |
| Scrubbing Org Volume | 0.00 | gal |
| Stripping Org Volume | 7321.63 | gal |
| Total Org Volume | 273624.67 | gal |

| **Other Indicators** | | | |
|---|---|---|---|
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 10.36 | m^3 | |
| Filter Press Area | 0.02 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM9: CM Precipitation*

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 0 | min | N/A |
| Precip Solid Yield | - | g/L | N/A |
| Precip Settlng Rate | 10 | ft/hr | N/A |
| Thickener UF % Solids | 25% | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 7.24 | g/L of Feed | N/A |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 0.00 | min | N/A |
| Thickener Capacity | 0.00 | ft^2/tph | N/A |

| Material Balance | | | |
|---|---|---|---|
| Slurry Feed Rate | 8302 | gpm | |
| Slurry Feed Rate | 31,428 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.041 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM10: CM Redissolution*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 2.0% | % | N/A |
| Dissolved Solids in Re-Leach | 100% | % | N/A |
| Leaching Retention Time | 45 | min | N/A |
| **Reagent Use** | | | |
| Acid Consumption | 441 | kg/ton | N/A |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 0.00 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 0 | gpm | |
| Liquid Density | 1.000 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | |
| Releach Conc. Factor | #DIV/0! | v/v | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | - | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

*PM11: CM Recovery*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 0.88 | g/L | WK2/WK2 Mass Balance-V1 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | | % | Not Applicable |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 2.78 | g/L | WK2/WK2 Mass Balance-V1 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Na2S Dose | 1.30 | g/L of Feed | WK2/WK2 Mass Balance-V1 |
| NaOH Dose | 8.70 | g/L of Feed | WK2/WK2 Mass Balance-V1 |
| HCL Dose | 4.93 | g/L of Feed | WK2/WK2 Mass Balance-V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 1676.54 | gpm | |
| Co Slurry Feed Rate | 6346.38 | lpm | |
| Co Precip Mass Flow | 0.37 | tph | |
| Co Thickener Feed % Solids (eff) | 0.09% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 1676.10 | gpm | |
| Mn Slurry Feed Rate | 6344.74 | lpm | |
| Mn Precip Mass Flow | 1.17 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.28% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.01 | g/cc | |
| Mn Liquid Density | 1.01 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 95.2 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 1.75 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.44 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 95.2 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 5.5 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 1.37 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

*PM12: Biooxidation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Requirec | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-2 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ 19,585,772 | $ - | $ 4,896,443 | $ 21,544,350 | $ 26,822,715 | $ 72,849,280 | IIIIIIII

# OPERATING COST ESTIMATE

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-2 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 20.76 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 7.97 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 233,537,701 | $ Total | $ 2,316.84 | $ 6,950.53 | $ 2,335,377 | $ 15.78 | $ 0.58 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 2.08 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 32,506,257 | $ Total | $ 161.24 | $ 483.72 | $ 162,531 | $ 1.10 | $ 0.04 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 3.11 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 233,537,701 | $ Total | $ 2,316.84 | $ 6,950.53 | $ 2,335,377 | $ 15.78 | $ 0.58 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 9,776.38** | **$ 29,329.13** | **$ 9,854,587** | **$ 66.58** | **$ 2.44** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 3.48 | $ 0.13 |
| PM2 | Roasting | PR | 198 | | $ 94.99 | $ 284.96 | $ 95,747 | $ 0.65 | $ 0.02 |
| PM3 | Leaching | PR | 5027 | | $ 2,409.90 | $ 7,229.70 | $ 2,429,178 | $ 16.41 | $ 0.60 |
| PM4 | Al Precipitation | PR | 941 | | $ 450.93 | $ 1,352.78 | $ 454,533 | $ 3.07 | $ 0.11 |
| PM5 | REE Precipitation | PR | 951 | | $ 455.73 | $ 1,367.20 | $ 459,380 | $ 3.10 | $ 0.11 |
| PM6 | RE-Redissolution | PR | 344 | | $ 164.75 | $ 494.26 | $ 166,072 | $ 1.12 | $ 0.04 |
| PM7 | REE Oxalate Precipitation | PR | 143 | | $ 68.33 | $ 205.00 | $ 68,881 | $ 0.47 | $ 0.02 |
| PM8 | Sc SX Separation | PR | 520 | | $ 249.21 | $ 747.64 | $ 251,206 | $ 1.70 | $ 0.06 |
| PM9 | CM Precipitation | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM10 | CM Redissolution | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM11 | CM Recovery | PR | 236 | | $ 113.08 | $ 339.24 | $ 113,985 | $ 0.77 | $ 0.03 |
| PM12 | Biooxidation | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM13 | Reserved | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM14 | Reserved | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM15 | Reserved | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | **9424.4** | | **$ 4,517.61** | **$ 13,552.82** | **$ 4,553,747.30** | **$ 30.77** | **$ 1.13** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.69 | $ 0.03 |
| PM2 | Roasting | CR | | $ 1,281.99 | $ 3,845.96 | $ 1,292,244 | $ 8.73 | $ 0.32 |
| PM3 | Leaching | CR | | $ 190,016.68 | $ 570,050.05 | $ 191,536,818 | $ 1,294.04 | $ 47.50 |
| PM4 | Al Precipitation | CR | | $ 5,572.39 | $ 16,717.17 | $ 5,616,970 | $ 37.95 | $ 1.39 |
| PM5 | REE Precipitation | CR | | $ 3,287.85 | $ 9,863.55 | $ 3,314,152 | $ 22.39 | $ 0.82 |
| PM6 | RE-Redissolution | CR | | $ 9,166.95 | $ 27,500.85 | $ 9,240,285 | $ 62.43 | $ 2.29 |
| PM7 | REE Oxalate Precipitation | CR | | $ 12,270.70 | $ 36,812.09 | $ 12,368,863 | $ 83.57 | $ 3.07 |
| PM8 | Sc SX Separation | CR | | $ 2,924.06 | $ 8,772.18 | $ 2,947,453 | $ 19.91 | $ 0.73 |
| PM9 | CM Precipitation | CR | | $ - | $ - | $ - | $ - | $ - |
| PM10 | CM Redissolution | CR | | $ - | $ - | $ - | $ - | $ - |
| PM11 | CM Recovery | CR | | $ 4,734.15 | $ 14,202.44 | $ 4,772,019 | $ 32.24 | $ 1.18 |
| PM12 | Biooxidation | CR | | $ - | $ - | $ - | $ - | $ - |
| PM13 | Reserved | CR | | $ - | $ - | $ - | $ - | $ - |
| PM14 | Reserved | CR | | $ - | $ - | $ - | $ - | $ - |
| PM15 | Reserved | CR | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ 229,355.57** | **$ 688,066.70** | **$ 231,190,409.56** | **$ 1,561.94** | **$ 57.34** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM8 | Sc SX Separation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM9 | CM Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM10 | CM Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM11 | CM Recovery | LS | | $ - | $ - | $ - | $ - | $ - |
| PM12 | Biooxidation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM13 | Reserved | LS | | $ - | $ - | $ - | $ - | $ - |
| PM14 | Reserved | LS | | $ - | $ - | $ - | $ - | $ - |
| PM15 | Reserved | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 5,365 | $ 0.10 | $ 178.83 | $ 536.49 | $ 180,259 | $ 1.22 | $ 0.04 |
| Non-Hazardous Precipitate | WD | Tons | 286 | $ 1.50 | $ 142.91 | $ 428.74 | $ 144,056 | $ 0.97 | $ 0.04 |
| Dust/Volatiles | WD | Tons | 28,800 | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 321.74** | **$ 965.23** | **$ 324,315.77** | **$ 2.19** | **$ 0.08** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 234,195** | **$ 702,585** | **$ 236,068,473** | **$ 1,595** | **$ 59** |

## Overhead Costs

| Allocation | Code | Rate | Basis | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% % of Direct Cost | $ 29,329 Daily Direct Cost | $ 1,466.46 | $ 4,399.37 | $ 1,478,188 | $ 9.99 | $ 0.37 |
| Variable Portion | OH | 15% % of Direct Cost | $ 702,585 Daily Direct Cost | $ 35,129.24 | $ 105,387.71 | $ 35,410,271 | $ 239.24 | $ 8.78 |
| **TOTAL OVERHEAD COSTS** | | | | **$ 36,595.69** | **$ 109,787.08** | **$ 36,888,458.93** | **$ 249.22** | **$ 9.15** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | WK No. 13 | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | WK-2 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 9,776.38 | $ 29,329.13 | $ 9,854,587 | $ 67 | $ 2.44 | I |
| Power | $ 4,517.61 | $ 13,552.82 | $ 4,553,747 | $ 31 | $ 1.13 | |
| Consumables | $ 229,355.57 | $ 688,066.70 | $ 231,190,410 | $ 1,562 | $ 57.34 | IIIIIIIIIIIIIIIIIIIIIIIIII |
| Leasing/Tolling | $ - | $ - | $ - | $ - | $ - | |
| Waste Disposal | $ 321.74 | $ 965.23 | $ 324,316 | $ 2 | $ 0.08 | |
| General Overhead | $ 36,595.69 | $ 109,787.08 | $ 36,888,459 | $ 249 | $ 9.15 | IIII |
| TOTAL | $ 280,567 | $ 841,701 | $ 282,811,518 | $ 1,911 | $ 70.14 | |

## Summary by Process Module

| | Process Module | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 611 | $ 1,834 | $ 616,370 | $ 4.16 | $ 0.15 | |
| PM2 | Roasting | $ 1,377 | $ 4,131 | $ 1,387,991 | $ 9.38 | $ 0.34 | |
| PM3 | Leaching | $ 192,427 | $ 577,280 | $ 193,965,996 | $ 1,310.45 | $ 48.11 | IIIIIIIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 6,023 | $ 18,070 | $ 6,071,503 | $ 41.02 | $ 1.51 | |
| PM5 | REE Precipitation | $ 3,744 | $ 11,231 | $ 3,773,532 | $ 25.49 | $ 0.94 | |
| PM6 | RE-Redissolution | $ 9,332 | $ 27,995 | $ 9,406,357 | $ 63.55 | $ 2.33 | I |
| PM7 | REE Oxalate Precipitation | $ 12,339 | $ 37,017 | $ 12,437,745 | $ 84.03 | $ 3.08 | I |
| PM8 | Sc SX Separation | $ 3,173 | $ 9,520 | $ 3,198,659 | $ 21.61 | $ 0.79 | |
| PM9 | CM Precipitation | $ - | $ - | $ - | $ - | $ - | |
| PM10 | CM Redissolution | $ - | $ - | $ - | $ - | $ - | |
| PM11 | CM Recovery | $ 4,847 | $ 14,542 | $ 4,886,005 | $ 33.01 | $ 1.21 | |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| NA | Unallocated | $ 46,694 | $ 140,081 | $ 47,067,362 | $ 318 | $ 11.67 | IIIII |
| | TOTAL | $ 280,567 | $ 841,701 | $ 282,811,518 | $ 1,911 | $ 70.14 | |

## Summary by Cost Center

| Cost Center | Code | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| Operating Labor | OL | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 21 | $ 0.76 | |
| Technical Labor | TL | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 8 | $ 0.29 | |
| Power/Utilities | PR | $ 4,517.61 | $ 13,552.82 | $ 4,553,747 | $ 31 | $ 1.13 | |
| Lease Agreements | LS | $ - | $ - | $ - | $ - | $ - | |
| Consumables/Reagents | CR | $ 229,355.57 | $ 688,066.70 | $ 231,190,410 | $ 1,562 | $ 57.34 | IIIIIIIIIIIIIIIIIIIIIIIIII |
| Waste Disposal | WD | $ 321.74 | $ 965.23 | $ 324,316 | $ 2 | $ 0.08 | |
| Plant Overhead | OH | $ 39,217.40 | $ 117,652.19 | $ 39,531,135 | $ 267 | $ 9.80 | IIII |
| General Sales, and Administration | GSA | $ 2,935.37 | $ 8,806.12 | $ 2,958,857 | $ 20 | $ 0.73 | |
| Reserved | | | | | | | |
| TOTAL | | $ 280,567 | $ 841,700.95 | $ 282,811,518 | $ 1,911 | $ 70.14 | |

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | WK No. 13 | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | WK-2 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 32,506,257

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 265 | 283 |
| Contained Value | ($/ton) | $ 5.9 | $ 111.2 |
| Feed Basket Price | ($/kg) | $ 24.7 | $ 432.9 |
| Product Basket Price | ($/kg) | $ 20.0 | $ 223.0 |
| Production Rate | (kg/day) | 426.59 | 440.52 |
| | (t/day) | 0.43 | 0.44 |
| | (t/yr) | 143.33 | 148.01 |
| | (% of US Demand) | 1.4% | |

## Revenue Contribution Table

| Element | REE/CM Price ($/kg) [1] | Product Revenue ($/hr) MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | REE Revenue ($/day) |
|---|---|---|---|---|---|---|---|
| Scandium | $ 4,200.00 | $ 1.1 | $ 3,735.1 | $ - | $ - | $ - | $ 89,668 |
| Yttrium | $ 6.00 | $ 3.4 | $ - | $ - | $ - | $ - | $ 82 |
| Lanthanum | $ 2.00 | $ 6.0 | $ - | $ - | $ - | $ - | $ 143 |
| Cerium | $ 2.00 | $ 13.7 | $ - | $ - | $ - | $ - | $ 328 |
| Praseodymium | $ 52.00 | $ 40.2 | $ - | $ - | $ - | $ - | $ 966 |
| Neodymium | $ 42.00 | $ 130.9 | $ - | $ - | $ - | $ - | $ 3,142 |
| Samarium | $ 2.11 | $ 1.3 | $ - | $ - | $ - | $ - | $ 31 |
| Europium | $ 34.00 | $ 3.6 | $ - | $ - | $ - | $ - | $ 86 |
| Gadolinium | $ 32.00 | $ 11.5 | $ - | $ - | $ - | $ - | $ 277 |
| Terbium | $ 400.00 | $ 8.9 | $ - | $ - | $ - | $ - | $ 215 |
| Dysprosium | $ 230.00 | $ 32.0 | $ - | $ - | $ - | $ - | $ 768 |
| Holmium | $ 52.96 | $ 1.2 | $ - | $ - | $ - | $ - | $ 28 |
| Erbium | $ 34.00 | $ 1.7 | $ - | $ - | $ - | $ - | $ 41 |
| Thulium | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Ytterbium | $ 28.72 | $ 0.9 | $ - | $ - | $ - | $ - | $ 22 |
| Lutetium | $ 796.64 | $ 3.6 | $ - | $ - | $ - | $ - | $ 87 |
| Manganese | $ 0.005 | $ - | $ - | $ 0.3 | $ - | $ - | $ 7 |
| Cobalt | $ 22.00 | $ - | $ - | $ - | $ 4.5 | $ - | $ 108 |
| Nickel | $ 18.00 | $ - | $ - | $ - | $ 5.8 | $ - | $ 139 |
| Zinc | $ 3.02 | $ - | $ - | $ - | $ 25.3 | $ - | $ 607 |
| Reserved | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Total: | | $ 260.1 | $ 3,735.1 | $ 0.3 | $ 35.6 | $ - | $ 96,745 |

[1] REEs listed as $/kg REO; Mn listed as $/kg 44% Mn Ore; Co listed as $/kg metal content

## Coal Revenue

List Coal Revenue Sources Here:

**Annual Coal Revenue:** $ -

## Other Revenue

List Other Revenue Sources Here:

**Annual Other Revenue:** $ -

## PRO FORMA CASH FLOW

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-2 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 35,520,465 | $ 36,586,078 | $ 37,683,661 | $ 38,814,171 | $ 39,978,596 | $ 41,177,954 | $ 42,413,292 | $ 43,685,691 | $ 44,996,262 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 35,520,465 | $ 36,586,078 | $ 37,683,661 | $ 38,814,171 | $ 39,978,596 | $ 41,177,954 | $ 42,413,292 | $ 43,685,691 | $ 44,996,262 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 700,613 | $ 5,002,470 | $ 315,907,399 | $ 324,597,368 | $ 333,530,603 | $ 342,713,356 | $ 352,152,004 | $ 361,853,048 | $ 371,823,115 | $ 382,068,954 | $ 392,597,439 |
| *- Operating Costs* | $ - | $ - | $ - | $ 309,035,782 | $ 318,306,856 | $ 327,856,061 | $ 337,691,743 | $ 347,822,495 | $ 358,257,170 | $ 369,004,885 | $ 380,075,032 | $ 391,477,283 |
| *- Loan #1 Interest* | $ - | $ 700,613 | $ 647,459 | $ 591,116 | $ 531,392 | $ 468,084 | $ 400,979 | $ 329,846 | $ 254,446 | $ 174,522 | $ 89,803 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 4,355,011 | $ 4,024,605 | $ 3,674,375 | $ 3,303,131 | $ 2,909,613 | $ 2,492,483 | $ 2,050,325 | $ 1,581,638 | $ 1,084,830 | $ 558,214 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 2,255,896 | $ 2,084,746 | $ 1,903,326 | $ 1,711,022 | $ 1,507,179 | $ 1,291,106 | $ 1,062,069 | $ 819,289 | $ 561,942 |
| Gross Profit | $ - | $ (700,613) | $ (5,002,470) | $ (280,386,934) | $ (288,011,290) | $ (295,846,942) | $ (303,899,186) | $ (312,173,408) | $ (320,675,095) | $ (329,409,822) | $ (338,383,263) | $ (347,601,177) |
| **Book Deductions** | $ - | $ 875,766 | $ 6,320,290 | $ 9,149,575 | $ 8,463,357 | $ 7,828,605 | $ 7,241,459 | $ 6,698,350 | $ 6,195,974 | $ 5,731,276 | $ 5,301,430 | $ 4,903,823 |
| *- CAPEX #1 Depreciation* | $ - | $ 875,766 | $ 810,084 | $ 749,328 | $ 693,128 | $ 641,143 | $ 593,058 | $ 548,578 | $ 507,435 | $ 469,377 | $ 434,174 | $ 401,611 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 5,510,206 | $ 5,096,941 | $ 4,714,670 | $ 4,361,070 | $ 4,033,990 | $ 3,731,441 | $ 3,451,583 | $ 3,192,714 | $ 2,953,260 | $ 2,731,766 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 3,303,306 | $ 3,055,558 | $ 2,826,391 | $ 2,614,412 | $ 2,418,331 | $ 2,236,956 | $ 2,069,184 | $ 1,913,996 | $ 1,770,446 |
| Taxable Before Depletion | $ - | $ (1,576,379) | $ (11,322,760) | $ (289,536,509) | $ (296,474,646) | $ (303,675,547) | $ (311,140,645) | $ (318,871,758) | $ (326,871,068) | $ (335,141,098) | $ (343,684,693) | $ (352,505,000) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ - | $ (788,190) | $ (5,661,380) | $ (144,768,254) | $ (148,237,323) | $ (151,837,774) | $ (155,570,323) | $ (159,435,879) | $ (163,435,534) | $ (167,570,549) | $ (171,842,346) | $ (176,252,500) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 4,972,865 | $ 5,122,051 | $ 5,275,713 | $ 5,433,984 | $ 5,597,003 | $ 5,764,914 | $ 5,937,861 | $ 6,115,997 | $ 6,299,477 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (1,576,379) | $ (11,322,760) | $ (289,536,509) | $ (296,474,646) | $ (303,675,547) | $ (311,140,645) | $ (318,871,758) | $ (326,871,068) | $ (335,141,098) | $ (343,684,693) | $ (352,505,000) |
| *- Income Tax* | $ - | $ (409,859) | $ (2,943,918) | $ (75,279,492) | $ (77,083,408) | $ (78,955,642) | $ (80,896,568) | $ (82,906,657) | $ (84,986,478) | $ (87,136,686) | $ (89,358,020) | $ (91,651,300) |
| Net Profit/Loss | $ - | $ (1,166,521) | $ (8,378,843) | $ (214,257,017) | $ (219,391,238) | $ (224,719,905) | $ (230,244,077) | $ (235,965,101) | $ (241,884,590) | $ (248,004,413) | $ (254,326,673) | $ (260,853,700) |
| **Book Deductions** | $ - | $ 875,766 | $ 6,320,290 | $ 9,149,575 | $ 8,463,357 | $ 7,828,605 | $ 7,241,459 | $ 6,698,350 | $ 6,195,974 | $ 5,731,276 | $ 5,301,430 | $ 4,903,823 |
| *+ CAPEX #1 Depreciation* | $ - | $ 875,766 | $ 810,084 | $ 749,328 | $ 693,128 | $ 641,143 | $ 593,058 | $ 548,578 | $ 507,435 | $ 469,377 | $ 434,174 | $ 401,611 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 5,510,206 | $ 5,096,941 | $ 4,714,670 | $ 4,361,070 | $ 4,033,990 | $ 3,731,441 | $ 3,451,583 | $ 3,192,714 | $ 2,953,260 | $ 2,731,766 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 3,303,306 | $ 3,055,558 | $ 2,826,391 | $ 2,614,412 | $ 2,418,331 | $ 2,236,956 | $ 2,069,184 | $ 1,913,996 | $ 1,770,446 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ - | $ (290,754) | $ (2,058,552) | $ (205,107,442) | $ (210,927,882) | $ (216,891,300) | $ (223,002,618) | $ (229,266,751) | $ (235,688,617) | $ (242,273,137) | $ (249,025,243) | $ (255,949,877) |
| **Captial Cost** | $ 11,676,885 | $ 73,469,419 | $ 44,044,081 | $ 9,685,071 | $ 10,266,176 | $ 10,882,146 | $ 11,535,075 | $ 12,227,179 | $ 12,960,810 | $ 13,738,459 | $ 14,562,766 | $ 13,850,018 |
| *- Fixed Capital Costs* | $ 23,353,770 | $ 145,167,035 | $ 75,196,524 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 11,676,885 | $ 72,583,518 | $ 37,598,262 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 885,901 | $ 939,056 | $ 995,399 | $ 1,055,123 | $ 1,118,430 | $ 1,185,536 | $ 1,256,668 | $ 1,332,068 | $ 1,411,992 | $ 1,496,712 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 5,506,763 | $ 5,837,169 | $ 6,187,399 | $ 6,558,643 | $ 6,952,162 | $ 7,369,291 | $ 7,811,449 | $ 8,280,136 | $ 8,776,944 | $ 9,303,561 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 2,852,503 | $ 3,023,654 | $ 3,205,073 | $ 3,397,377 | $ 3,601,220 | $ 3,817,293 | $ 4,046,331 | $ 4,289,110 | $ 4,546,457 |
| **Net Cash Flow** | $ (11,676,885) | $ (73,760,173) | $ (46,102,633) | $ (214,792,513) | $ (221,194,057) | $ (227,773,446) | $ (234,537,693) | $ (241,493,931) | $ (248,649,427) | $ (256,011,596) | $ (263,588,009) | $ (269,799,895) |
| Discounted Cash Flow | $ (11,676,885) | $ (67,054,703) | $ (38,101,350) | $ (161,376,794) | $ (151,078,517) | $ (141,429,390) | $ (132,390,413) | $ (123,924,571) | $ (115,996,793) | $ (108,573,908) | $ (101,624,588) | $ (94,563,217) |
| **Cumulative DCF** | $ (11,676,885) | $ (78,731,588) | $ (116,832,938) | $ (278,209,732) | $ (429,288,250) | $ (570,717,639) | $ (703,108,053) | $ (827,032,623) | $ (943,029,416) | $ (1,051,603,324) | $ (1,153,227,912) | $ (1,247,791,129) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 233,537,701 | Payback Period | 25.5 |
| Constant Dollar OPEX | $ 282,811,518 | Internal Rate of Return | N/A |
| Constant Dollar REE Rev | $ 32,506,257 | Net Present Value | $ (1,928,199,815) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ - |

## PRO FORMA CASH FLOW

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-2 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 46,346,150** | **$ 47,736,534** | **$ 49,168,630** | **$ 50,643,689** | **$ 52,163,000** | **$ 53,727,890** | **$ 55,339,726** | **$ 56,999,918** | **$ 58,709,916** | **$ 60,471,213** | **$ 62,285,350** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 46,346,150 | $ 47,736,534 | $ 49,168,630 | $ 50,643,689 | $ 52,163,000 | $ 53,727,890 | $ 55,339,726 | $ 56,999,918 | $ 58,709,916 | $ 60,471,213 | $ 62,285,350 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 403,510,756** | **$ 415,318,249** | **$ 427,777,797** | **$ 440,611,131** | **$ 453,829,465** | **$ 467,444,349** | **$ 481,467,679** | **$ 495,911,710** | **$ 510,789,061** | **$ 526,112,733** | **$ 541,896,115** |
| *- Operating Costs* | $ 403,221,601 | $ 415,318,249 | $ 427,777,797 | $ 440,611,131 | $ 453,829,465 | $ 467,444,349 | $ 481,467,679 | $ 495,911,710 | $ 510,789,061 | $ 526,112,733 | $ 541,896,115 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 289,155 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ (357,164,607) | $ (367,581,715) | $ (378,609,167) | $ (389,967,442) | $ (401,666,465) | $ (413,716,459) | $ (426,127,953) | $ (438,911,791) | $ (452,079,145) | $ (465,641,520) | $ (479,610,765) |
| **Book Deductions** | **$ 4,536,036** | **$ 4,195,833** | **$ 3,881,146** | **$ 3,590,060** | **$ 3,320,805** | **$ 3,071,745** | **$ 2,841,364** | **$ 2,628,262** | **$ 2,431,142** | **$ 2,064,635** | **$ 751,003** |
| *- CAPEX #1 Depreciation* | $ 371,490 | $ 343,628 | $ 317,856 | $ 294,017 | $ 271,966 | $ 251,568 | $ 232,701 | $ 215,248 | $ 199,105 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 2,526,883 | $ 2,337,367 | $ 2,162,065 | $ 1,999,910 | $ 1,849,917 | $ 1,711,173 | $ 1,582,835 | $ 1,464,122 | $ 1,354,313 | $ 1,252,740 | $ - |
| *- CAPEX #3 Depreciation* | $ 1,637,663 | $ 1,514,838 | $ 1,401,225 | $ 1,296,133 | $ 1,198,923 | $ 1,109,004 | $ 1,025,829 | $ 948,891 | $ 877,725 | $ 811,895 | $ 751,003 |
| Taxable Before Depletion | $ (361,700,643) | $ (371,777,549) | $ (382,490,313) | $ (393,557,502) | $ (404,987,271) | $ (416,788,204) | $ (428,969,317) | $ (441,540,053) | $ (454,510,287) | $ (467,706,154) | $ (480,361,768) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ (180,850,321) | $ (185,888,774) | $ (191,245,156) | $ (196,778,751) | $ (202,493,635) | $ (208,394,102) | $ (214,484,658) | $ (220,770,027) | $ (227,255,144) | $ (233,853,077) | $ (240,180,884) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 6,488,461 | $ 6,683,115 | $ 6,883,608 | $ 7,090,116 | $ 7,302,820 | $ 7,521,905 | $ 7,747,562 | $ 7,979,989 | $ 8,219,388 | $ 8,465,970 | $ 8,719,949 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ (361,700,643)** | **$ (371,777,549)** | **$ (382,490,313)** | **$ (393,557,502)** | **$ (404,987,271)** | **$ (416,788,204)** | **$ (428,969,317)** | **$ (441,540,053)** | **$ (454,510,287)** | **$ (467,706,154)** | **$ (480,361,768)** |
| *- Income Tax* | $ (94,042,167) | $ (96,662,163) | $ (99,447,481) | $ (102,324,950) | $ (105,296,690) | $ (108,364,933) | $ (111,532,022) | $ (114,800,414) | $ (118,172,675) | $ (121,603,600) | $ (124,894,060) |
| Net Profit/Loss | $ (267,658,475) | $ (275,115,386) | $ (283,042,831) | $ (291,232,551) | $ (299,690,580) | $ (308,423,271) | $ (317,437,295) | $ (326,739,639) | $ (336,337,613) | $ (346,102,554) | $ (355,467,708) |
| **Book Deductions** | **$ 4,536,036** | **$ 4,195,833** | **$ 3,881,146** | **$ 3,590,060** | **$ 3,320,805** | **$ 3,071,745** | **$ 2,841,364** | **$ 2,628,262** | **$ 2,431,142** | **$ 2,064,635** | **$ 751,003** |
| *+ CAPEX #1 Depreciation* | $ 371,490 | $ 343,628 | $ 317,856 | $ 294,017 | $ 271,966 | $ 251,568 | $ 232,701 | $ 215,248 | $ 199,105 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 2,526,883 | $ 2,337,367 | $ 2,162,065 | $ 1,999,910 | $ 1,849,917 | $ 1,711,173 | $ 1,582,835 | $ 1,464,122 | $ 1,354,313 | $ 1,252,740 | $ - |
| *+ CAPEX #3 Depreciation* | $ 1,637,663 | $ 1,514,838 | $ 1,401,225 | $ 1,296,133 | $ 1,198,923 | $ 1,109,004 | $ 1,025,829 | $ 948,891 | $ 877,725 | $ 811,895 | $ 751,003 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ (263,122,439) | $ (270,919,553) | $ (279,161,686) | $ (287,642,491) | $ (296,369,775) | $ (305,351,526) | $ (314,595,930) | $ (324,111,378) | $ (333,906,470) | $ (344,037,919) | $ (354,716,705) |
| **Captial Cost** | **$ 4,819,244** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 4,819,244 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ (267,941,684) | $ (270,919,553) | $ (279,161,686) | $ (287,642,491) | $ (296,369,775) | $ (305,351,526) | $ (314,595,930) | $ (324,111,378) | $ (333,906,470) | $ (344,037,919) | $ (354,716,705) |
| Discounted Cash Flow | $ (85,374,478) | $ (78,475,744) | $ (73,511,997) | $ (68,859,326) | $ (64,498,698) | $ (60,412,172) | $ (56,582,843) | $ (52,994,800) | $ (49,633,069) | $ (46,490,040) | $ (43,575,519) |
| **Cumulative DCF** | $ (1,333,165,607) | $ (1,411,641,351) | $ (1,485,153,348) | $ (1,554,012,674) | $ (1,618,511,372) | $ (1,678,923,543) | $ (1,735,506,387) | $ (1,788,501,187) | $ (1,838,134,256) | $ (1,884,624,296) | $ (1,928,199,815) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 233,537,701 | Payback Period | 0 | 25.5 |
| Constant Dollar OPEX | $ 282,811,518 | Internal Rate of Return | 0 | N/A |
| Constant Dollar REE Rev | $ 32,506,257 | Net Present Value | 0 | $ (1,928,199,815) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ - |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-2 | By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 1.96 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 1.22 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.09 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.10 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.10 | $ 0.00 |
| 7 | Conveyor | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 3.48** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.69 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.69** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 611** **$ 1,834** **$ 616,370** **$ 4.16** **$ 0.15**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-2 | By: ks

## Module Information

Module ID: PM2
Module Name: Roasting
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 500 | tph |
| External Heat | 10.0% | % |
| Product Rate | 480 | tph |
| Roasting Temperature | 575 | deg C |
| Bin Storage Capacity | 20% | % of hourly prod. |
| Material Heat Capacity | 0.80 | kg/kg-K |
| Reaction Enthalpy | 1783.00 | kJ/kg |
| % of Feed Reacted | 50% | % |
| Scurbber Airflow | 20.00 | cfm/tph |
| Quencher Aiflow | 20.00 | cfm/tph |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Storage Bins | 2 | 100% | 100 | Tons | $ 115,387 | $ 230,773 | $ - | |
| 2 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 3 | Roaster | 1 | 100% | 581 | MBTU/hr | $ 18,994,975 | $ 18,994,975 | $ - | |
| 4 | Gas Scrubber | 1 | 100% | 10,000 | cfm | $ 57,958 | $ 57,958 | $ - | |
| 5 | Spray Chamber Quencher | 1 | 100% | 10,000 | cfm | $ 206,510 | $ 206,510 | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 19,585,772** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Storage Bins | 10.00 | Power Cost: | $ 0.07 | $/kWhr | | $ 4.79 | $ 14.38 | $ 4,832 | $ 0.03 | $ 0.00 |
| 2 | Conveyor | 31.00 | Effiicency: | 0.85 | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.10 | $ 0.00 |
| 3 | Roaster | 87.16 | | | | | $ 41.78 | $ 125.34 | $ 42,113 | $ 0.28 | $ 0.01 |
| 4 | Gas Scrubber | 50.00 | | | | | $ 23.97 | $ 71.90 | $ 24,159 | $ 0.16 | $ 0.01 |
| 5 | Spray Chamber Quencher | 20.00 | | | | | $ 9.59 | $ 28.76 | $ 9,664 | $ 0.07 | $ 0.00 |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **198** | | | | | **$ 94.99** | **$ 284.96** | **$ 95,747** | **$ 0.65** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Natural Gas | 56.03 | MCF/hr | $ 2.860 | / MCF | $ 1,281.99 | $ 3,845.96 | $ 1,292,244 | $ 8.73 | $ 0.32 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 1,281.99** | **$ 3,845.96** | **$ 1,292,244** | **$ 8.73** | **$ 0.32** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 1,377** **$ 4,131** **$ 1,387,991** **$ 9.38** **$ 0.34**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-2 | By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 13.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 480 | tph |
| Residue Rate | 224 | tph |
| Slurry Flow | 8880 | gpm |
| Leaching Time | 300 | min |
| Acid Consumption | 224.11 | kg/ton |
| Bioleaching Sol'n Consumption | 0 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 2 | -- |
| Thickener Capacity | 11.72 | ft^2/tph |
| Polymer Dose | 0.016 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 52 | 100% | 51,231 | gal | $ 179,830 | $ 9,351,134 | $ - | |
| 2 | Tank Mixer | 52 | 100% | 77 | HP | $ 185,415 | $ 9,641,604 | $ - | |
| 3 | Process Pump | 26 | 100% | 683 | gpm | $ 78,089 | $ 2,030,323 | $ - | |
| 4 | Conveyor | 1 | 100% | 480 | TPH | $ 93,438 | $ 93,438 | $ - | |
| 5 | Belt Filter | 2 | 100% | 112 | TPH | $ 1,013,308 | $ 2,026,616 | $ - | |
| 6 | Solution Heater | 1 | 100% | 5 | MBTU/hr | $ 181,582 | $ 181,582 | $ - | |
| 7 | Thickener | 13 | 100% | 201 | ft^2 | $ 29,731 | $ 386,505 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 23,711,204** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 3,996.00 | Effiicency: | 0.85 | | $ 1,915.50 | $ 5,746.49 | $ 1,930,820 | $ 13.04 | $ 0.48 |
| 3 | Process Pump | 444.00 | | | | $ 212.83 | $ 638.50 | $ 214,536 | $ 1.45 | $ 0.05 |
| 4 | Conveyor | 29.76 | | | | $ 14.27 | $ 42.80 | $ 14,380 | $ 0.10 | $ 0.00 |
| 5 | Belt Filter | 54.21 | | | | $ 25.98 | $ 77.95 | $ 26,192 | $ 0.18 | $ 0.01 |
| 6 | Solution Heater | 501.76 | | | | $ 240.52 | $ 721.56 | $ 242,445 | $ 1.64 | $ 0.06 |
| 7 | Thickener | 1.67 | | | | $ 0.80 | $ 2.40 | $ 805 | $ 0.01 | $ 0.00 |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **5027** | | | | **$ 2,409.90** | **$ 7,229.70** | **$ 2,429,178** | **$ 16.41** | **$ 0.60** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | - | L/hr | $ 0.008 | / L | $ - | $ - | $ - | $ - | $ - |
| 2 | H2SO4 | 107.57 | mt/hr | $ 220.000 | / mt | $ 189,325.12 | $ 567,975.36 | $ 190,839,720 | $ 1,289.33 | $ 47.33 |
| 3 | Natural Gas | 4.84 | MCF/hr | $ 2.860 | / MCF | $ 110.71 | $ 332.12 | $ 111,593 | $ 0.75 | $ 0.03 |
| 4 | Polymer | 32.27 | kg/hr | $ 2.250 | / kg | $ 580.86 | $ 1,742.57 | $ 585,505 | $ 3.96 | $ 0.15 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 190,016.68** | **$ 570,050.05** | **$ 191,536,818** | **$ 1,294.04** | **$ 47.50** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 192,427** | **$ 577,280** | **$ 193,965,996** | **$ 1,310.45** | **$ 48.11** | |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 Date: 3/18/2023
Case: WK-2 By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 12.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8322 | gpm |
| Feed Rate | 31502 | lpm |
| Precip Mass Flow | 11.91 | tph |
| Precip Conditioning Time | 72 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 897.48 | ft^2/tpd |
| NaOH Dose | 2.95 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 12 | 100% | 49,932 | gal | $ 118,167 | $ 1,418,007 | $ - | |
| 2 | Tank Mixer | 12 | 100% | 25 | HP | $ 98,347 | $ 1,180,158 | $ - | |
| 3 | Process Pump | 36 | 100% | 693 | gpm | $ 39,271 | $ 1,413,749 | $ - | |
| 4 | Filter Press | 1 | 100% | 56 | ft^3 | $ 165,980 | $ 165,980 | $ - | |
| 5 | Conveyor | 1 | 100% | 12 | TPH | $ 12,275 | $ 12,275 | $ - | |
| 6 | Thickener | 12 | 100% | 891 | ft^2 | $ 97,972 | $ 1,175,659 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 5,365,827** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 299.59 | Effiicency: | 0.85 | | $ 143.61 | $ 430.83 | $ 144,759 | $ 0.98 | $ 0.04 |
| 3 | Process Pump | 624.15 | | | | $ 299.19 | $ 897.56 | $ 301,581 | $ 2.04 | $ 0.07 |
| 4 | Filter Press | 9.42 | | | | $ 4.52 | $ 13.55 | $ 4,552 | $ 0.03 | $ 0.00 |
| 5 | Conveyor | 0.74 | | | | $ 0.35 | $ 1.06 | $ 357 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 6.80 | | | | $ 3.26 | $ 9.78 | $ 3,285 | $ 0.02 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **941** | | | | **$ 450.93** | **$ 1,352.78** | **$ 454,533** | **$ 3.07** | **$ 0.11** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 5.57 | mt/hr | $ 125.000 | / mt | $ 5,572.39 | $ 16,717.17 | $ 5,616,970 | $ 37.95 | $ 1.39 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 5,572.39** | **$ 16,717.17** | **$ 5,616,970** | **$ 37.95** | **$ 1.39** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 6,023** **$ 18,070** **$ 6,071,503** **$ 41.02** **$ 1.51**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-2 | By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 5.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 8308 | gpm |
| Feed Rate | 31449 | lpm |
| Precip Mass Flow | 4.86 | tph |
| Precip Conditioning Time | 72 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 7892.66 | ft^2/tpd |
| NaOH Dose | 1.74 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 5 | 100% | 119,635 | gal | $ 193,160 | $ 965,799 | $ - | |
| 2 | Tank Mixer | 5 | 100% | 60 | HP | $ 160,985 | $ 804,926 | $ - | |
| 3 | Process Pump | 15 | 100% | 1,662 | gpm | $ 54,803 | $ 822,044 | $ - | |
| 4 | Filter Press | 1 | 100% | 23 | ft^3 | $ 87,078 | $ 87,078 | $ - | |
| 5 | Conveyor | 1 | 100% | 5 | TPH | $ 7,504 | $ 7,504 | $ - | |
| 6 | Thickener | 5 | 100% | 7,671 | ft^2 | $ 551,254 | $ 2,756,269 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 5,443,620** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 299.09 | Effiicency: | 0.85 | | | $ 143.37 | $ 430.11 | $ 144,515 | $ 0.98 | $ 0.04 |
| 3 | Process Pump | 623.10 | | | | | $ 298.68 | $ 896.05 | $ 301,074 | $ 2.03 | $ 0.07 |
| 4 | Filter Press | 3.85 | | | | | $ 1.84 | $ 5.53 | $ 1,858 | $ 0.01 | $ 0.00 |
| 5 | Conveyor | 0.30 | | | | | $ 0.14 | $ 0.43 | $ 146 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 24.39 | | | | | $ 11.69 | $ 35.08 | $ 11,787 | $ 0.08 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **951** | | | | | **$ 455.73** | **$ 1,367.20** | **$ 459,380** | **$ 3.10** | **$ 0.11** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 3.29 | mt/hr | $ 125.000 | / mt | $ 3,287.85 | $ 9,863.55 | $ 3,314,152 | $ 22.39 | $ 0.82 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 3,287.85** | **$ 9,863.55** | **$ 3,314,152** | **$ 22.39** | **$ 0.82** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | **Total Operating Cost:** | | | **$ 3,744** | **$ 11,231** | **$ 3,773,532** | **$ 25.49** | **$ 0.94** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-2 | By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 4.86 | tph |
| Slurry Feed Rate | 1677 | gpm |
| Leaching Time | 120 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 873.31 | kg/ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 50,298 | gal | $ 60,350 | $ 241,401 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 75 | HP | $ 183,503 | $ 734,013 | $ - | |
| 3 | Process Pump | 1 | 100% | 1,677 | gpm | $ 109,982 | $ 109,982 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,085,396** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 301.79 | Effiicency: | 0.85 | | | $ 144.66 | $ 433.99 | $ 145,819 | $ 0.99 | $ 0.04 |
| 3 | Process Pump | 41.91 | | | | | $ 20.09 | $ 60.28 | $ 20,253 | $ 0.14 | $ 0.01 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **344** | | | | | **$ 164.75** | **$ 494.26** | **$ 166,072** | **$ 1.12** | **$ 0.04** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 4.24 | mt/hr | $ 270.000 | / mt | $ 9,166.95 | $ 27,500.85 | $ 9,240,285 | $ 62.43 | $ 2.29 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 9,166.95** | **$ 27,500.85** | **$ 9,240,285** | **$ 62.43** | **$ 2.29** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 9,332** | **$ 27,995** | **$ 9,406,357** | **$ 63.55** | **$ 2.33** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-2 | By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 1677 | gpm |
| Feed Rate | 6347 | lpm |
| Precip Mass Flow | 0.04 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 2.11 | g/L of Feed |
| Oxalic Acid Dose | 6.27 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.03 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 8,383 | gal | $ 11,017 | $ 44,067 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 4 | HP | $ 35,948 | $ 143,790 | $ - | |
| 3 | Process Pump | 3 | 100% | 1,677 | gpm | $ 109,982 | $ 329,946 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 0.21 | ft^3 | $ 2,985 | $ 2,985 | $ - | |
| 6 | Roaster | 1 | 100% | 0.07 | MBTU/hr | $ 248,798 | $ 248,798 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 769,587** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 16.77 | Effiicency: | 0.85 | | | $ 8.04 | $ 24.11 | $ 8,101 | $ 0.05 | $ 0.00 |
| 3 | Process Pump | 125.74 | | | | | $ 60.28 | $ 180.83 | $ 60,758 | $ 0.41 | $ 0.02 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.04 | | | | | $ 0.02 | $ 0.05 | $ 17 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.01 | | | | | $ 0.00 | $ 0.01 | $ 5 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **143** | | | | | **$ 68.33** | **$ 205.00** | **$ 68,881** | **$ 0.47** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 0.80 | mt/hr | $ 125.000 | / mt | $ 802.52 | $ 2,407.56 | $ 808,939 | $ 5.47 | $ 0.20 |
| 2 | Oxalic Acid | 2,388.88 | kg/hr | $ 0.600 | / kg | $ 11,466.65 | $ 34,399.94 | $ 11,558,379 | $ 78.09 | $ 2.87 |
| 3 | Natural Gas | 0.07 | MCF/hr | $ 2.860 | / MCF | $ 1.53 | $ 4.60 | $ 1,545 | $ 0.01 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 12,270.70** | **$ 36,812.09** | **$ 12,368,863** | **$ 83.57** | **$ 3.07** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 12,339** **$ 37,017** **$ 12,437,745** **$ 84.03** **$ 3.08**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-2 | By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 8322 | gpm |
| Raffinate Rate: | 8322 | gpm |
| Extraction Feed (O+A) | 16644 | gpm |
| Scrub Feed (O+A) | 229 | gpm |
| Strip Feed (O+A) | 458 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 229 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 273625 | gal |
| Precip Mixing Intensity | 0.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 60.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 16,644 | gal | $ 95,555 | $ 191,111 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 42 | HP | $ 131,184 | $ 262,369 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 249,659 | gal | $ 438,217 | $ 876,434 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 8,322 | gpm | $ 202,628 | $ 405,256 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 229 | gal | $ 8,574 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,690 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,432 | gal | $ 39,320 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 458 | gal | $ 12,661 | $ 25,323 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 17,283 | $ 34,565 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,864 | gal | $ 58,065 | $ 116,130 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,615 | $ 19,615 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 59,557 | kg | $ 3,728,849 | $ - | $ 3,728,849 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 1,033,193 | kg | $ 1,118,789 | $ - | $ 1,118,789 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,738 | gal | $ 5,713 | $ 22,850 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 1.37 | HP | $ 19,123 | $ 76,493 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,615 | $ 39,230 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.02 | ft^3 | $ 462 | $ 462 | $ - | |
| 20 | Process Pump | 2 | 100% | 229 | gpm | $ 51,454 | $ 102,909 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,172,744** | **$ 4,847,638** | |

## Power/Utilities

Power Cost: $ 0.07 $/kWhr
Effiicency: 0.85

| ID | Unit Operation | Total HP | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 83.22 | $ 39.89 | $ 119.67 | $ 40,211 | $ 0.27 | $ 0.01 |
| 3 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 416.10 | $ 199.46 | $ 598.37 | $ 201,054 | $ 1.36 | $ 0.05 |
| 5 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.29 | $ 1.10 | $ 3.29 | $ 1,106 | $ 0.01 | $ 0.00 |
| 11 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.46 | $ 0.22 | $ 0.66 | $ 220 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 5.48 | $ 2.62 | $ 7.87 | $ 2,646 | $ 0.02 | $ 0.00 |
| 17 | Thickener | - | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.91 | $ 0.44 | $ 1.31 | $ 441 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 11.44 | $ 5.48 | $ 16.45 | $ 5,528 | $ 0.04 | $ 0.00 |
| | **TOTAL:** | **520** | **$ 249.21** | **$ 747.64** | **$ 251,206** | **$ 1.70** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 107.74 | kg/hr | $ 0.940 | / kg | $ 810.18 | $ 2,430.55 | $ 816,665 | $ 5.52 | $ 0.20 |
| 2 | D2EHPA | 5.67 | kg/hr | $ 5.000 | / kg | $ 226.82 | $ 680.45 | $ 228,630 | $ 1.54 | $ 0.06 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.87 | mt/hr | $ 270.000 | / mt | $ 1,887.06 | $ 5,661.19 | $ 1,902,159 | $ 12.85 | $ 0.47 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,924.06** | **$ 8,772.18** | **$ 2,947,453** | **$ 19.91** | **$ 0.73** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,173** **$ 9,520** **$ 3,198,659** **$ 21.61** **$ 0.79**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-2 | By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 1676.54 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 6346.38 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 1676.10 | gpm | HCL Dose | 4.93 | g/L of Feed |
| Mn Feed Rate | 6344.74 | lpm | | | |
| Co Precip Mass Flow | 0.37 | tph | | | |
| Mn Precip Mass Flow | 1.17 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 1.30 | g/L of Feed | | | |
| NaOH Dose | 8.70 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 25,148 | gal | $ 31,256 | $ 62,511 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 38 | HP | $ 124,124 | $ 248,248 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 1,677 | gpm | $ 109,981 | $ 109,981 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 1.75 | ft^3 | $ 13,662 | $ 13,662 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 25,142 | gal | $ 31,248 | $ 62,496 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 38 | HP | $ 124,106 | $ 248,211 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 1,676 | gpm | $ 109,970 | $ 109,970 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 6 | ft^3 | $ 31,200 | $ 31,200 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 886,279** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 75.44 | Effiicency: | 0.85 | | | $ 36.16 | $ 108.49 | $ 36,454 | $ 0.25 | $ 0.01 |
| 3 | Process Pump | 41.91 | | | | | $ 20.09 | $ 60.27 | $ 20,252 | $ 0.14 | $ 0.01 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.29 | | | | | $ 0.14 | $ 0.42 | $ 142 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 75.42 | | | | | $ 36.16 | $ 108.47 | $ 36,444 | $ 0.25 | $ 0.01 |
| 8 | Process Pump | 41.90 | | | | | $ 20.09 | $ 60.26 | $ 20,247 | $ 0.14 | $ 0.01 |
| 9 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 0.92 | | | | | $ 0.44 | $ 1.33 | $ 447 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **236** | | | | | **$ 113.08** | **$ 339.24** | **$ 113,985** | **$ 0.77** | **$ 0.03** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 0.49 | mt/hr | $ 360.000 | / mt | $ 1,422.79 | $ 4,268.38 | $ 1,434,176 | $ 9.69 | $ 0.36 |
| 2 | NaOH | 3.31 | mt/hr | $ 125.000 | / mt | $ 3,311.11 | $ 9,933.34 | $ 3,337,604 | $ 22.55 | $ 0.83 |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 0.24 | $ 0.71 | $ 239 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 4,734.15** | **$ 14,202.44** | **$ 4,772,019** | **$ 32.24** | **$ 1.18** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 4,847** **$ 14,542** **$ 4,886,005** **$ 33.01** **$ 1.21**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | WK No. 13 | Date: | 3/18/2023 |
| Case: | WK-3 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | an |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | N/A |
| *(unit conversion)* | 12000 | TPD | Project NPV: | (1,557,968,139) |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 22.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | WK No. 13 | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | WK-3 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 256,070,117 | | | | | |
| Constant Dollar Annual OPEX | $ | 262,296,539 | $ | 65.05 | /ton | $ 1,541.5 | /kg |
| Constant Dollar Annual REE Rev | $ | 67,492,184 | $ | 16.74 | /ton | $ 396.66 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | | |
|---|---|---|---|---|---|
| Payback Period | | 25.5 | years | *22.5* | *operating years* |
| Internal Rate of Return | | N/A | | | |
| Net Present Value | $ | (1,557,968,139) | | | |
| Total Severence Tax Paid | $ | - | | | |
| Total State and Federal Tax Paid | $ | - | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-3 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 306 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 6.62 | 20.23 | 9.18 | | | | | | | 9.18 |
| Yttrium | 7.15 | 21.86 | 9.91 | | | | | | | 9.91 |
| Lanthanum | 16.88 | 51.62 | 23.41 | | | | | | | 23.41 |
| Cerium | 38.16 | 116.65 | 52.91 | | | | | | | 52.91 |
| Praseodymium | 4.58 | 13.99 | 6.34 | | | | | | | 6.34 |
| Neodymium | 14.97 | 45.77 | 20.76 | | | | | | | 20.76 |
| Samarium | 3.75 | 11.47 | 5.20 | | | | | | | 5.20 |
| Europium | 0.56 | 1.71 | 0.78 | | | | | | | 0.78 |
| Gadolinium | 3.12 | 9.53 | 4.32 | | | | | | | 4.32 |
| Terbium | 0.08 | 0.23 | 0.11 | | | | | | | 0.11 |
| Dysprosium | 1.06 | 3.24 | 1.47 | | | | | | | 1.47 |
| Holmium | 0.54 | 1.66 | 0.75 | | | | | | | 0.75 |
| Erbium | 1.03 | 3.14 | 1.43 | | | | | | | 1.43 |
| Thulium | 0.15 | 0.45 | 0.20 | | | | | | | 0.20 |
| Ytterbium | 1.05 | 3.21 | 1.46 | | | | | | | 1.46 |
| Lutetium | 0.31 | 0.93 | 0.42 | | | | | | | 0.42 |
| Manganese | -- | 161.60 | 73.30 | | | 0.00 | -- | | | 73.30 |
| Cobalt | -- | 21.90 | 9.93 | | | 0.00 | -- | | | 9.93 |
| Nickel | -- | 49.10 | 22.27 | | | 0.00 | -- | | | 22.27 |
| Zinc | -- | 113.40 | 51.44 | | | 0.00 | -- | | | 51.44 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 305.69 | 138.66 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 138.7 |
| Total non-REE CM | -- | 346.00 | 156.94 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 156.9 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 9.18 | 0.00 | 1.25 | 0.00 | 0.00 | 0.00 | 1.25 | 7.93 | 13.6% | |
| Yttrium | 9.91 | 0.87 | 0.00 | 0.06 | 0.00 | 0.00 | 0.93 | 8.98 | 9.4% | |
| Lanthanum | 23.41 | 3.58 | 0.00 | 0.05 | 0.01 | 0.00 | 3.64 | 19.77 | 15.5% | |
| Cerium | 52.91 | 8.07 | 0.00 | 0.08 | 0.02 | 0.00 | 8.18 | 44.73 | 15.5% | |
| Praseodymium | 6.34 | 0.99 | 0.00 | 0.01 | 0.00 | 0.00 | 1.00 | 5.34 | 15.8% | |
| Neodymium | 20.76 | 4.15 | 0.00 | 0.04 | 0.01 | 0.00 | 4.20 | 16.56 | 20.2% | |
| Samarium | 5.20 | 0.81 | 0.00 | 0.01 | 0.00 | 0.00 | 0.82 | 4.38 | 15.7% | |
| Europium | 0.78 | 0.14 | 0.00 | 0.00 | 0.00 | 0.00 | 0.14 | 0.63 | 18.2% | |
| Gadolinium | 4.32 | 0.50 | 0.00 | 0.01 | 0.00 | 0.00 | 0.51 | 3.82 | 11.7% | |
| Terbium | 0.11 | 0.03 | 0.00 | 0.00 | 0.00 | 0.00 | 0.03 | 0.07 | 30.8% | |
| Dysprosium | 1.47 | 0.20 | 0.00 | 0.01 | 0.00 | 0.00 | 0.21 | 1.26 | 14.1% | |
| Holmium | 0.75 | 0.04 | 0.00 | 0.00 | 0.00 | 0.00 | 0.04 | 0.71 | 4.9% | |
| Erbium | 1.43 | 0.08 | 0.00 | 0.00 | 0.00 | 0.00 | 0.08 | 1.34 | 5.8% | |
| Thulium | 0.20 | 0.01 | 0.00 | 0.00 | 0.00 | 0.00 | 0.01 | 0.19 | 5.4% | |
| Ytterbium | 1.46 | 0.05 | 0.00 | 0.00 | 0.00 | 0.00 | 0.06 | 1.40 | 4.0% | |
| Lutetium | 0.42 | 0.01 | 0.00 | 0.00 | 0.00 | 0.00 | 0.01 | 0.41 | 2.1% | |
| Manganese | 73.30 | 0.53 | 0.00 | 13.66 | 7.46 | 0.00 | 21.64 | 51.66 | 29.5% | |
| Cobalt | 9.93 | 0.23 | 0.00 | 0.19 | 0.63 | 0.00 | 1.06 | 8.88 | 10.7% | |
| Nickel | 22.27 | 0.52 | 0.00 | 0.92 | 1.34 | 0.00 | 2.78 | 19.49 | 12.5% | |
| Zinc | 51.44 | 2.20 | 0.00 | 0.04 | 2.97 | 0.00 | 5.21 | 46.23 | 10.1% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 138.66 | 19.52 | 1.25 | 0.27 | 0.07 | 0.00 | 21.10 | 117.56 | 15.2% | <---- |
| Total non-REE CM | 156.94 | 3.48 | 0.00 | 14.81 | 12.40 | 0.00 | 30.69 | 126.26 | 19.6% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 355 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 50 | ton | $ | 1.50 |
| Dust/Volatiles | 20 | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | | |
|---|---|---|
| 0.47 | TPD REE | |
| 0.030 | TPD Sc | |
| 0.65 | TPD CM | |
| 0.14 | REE | |
| 13.6% | Sc | |
| 18.6% | Mn | |
| 5.9% | Metal-S | |
| 500 | Feed | |
| 306 | Grade | |
| 25.5 | PP | |
| N/A | IRR | |
| -1557968139 | NPV | |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-3 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 4.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 12.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | Based on Design |
| Dissolved Solids in Leach | 26.1% | % | WK3/WK3 Mass Balance-V1 |
| Leaching Retention Time | 300 | min | Based on Design |
| Thickener UF % Solids | 45% | % | VT Override from 7.1% \| WK3/WK3 Samples |
| Residue Settling Rate | 15.0 | ft/hr | Not Available |
| **Reagent Use** | | | |
| Acid Consumption | 0.0 | kg/ton | |
| Bioleaching Sol'n Consumption | 4361 | L/ton | WK3/WK3 Mass Balance-V1 |
| Polymer Dosage | 0.028 | g/L | WK3/WK3 Mass Balance-V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 9.49 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 11.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 72.0 | min | Based on Design |
| Precip Solid Yield | 24.99 | g/L | WK3/WK3 Mass Balance-V1 |
| Precip Settling Rate | 2.24 | ft/hr | Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 10.0% | % | VT Override from 1.65% \| WK3/WK3 Samples |
| **Reagent Use** | | | |
| NaOH Dose | 11.39 | g/L of Feed | WK3/WK3 Mass Balance-V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 440.43 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 306 | ppm | |
| Grade After Sorter | 306 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 480 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 581 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 480 | tph | |
| Leaching Residue Rate | 354.576 | tph | |
| Slurry Flow (feed) | 8880 | gpm | |
| Product Flow | 7,995 | gpm | |
| Thickener Feed % Solids (eff) | 15% | % | |
| Liquid Density | 1.023 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 65.22 | g/L | |
| Leaching Vessel Size | 55500.00 | gal | |
| Leaching Vessel Size | 210.09 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 177.29 | tph | 70 to 200 tph |
| Thickener Area | 280.47 | ft^2 | |
| Thickener Diameter | 18.90 | ft | ideal < 100' |
| Liquid Balance | 313.11 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 7,995 | gpm | |
| Slurry Feed Rate | 30,264 | lpm | |
| Precip Mass Flow | 50.01 | tph | |
| Thickener Feed % Solids (eff) | 2.44% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.006 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 198.09 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 235.62 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 58.75 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 2,002 | ft^2 | |
| Thickener Diameter | 50.51 | ft | ideal < 100' |

*PM5: REE Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 11.0 | -- | VT: Process Design |
| | | | |
| **Process Parameters** | | | |
| Precip Conditioning Time | 72 | min | Based on Design |
| Precip Solid Yield | 8.57 | g/L | WK3/WK3 Mass Balance-V1 |
| Precip Settling Rate | 0.044 | ft/hr | Lily Banks/Settling Tests |
| Thickener UF % Solids | 0.90% | % | VT Override from 0.35% \| WK3/WK3 Samples |
| | | | |
| **Reagent Use** | | | |
| NaOH Dose | 3.01 | g/L of Feed | WK3/WK3 Mass Balance-V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| | | | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 4579.50 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 7936 | gpm | |
| Slurry Feed Rate | 30,041 | lpm | |
| Precip Mass Flow | 17.02 | tph | |
| Thickener Feed % Solids (eff) | 0.9% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.000 | g/cc | |
| | | | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 196.63 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 160.48 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 20.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 7087 | ft^2 | |
| Thickener Diameter | 95.02 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | 28.20 | m/m | target = 6.35 |

*PM6: RE-Redissolution*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 2.2% | % | VT Overide from 10.73% \|\| WK3/WK3 Mass Balance-V1 |
| Dissolved Solids in Re-Leach | 25% | % | WK3/WK3 Mass Balance-V1 |
| Leaching Retention Time | 120 | min | Based on Design |
| **Reagent Use** | | | |
| Acid Consumption | 409.5 | kg/ton | WK3/WK3 Mass Balance-V1 |
| | 0.0 | | |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Solid Feed Rate | 17.02 | tph | |
| Slurry Flow (feed) | 3,047 | gpm | |
| Product Flow | 3,047 | gpm | |
| Liquid Density | 1.016 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 346.0 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 2.62 | v/v | target = 2.86, based on test data |
| Liquid Balance | 20.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 22.3 | g/L | |

*PM7: REE Oxalate Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | Based on Design |
| Precip Solid Yield | 0.2 | g/L | WK3/WK3 Mass Balance-V1 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Thickener UF %Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| NaOH Dose | 2.31 | g/L | WK3/WK3 Mass Balance-V1 |
| Oxalic Acid Dose | 6.17 | g/L | WK3/WK3 Mass Balance-V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 3047.17 | gpm | |
| Slurry Feed Rate | 11534.78 | lpm | |
| Precip Mass Flow | 0.16 | tph | |
| Thickener Feed % Solids (eff) | 0.02% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.10 | tph | |
| Liquid Density | 1.016 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 57.67 | m^3 | |
| Filter Press Area | 0.78 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.19 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 4.89 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.68 | m/m | |

*PM8: Sc SX Separation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | UK: Design |
| Settler Residence Time | 15.00 | min | UK: Design |
| Loading: Number of Stages | 2.00 | - | UK: Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Loading O:A Advance Ratio | 0.03 | v/v | UK: Design |
| Scrubbing: Number of Stages | | - | N/A |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | N/A |
| Scrubbing O:A Advance Ratio | | v/v | N/A |
| Stripping: Number of Stages | 2.00 | - | UK: Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | UK: Design |
| Sc2O3 Grade | 2% | % | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Precip Solid Yield | 0.4 | g/L | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance (SX)** | | | |
| PLS Flow | 7994.8 | gpm | |
| Raffinate Rate: | 7994.8 | gpm | |
| Load Org Recycle | 7775.0 | gpm | |
| Load Org Feed | 7994.8 | gpm | |
| Loag Org Advance | 219.8 | gpm | |
| Scrub Aq Product | 0.0 | gpm | |
| Scrub Aq Recycle | 0.0 | gpm | |
| Scrub Aq Feed | 0.0 | gpm | |
| Scurb Org Product | 219.8 | gpm | |
| Strip Aq Product | 17.5 | gpm | |
| Strip Aq Recycle | 202.3 | gpm | |
| Strip Aq Feed | 219.8 | gpm | |
| Strip Org Product | 219.8 | gpm | |
| Extraction Feed (O+A): | 15989.7 | gpm | |
| Scrub Feed (O+A): | 219.8 | gpm | |
| Strip Feed (O+A): | 439.6 | gpm | |
| **Material Balance (Precip)** | | | |
| Strip Solution Flow | 17.5 | gpm | |
| Solid Density | 3.4 | g/cc | |
| Solid Product Rate | 0.00 | tph | |
| Liquid Density (after precip) | 1.13 | g/cc | |
| Thickener Feed % Solids (eff) | 0.03% | % | |
| **Total Volumes** | | | |
| Total Extraction Volume | 511670 | gal | |
| Total Scrubbing Volume | 0 | gal | |
| Total Stripping Volume | 14068 | gal | |
| Extraction Org Volume | 255835.00 | gal | |
| Scrubbing Org Volume | 0.00 | gal | |
| Stripping Org Volume | 7033.82 | gal | |
| Total Org Volume | 262868.82 | gal | |
| **Other Indicators** | | | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| Conditioning Vessel Size | 9.96 | m^3 | |
| Filter Press Area | 0.02 | ft^3 | ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM9: CM Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 0 | min | N/A |
| Precip Solid Yield | - | g/L | N/A |
| Precip Settlng Rate | 10 | ft/hr | N/A |
| Thickener UF % Solids | 25% | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 7.24 | g/L of Feed | N/A |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 0.00 | min | N/A |
| Thickener Capacity | 0.00 | ft^2/tph | N/A |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 7916 | gpm | |
| Slurry Feed Rate | 29,966 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.000 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

## PM10: CM Redissolution

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 2.0% | % | N/A |
| Dissolved Solids in Re-Leach | 100% | % | N/A |
| Leaching Retention Time | 45 | min | N/A |
| **Reagent Use** | | | |
| Acid Consumption | 441 | kg/ton | N/A |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 0.00 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 0 | gpm | |
| Liquid Density | 1.000 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | |
| Releach Conc. Factor | #DIV/0! | v/v | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | - | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

## PM11: CM Recovery

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 0.46 | g/L | WK3/WK3 Mass Balance-V1 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | | % | Not Applicable |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 2.10 | g/L | WK3/WK3 Mass Balance-V1 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Na2S Dose | 4.13 | g/L of Feed | WK3/WK3 Mass Balance-V1 |
| NaOH Dose | 8.85 | g/L of Feed | WK3/WK3 Mass Balance-V1 |
| HCL Dose | 9.74 | g/L of Feed | WK3/WK3 Mass Balance-V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 3046.97 | gpm | |
| Co Slurry Feed Rate | 11534.05 | lpm | |
| Co Precip Mass Flow | 0.35 | tph | |
| Co Thickener Feed % Solids (eff) | 0.05% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 3046.56 | gpm | |
| Mn Slurry Feed Rate | 11532.48 | lpm | |
| Mn Precip Mass Flow | 1.60 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.21% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.02 | g/cc | |
| Mn Liquid Density | 1.01 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 173.0 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 1.66 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 0.41 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 173.0 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 7.6 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 1.88 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

## PM12: Biooxidation

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Requirec | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-3 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ 19,585,772 | $ - | $ 4,896,443 | $ 21,544,350 | $ 26,822,715 | $ 72,849,280 | IIIIIIII

# OPERATING COST ESTIMATE

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-3 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 18.06 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 6.94 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 256,070,117 | $ Total | $ 2,540.38 | $ 7,621.13 | $ 2,560,701 | $ 15.05 | $ 0.64 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 1.81 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 67,492,184 | $ Total | $ 334.78 | $ 1,004.35 | $ 337,461 | $ 1.98 | $ 0.08 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 2.71 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 256,070,117 | $ Total | $ 2,540.38 | $ 7,621.13 | $ 2,560,701 | $ 15.05 | $ 0.64 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 10,396.99** | **$ 31,190.97** | **$ 10,480,165** | **$ 61.59** | **$ 2.60** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 3.03 | $ 0.13 |
| PM2 | Roasting | PR | 198 | | $ 94.99 | $ 284.96 | $ 95,747 | $ 0.56 | $ 0.02 |
| PM3 | Leaching | PR | 5046 | | $ 2,419.03 | $ 7,257.08 | $ 2,438,379 | $ 14.33 | $ 0.60 |
| PM4 | Al Precipitation | PR | 944 | | $ 452.56 | $ 1,357.67 | $ 456,177 | $ 2.68 | $ 0.11 |
| PM5 | REE Precipitation | PR | 958 | | $ 459.46 | $ 1,378.37 | $ 463,131 | $ 2.72 | $ 0.11 |
| PM6 | RE-Redissolution | PR | 625 | | $ 299.44 | $ 898.31 | $ 301,833 | $ 1.77 | $ 0.07 |
| PM7 | REE Oxalate Precipitation | PR | 259 | | $ 124.24 | $ 372.71 | $ 125,232 | $ 0.74 | $ 0.03 |
| PM8 | Sc SX Separation | PR | 499 | | $ 239.42 | $ 718.25 | $ 241,332 | $ 1.42 | $ 0.06 |
| PM9 | CM Precipitation | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM10 | CM Redissolution | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM11 | CM Recovery | PR | 428 | | $ 205.21 | $ 615.63 | $ 206,851 | $ 1.22 | $ 0.05 |
| PM12 | Biooxidation | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM13 | Reserved | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM14 | Reserved | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM15 | Reserved | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | **10023.9** | | **$ 4,805.00** | **$ 14,415.01** | **$ 4,843,444.00** | **$ 28.47** | **$ 1.20** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.60 | $ 0.03 |
| PM2 | Roasting | CR | | $ 1,281.99 | $ 3,845.96 | $ 1,292,244 | $ 7.59 | $ 0.32 |
| PM3 | Leaching | CR | | $ 128,386.53 | $ 385,159.59 | $ 129,413,624 | $ 760.57 | $ 32.10 |
| PM4 | Al Precipitation | CR | | $ 20,680.67 | $ 62,042.00 | $ 20,846,114 | $ 122.51 | $ 5.17 |
| PM5 | REE Precipitation | CR | | $ 5,417.88 | $ 16,253.65 | $ 5,461,227 | $ 32.10 | $ 1.35 |
| PM6 | RE-Redissolution | CR | | $ 15,057.91 | $ 45,173.72 | $ 15,178,368 | $ 89.20 | $ 3.76 |
| PM7 | REE Oxalate Precipitation | CR | | $ 22,087.60 | $ 66,262.81 | $ 22,264,305 | $ 130.85 | $ 5.52 |
| PM8 | Sc SX Separation | CR | | $ 2,809.12 | $ 8,427.36 | $ 2,831,592 | $ 16.64 | $ 0.70 |
| PM9 | CM Precipitation | CR | | $ - | $ - | $ - | $ - | $ - |
| PM10 | CM Redissolution | CR | | $ - | $ - | $ - | $ - | $ - |
| PM11 | CM Recovery | CR | | $ 14,365.46 | $ 43,096.37 | $ 14,480,380 | $ 85.10 | $ 3.59 |
| PM12 | Biooxidation | CR | | $ - | $ - | $ - | $ - | $ - |
| PM13 | Reserved | CR | | $ - | $ - | $ - | $ - | $ - |
| PM14 | Reserved | CR | | $ - | $ - | $ - | $ - | $ - |
| PM15 | Reserved | CR | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ 210,187.96** | **$ 630,563.87** | **$ 211,869,460.60** | **$ 1,245.17** | **$ 52.55** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM8 | Sc SX Separation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM9 | CM Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM10 | CM Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM11 | CM Recovery | LS | | $ - | $ - | $ - | $ - | $ - |
| PM12 | Biooxidation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM13 | Reserved | LS | | $ - | $ - | $ - | $ - | $ - |
| PM14 | Reserved | LS | | $ - | $ - | $ - | $ - | $ - |
| PM15 | Reserved | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 8,510 | $ 0.10 | $ 283.66 | $ 850.98 | $ 285,930 | $ 1.68 | $ 0.07 |
| Non-Hazardous Precipitate | WD | Tons | 1,200 | $ 1.50 | $ 600.15 | $ 1,800.44 | $ 604,947 | $ 3.56 | $ 0.15 |
| Dust/Volatiles | WD | Tons | 28,800 | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 883.81** | **$ 2,651.42** | **$ 890,877.48** | **$ 5.24** | **$ 0.22** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 215,877** | **$ 647,630** | **$ 217,603,782** | **$ 1,279** | **$ 54** |

## Overhead Costs

| Allocation | Code | Rate | Basis | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% % of Direct Cost | $ 31,191 | Daily Direct Cost | $ 1,559.55 | $ 4,678.65 | $ 1,572,025 | $ 9.24 | $ 0.39 |
| Variable Portion | OH | 15% % of Direct Cost | $ 647,630 | Daily Direct Cost | $ 32,381.52 | $ 97,144.55 | $ 32,640,567 | $ 191.83 | $ 8.10 |
| **TOTAL OVERHEAD COSTS** | | | | | **$ 33,941.06** | **$ 101,823.19** | **$ 34,212,592.04** | **$ 201.07** | **$ 8.49** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | WK No. 13 | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | WK-3 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 10,396.99 | $ 31,190.97 | $ 10,480,165 | $ 62 | $ 2.60 | I |
| Power | $ 4,805.00 | $ 14,415.01 | $ 4,843,444 | $ 28 | $ 1.20 | |
| Consumables | $ 210,187.96 | $ 630,563.87 | $ 211,869,461 | $ 1,245 | $ 52.55 | IIIIIIIIIIIIIIIIIIIIII |
| Leasing/Tolling | $ - | $ - | $ - | $ - | $ - | |
| Waste Disposal | $ 883.81 | $ 2,651.42 | $ 890,877 | $ 5 | $ 0.22 | |
| General Overhead | $ 33,941.06 | $ 101,823.19 | $ 34,212,592 | $ 201 | $ 8.49 | IIII |
| TOTAL | $ 260,215 | $ 780,644 | $ 262,296,539 | $ 1,542 | $ 65.05 | |

## Summary by Process Module

| | Process Module | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 611 | $ 1,834 | $ 616,370 | $ 3.62 | $ 0.15 | |
| PM2 | Roasting | $ 1,377 | $ 4,131 | $ 1,387,991 | $ 8.16 | $ 0.34 | |
| PM3 | Leaching | $ 130,806 | $ 392,417 | $ 131,852,003 | $ 774.90 | $ 32.70 | IIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 21,133 | $ 63,400 | $ 21,302,291 | $ 125.19 | $ 5.28 | II |
| PM5 | REE Precipitation | $ 5,877 | $ 17,632 | $ 5,924,358 | $ 34.82 | $ 1.47 | |
| PM6 | RE-Redissolution | $ 15,357 | $ 46,072 | $ 15,480,201 | $ 90.98 | $ 3.84 | I |
| PM7 | REE Oxalate Precipitation | $ 22,212 | $ 66,636 | $ 22,389,537 | $ 131.58 | $ 5.55 | II |
| PM8 | Sc SX Separation | $ 3,049 | $ 9,146 | $ 3,072,924 | $ 18.06 | $ 0.76 | |
| PM9 | CM Precipitation | $ - | $ - | $ - | $ - | $ - | |
| PM10 | CM Redissolution | $ - | $ - | $ - | $ - | $ - | |
| PM11 | CM Recovery | $ 14,571 | $ 43,712 | $ 14,687,231 | $ 86.32 | $ 3.64 | I |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| NA | Unallocated | $ 45,222 | $ 135,666 | $ 45,583,634 | $ 268 | $ 11.31 | IIIII |
| | TOTAL | $ 260,215 | $ 780,644 | $ 262,296,539 | $ 1,542 | $ 65.05 | |

## Summary by Cost Center

| Cost Center | Code | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| Operating Labor | OL | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 18 | $ 0.76 | |
| Technical Labor | TL | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 7 | $ 0.29 | |
| Power/Utilities | PR | $ 4,805.00 | $ 14,415.01 | $ 4,843,444 | $ 28 | $ 1.20 | |
| Lease Agreements | LS | $ - | $ - | $ - | $ - | $ - | |
| Consumables/Reagents | CR | $ 210,187.96 | $ 630,563.87 | $ 211,869,461 | $ 1,245 | $ 52.55 | IIIIIIIIIIIIIIIIIIIIII |
| Waste Disposal | WD | $ 883.81 | $ 2,651.42 | $ 890,877 | $ 5 | $ 0.22 | |
| Plant Overhead | OH | $ 36,786.30 | $ 110,358.91 | $ 37,080,592 | $ 218 | $ 9.20 | IIII |
| General Sales, and Administration | GSA | $ 3,332.45 | $ 9,997.35 | $ 3,359,110 | $ 20 | $ 0.83 | |
| Reserved | | | | | | | |
| TOTAL | | $ 260,215 | $ 780,644.46 | $ 262,296,539 | $ 1,542 | $ 65.05 | |

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | WK No. 13 | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | WK-3 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 67,492,184

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 285 | 306 |
| Contained Value | ($/ton) | $ 5.7 | $ 123.9 |
| Feed Basket Price | ($/kg) | $ 22.0 | $ 446.9 |
| Product Basket Price | ($/kg) | $ 21.0 | $ 400.5 |
| Production Rate | (kg/day) | 476.48 | 506.41 |
| | (t/day) | 0.48 | 0.51 |
| | (t/yr) | 160.10 | 170.15 |
| | (% of US Demand) | 1.6% | |

## Revenue Contribution Table

| | | Product Revenue ($/hr) | | | | | |
|---|---|---|---|---|---|---|---|
| **Element** | **REE/CM Price ($/kg) [1]** | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | **REE Revenue ($/day)** |
| Scandium | $ 4,200.00 | $ 0.2 | $ 8,028.6 | $ - | $ - | $ - | $ 192,691 |
| Yttrium | $ 6.00 | $ 5.0 | $ - | $ - | $ - | $ - | $ 119 |
| Lanthanum | $ 2.00 | $ 6.3 | $ - | $ - | $ - | $ - | $ 151 |
| Cerium | $ 2.00 | $ 14.9 | $ - | $ - | $ - | $ - | $ 357 |
| Praseodymium | $ 52.00 | $ 46.7 | $ - | $ - | $ - | $ - | $ 1,120 |
| Neodymium | $ 42.00 | $ 152.5 | $ - | $ - | $ - | $ - | $ 3,659 |
| Samarium | $ 2.11 | $ 1.5 | $ - | $ - | $ - | $ - | $ 36 |
| Europium | $ 34.00 | $ 4.2 | $ - | $ - | $ - | $ - | $ 100 |
| Gadolinium | $ 32.00 | $ 13.8 | $ - | $ - | $ - | $ - | $ 330 |
| Terbium | $ 400.00 | $ 11.0 | $ - | $ - | $ - | $ - | $ 264 |
| Dysprosium | $ 230.00 | $ 39.1 | $ - | $ - | $ - | $ - | $ 940 |
| Holmium | $ 52.96 | $ 1.6 | $ - | $ - | $ - | $ - | $ 39 |
| Erbium | $ 34.00 | $ 2.3 | $ - | $ - | $ - | $ - | $ 55 |
| Thulium | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Ytterbium | $ 28.72 | $ 1.3 | $ - | $ - | $ - | $ - | $ 32 |
| Lutetium | $ 796.64 | $ 5.3 | $ - | $ - | $ - | $ - | $ 127 |
| Manganese | $ 0.005 | $ - | $ - | $ 0.1 | $ - | $ - | $ 3 |
| Cobalt | $ 22.00 | $ - | $ - | $ - | $ 10.5 | $ - | $ 251 |
| Nickel | $ 18.00 | $ - | $ - | $ - | $ 18.1 | $ - | $ 435 |
| Zinc | $ 3.02 | $ - | $ - | $ - | $ 6.7 | $ - | $ 162 |
| Reserved | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Total: | | $ 305.6 | $ 8,028.6 | $ 0.1 | $ 35.3 | $ - | $ 200,870 |

[1] REEs listed as $/kg REO; Mn listed as $/kg 44% Mn Ore; Co listed as $/kg metal content

## Coal Revenue

List Coal Revenue Sources Here:

**Annual Coal Revenue:** $ -

## Other Revenue

List Other Revenue Sources Here:

**Annual Other Revenue:** $ -

## PRO FORMA CASH FLOW

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-3 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 73,750,532 | $ 75,963,048 | $ 78,241,939 | $ 80,589,197 | $ 83,006,873 | $ 85,497,079 | $ 88,061,992 | $ 90,703,851 | $ 93,424,967 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 73,750,532 | $ 75,963,048 | $ 78,241,939 | $ 80,589,197 | $ 83,006,873 | $ 85,497,079 | $ 88,061,992 | $ 90,703,851 | $ 93,424,967 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 768,210 | $ 5,485,123 | $ 294,153,121 | $ 302,114,505 | $ 310,295,616 | $ 318,701,898 | $ 327,338,891 | $ 336,212,227 | $ 345,327,631 | $ 354,690,916 | $ 364,307,985 |
| *- Operating Costs* | $ - | $ - | $ - | $ 286,618,510 | $ 295,217,065 | $ 304,073,577 | $ 313,195,785 | $ 322,591,658 | $ 332,269,408 | $ 342,237,490 | $ 352,504,615 | $ 363,079,753 |
| *- Loan #1 Interest* | $ - | $ 768,210 | $ 709,928 | $ 648,148 | $ 582,662 | $ 513,247 | $ 439,666 | $ 361,671 | $ 278,996 | $ 191,361 | $ 98,467 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 4,775,196 | $ 4,412,911 | $ 4,028,890 | $ 3,621,827 | $ 3,190,341 | $ 2,732,965 | $ 2,248,147 | $ 1,734,240 | $ 1,189,498 | $ 612,072 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 2,473,551 | $ 2,285,888 | $ 2,086,965 | $ 1,876,106 | $ 1,652,596 | $ 1,415,676 | $ 1,164,540 | $ 898,336 | $ 616,160 |
| Gross Profit | $ - | $ (768,210) | $ (5,485,123) | $ (220,402,589) | $ (226,151,458) | $ (232,053,677) | $ (238,112,701) | $ (244,332,018) | $ (250,715,148) | $ (257,265,639) | $ (263,987,065) | $ (270,883,018) |
| **Book Deductions** | $ - | $ 960,263 | $ 6,930,091 | $ 10,032,353 | $ 9,279,926 | $ 8,583,932 | $ 7,940,137 | $ 7,344,627 | $ 6,793,780 | $ 6,284,246 | $ 5,812,928 | $ 5,376,958 |
| *- CAPEX #1 Depreciation* | $ - | $ 960,263 | $ 888,243 | $ 821,625 | $ 760,003 | $ 703,003 | $ 650,278 | $ 601,507 | $ 556,394 | $ 514,664 | $ 476,064 | $ 440,360 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 6,041,848 | $ 5,588,709 | $ 5,169,556 | $ 4,781,839 | $ 4,423,201 | $ 4,091,461 | $ 3,784,602 | $ 3,500,756 | $ 3,238,200 | $ 2,995,335 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 3,622,019 | $ 3,350,367 | $ 3,099,090 | $ 2,866,658 | $ 2,651,659 | $ 2,452,784 | $ 2,268,826 | $ 2,098,664 | $ 1,941,264 |
| Taxable Before Depletion | $ - | $ (1,728,473) | $ (12,415,214) | $ (230,434,942) | $ (235,431,384) | $ (240,637,609) | $ (246,052,838) | $ (251,676,644) | $ (257,508,927) | $ (263,549,885) | $ (269,799,993) | $ (276,259,976) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ - | $ (864,237) | $ (6,207,607) | $ (115,217,471) | $ (117,715,692) | $ (120,318,804) | $ (123,026,419) | $ (125,838,322) | $ (128,754,464) | $ (131,774,943) | $ (134,899,996) | $ (138,129,988) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 10,325,074 | $ 10,634,827 | $ 10,953,871 | $ 11,282,488 | $ 11,620,962 | $ 11,969,591 | $ 12,328,679 | $ 12,698,539 | $ 13,079,495 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (1,728,473) | $ (12,415,214) | $ (230,434,942) | $ (235,431,384) | $ (240,637,609) | $ (246,052,838) | $ (251,676,644) | $ (257,508,927) | $ (263,549,885) | $ (269,799,993) | $ (276,259,976) |
| *- Income Tax* | $ - | $ (449,403) | $ (3,227,956) | $ (59,913,085) | $ (61,212,160) | $ (62,565,778) | $ (63,973,738) | $ (65,435,928) | $ (66,952,321) | $ (68,522,970) | $ (70,147,998) | $ (71,827,594) |
| Net Profit/Loss | $ - | $ (1,279,070) | $ (9,187,258) | $ (170,521,857) | $ (174,219,224) | $ (178,071,831) | $ (182,079,100) | $ (186,240,717) | $ (190,556,606) | $ (195,026,915) | $ (199,651,995) | $ (204,432,383) |
| **Book Deductions** | $ - | $ 960,263 | $ 6,930,091 | $ 10,032,353 | $ 9,279,926 | $ 8,583,932 | $ 7,940,137 | $ 7,344,627 | $ 6,793,780 | $ 6,284,246 | $ 5,812,928 | $ 5,376,958 |
| *+ CAPEX #1 Depreciation* | $ - | $ 960,263 | $ 888,243 | $ 821,625 | $ 760,003 | $ 703,003 | $ 650,278 | $ 601,507 | $ 556,394 | $ 514,664 | $ 476,064 | $ 440,360 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 6,041,848 | $ 5,588,709 | $ 5,169,556 | $ 4,781,839 | $ 4,423,201 | $ 4,091,461 | $ 3,784,602 | $ 3,500,756 | $ 3,238,200 | $ 2,995,335 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 3,622,019 | $ 3,350,367 | $ 3,099,090 | $ 2,866,658 | $ 2,651,659 | $ 2,452,784 | $ 2,268,826 | $ 2,098,664 | $ 1,941,264 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ - | $ (318,807) | $ (2,257,168) | $ (160,489,504) | $ (164,939,298) | $ (169,487,899) | $ (174,138,963) | $ (178,896,090) | $ (183,762,826) | $ (188,742,669) | $ (193,839,067) | $ (199,055,424) |
| **Captial Cost** | $ 12,803,506 | $ 80,557,968 | $ 48,293,586 | $ 10,619,516 | $ 11,256,687 | $ 11,932,088 | $ 12,648,013 | $ 13,406,894 | $ 14,211,308 | $ 15,063,986 | $ 15,967,826 | $ 15,186,309 |
| *- Fixed Capital Costs* | $ 25,607,012 | $ 159,173,185 | $ 82,451,710 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 12,803,506 | $ 79,586,592 | $ 41,225,855 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 971,376 | $ 1,029,658 | $ 1,091,438 | $ 1,156,924 | $ 1,226,340 | $ 1,299,920 | $ 1,377,915 | $ 1,460,590 | $ 1,548,226 | $ 1,641,119 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 6,038,072 | $ 6,400,357 | $ 6,784,378 | $ 7,191,441 | $ 7,622,927 | $ 8,080,303 | $ 8,565,121 | $ 9,079,028 | $ 9,623,770 | $ 10,201,196 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 3,127,721 | $ 3,315,385 | $ 3,514,308 | $ 3,725,166 | $ 3,948,676 | $ 4,185,597 | $ 4,436,733 | $ 4,702,937 | $ 4,985,113 |
| **Net Cash Flow** | $ (12,803,506) | $ (80,876,776) | $ (50,550,753) | $ (171,109,020) | $ (176,195,985) | $ (181,419,987) | $ (186,786,976) | $ (192,302,984) | $ (197,974,134) | $ (203,806,655) | $ (209,806,892) | $ (214,241,733) |
| Discounted Cash Flow | $ (12,803,506) | $ (73,524,341) | $ (41,777,482) | $ (128,556,739) | $ (120,344,228) | $ (112,647,538) | $ (105,436,379) | $ (98,681,838) | $ (92,356,395) | $ (86,433,917) | $ (80,889,639) | $ (75,090,420) |
| **Cumulative DCF** | $ (12,803,506) | $ (86,327,847) | $ (128,105,329) | $ (256,662,069) | $ (377,006,297) | $ (489,653,835) | $ (595,090,214) | $ (693,772,051) | $ (786,128,446) | $ (872,562,363) | $ (953,452,003) | $ (1,028,542,423) |

### Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

### Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 256,070,117 | Payback Period | 25.5 |
| Constant Dollar OPEX | $ 262,296,539 | Internal Rate of Return | N/A |
| Constant Dollar REE Rev | $ 67,492,184 | Net Present Value | $ (1,557,968,139) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ - |

# PRO FORMA CASH FLOW

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-3 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 96,227,716** | **$ 99,114,548** | **$ 102,087,984** | **$ 105,150,623** | **$ 108,305,142** | **$ 111,554,296** | **$ 114,900,925** | **$ 118,347,953** | **$ 121,898,392** | **$ 125,555,343** | **$ 129,322,004** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 96,227,716 | $ 99,114,548 | $ 102,087,984 | $ 105,150,623 | $ 108,305,142 | $ 111,554,296 | $ 114,900,925 | $ 118,347,953 | $ 121,898,392 | $ 125,555,343 | $ 129,322,004 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 374,289,199** | **$ 385,191,310** | **$ 396,747,050** | **$ 408,649,461** | **$ 420,908,945** | **$ 433,536,213** | **$ 446,542,300** | **$ 459,938,569** | **$ 473,736,726** | **$ 487,948,828** | **$ 502,587,292** |
| *- Operating Costs* | $ 373,972,146 | $ 385,191,310 | $ 396,747,050 | $ 408,649,461 | $ 420,908,945 | $ 433,536,213 | $ 446,542,300 | $ 459,938,569 | $ 473,736,726 | $ 487,948,828 | $ 502,587,292 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 317,053 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ (278,061,483) | $ (286,076,763) | $ (294,659,066) | $ (303,498,838) | $ (312,603,803) | $ (321,981,917) | $ (331,641,374) | $ (341,590,616) | $ (351,838,334) | $ (362,393,484) | $ (373,265,289) |
| **Book Deductions** | **$ 4,973,686** | **$ 4,600,660** | **$ 4,255,610** | **$ 3,936,440** | **$ 3,641,207** | **$ 3,368,116** | **$ 3,115,507** | **$ 2,881,844** | **$ 2,665,706** | **$ 2,263,837** | **$ 823,462** |
| *- CAPEX #1 Depreciation* | $ 407,333 | $ 376,783 | $ 348,524 | $ 322,385 | $ 298,206 | $ 275,840 | $ 255,152 | $ 236,016 | $ 218,315 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 2,770,685 | $ 2,562,883 | $ 2,370,667 | $ 2,192,867 | $ 2,028,402 | $ 1,876,272 | $ 1,735,551 | $ 1,605,385 | $ 1,484,981 | $ 1,373,608 | $ - |
| *- CAPEX #3 Depreciation* | $ 1,795,669 | $ 1,660,994 | $ 1,536,419 | $ 1,421,188 | $ 1,314,599 | $ 1,216,004 | $ 1,124,804 | $ 1,040,443 | $ 962,410 | $ 890,229 | $ 823,462 |
| Taxable Before Depletion | $ (283,035,170) | $ (290,677,423) | $ (298,914,676) | $ (307,435,277) | $ (316,245,010) | $ (325,350,033) | $ (334,756,882) | $ (344,472,460) | $ (354,504,040) | $ (364,657,321) | $ (374,088,751) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ (141,517,585) | $ (145,338,711) | $ (149,457,338) | $ (153,717,639) | $ (158,122,505) | $ (162,675,017) | $ (167,378,441) | $ (172,236,230) | $ (177,252,020) | $ (182,328,661) | $ (187,044,375) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 13,471,880 | $ 13,876,037 | $ 14,292,318 | $ 14,721,087 | $ 15,162,720 | $ 15,617,601 | $ 16,086,130 | $ 16,568,713 | $ 17,065,775 | $ 17,577,748 | $ 18,105,081 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ (283,035,170)** | **$ (290,677,423)** | **$ (298,914,676)** | **$ (307,435,277)** | **$ (316,245,010)** | **$ (325,350,033)** | **$ (334,756,882)** | **$ (344,472,460)** | **$ (354,504,040)** | **$ (364,657,321)** | **$ (374,088,751)** |
| *- Income Tax* | $ (73,589,144) | $ (75,576,130) | $ (77,717,816) | $ (79,933,172) | $ (82,223,702) | $ (84,591,009) | $ (87,036,789) | $ (89,562,840) | $ (92,171,050) | $ (94,810,904) | $ (97,263,075) |
| Net Profit/Loss | $ (209,446,025) | $ (215,101,293) | $ (221,196,860) | $ (227,502,105) | $ (234,021,307) | $ (240,759,025) | $ (247,720,093) | $ (254,909,620) | $ (262,332,990) | $ (269,846,418) | $ (276,825,676) |
| **Book Deductions** | **$ 4,973,686** | **$ 4,600,660** | **$ 4,255,610** | **$ 3,936,440** | **$ 3,641,207** | **$ 3,368,116** | **$ 3,115,507** | **$ 2,881,844** | **$ 2,665,706** | **$ 2,263,837** | **$ 823,462** |
| *+ CAPEX #1 Depreciation* | $ 407,333 | $ 376,783 | $ 348,524 | $ 322,385 | $ 298,206 | $ 275,840 | $ 255,152 | $ 236,016 | $ 218,315 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 2,770,685 | $ 2,562,883 | $ 2,370,667 | $ 2,192,867 | $ 2,028,402 | $ 1,876,272 | $ 1,735,551 | $ 1,605,385 | $ 1,484,981 | $ 1,373,608 | $ - |
| *+ CAPEX #3 Depreciation* | $ 1,795,669 | $ 1,660,994 | $ 1,536,419 | $ 1,421,188 | $ 1,314,599 | $ 1,216,004 | $ 1,124,804 | $ 1,040,443 | $ 962,410 | $ 890,229 | $ 823,462 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ (204,472,339) | $ (210,500,633) | $ (216,941,250) | $ (223,565,666) | $ (230,380,100) | $ (237,390,908) | $ (244,604,585) | $ (252,027,776) | $ (259,667,284) | $ (267,582,581) | $ (276,002,214) |
| **Captial Cost** | **$ 5,284,220** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 5,284,220 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ (209,756,559) | $ (210,500,633) | $ (216,941,250) | $ (223,565,666) | $ (230,380,100) | $ (237,390,908) | $ (244,604,585) | $ (252,027,776) | $ (259,667,284) | $ (267,582,581) | $ (276,002,214) |
| Discounted Cash Flow | $ (66,834,904) | $ (60,974,535) | $ (57,127,411) | $ (53,519,843) | $ (50,137,422) | $ (46,966,526) | $ (43,994,285) | $ (41,208,555) | $ (38,597,887) | $ (36,158,587) | $ (33,905,761) |
| **Cumulative DCF** | $ (1,095,377,327) | $ (1,156,351,862) | $ (1,213,479,274) | $ (1,266,999,116) | $ (1,317,136,539) | $ (1,364,103,064) | $ (1,408,097,349) | $ (1,449,305,904) | $ (1,487,903,791) | $ (1,524,062,378) | $ (1,557,968,139) |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

## Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 256,070,117 | Payback Period | 0 | 25.5 |
| Constant Dollar OPEX | $ 262,296,539 | Internal Rate of Return | 0 | N/A |
| Constant Dollar REE Rev | $ 67,492,184 | Net Present Value | 0 | $ (1,557,968,139) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ - |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-3 | By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 1.70 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 1.06 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.08 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.09 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.09 | $ 0.00 |
| 7 | Conveyor | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 3.03** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.60 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.60** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 611** | **$ 1,834** | **$ 616,370** | **$ 3.62** | **$ 0.15** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-3 | By: ks

## Module Information

Module ID: PM2
Module Name: Roasting
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 500 | tph |
| External Heat | 10.0% | % |
| Product Rate | 480 | tph |
| Roasting Temperature | 575 | deg C |
| Bin Storage Capacity | 20% | % of hourly prod. |
| Material Heat Capacity | 0.80 | kg/kg-K |
| Reaction Enthalpy | 1783.00 | kJ/kg |
| % of Feed Reacted | 50% | % |
| Scurbber Airflow | 20.00 | cfm/tph |
| Quencher Aiflow | 20.00 | cfm/tph |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Storage Bins | 2 | 100% | 100 | Tons | $ 115,387 | $ 230,773 | $ - | |
| 2 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 3 | Roaster | 1 | 100% | 581 | MBTU/hr | $ 18,994,975 | $ 18,994,975 | $ - | |
| 4 | Gas Scrubber | 1 | 100% | 10,000 | cfm | $ 57,958 | $ 57,958 | $ - | |
| 5 | Spray Chamber Quencher | 1 | 100% | 10,000 | cfm | $ 206,510 | $ 206,510 | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 19,585,772** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Storage Bins | 10.00 | Power Cost: | $ 0.07 | $/kWhr | | $ 4.79 | $ 14.38 | $ 4,832 | $ 0.03 | $ 0.00 |
| 2 | Conveyor | 31.00 | Effiicency: | 0.85 | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.09 | $ 0.00 |
| 3 | Roaster | 87.16 | | | | | $ 41.78 | $ 125.34 | $ 42,113 | $ 0.25 | $ 0.01 |
| 4 | Gas Scrubber | 50.00 | | | | | $ 23.97 | $ 71.90 | $ 24,159 | $ 0.14 | $ 0.01 |
| 5 | Spray Chamber Quencher | 20.00 | | | | | $ 9.59 | $ 28.76 | $ 9,664 | $ 0.06 | $ 0.00 |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **198** | | | | | **$ 94.99** | **$ 284.96** | **$ 95,747** | **$ 0.56** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Natural Gas | 56.03 | MCF/hr | $ 2.860 | / MCF | $ 1,281.99 | $ 3,845.96 | $ 1,292,244 | $ 7.59 | $ 0.32 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 1,281.99** | **$ 3,845.96** | **$ 1,292,244** | **$ 7.59** | **$ 0.32** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 1,377** **$ 4,131** **$ 1,387,991** **$ 8.16** **$ 0.34**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-3 | By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 12.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 480 | tph |
| Residue Rate | 355 | tph |
| Slurry Flow | 8880 | gpm |
| Leaching Time | 300 | min |
| Acid Consumption | 0.00 | kg/ton |
| Bioleaching Sol'n Consumption | 4360.685244 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 2 | -- |
| Thickener Capacity | 9.49 | ft^2/tph |
| Polymer Dose | 0.028 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 48 | 100% | 55,500 | gal | $ 188,110 | $ 9,029,265 | $ - | |
| 2 | Tank Mixer | 48 | 100% | 83 | HP | $ 193,978 | $ 9,310,929 | $ - | |
| 3 | Process Pump | 24 | 100% | 740 | gpm | $ 80,510 | $ 1,932,241 | $ - | |
| 4 | Conveyor | 1 | 100% | 480 | TPH | $ 93,438 | $ 93,438 | $ - | |
| 5 | Belt Filter | 2 | 100% | 177 | TPH | $ 1,136,742 | $ 2,273,485 | $ - | |
| 6 | Solution Heater | 1 | 100% | 5 | MBTU/hr | $ 177,023 | $ 177,023 | $ - | |
| 7 | Thickener | 12 | 100% | 280 | ft^2 | $ 38,768 | $ 465,210 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 23,281,592** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 3,996.00 | Effiicency: | 0.85 | | | $ 1,915.50 | $ 5,746.49 | $ 1,930,820 | $ 11.35 | $ 0.48 |
| 3 | Process Pump | 444.00 | | | | | $ 212.83 | $ 638.50 | $ 214,536 | $ 1.26 | $ 0.05 |
| 4 | Conveyor | 29.76 | | | | | $ 14.27 | $ 42.80 | $ 14,380 | $ 0.08 | $ 0.00 |
| 5 | Belt Filter | 85.98 | | | | | $ 41.22 | $ 123.65 | $ 41,547 | $ 0.24 | $ 0.01 |
| 6 | Solution Heater | 488.55 | | | | | $ 234.19 | $ 702.57 | $ 236,062 | $ 1.39 | $ 0.06 |
| 7 | Thickener | 2.14 | | | | | $ 1.03 | $ 3.08 | $ 1,034 | $ 0.01 | $ 0.00 |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **5046** | | | | | **$ 2,419.03** | **$ 7,257.08** | **$ 2,438,379** | **$ 14.33** | **$ 0.60** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | 2,093,128.92 | L/hr | $ 0.008 | / L | $ 127,262.24 | $ 381,786.71 | $ 128,280,336 | $ 753.91 | $ 31.82 |
| 2 | H2SO4 | - | mt/hr | $ 220.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Natural Gas | 4.71 | MCF/hr | $ 2.860 | / MCF | $ 107.79 | $ 323.38 | $ 108,655 | $ 0.64 | $ 0.03 |
| 4 | Polymer | 56.47 | kg/hr | $ 2.250 | / kg | $ 1,016.50 | $ 3,049.50 | $ 1,024,633 | $ 6.02 | $ 0.25 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 128,386.53** | **$ 385,159.59** | **$ 129,413,624** | **$ 760.57** | **$ 32.10** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 130,806** **$ 392,417** **$ 131,852,003** **$ 774.90** **$ 32.70**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-3 | By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 11.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 7995 | gpm |
| Feed Rate | 30264 | lpm |
| Precip Mass Flow | 50.01 | tph |
| Precip Conditioning Time | 72 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 440.43 | ft^2/tpd |
| NaOH Dose | 11.39 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 11 | 100% | 52,330 | gal | $ 121,326 | $ 1,334,588 | $ - | |
| 2 | Tank Mixer | 11 | 100% | 26 | HP | $ 100,983 | $ 1,110,815 | $ - | |
| 3 | Process Pump | 33 | 100% | 727 | gpm | $ 39,980 | $ 1,319,331 | $ - | |
| 4 | Filter Press | 1 | 100% | 236 | ft^3 | $ 466,389 | $ 466,389 | $ - | |
| 5 | Conveyor | 1 | 100% | 50 | TPH | $ 26,991 | $ 26,991 | $ - | |
| 6 | Thickener | 11 | 100% | 2,002 | ft^2 | $ 187,659 | $ 2,064,254 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 6,322,367** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 287.81 | Effiicency: | 0.85 | | $ 137.96 | $ 413.89 | $ 139,069 | $ 0.82 | $ 0.03 |
| 3 | Process Pump | 599.61 | | | | $ 287.43 | $ 862.28 | $ 289,726 | $ 1.70 | $ 0.07 |
| 4 | Filter Press | 39.56 | | | | $ 18.96 | $ 56.89 | $ 19,115 | $ 0.11 | $ 0.00 |
| 5 | Conveyor | 3.10 | | | | $ 1.49 | $ 4.46 | $ 1,498 | $ 0.01 | $ 0.00 |
| 6 | Thickener | 14.01 | | | | $ 6.72 | $ 20.15 | $ 6,769 | $ 0.04 | $ 0.00 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **944** | | | | **$ 452.56** | **$ 1,357.67** | **$ 456,177** | **$ 2.68** | **$ 0.11** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 20.68 | mt/hr | $ 125.000 | / mt | $ 20,680.67 | $ 62,042.00 | $ 20,846,114 | $ 122.51 | $ 5.17 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 20,680.67** | **$ 62,042.00** | **$ 20,846,114** | **$ 122.51** | **$ 5.17** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 21,133** | **$ 63,400** | **$ 21,302,291** | **$ 125.19** | **$ 5.28** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-3 | By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 11.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 7936 | gpm |
| Feed Rate | 30041 | lpm |
| Precip Mass Flow | 17.02 | tph |
| Precip Conditioning Time | 72 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 60.00 | min |
| Thickener Capacity | 4579.50 | ft^2/tpd |
| NaOH Dose | 3.01 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 11 | 100% | 51,945 | gal | $ 120,824 | $ 1,329,063 | $ - | |
| 2 | Tank Mixer | 11 | 100% | 26 | HP | $ 100,564 | $ 1,106,204 | $ - | |
| 3 | Process Pump | 33 | 100% | 721 | gpm | $ 39,867 | $ 1,315,624 | $ - | |
| 4 | Filter Press | 1 | 100% | 160 | ft^3 | $ 353,717 | $ 353,717 | $ - | |
| 5 | Conveyor | 1 | 100% | 17 | TPH | $ 14,936 | $ 14,936 | $ - | |
| 6 | Thickener | 11 | 100% | 7,087 | ft^2 | $ 517,334 | $ 5,690,678 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 9,810,222** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 285.70 | Effiicency: | 0.85 | | $ 136.95 | $ 410.85 | $ 138,047 | $ 0.81 | $ 0.03 |
| 3 | Process Pump | 595.21 | | | | $ 285.31 | $ 855.94 | $ 287,597 | $ 1.69 | $ 0.07 |
| 4 | Filter Press | 26.94 | | | | $ 12.92 | $ 38.75 | $ 13,019 | $ 0.08 | $ 0.00 |
| 5 | Conveyor | 1.06 | | | | $ 0.51 | $ 1.52 | $ 510 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 49.58 | | | | $ 23.77 | $ 71.30 | $ 23,958 | $ 0.14 | $ 0.01 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **958** | | | | **$ 459.46** | **$ 1,378.37** | **$ 463,131** | **$ 2.72** | **$ 0.11** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 5.42 | mt/hr | $ 125.000 | / mt | $ 5,417.88 | $ 16,253.65 | $ 5,461,227 | $ 32.10 | $ 1.35 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 5,417.88** | **$ 16,253.65** | **$ 5,461,227** | **$ 32.10** | **$ 1.35** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 5,877** **$ 17,632** **$ 5,924,358** **$ 34.82** **$ 1.47**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-3 | By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 17.02 | tph |
| Slurry Feed Rate | 3047 | gpm |
| Leaching Time | 120 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 409.50 | kg/ton |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 91,415 | gal | $ 106,406 | $ 425,625 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 137 | HP | $ 257,030 | $ 1,028,121 | $ - | |
| 3 | Process Pump | 1 | 100% | 3,047 | gpm | $ 138,128 | $ 138,128 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,591,875** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 548.49 | Effiicency: | 0.85 | | | $ 262.92 | $ 788.76 | $ 265,024 | $ 1.56 | $ 0.07 |
| 3 | Process Pump | 76.18 | | | | | $ 36.52 | $ 109.55 | $ 36,809 | $ 0.22 | $ 0.01 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **625** | | | | | **$ 299.44** | **$ 898.31** | **$ 301,833** | **$ 1.77** | **$ 0.07** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 6.97 | mt/hr | $ 270.000 | / mt | $ 15,057.91 | $ 45,173.72 | $ 15,178,368 | $ 89.20 | $ 3.76 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 15,057.91** | **$ 45,173.72** | **$ 15,178,368** | **$ 89.20** | **$ 3.76** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 15,357** **$ 46,072** **$ 15,480,201** **$ 90.98** **$ 3.84**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-3 | By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 3047 | gpm |
| Feed Rate | 11535 | lpm |
| Precip Mass Flow | 0.16 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 2.31 | g/L of Feed |
| Oxalic Acid Dose | 6.17 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.10 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 15,236 | gal | $ 19,424 | $ 77,697 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 8 | HP | $ 50,351 | $ 201,405 | $ - | |
| 3 | Process Pump | 3 | 100% | 3,047 | gpm | $ 138,128 | $ 414,385 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 0.78 | ft^3 | $ 7,610 | $ 7,610 | $ - | |
| 6 | Roaster | 1 | 100% | 0.25 | MBTU/hr | $ 464,267 | $ 464,267 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,165,365** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 30.47 | Effiicency: | 0.85 | | | $ 14.61 | $ 43.82 | $ 14,724 | $ 0.09 | $ 0.00 |
| 3 | Process Pump | 228.54 | | | | | $ 109.55 | $ 328.65 | $ 110,427 | $ 0.65 | $ 0.03 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.13 | | | | | $ 0.06 | $ 0.19 | $ 63 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.04 | | | | | $ 0.02 | $ 0.05 | $ 18 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **259** | | | | | **$ 124.24** | **$ 372.71** | **$ 125,232** | **$ 0.74** | **$ 0.03** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 1.60 | mt/hr | $ 125.000 | / mt | $ 1,599.96 | $ 4,799.87 | $ 1,612,758 | $ 9.48 | $ 0.40 |
| 2 | Oxalic Acid | 4,267.09 | kg/hr | $ 0.600 | / kg | $ 20,482.02 | $ 61,446.07 | $ 20,645,881 | $ 121.34 | $ 5.12 |
| 3 | Natural Gas | 0.25 | MCF/hr | $ 2.860 | / MCF | $ 5.62 | $ 16.86 | $ 5,666 | $ 0.03 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 22,087.60** | **$ 66,262.81** | **$ 22,264,305** | **$ 130.85** | **$ 5.52** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 22,212** | **$ 66,636** | **$ 22,389,537** | **$ 131.58** | **$ 5.55** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-3 | By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 7995 | gpm |
| Raffinate Rate: | 7995 | gpm |
| Extraction Feed (O+A) | 15990 | gpm |
| Scrub Feed (O+A) | 220 | gpm |
| Strip Feed (O+A) | 440 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 220 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 262869 | gal |
| Precip Mixing Intensity | 0.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 60.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 15,990 | gal | $ 93,424 | $ 186,849 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 40 | HP | $ 128,250 | $ 256,501 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 239,845 | gal | $ 428,444 | $ 856,888 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 7,995 | gpm | $ 199,552 | $ 399,105 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 220 | gal | $ 8,383 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,429 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,297 | gal | $ 38,443 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 440 | gal | $ 12,379 | $ 24,758 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 16,896 | $ 33,792 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,594 | gal | $ 56,770 | $ 113,540 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,317 | $ 19,317 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 57,216 | kg | $ 3,582,273 | $ - | $ 3,582,273 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 992,579 | kg | $ 1,074,811 | $ - | $ 1,074,811 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,630 | gal | $ 5,499 | $ 21,997 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 1.32 | HP | $ 18,696 | $ 74,782 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,317 | $ 38,634 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.02 | ft^3 | $ 448 | $ 448 | $ - | |
| 20 | Process Pump | 2 | 100% | 220 | gpm | $ 50,673 | $ 101,347 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,127,957** | **$ 4,657,083** | |

## Power/Utilities

Power Cost: $ 0.07 $/kWhr
Effiicency: 0.85

| ID | Unit Operation | Total HP | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 79.95 | $ 38.32 | $ 114.97 | $ 38,630 | $ 0.23 | $ 0.01 |
| 3 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 399.74 | $ 191.62 | $ 574.85 | $ 193,151 | $ 1.14 | $ 0.05 |
| 5 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.20 | $ 1.05 | $ 3.16 | $ 1,062 | $ 0.01 | $ 0.00 |
| 11 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.44 | $ 0.21 | $ 0.63 | $ 212 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 5.26 | $ 2.52 | $ 7.56 | $ 2,542 | $ 0.01 | $ 0.00 |
| 17 | Thickener | - | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.88 | $ 0.42 | $ 1.26 | $ 424 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 10.99 | $ 5.27 | $ 15.80 | $ 5,310 | $ 0.03 | $ 0.00 |
| | **TOTAL:** | **499** | **$ 239.42** | **$ 718.25** | **$ 241,332** | **$ 1.42** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 103.50 | kg/hr | $ 0.940 | / kg | $ 778.34 | $ 2,335.01 | $ 784,563 | $ 4.61 | $ 0.19 |
| 2 | D2EHPA | 5.45 | kg/hr | $ 5.000 | / kg | $ 217.90 | $ 653.70 | $ 219,642 | $ 1.29 | $ 0.05 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.84 | mt/hr | $ 270.000 | / mt | $ 1,812.88 | $ 5,438.65 | $ 1,827,387 | $ 10.74 | $ 0.45 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,809.12** | **$ 8,427.36** | **$ 2,831,592** | **$ 16.64** | **$ 0.70** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,049** **$ 9,146** **$ 3,072,924** **$ 18.06** **$ 0.76**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-3 | By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 3046.97 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 11534.05 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 3046.56 | gpm | HCL Dose | 9.74 | g/L of Feed |
| Mn Feed Rate | 11532.48 | lpm | | | |
| Co Precip Mass Flow | 0.35 | tph | | | |
| Mn Precip Mass Flow | 1.60 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 4.13 | g/L of Feed | | | |
| NaOH Dose | 8.85 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 45,705 | gal | $ 55,107 | $ 110,213 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 69 | HP | $ 173,855 | $ 347,711 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 3,047 | gpm | $ 138,125 | $ 138,125 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 1.66 | ft^3 | $ 13,178 | $ 13,178 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 45,698 | gal | $ 55,100 | $ 110,199 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 69 | HP | $ 173,842 | $ 347,684 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 3,047 | gpm | $ 138,118 | $ 138,118 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 8 | ft^3 | $ 39,196 | $ 39,196 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,244,424** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 137.11 | Effiicency: | 0.85 | | $ 65.73 | $ 197.18 | $ 66,252 | $ 0.39 | $ 0.02 |
| 3 | Process Pump | 76.17 | | | | $ 36.51 | $ 109.54 | $ 36,807 | $ 0.22 | $ 0.01 |
| 4 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.28 | | | | $ 0.13 | $ 0.40 | $ 135 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 137.10 | | | | $ 65.72 | $ 197.15 | $ 66,243 | $ 0.39 | $ 0.02 |
| 8 | Process Pump | 76.16 | | | | $ 36.51 | $ 109.53 | $ 36,802 | $ 0.22 | $ 0.01 |
| 9 | Thickener | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 1.27 | | | | $ 0.61 | $ 1.83 | $ 613 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **428** | | | | **$ 205.21** | **$ 615.63** | **$ 206,851** | **$ 1.22** | **$ 0.05** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 2.86 | mt/hr | $ 360.000 | / mt | $ 8,240.16 | $ 24,720.48 | $ 8,306,081 | $ 48.82 | $ 2.06 |
| 2 | NaOH | 6.12 | mt/hr | $ 125.000 | / mt | $ 6,124.85 | $ 18,374.55 | $ 6,173,850 | $ 36.28 | $ 1.53 |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 0.45 | $ 1.34 | $ 449 | $ 0.00 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 14,365.46** | **$ 43,096.37** | **$ 14,480,380** | **$ 85.10** | **$ 3.59** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 14,571** **$ 43,712** **$ 14,687,231** **$ 86.32** **$ 3.64**

# *Process Techno-Economic Assessment Worksheet*

## Analysis Information

| | | | |
|---|---|---|---|
| Feedstock Material: | WK No. 13 | Date: | 3/18/2023 |
| Case: | WK-4 | Performed by: | ks |
| Project | DE-FE0031827 | Checked by: | an |

## Operational Information

| | Total Plant | | | |
|---|---|---|---|---|
| Nominal Plant Size | 500 | TPH | Project IRR | N/A |
| *(unit conversion)* | 12000 | TPD | Project NPV: | (1,598,005,780) |
| *(unit conversion)* | 1000000 | lb/hr | Payback (operating years): | 22.50 |

## Document Information

| | *Process Codes* | | *Accounting Codes* |
|---|---|---|---|
| PM1 | Feed Preparation | OL | Operating Labor |
| PM2 | Roasting | TL | Technical Labor |
| PM3 | Leaching | PR | Power/Utilities |
| PM4 | Al Precipitation | LS | Lease Agreements |
| PM5 | REE Precipitation | CR | Consumables/Reagents |
| PM6 | RE-Redissolution | WD | Waste Disposal |
| PM7 | REE Oxalate Precipitation | OH | Plant Overhead |
| PM8 | Sc SX Separation | GSA | General Sales, and Administration |
| PM9 | CM Precipitation | | |
| PM10 | CM Redissolution | | |
| PM11 | CM Recovery | | |
| PM12 | Biooxidation | | |
| PM13 | Reserved | | |
| PM14 | Reserved | | |
| PM15 | Reserved | | |

*Ver: 5.4*

# ECONOMIC OUTPUT SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | WK No. 13 | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | WK-4 | | | By: | ks |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | |
|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % |
| % Financed | 50 | % | Depreciation Term | 20 | years |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % |
| | | | REE Royalty Rate | 0 | % |
| | | | Other Royalty Rate | 0 | % |

| *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|
| Operating Period | 20 | years | Price Escalation | 3 | % |
| Discount Rate | 10 | % | Cost Escalation | 3 | % |

## Output Metrics

| *Cost Analysis* | | | | | | | |
|---|---|---|---|---|---|---|---|
| Overnight Capital Cost | $ | 264,059,303 | | | | | |
| Constant Dollar Annual OPEX | $ | 277,175,172 | $ | 68.74 | /ton | $ 2,099.7 | /kg |
| Constant Dollar Annual REE Rev | $ | 77,520,156 | $ | 19.23 | /ton | $ 587.25 | /kg |
| Constant Dollar Coal Rev | $ | - | | | | | |
| Constant Dollar Other Rev | $ | - | | | | | |

| *Economic Indicators* | | | | | |
|---|---|---|---|---|---|
| Payback Period | | 25.5 | years | *22.5* | *operating years* |
| Internal Rate of Return | | N/A | | | |
| Net Present Value | $ | (1,598,005,780) | | | |
| Total Severence Tax Paid | $ | - | | | |
| Total State and Federal Tax Paid | $ | - | | | |

# TECHNICAL OUTPUT SUMMARY

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-4 | By: ks

## Mass Balance Information

### *Feed Inforatmion*

| | Solid Feed | | | Liquid Feed | | | | | | Combined |
|---|---|---|---|---|---|---|---|---|---|---|
| Description: | Shakespear Lignight | | | | | | Reserved | | | |
| Circuit Entry Point: | Entering Leaching | | | | | | | | | |
| Feed Rate: | 500 | tph (short tons) | | | LPM | | | | | |
| REE Grade: | 329 | ppm or g/t or mg/kg | | | mg/L or ppm | | | | | |
| | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/L) | Flux (kg/hr) | REE Dist (%) [1] | Grade (mg/kg) | Flux (kg/hr) | Flux (kg/hr) |
| Scandium | 6.89 | 22.69 | 10.29 | | | | | | | 10.29 |
| Yttrium | 7.82 | 25.76 | 11.68 | | | | | | | 11.68 |
| Lanthanum | 16.60 | 54.70 | 24.81 | | | | | | | 24.81 |
| Cerium | 38.12 | 125.57 | 56.96 | | | | | | | 56.96 |
| Praseodymium | 4.44 | 14.63 | 6.64 | | | | | | | 6.64 |
| Neodymium | 14.89 | 49.05 | 22.25 | | | | | | | 22.25 |
| Samarium | 3.32 | 10.95 | 4.97 | | | | | | | 4.97 |
| Europium | 0.57 | 1.88 | 0.85 | | | | | | | 0.85 |
| Gadolinium | 3.16 | 10.40 | 4.72 | | | | | | | 4.72 |
| Terbium | 0.08 | 0.25 | 0.11 | | | | | | | 0.11 |
| Dysprosium | 1.11 | 3.67 | 1.66 | | | | | | | 1.66 |
| Holmium | 0.25 | 0.82 | 0.37 | | | | | | | 0.37 |
| Erbium | 1.18 | 3.89 | 1.76 | | | | | | | 1.76 |
| Thulium | 0.17 | 0.56 | 0.26 | | | | | | | 0.26 |
| Ytterbium | 1.09 | 3.60 | 1.63 | | | | | | | 1.63 |
| Lutetium | 0.30 | 0.99 | 0.45 | | | | | | | 0.45 |
| Manganese | -- | 142.00 | 64.41 | | | 0.00 | -- | | | 64.41 |
| Cobalt | -- | 21.70 | 9.84 | | | 0.00 | -- | | | 9.84 |
| Nickel | -- | 46.20 | 20.96 | | | 0.00 | -- | | | 20.96 |
| Zinc | -- | 102.40 | 46.45 | | | 0.00 | -- | | | 46.45 |
| Reserved | -- | | 0.00 | | | 0.00 | -- | | | 0.00 |
| Total REE | 100.00 | 329.42 | 149.42 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 149.4 |
| Total non-REE CM | -- | 312.30 | 141.66 | -- | 0.00 | 0.00 | -- | 0.00 | 0.00 | 141.7 |

Notes/Instructions:

[1] Enter the feed rate and feed grade entering the process. The model allows for up to three simultaneous and distinct circuit feeds, each with a different feed distribution and a different processing route.
[2] Calculations are performed by assuming a fixed REE distribution with a user inputted feed grade and feed rate. This approach provides a simple mechanism for sensitivity analysis. Data are summed to calculate a combined flux.

### *Product Information*

| | Combined Feed | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | Total | Calc. Loss | Overall | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Rate (kg/hr) | Product (kg/hr) | Rate (kg/hr) | Recovery (%) | |
| Scandium | 10.29 | 0.00 | 1.44 | 0.00 | 0.00 | 0.00 | 1.45 | 8.85 | 14.1% | |
| Yttrium | 11.68 | 0.54 | 0.00 | 0.00 | 0.00 | 0.00 | 0.54 | 11.14 | 4.7% | |
| Lanthanum | 24.81 | 3.12 | 0.00 | 0.00 | 0.01 | 0.00 | 3.14 | 21.67 | 12.6% | |
| Cerium | 56.96 | 6.39 | 0.00 | 0.01 | 0.01 | 0.00 | 6.41 | 50.55 | 11.3% | |
| Praseodymium | 6.64 | 0.67 | 0.00 | 0.00 | 0.01 | 0.00 | 0.68 | 5.96 | 10.2% | |
| Neodymium | 22.25 | 2.84 | 0.00 | 0.00 | 0.01 | 0.00 | 2.85 | 19.39 | 12.8% | |
| Samarium | 4.97 | 0.59 | 0.00 | 0.01 | 0.00 | 0.00 | 0.60 | 4.37 | 12.0% | |
| Europium | 0.85 | 0.09 | 0.00 | 0.00 | 0.00 | 0.00 | 0.09 | 0.76 | 10.4% | |
| Gadolinium | 4.72 | 0.34 | 0.00 | 0.00 | 0.00 | 0.00 | 0.34 | 4.37 | 7.3% | |
| Terbium | 0.11 | 0.02 | 0.00 | 0.00 | 0.00 | 0.00 | 0.02 | 0.09 | 19.5% | |
| Dysprosium | 1.66 | 0.15 | 0.00 | 0.00 | 0.00 | 0.00 | 0.15 | 1.51 | 9.0% | |
| Holmium | 0.37 | 0.02 | 0.00 | 0.00 | 0.00 | 0.00 | 0.02 | 0.36 | 4.9% | |
| Erbium | 1.76 | 0.04 | 0.00 | 0.00 | 0.00 | 0.00 | 0.04 | 1.72 | 2.5% | |
| Thulium | 0.26 | 0.01 | 0.00 | 0.00 | 0.00 | 0.00 | 0.01 | 0.25 | 2.4% | |
| Ytterbium | 1.63 | 0.02 | 0.00 | 0.00 | 0.00 | 0.00 | 0.02 | 1.61 | 1.5% | |
| Lutetium | 0.45 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.45 | 1.0% | |
| Manganese | 64.41 | 0.14 | 0.00 | 6.17 | 13.65 | 0.00 | 19.96 | 44.45 | 31.0% | |
| Cobalt | 9.84 | 0.08 | 0.00 | 0.23 | 1.28 | 0.00 | 1.59 | 8.25 | 16.2% | |
| Nickel | 20.96 | 0.16 | 0.00 | 0.88 | 3.67 | 0.00 | 4.72 | 16.24 | 22.5% | |
| Zinc | 46.45 | 1.09 | 0.00 | 0.10 | 9.59 | 0.00 | 10.79 | 35.66 | 23.2% | |
| Reserved | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.0% | |
| Total REE | 149.42 | 14.85 | 1.44 | 0.03 | 0.04 | 0.00 | 16.37 | 133.05 | 11.0% | <---- |
| Total non-REE CM | 141.66 | 1.48 | 0.00 | 7.38 | 28.20 | 0.00 | 37.06 | 104.60 | 26.2% | <---- |

| Waste Stream | Production Rate | Unit / [hr] | | Disposal Cost ($/Unit) |
|---|---|---|---|---|
| Sorter Waste | - | ton | $ | - |
| Non-Hazardous Solid Waste | 355 | ton | $ | 0.10 |
| Non-Hazardous Precipitate | 35 | ton | $ | 1.50 |
| Dust/Volatiles | 20 | ton | $ | - |
| Waste Water | - | ton | $ | 0.001 |
| Reserved | - | ton | $ | - |

| | | |
|---|---|---|
| 0.36 | TPD REE | |
| 0.035 | TPD Sc | |
| 0.85 | TPD CM | |
| 0.10 | REE | |
| 14.0% | Sc | |
| 9.6% | Mn | |
| 18.8% | Metal-S | |
| 500 | Feed | |
| 329 | Grade | |
| 25.5 | PP | |
| N/A | IRR | |
| -1598005780 | NPV | |

Notes/Instructions:

[1] Enter the mass flowrate of each valueable element into each of up to five saleable products streams. Recovery panel to right is can be used to link product flow rates to technical factors.

## Other Global Paramters

**Operating Schedule Matrix**

| | Hours | Shifts | Day | Week | Year |
|---|---|---|---|---|---|
| Hours | 1 | 8 | 24 | 168 | 8064 |
| Shifts | -- | 1 | 3 | 21 | 1008 |
| Day | -- | -- | 1 | 7 | 336 |
| Week | -- | -- | -- | 1 | 48 |
| Year | -- | -- | -- | -- | 1 |

# TECHNICAL OUTPUT SUMMARY

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-4 | By: ks

## Technical Design Factors

| Parameter | Value | Unit | Source |
|---|---|---|---|
| *PM1: Feed Preparation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Sorter Yield | 100.0% | m/m | (enter 100% for no sorting) |
| Sorter Conc. Factor (c/f) | 100.0% | m/m | |
| **Equipment Specifications** | | | |
| Loader Capacity | 500 | tph/10 yd Unit | VT: Design |
| VSI Efficiency | 1.2 | HP/TPH | VT: Assumed |
| Roll Crusher Efficiency | 0.75 | HP/TPH | VT: Assumed |
| Screen Opening | 1.00 | Inch | VT: Design |
| Diesel Consumption | 6.3 | gal/hr/loader | VT: MAMEC Design |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton | VT: Industrial sources |
| *PM2: Roasting* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| % of heat from Gas | 10.0% | E/E | VT: Assumed |
| Volatile Loss | 4.0% | m/m | UK: Measured |
| Temperature Change | 575 | deg C | UK: Design |
| **Equipment Specifications** | | | |
| Bin Storage Capacity | 20% | % of hourly prod. | VT: Design |
| Material Heat Capacity | 0.80 | kg/kg-K | VT: Assumed |
| Reaction Enthalpy | 1783.00 | kJ/kg | VT: Assumed |
| % of Feed Reacted | 50% | % | VT: Assumed |
| Scurbber Airflow | 20.00 | cfm/tph | VT: Design |
| Quencher Aiflow | 20.00 | cfm/tph | VT: Design |
| *PM3: Leaching* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 13.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Leach % Solids | 20% | % | Based on Design |
| Dissolved Solids in Leach | 26.1% | % | WK4/WK4 Mass Balance-V1 |
| Leaching Retention Time | 300 | min | Based on Design |
| Thickener UF % Solids | 20% | % | VT Override from 7.1% \| WK4/WK4 Mass Balance-V1 |
| Residue Settling Rate | 15.0 | ft/hr | Not Available |
| **Reagent Use** | | | |
| Acid Consumption | 0.0 | kg/ton | |
| Bioleaching Sol'n Consumption | 4656.9 | L/ton | WK4/WK4 Mass Balance-V1 |
| Polymer Dosage | 0.028 | g/L | WK4/WK4 Mass Balance-V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Leach Tanks H/D | 1.00 | ft/ft | VT: Assumed |
| Surface Heat Loss | 230 | BTU/ft^2/hr | VT: Assumed |
| Wall Heat Loss | 15 | BTU/ft^2/hr | VT: Assumed |
| Number of Belt Filters | 2.00 | -- | VT: Design |
| Thickener Capacity | 3.69 | ft^2/tph | VT: Calculated |
| *PM4: Al Precipitation* | | | |
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 11.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 72.0 | min | Based on Design |
| Precip Solid Yield | 17.28 | g/L | WK4/WK4 Mass Balance-V1 |
| Precip Settling Rate | 2.24 | ft/hr | Lily Banks/Flocculant Tests |
| Thickener UF %Solids | 4.0% | % | VT Override from 1.65% \| WK4/WK4 Mass Balance-V1 |
| **Reagent Use** | | | |
| NaOH Dose | 6.65 | g/L of Feed | WK4/WK4 Mass Balance-V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 484.29 | ft^2/tph | VT: Calculated |

## Watch Factors

| Parameter | Value | Unit | Desired Range |
|---|---|---|---|
| **Material Balance** | | | |
| Sorter Feed Rate | 500 | tph | |
| Sorter Product Rate | 500 | tph | |
| Sorter Wate Rate | 0 | tph | |
| Sorter Ejected Tons | 0 | tph | |
| **Other Indicators** | | | |
| Grade Before Sorter | 329 | ppm | |
| Grade After Sorter | 329 | ppm | |
| **Material Balance** | | | |
| Roaster Feed Rate | 500 | tph | |
| Roaster Product Rate | 480 | tph | |
| **Other Indicators** | | | |
| Roaster Size | 581 | MBTU/hr | 1 to 100 MMBTU/hr |
| **Material Balance** | | | |
| Leaching Feed Rate | 480 | tph | |
| Leaching Residue Rate | 354.576 | tph | |
| Slurry Flow (feed) | 8880 | gpm | |
| Product Flow | 7,995 | gpm | |
| Thickener Feed % Solids (eff) | 15% | % | |
| Liquid Density | 1.023 | g/cc | |
| **Other Indicators** | | | |
| Dissolved Solid Conc. | 65.22 | g/L | |
| Leaching Vessel Size | 51230.77 | gal | |
| Leaching Vessel Size | 193.93 | m^3 | ideally < 200 m^2 |
| Belt Filter Size | 177.29 | tph | 70 to 200 tph |
| Thickener Area | 100.71 | ft^2 | |
| Thickener Diameter | 11.33 | ft | ideal < 100' |
| Liquid Balance | 313.11 | gpm | must be > 0; adjust liquid density |
| Slurry Balance | 0.00 | gpm | should be close to zero |
| **Material Balance** | | | |
| Slurry Feed Rate | 7,995 | gpm | |
| Slurry Feed Rate | 30,264 | lpm | |
| Precip Mass Flow | 34.59 | tph | |
| Thickener Feed % Solids (eff) | 1.69% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.011 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 198.09 | m^3 | ideally < 200 m^2 |
| Filter Press Area | 162.95 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 40.63 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 1,523 | ft^2 | |
| Thickener Diameter | 44.04 | ft | ideal < 100' |

*PM5: REE Precipitation*

| | | | |
|---|---|---|---|
| **Flowsheet Design** | | | |
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 12.0 | -- | VT: Process Design |
| | | | |
| **Process Parameters** | | | |
| Precip Conditioning Time | 72 | min | Based on Design |
| Precip Solid Yield | 9.35 | g/L | WK4/WK4 Mass Balance-V1 |
| Precip Settling Rate | 0.044 | ft/hr | Lily Banks/Settling Tests |
| Thickener UF % Solids | 1.00% | % | VT Override from 1.58% \| WK4/WK4 Mass Balance-V1 |
| | | | |
| **Reagent Use** | | | |
| NaOH Dose | 3.00 | g/L of Feed | WK4/WK4 Mass Balance-V1 |
| CaCO3 Dose | 0.00 | g/L of Feed | |
| Coal Calcite Dose | 0.00 | g/L of Feed | |
| | | | |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 60.00 | min | VT: Assumed |
| Thickener Capacity | 5915.84 | ft^2/tph | VT: Calculated |

| | | | |
|---|---|---|---|
| **Material Balance** | | | |
| Slurry Feed Rate | 7954 | gpm | |
| Slurry Feed Rate | 30,110 | lpm | |
| Precip Mass Flow | 18.62 | tph | |
| Thickener Feed % Solids (eff) | 0.9% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.005 | g/cc | |
| | | | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 180.66 | m^3 | ideally < 200 m^3 |
| Filter Press Area | 175.53 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 21.87 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 9180 | ft^2 | |
| Thickener Diameter | 108.14 | ft | ideal < 100' |
| | | | |
| Tons of Feed/ton precip | 25.78 | m/m | target = 6.35 |

*PM6: RE-Redissolution*

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 2.2% | % | VT Override from 10.73% \|\| WK4/WK4 Mass Balance-V1 |
| Dissolved Solids in Re-Leach | 13% | % | WK4/WK4 Mass Balance-V1 |
| Leaching Retention Time | 120 | min | Based on Design |
| **Reagent Use** | | | |
| Acid Consumption | 422.6 | kg/ton | WK4/WK4 Mass Balance-V1 |
| | 0.0 | | |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |

| Material Balance | | | |
|---|---|---|---|
| Solid Feed Rate | 18.62 | tph | |
| Slurry Flow (feed) | 3,333 | gpm | |
| Product Flow | 3,333 | gpm | |
| Liquid Density | 1.016 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 378.5 | m^3 | ideally < 200 m^3 |
| Releach Conc. Factor | 2.40 | v/v | target = 2.37, based on test data |
| Liquid Balance | 21.88 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | 0.0 | gpm | should be close to zero |
| Dissolved Solids | 22.3 | g/L | |

*PM7: REE Oxalate Precipitation*

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | 20 | min | Based on Design |
| Precip Solid Yield | 0.1 | g/L | WK4/WK4 Mass Balance-V1 |
| Precip Settling Rate | 10.0 | ft/hr | Not Available |
| Thickener UF %Solids | | % | Not Applicable |
| **Reagent Use** | | | |
| NaOH Dose | 2.35 | g/L | WK4/WK4 Mass Balance-V1 |
| Oxalic Acid Dose | 7.63 | g/L | WK4/WK4 Mass Balance-V1 |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Temperature Change | 1000.00 | deg C | VT: Design |
| Heat Capacity | 0.80 | kJ/kg-K | VT: Assumed |
| Enthalpy | 1000.00 | kJ/kg | VT: Assumed |
| % Reacted | 100% | % | VT: Assumed |

| Material Balance | | | |
|---|---|---|---|
| Slurry Feed Rate | 3332.93 | gpm | |
| Slurry Feed Rate | 12616.50 | lpm | |
| Precip Mass Flow | 0.05 | tph | |
| Thickener Feed % Solids (eff) | 0.01% | % | |
| Solid Density | 3.40 | g/cc | |
| REO Mass | 0.03 | tph | |
| Liquid Density | 1.016 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | 63.08 | m^3 | |
| Filter Press Area | 0.25 | ft^3 | ideal 30 to 800 |
| Liquid Balance | 0.06 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |
| Ratio of Conc. | 15.27 | Ton feed/kg product | |
| Oxide to Oxalate Ratio | 1.68 | m/m | |

*PM8: Sc SX Separation*

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.00 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Mixer Residence Time | 1.00 | min | UK: Design |
| Settler Residence Time | 15.00 | min | UK: Design |
| Loading: Number of Stages | 2.00 | - | UK: Design |
| Loading O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Loading O:A Advance Ratio | 0.03 | v/v | UK: Design |
| Scrubbing: Number of Stages | | - | N/A |
| Scrubbing O:A Mixer Ratio | 1.00 | v/v | N/A |
| Scrubbing O:A Advance Ratio | | v/v | N/A |
| Stripping: Number of Stages | 2.00 | - | UK: Design |
| Stripping O:A Mixer Ratio | 1.00 | v/v | UK: Design |
| Stripping O:A Advance Ratio | 12.54 | v/v | UK: Design |
| Sc2O3 Grade | 2% | % | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Conditioning Time | 600.00 | min | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Precip Settling Rate | 10.0 | ft/hr | N/A |
| Precip Solid Yield | 0.4 | g/L | UK: Daniel Dailey/Experiments/Scandium Cleaning/2022 DD-SX-P2 |
| Strip Solution Density | 1.13 | g/cc | UK: Data |
| Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Organic Entrainment (loss) | 60 | mg/L | VT: Assumed |
| Diluent Density | 0.84 | g/ml | VT: Assumed |
| Extractant Density | 0.92 | g/ml | VT: Assumed |
| Extractant Concentration | 5% | % Volume | UK: Design |
| Scrubbing Acid Molarity | | mol/L | N/A |
| Srubbing Acid Molar Mass | | g/mol | N/A |
| Scrub Acid Stock Conc | 37% | % | N/A |
| Stripping Solution Molarity | 2.00 | Mol/L | UK: Design |
| Stripping Solution Acid Molar M | 98.000 | g/mol | UK: Design |
| Strip Acid Stock Conc | 93% | % | UK: Design |
| Neutralizaiton : Strip Ratio | | mol base / mol ac | N/A |
| Neutralization Molar Mass | | g/mol | N/A |
| Neutralization Stock Conc | 50% | % | N/A |
| **Equipment Specifications** | | | |
| SX Mixing Power Intensity | 2.50 | HP/1000 gal | VT: Assumed |
| Precip Mixing Power Intensity | 0.50 | HP/1000 gal | VT: Assumed |
| Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

| Material Balance (SX) | | |
|---|---|---|
| PLS Flow | 7994.8 | gpm |
| Raffinate Rate: | 7994.8 | gpm |
| Load Org Recycle | 7775.0 | gpm |
| Load Org Feed | 7994.8 | gpm |
| Loag Org Advance | 219.8 | gpm |
| Scrub Aq Product | 0.0 | gpm |
| Scrub Aq Recycle | 0.0 | gpm |
| Scrub Aq Feed | 0.0 | gpm |
| Scurb Org Product | 219.8 | gpm |
| Strip Aq Product | 17.5 | gpm |
| Strip Aq Recycle | 202.3 | gpm |
| Strip Aq Feed | 219.8 | gpm |
| Strip Org Product | 219.8 | gpm |
| Extraction Feed (O+A): | 15989.7 | gpm |
| Scrub Feed (O+A): | 219.8 | gpm |
| Strip Feed (O+A): | 439.6 | gpm |
| **Material Balance (Precip)** | | |
| Strip Solution Flow | 17.5 | gpm |
| Solid Density | 3.4 | g/cc |
| Solid Product Rate | 0.00 | tph |
| Liquid Density (after precip) | 1.13 | g/cc |
| Thickener Feed % Solids (eff) | 0.03% | % |
| **Total Volumes** | | |
| Total Extraction Volume | 511670 | gal |
| Total Scrubbing Volume | 0 | gal |
| Total Stripping Volume | 14068 | gal |
| Extraction Org Volume | 255835.00 | gal |
| Scrubbing Org Volume | 0.00 | gal |
| Stripping Org Volume | 7033.82 | gal |
| Total Org Volume | 262868.82 | gal |
| **Other Indicators** | | |
| Liquid Balance | 0.00 | gpm must be > 0; adjust slurry density |
| Slurry Balance | 0.00 | gpm should be close to zero |
| Conditioning Vessel Size | 9.96 | m^3 |
| Filter Press Area | 0.01 | ft^3 ideal 30 to 800 |
| Thickener Area | 0.00 | ft^2 |
| Thickener Diameter | 0.00 | ft ideal < 100' |

*PM9: CM Precipitation*

| Flowsheet Design | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 4.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Precip Conditioning Time | | min | N/A |
| Precip Solid Yield | - | g/L | N/A |
| Precip Settlng Rate | 10 | ft/hr | N/A |
| Thickener UF % Solids | 25% | % | N/A |
| **Reagent Use** | | | |
| NaOH Dose | 7.24 | g/L of Feed | N/A |
| CaCO3 Dose | 0.00 | g/L of Feed | N/A |
| Coal Calcite Dose | 0.00 | g/L of Feed | N/A |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |
| Filter Press Cycle Time | 0.00 | min | N/A |
| Thickener Capacity | 0.00 | ft^2/tph | N/A |

| Material Balance | | | |
|---|---|---|---|
| Slurry Feed Rate | 7932 | gpm | |
| Slurry Feed Rate | 30,027 | lpm | |
| Precip Mass Flow | - | tph | |
| Thickener Feed % Solids (eff) | 0.0% | % | |
| Solid Density | 3.40 | g/cc | |
| Liquid Density | 1.005 | g/cc | |
| **Other Indicators** | | | |
| Conditioning Vessel Size | - | m^3 | ideally < 200 m^2 |
| Filter Press Area | - | ft^3 | ideal 30 to 800 |
| Liquid Balance | (0.00) | gpm | must be > 0; adjust slurry density |
| Slurry Balance | (0.00) | gpm | should be close to zero |
| Thickener Area | 0 | ft^2 | |
| Thickener Diameter | 0.00 | ft | ideal < 100' |

*PM10: CM Redissolution*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Re-Leach %Solids | 2.0% | % | N/A |
| Dissolved Solids in Re-Leach | 100% | % | N/A |
| Leaching Retention Time | 45 | min | N/A |
| **Reagent Use** | | | |
| Acid Consumption | 441 | kg/ton | N/A |
| **Equipment Specification** | | | |
| Mixing Power Intensity | 0.00 | HP/1000 gal | N/A |

**Material Balance**

| | | | |
|---|---|---|---|
| Solid Feed Rate | 0.00 | tph | |
| Slurry Flow (feed) | 0 | gpm | |
| Product Flow | 0 | gpm | |
| Liquid Density | 1.000 | g/cc | |
| **Other Indicators** | | | |
| Releach Vessel Size | 0.0 | m^3 | |
| Releach Conc. Factor | #DIV/0! | v/v | |
| Liquid Balance | 0.00 | gpm | must be > 0; adjust slurry density |
| Slurry Balance | - | gpm | should be close to zero |
| Dissolved Solids | #DIV/0! | g/L | |

*PM11: CM Recovery*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | Yes | txt | UK: Test Flowsheet |
| Number of Parallel PM | 1.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Co Precip Conditioning Time | 30.00 | min | Based on design |
| Co Precip Solid Yield | 1.67 | g/L | WK4/WK4 Mass Balance-V1 |
| Co Precip Settling Rate | | ft/hr | Not Available |
| Co Thickener UF % Solids | | % | Not Applicable |
| Mn Precip Conditioning Time | 30.00 | min | Based on design |
| Mn Precip Solid Yield | 1.73 | g/L | WK4/WK4 Mass Balance-V1 |
| Mn Precip Settling Rate | | ft/hr | Not Applicable |
| Mn Thickener UF % Solids | | % | N/A |
| **Reagent Use** | | | |
| Na2S Dose | 6.70 | g/L of Feed | WK4/WK4 Mass Balance-V1 |
| NaOH Dose | 4.08 | g/L of Feed | WK4/WK4 Mass Balance-V1 |
| HCL Dose | 21.59 | g/L of Feed | WK4/WK4 Mass Balance-V1 |
| **Equipment Specifications** | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | VT: Assumed |
| Co Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Co Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |
| Mn Filter Press Cycle Time | 30.00 | min | VT: Assumed |
| Mn Thickener Capacity | 0.00 | ft^2/tph | VT: Calculated |

**Material Balance**

| | | | |
|---|---|---|---|
| Co Slurry Feed Rate | 3332.87 | gpm | |
| Co Slurry Feed Rate | 12616.27 | lpm | |
| Co Precip Mass Flow | 1.39 | tph | |
| Co Thickener Feed % Solids (eff) | 0.16% | % | |
| Co Solid Density | 3.4 | g/cc | |
| Mn Slurry Feed Rate | 3331.23 | gpm | |
| Mn Slurry Feed Rate | 12610.07 | lpm | |
| Mn Precip Mass Flow | 1.44 | tph | |
| Mn Thickener Feed % Solids (eff) | 0.17% | % | |
| Mn Solid Density | 3.4 | g/cc | |
| Co Liquid Density | 1.01 | g/cc | |
| Mn Liquid Density | 1.01 | g/cc | |
| **Other Indicators** | | | |
| Co Conditioning Vessel Size | 189.2 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 6.57 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 1.64 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |
| Co Conditioning Vessel Size | 189.2 | m^3 | ideally < 200 m^2 |
| Co Filter Press Area | 6.8 | ft^3 | ideal 30 to 800 |
| Co Liquid Balance | 1.70 | gpm | must be > 0; adjust slurry density |
| Co Slurry Balance | 0.00 | gpm | should be close to zero |
| Co Thickener Area | 0.0 | f^2 | |
| Co Co Thickener Diameter | 0.00 | ft^3 | ideal < 100' |

*PM12: Biooxidation*

**Flowsheet Design**

| | | | |
|---|---|---|---|
| Include? | No | txt | |
| Number of Parallel PM | 5.0 | -- | VT: Process Design |
| **Process Parameters** | | | |
| Raw Leaching Solution Requirec | 250 | gpm | |
| BioLeach Retention Time 1 | 36 | hr | 2.5 |
| BioLeach Retention Time 2 | 18 | hr | 60 |
| BioLeach Retention Time 3 | 18 | hr | |
| Output Acid Molarity | 0.7 | Mol/L | |
| Acid Molar Mass | 98 | g/mol | |
| Acid Solution Density | 1.2 | g/mL | |
| Pyrite Density | 5.1 | g/cc | |
| Solution Heat Capacity | 4.2 | kJ/kg-K | |
| Temperature Drop | 6 | deg C | |
| Chiller Efficiency | 55% | % | |
| **Reagent Use** | | | |
| Pyrite Consumption | 0.1 | kg/L | |
| Ammonium Sulfate Consumptic | 1.5 | g/L | |
| Nutrient Selection | K2HPO4 | txt | |
| Nutrient Dose | 0.6 | g/L | |
| **Equipment Specifications** | | | |
| Airflow requirement | 0.1 | lpm air/lpm solution | |

**Material Balance**

| | | | |
|---|---|---|---|
| Raw Leach Solution Req'd | 250.0 | gpm | |
| Raw Leach Solution Req'd | 946.4 | lpm | |
| Yearly Acid Production | 457,883,409 | liters solution/yr | |
| Yearly Acid Production | 549,460 | mt solution/yr | |
| Acid Concentration | 0.0686 | kg/L | |
| **Other Indicators** | | | |
| Largest Bioleach Vesel Size | 0.00 | m^3 | |
| Mass Pyrite to Acid | 68.6% | % | |
| **Cost Indicators** | | | |
| Annual OPEX | $ - | $/yr | |
| Solution Cost (OPEX) | $ - | $/L | |
| Solution Cost (OPEX) | $ - | $/mt | |
| Acid Cost (OPEX) | $ - | $/kg acid equivalent | |
| Total CAPEX (Installed) | $ - | $ | |
| Annualized CAPEX | $ - | $/yr | |
| Solution Cost (CAPEX) | $ - | $/L | |
| Solution Cost (CAPEX) | $ - | $/mt | |
| Acid Cost (CAPEX) | $ - | $/kg acid equivalent | |
| Solution Cost (Total) | $ - | $/L | |
| Solution Cost (Total) | $ - | $/mt | |
| Acid Cost (Total) | $ - | $/kg acid equivalent | |

# CAPITAL COST ESTIMATE

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-4 | By: ks

## CAPEX Summary

| No. | Process Module | Equipment Cost | Reagent Fills | Installation | Other Direct | Indirect | TOTAL PLANT COST | Relative Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 2,463,659 | $ - | $ 615,915 | $ 2,710,025 | $ 3,373,981 | $ 9,163,581 | IIIIIIIIII |
| PM2 | Roasting | $ 19,585,772 | $ - | $ 4,896,443 | $ 21,544,350 | $ 26,822,715 | $ 72,849,280 | IIIIIIII

# OPERATING COST ESTIMATE

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-4 | By: ks

## Fixed Operating Costs

*Operating & Maintenance Labor Requirement*

| Labor Category | Code | Labor Rate ($/hr.) | Labor Burden | No. Per Shift | Shift Cost | Source/Notes |
|---|---|---|---|---|---|---|
| Skilled Operator | OL | $ 27.90 | 25% | 2.00 | $ 558.00 | Occupation Code: 51-9012 |
| Operator | OL | $ 23.26 | 25% | 5.00 | $ 1,163.00 | Occupation Code: 51-9011 |
| Line Supervisor | OL | $ 30.29 | 25% | 2.00 | $ 605.80 | Occupation Code: 51-1011 |
| Maintenance/Electrician | OL | $ 24.06 | 25% | 3.00 | $ 721.80 | Occupation Code: 49-9043 |
| Lab Technician | TL | $ 23.43 | 25% | 1.00 | $ 234.30 | Occupation Code: 19-4031 |
| Metallurgist/Chemical Engineer | TL | $ 46.82 | 25% | 2.00 | $ 936.40 | Occupation Code: 17-2041 |
| Reserved | | | | | | |
| Reserved | | | | | | |
| **Sub-Total** | | | | **15.00** | **$ 4,219.30** | |

*Itemized Fixed Costs*

| Item | Code | Basis | Units | Base | Units | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| Operating Labor | OL | Itemized | | | | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 23.28 | $ 0.76 |
| Engineering & Tech. Labor | TL | Itemized | | | | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 8.94 | $ 0.29 |
| Maintenance and Supply Material | OH | 1 | % of Capital / Anum | $ 264,059,303 | $ Total | $ 2,619.64 | $ 7,858.91 | $ 2,640,593 | $ 20.00 | $ 0.65 |
| Sample Analysis, QAQC | OH | 10 | % of operating labor | $ 3,049 | $ / shift | $ 304.86 | $ 914.58 | $ 307,299 | $ 2.33 | $ 0.08 |
| Sales, IP, and R&D Costs | GSA | 0.5 | % of Sales / Anum | $ 77,520,156 | $ Total | $ 384.52 | $ 1,153.57 | $ 387,601 | $ 2.94 | $ 0.10 |
| Admin. & Support Labor | GSA | 15 | % of Direct Labor | $ 3,072,989 | $ / Anum | $ 457.29 | $ 1,371.87 | $ 460,948 | $ 3.49 | $ 0.11 |
| Property Taxes & Insurance | GSA | 1 | % of Capital / Anum | $ 264,059,303 | $ Total | $ 2,619.64 | $ 7,858.91 | $ 2,640,593 | $ 20.00 | $ 0.65 |
| Reserved | | | | | | | | | | |
| Reserved | | | | | | | | | | |
| **TOTAL FIXED COSTS** | | | | | | **$ 10,605.25** | **$ 31,815.74** | **$ 10,690,088** | **$ 80.98** | **$ 2.65** |

## Variable Operating Cost

*Power/Utilities*

| No. | Process Module | Code | Total HP | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | PR | 1065 | | $ 510.68 | $ 1,532.03 | $ 514,764 | $ 3.90 | $ 0.13 |
| PM2 | Roasting | PR | 198 | | $ 94.99 | $ 284.96 | $ 95,747 | $ 0.73 | $ 0.02 |
| PM3 | Leaching | PR | 5058 | | $ 2,424.73 | $ 7,274.20 | $ 2,444,130 | $ 18.52 | $ 0.61 |
| PM4 | Al Precipitation | PR | 928 | | $ 444.64 | $ 1,333.92 | $ 448,198 | $ 3.40 | $ 0.11 |
| PM5 | REE Precipitation | PR | 984 | | $ 471.49 | $ 1,414.48 | $ 475,265 | $ 3.60 | $ 0.12 |
| PM6 | RE-Redissolution | PR | 683 | | $ 327.52 | $ 982.55 | $ 330,138 | $ 2.50 | $ 0.08 |
| PM7 | REE Oxalate Precipitation | PR | 283 | | $ 135.83 | $ 407.48 | $ 136,913 | $ 1.04 | $ 0.03 |
| PM8 | Sc SX Separation | PR | 499 | | $ 239.42 | $ 718.25 | $ 241,331 | $ 1.83 | $ 0.06 |
| PM9 | CM Precipitation | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM10 | CM Redissolution | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM11 | CM Recovery | PR | 469 | | $ 224.69 | $ 674.07 | $ 226,486 | $ 1.72 | $ 0.06 |
| PM12 | Biooxidation | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM13 | Reserved | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM14 | Reserved | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| PM15 | Reserved | PR | 0 | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | **10167.8** | | **$ 4,873.98** | **$ 14,621.94** | **$ 4,912,971.68** | **$ 37.22** | **$ 1.22** |

*Consumables/Reagent*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | CR | | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.77 | $ 0.03 |
| PM2 | Roasting | CR | | $ 1,281.99 | $ 3,845.96 | $ 1,292,244 | $ 9.79 | $ 0.32 |
| PM3 | Leaching | CR | | $ 137,033.85 | $ 411,101.54 | $ 138,130,117 | $ 1,046.40 | $ 34.26 |
| PM4 | Al Precipitation | CR | | $ 12,081.25 | $ 36,243.76 | $ 12,177,902 | $ 92.25 | $ 3.02 |
| PM5 | REE Precipitation | CR | | $ 5,415.93 | $ 16,247.80 | $ 5,459,260 | $ 41.36 | $ 1.35 |
| PM6 | RE-Redissolution | CR | | $ 16,998.83 | $ 50,996.48 | $ 17,134,818 | $ 129.80 | $ 4.25 |
| PM7 | REE Oxalate Precipitation | CR | | $ 29,513.09 | $ 88,539.27 | $ 29,749,194 | $ 225.36 | $ 7.38 |
| PM8 | Sc SX Separation | CR | | $ 2,809.12 | $ 8,427.36 | $ 2,831,592 | $ 21.45 | $ 0.70 |
| PM9 | CM Precipitation | CR | | $ - | $ - | $ - | $ - | $ - |
| PM10 | CM Redissolution | CR | | $ - | $ - | $ - | $ - | $ - |
| PM11 | CM Recovery | CR | | $ 17,696.23 | $ 53,088.70 | $ 17,837,803 | $ 135.13 | $ 4.42 |
| PM12 | Biooxidation | CR | | $ - | $ - | $ - | $ - | $ - |
| PM13 | Reserved | CR | | $ - | $ - | $ - | $ - | $ - |
| PM14 | Reserved | CR | | $ - | $ - | $ - | $ - | $ - |
| PM15 | Reserved | CR | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ 222,931.09** | **$ 668,793.27** | **$ 224,714,537.09** | **$ 1,702.32** | **$ 55.73** |

*Leasing/Tolling*

| No. | Process Module | Code | Notes | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM2 | Roasting | LS | | $ - | $ - | $ - | $ - | $ - |
| PM3 | Leaching | LS | | $ - | $ - | $ - | $ - | $ - |
| PM4 | Al Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM5 | REE Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM6 | RE-Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM7 | REE Oxalate Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM8 | Sc SX Separation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM9 | CM Precipitation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM10 | CM Redissolution | LS | | $ - | $ - | $ - | $ - | $ - |
| PM11 | CM Recovery | LS | | $ - | $ - | $ - | $ - | $ - |
| PM12 | Biooxidation | LS | | $ - | $ - | $ - | $ - | $ - |
| PM13 | Reserved | LS | | $ - | $ - | $ - | $ - | $ - |
| PM14 | Reserved | LS | | $ - | $ - | $ - | $ - | $ - |
| PM15 | Reserved | LS | | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

*Waste Disposal*

| Waste Stream | Code | Unit | Daily Generation [units] / day | Disposal Cost $ / [unit] | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|
| Sorter Waste | WD | Tons | - | $ - | $ - | $ - | $ - | $ - | $ - |
| Non-Hazardous Solid Waste | WD | Tons | 8,510 | $ 0.10 | $ 283.66 | $ 850.98 | $ 285,930 | $ 2.17 | $ 0.07 |
| Non-Hazardous Precipitate | WD | Tons | 830 | $ 1.50 | $ 415.04 | $ 1,245.12 | $ 418,361 | $ 3.17 | $ 0.10 |
| Dust/Volatiles | WD | Tons | 28,800 | $ - | $ - | $ - | $ - | $ - | $ - |
| Waste Water | WD | NA | - | $ 0.001 | $ - | $ - | $ - | $ - | $ - |
| Reserved | WD | NA | - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Sub-Total** | | | | | **$ 698.70** | **$ 2,096.10** | **$ 704,291.17** | **$ 5.34** | **$ 0.17** |
| **TOTAL VARIABLE COSTS** | | | | | **$ 228,504** | **$ 685,511** | **$ 230,331,800** | **$ 1,745** | **$ 57** |

## Overhead Costs

| Allocation | Code | Rate | Basis | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| Fixed Portion | OH | 15% % of Direct Cost | $ 31,816 Daily Direct Cost | $ 1,590.79 | $ 4,772.36 | $ 1,603,513 | $ 12.15 | $ 0.40 |
| Variable Portion | OH | 15% % of Direct Cost | $ 685,511 Daily Direct Cost | $ 34,275.57 | $ 102,826.70 | $ 34,549,770 | $ 261.73 | $ 8.57 |
| **TOTAL OVERHEAD COSTS** | | | | **$ 35,866.35** | **$ 107,599.06** | **$ 36,153,283.26** | **$ 273.88** | **$ 8.97** |

# OPERATING COST ESTIMATE, COST SUMMARY

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | WK No. 13 | Plant Size: | 12000 TPD | Date: | 3/18/2023 |
| Case: | WK-4 | | | By: | ks |

## Summary by Cost Type

| Cost Type | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|
| Fixed | $ 10,605.25 | $ 31,815.74 | $ 10,690,088 | $ 81 | $ 2.65 | I |
| Power | $ 4,873.98 | $ 14,621.94 | $ 4,912,972 | $ 37 | $ 1.22 | |
| Consumables | $ 222,931.09 | $ 668,793.27 | $ 224,714,537 | $ 1,702 | $ 55.73 | IIIIIIIIIIIIIIIIIIIIIIII |
| Leasing/Tolling | $ - | $ - | $ - | $ - | $ - | |
| Waste Disposal | $ 698.70 | $ 2,096.10 | $ 704,291 | $ 5 | $ 0.17 | |
| General Overhead | $ 35,866.35 | $ 107,599.06 | $ 36,153,283 | $ 274 | $ 8.97 | IIII |
| TOTAL | $ 274,975 | $ 824,926 | $ 277,175,172 | $ 2,100 | $ 68.74 | |

## Summary by Process Module

| | Process Module | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| PM1 | Feed Preparation | $ 611 | $ 1,834 | $ 616,370 | $ 4.67 | $ 0.15 | |
| PM2 | Roasting | $ 1,377 | $ 4,131 | $ 1,387,991 | $ 10.51 | $ 0.34 | |
| PM3 | Leaching | $ 139,459 | $ 418,376 | $ 140,574,247 | $ 1,064.92 | $ 34.86 | IIIIIIIIIIIIIII |
| PM4 | Al Precipitation | $ 12,526 | $ 37,578 | $ 12,626,100 | $ 95.65 | $ 3.13 | I |
| PM5 | REE Precipitation | $ 5,887 | $ 17,662 | $ 5,934,525 | $ 44.96 | $ 1.47 | |
| PM6 | RE-Redissolution | $ 17,326 | $ 51,979 | $ 17,464,957 | $ 132.31 | $ 4.33 | II |
| PM7 | REE Oxalate Precipitation | $ 29,649 | $ 88,947 | $ 29,886,107 | $ 226.40 | $ 7.41 | III |
| PM8 | Sc SX Separation | $ 3,049 | $ 9,146 | $ 3,072,923 | $ 23.28 | $ 0.76 | |
| PM9 | CM Precipitation | $ - | $ - | $ - | $ - | $ - | |
| PM10 | CM Redissolution | $ - | $ - | $ - | $ - | $ - | |
| PM11 | CM Recovery | $ 17,921 | $ 53,763 | $ 18,064,289 | $ 136.85 | $ 4.48 | II |
| PM12 | Biooxidation | $ - | $ - | $ - | $ - | $ - | |
| PM13 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM14 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| PM15 | Reserved | $ - | $ - | $ - | $ - | $ - | |
| NA | Unallocated | $ 47,170 | $ 141,511 | $ 47,547,663 | $ 360 | $ 11.79 | IIIII |
| | TOTAL | $ 274,975 | $ 824,926 | $ 277,175,172 | $ 2,100 | $ 68.74 | |

## Summary by Cost Center

| Cost Center | Code | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost | Relative Cost |
|---|---|---|---|---|---|---|---|
| Operating Labor | OL | $ 3,048.60 | $ 9,145.80 | $ 3,072,989 | $ 23 | $ 0.76 | |
| Technical Labor | TL | $ 1,170.70 | $ 3,512.10 | $ 1,180,066 | $ 9 | $ 0.29 | |
| Power/Utilities | PR | $ 4,873.98 | $ 14,621.94 | $ 4,912,972 | $ 37 | $ 1.22 | |
| Lease Agreements | LS | $ - | $ - | $ - | $ - | $ - | |
| Consumables/Reagents | CR | $ 222,931.09 | $ 668,793.27 | $ 224,714,537 | $ 1,702 | $ 55.73 | IIIIIIIIIIIIIIIIIIIIIIII |
| Waste Disposal | WD | $ 698.70 | $ 2,096.10 | $ 704,291 | $ 5 | $ 0.17 | |
| Plant Overhead | OH | $ 38,790.85 | $ 116,372.55 | $ 39,101,175 | $ 296 | $ 9.70 | IIII |
| General Sales, and Administration | GSA | $ 3,461.45 | $ 10,384.35 | $ 3,489,142 | $ 26 | $ 0.87 | |
| Reserved | | | | | | | |
| Reserved | | | | | | | |
| TOTAL | | $ 274,975 | $ 824,926.11 | $ 277,175,172 | $ 2,100 | $ 68.74 | |

# REVENUE ESTIMATE

| | | | | | |
|---|---|---|---|---|---|
| Feedstock: | WK No. 13 | Plant Size: | 500 TPH | Date: | 3/18/2023 |
| Case: | WK-4 | | | By: | ks |

## REE/CM Revenue

REE Price Deck: NETL Oxides (Dec 2020)

**Annual REE Revenue:** $ 77,520,156

| | | Y + Lanthanides | REE + Sc |
|---|---|---|---|
| Feed Grade | (mg/kg) | 307 | 329 |
| Contained Value | ($/ton) | $ 6.1 | $ 138.7 |
| Feed Basket Price | ($/kg) | $ 22.0 | $ 464.3 |
| Product Basket Price | ($/kg) | $ 19.1 | $ 586.7 |
| Production Rate | (kg/day) | 358.15 | 392.87 |
| | (t/day) | 0.36 | 0.39 |
| | (t/yr) | 120.34 | 132.00 |
| | (% of US Demand) | 1.2% | |

## Revenue Contribution Table

| | | Product Revenue ($/hr) | | | | | |
|---|---|---|---|---|---|---|---|
| **Element** | **REE/CM Price ($/kg) [1]** | MREO Product 1 | Sc2O3 Product 2 | Mn Product 3 | CoS Product 4 | Reserved Product 5 | **REE Revenue ($/day)** |
| Scandium | $ 4,200.00 | $ 0.5 | $ 9,308.4 | $ - | $ - | $ - | $ 223,412 |
| Yttrium | $ 6.00 | $ 3.1 | $ - | $ - | $ - | $ - | $ 74 |
| Lanthanum | $ 2.00 | $ 5.5 | $ - | $ - | $ - | $ - | $ 132 |
| Cerium | $ 2.00 | $ 11.8 | $ - | $ - | $ - | $ - | $ 282 |
| Praseodymium | $ 52.00 | $ 31.7 | $ - | $ - | $ - | $ - | $ 761 |
| Neodymium | $ 42.00 | $ 104.5 | $ - | $ - | $ - | $ - | $ 2,507 |
| Samarium | $ 2.11 | $ 1.1 | $ - | $ - | $ - | $ - | $ 26 |
| Europium | $ 34.00 | $ 2.6 | $ - | $ - | $ - | $ - | $ 63 |
| Gadolinium | $ 32.00 | $ 9.4 | $ - | $ - | $ - | $ - | $ 225 |
| Terbium | $ 400.00 | $ 7.8 | $ - | $ - | $ - | $ - | $ 188 |
| Dysprosium | $ 230.00 | $ 29.4 | $ - | $ - | $ - | $ - | $ 707 |
| Holmium | $ 52.96 | $ 0.8 | $ - | $ - | $ - | $ - | $ 20 |
| Erbium | $ 34.00 | $ 1.3 | $ - | $ - | $ - | $ - | $ 30 |
| Thulium | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Ytterbium | $ 28.72 | $ 0.6 | $ - | $ - | $ - | $ - | $ 13 |
| Lutetium | $ 796.64 | $ 2.3 | $ - | $ - | $ - | $ - | $ 54 |
| Manganese | $ 0.005 | $ - | $ - | $ 0.1 | $ - | $ - | $ 1 |
| Cobalt | $ 22.00 | $ - | $ - | $ - | $ 21.1 | $ - | $ 507 |
| Nickel | $ 18.00 | $ - | $ - | $ - | $ 49.6 | $ - | $ 1,190 |
| Zinc | $ 3.02 | $ - | $ - | $ - | $ 21.7 | $ - | $ 522 |
| Reserved | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Total: | | $ 212.2 | $ 9,308.4 | $ 0.1 | $ 92.5 | $ - | $ 230,715 |

[1] REEs listed as $/kg REO; Mn listed as $/kg 44% Mn Ore; Co listed as $/kg metal content

## Coal Revenue

List Coal Revenue Sources Here:

**Annual Coal Revenue:** $ -

## Other Revenue

List Other Revenue Sources Here:

**Annual Other Revenue:** $ -

# PRO FORMA CASH FLOW

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-4 | Years: 1 through 12 | By: ks

| Period | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | $ - | $ - | $ - | $ 84,708,368 | $ 87,249,619 | $ 89,867,107 | $ 92,563,120 | $ 95,340,014 | $ 98,200,215 | $ 101,146,221 | $ 104,180,608 | $ 107,306,026 |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ - | $ - | $ - | $ 84,708,368 | $ 87,249,619 | $ 89,867,107 | $ 92,563,120 | $ 95,340,014 | $ 98,200,215 | $ 101,146,221 | $ 104,180,608 | $ 107,306,026 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | $ - | $ 792,178 | $ 5,656,255 | $ 310,646,479 | $ 319,075,732 | $ 327,738,152 | $ 336,639,550 | $ 345,785,842 | $ 355,183,046 | $ 364,837,282 | $ 374,754,765 | $ 384,941,812 |
| *- Operating Costs* | $ - | $ - | $ - | $ 302,876,794 | $ 311,963,098 | $ 321,321,991 | $ 330,961,650 | $ 340,890,500 | $ 351,117,215 | $ 361,650,731 | $ 372,500,253 | $ 383,675,261 |
| *- Loan #1 Interest* | $ - | $ 792,178 | $ 732,077 | $ 668,370 | $ 600,841 | $ 529,259 | $ 453,383 | $ 372,955 | $ 287,701 | $ 197,331 | $ 101,539 | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ 4,924,178 | $ 4,550,591 | $ 4,154,588 | $ 3,734,825 | $ 3,289,877 | $ 2,818,231 | $ 2,318,287 | $ 1,788,347 | $ 1,226,609 | $ 631,168 |
| *- Loan #3 Interest* | $ - | $ - | $ - | $ 2,550,724 | $ 2,357,206 | $ 2,152,077 | $ 1,934,640 | $ 1,704,156 | $ 1,459,844 | $ 1,200,873 | $ 926,364 | $ 635,384 |
| Gross Profit | $ - | $ (792,178) | $ (5,656,255) | $ (225,938,111) | $ (231,826,113) | $ (237,871,045) | $ (244,076,429) | $ (250,445,828) | $ (256,982,832) | $ (263,691,061) | $ (270,574,158) | $ (277,635,786) |
| **Book Deductions** | $ - | $ 990,222 | $ 7,146,304 | $ 10,345,354 | $ 9,569,453 | $ 8,851,744 | $ 8,187,863 | $ 7,573,773 | $ 7,005,740 | $ 6,480,310 | $ 5,994,287 | $ 5,544,715 |
| *- CAPEX #1 Depreciation* | $ - | $ 990,222 | $ 915,956 | $ 847,259 | $ 783,715 | $ 724,936 | $ 670,566 | $ 620,273 | $ 573,753 | $ 530,721 | $ 490,917 | $ 454,099 |
| *- CAPEX #2 Depreciation* | $ - | $ - | $ 6,230,349 | $ 5,763,072 | $ 5,330,842 | $ 4,931,029 | $ 4,561,202 | $ 4,219,112 | $ 3,902,678 | $ 3,609,977 | $ 3,339,229 | $ 3,088,787 |
| *- CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 3,735,023 | $ 3,454,896 | $ 3,195,779 | $ 2,956,096 | $ 2,734,388 | $ 2,529,309 | $ 2,339,611 | $ 2,164,140 | $ 2,001,830 |
| Taxable Before Depletion | $ - | $ (1,782,400) | $ (12,802,559) | $ (236,283,465) | $ (241,395,566) | $ (246,722,788) | $ (252,264,293) | $ (258,019,601) | $ (263,988,572) | $ (270,171,370) | $ (276,568,444) | $ (283,180,502) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ - | $ (891,200) | $ (6,401,280) | $ (118,141,733) | $ (120,697,783) | $ (123,361,394) | $ (126,132,146) | $ (129,009,801) | $ (131,994,286) | $ (135,085,685) | $ (138,284,222) | $ (141,590,251) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ - | $ - | $ - | $ 11,859,171 | $ 12,214,947 | $ 12,581,395 | $ 12,958,837 | $ 13,347,602 | $ 13,748,030 | $ 14,160,471 | $ 14,585,285 | $ 15,022,844 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | $ - | $ (1,782,400) | $ (12,802,559) | $ (236,283,465) | $ (241,395,566) | $ (246,722,788) | $ (252,264,293) | $ (258,019,601) | $ (263,988,572) | $ (270,171,370) | $ (276,568,444) | $ (283,180,502) |
| *- Income Tax* | $ - | $ (463,424) | $ (3,328,665) | $ (61,433,701) | $ (62,762,847) | $ (64,147,925) | $ (65,588,716) | $ (67,085,096) | $ (68,637,029) | $ (70,244,556) | $ (71,907,796) | $ (73,626,930) |
| Net Profit/Loss | $ - | $ (1,318,976) | $ (9,473,894) | $ (174,849,764) | $ (178,632,719) | $ (182,574,863) | $ (186,675,577) | $ (190,934,505) | $ (195,351,544) | $ (199,926,814) | $ (204,660,649) | $ (209,553,571) |
| **Book Deductions** | $ - | $ 990,222 | $ 7,146,304 | $ 10,345,354 | $ 9,569,453 | $ 8,851,744 | $ 8,187,863 | $ 7,573,773 | $ 7,005,740 | $ 6,480,310 | $ 5,994,287 | $ 5,544,715 |
| *+ CAPEX #1 Depreciation* | $ - | $ 990,222 | $ 915,956 | $ 847,259 | $ 783,715 | $ 724,936 | $ 670,566 | $ 620,273 | $ 573,753 | $ 530,721 | $ 490,917 | $ 454,099 |
| *+ CAPEX #2 Depreciation* | $ - | $ - | $ 6,230,349 | $ 5,763,072 | $ 5,330,842 | $ 4,931,029 | $ 4,561,202 | $ 4,219,112 | $ 3,902,678 | $ 3,609,977 | $ 3,339,229 | $ 3,088,787 |
| *+ CAPEX #3 Depreciation* | $ - | $ - | $ - | $ 3,735,023 | $ 3,454,896 | $ 3,195,779 | $ 2,956,096 | $ 2,734,388 | $ 2,529,309 | $ 2,339,611 | $ 2,164,140 | $ 2,001,830 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ - | $ (328,754) | $ (2,327,590) | $ (164,504,410) | $ (169,063,266) | $ (173,723,120) | $ (178,487,713) | $ (183,360,732) | $ (188,345,803) | $ (193,446,504) | $ (198,666,362) | $ (204,008,856) |
| **Captial Cost** | $ 13,202,965 | $ 83,071,314 | $ 49,800,307 | $ 10,950,836 | $ 11,607,887 | $ 12,304,360 | $ 13,042,621 | $ 13,825,179 | $ 14,654,689 | $ 15,533,971 | $ 16,466,009 | $ 15,660,110 |
| *- Fixed Capital Costs* | $ 26,405,930 | $ 164,139,263 | $ 85,024,138 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ 13,202,965 | $ 82,069,631 | $ 42,512,069 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ 1,001,682 | $ 1,061,783 | $ 1,125,490 | $ 1,193,019 | $ 1,264,600 | $ 1,340,476 | $ 1,420,905 | $ 1,506,159 | $ 1,596,529 | $ 1,692,321 | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ 6,226,455 | $ 6,600,043 | $ 6,996,045 | $ 7,415,808 | $ 7,860,756 | $ 8,332,402 | $ 8,832,346 | $ 9,362,287 | $ 9,924,024 | $ 10,519,465 |
| *- Loan #3 Principle Payments* | $ - | $ - | $ - | $ 3,225,304 | $ 3,418,822 | $ 3,623,951 | $ 3,841,389 | $ 4,071,872 | $ 4,316,184 | $ 4,575,155 | $ 4,849,665 | $ 5,140,644 |
| **Net Cash Flow** | $ (13,202,965) | $ (83,400,067) | $ (52,127,897) | $ (175,455,246) | $ (180,671,153) | $ (186,027,479) | $ (191,530,335) | $ (197,185,910) | $ (203,000,493) | $ (208,980,475) | $ (215,132,371) | $ (219,668,966) |
| Discounted Cash Flow | $ (13,202,965) | $ (75,818,243) | $ (43,080,907) | $ (131,822,123) | $ (123,400,828) | $ (115,508,429) | $ (108,113,881) | $ (101,187,551) | $ (94,701,228) | $ (88,628,122) | $ (82,942,842) | $ (76,992,632) |
| **Cumulative DCF** | $ (13,202,965) | $ (89,021,208) | $ (132,102,115) | $ (263,924,238) | $ (387,325,067) | $ (502,833,495) | $ (610,947,376) | $ (712,134,927) | $ (806,836,155) | $ (895,464,277) | $ (978,407,119) | $ (1,055,399,751) |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

## Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | |
|---|---|---|---|
| Overnight Capital Cost | $ 264,059,303 | Payback Period | 25.5 |
| Constant Dollar OPEX | $ 277,175,172 | Internal Rate of Return | N/A |
| Constant Dollar REE Rev | $ 77,520,156 | Net Present Value | $ (1,598,005,780) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal Tax Paid | $ - |

# PRO FORMA CASH FLOW

Feedstock: WK No. 13 | Plant Size: 500 TPH | Date: 3/18/2023
Case: WK-4 | Years: 13 through 23 | By: ks

| Period | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Sales Revenue** | **$ 110,525,207** | **$ 113,840,963** | **$ 117,256,192** | **$ 120,773,877** | **$ 124,397,094** | **$ 128,129,007** | **$ 131,972,877** | **$ 135,932,063** | **$ 140,010,025** | **$ 144,210,326** | **$ 148,536,635** |
| *+ Dry Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ REE and CM Products (Combined)* | $ 110,525,207 | $ 113,840,963 | $ 117,256,192 | $ 120,773,877 | $ 124,397,094 | $ 128,129,007 | $ 131,972,877 | $ 135,932,063 | $ 140,010,025 | $ 144,210,326 | $ 148,536,635 |
| *+ Other Byproducts* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Royalties/Local Taxes** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Coal Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Coal Severance Tax* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- REE Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Other Royalties* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Operating & Interest Expenses** | **$ 395,512,464** | **$ 407,041,084** | **$ 419,252,317** | **$ 431,829,886** | **$ 444,784,783** | **$ 458,128,326** | **$ 471,872,176** | **$ 486,028,341** | **$ 500,609,191** | **$ 515,627,467** | **$ 531,096,291** |
| *- Operating Costs* | $ 395,185,519 | $ 407,041,084 | $ 419,252,317 | $ 431,829,886 | $ 444,784,783 | $ 458,128,326 | $ 471,872,176 | $ 486,028,341 | $ 500,609,191 | $ 515,627,467 | $ 531,096,291 |
| *- Loan #1 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Interest* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Interest* | $ 326,945 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Gross Profit | $ (284,987,257) | $ (293,200,121) | $ (301,996,125) | $ (311,056,009) | $ (320,387,689) | $ (329,999,320) | $ (339,899,299) | $ (350,096,278) | $ (360,599,167) | $ (371,417,142) | $ (382,559,656) |
| **Book Deductions** | **$ 5,128,861** | **$ 4,744,197** | **$ 4,388,382** | **$ 4,059,253** | **$ 3,754,809** | **$ 3,473,199** | **$ 3,212,709** | **$ 2,971,756** | **$ 2,748,874** | **$ 2,334,467** | **$ 849,154** |
| *- CAPEX #1 Depreciation* | $ 420,041 | $ 388,538 | $ 359,398 | $ 332,443 | $ 307,510 | $ 284,446 | $ 263,113 | $ 243,379 | $ 225,126 | $ - | $ - |
| *- CAPEX #2 Depreciation* | $ 2,857,128 | $ 2,642,843 | $ 2,444,630 | $ 2,261,283 | $ 2,091,687 | $ 1,934,810 | $ 1,789,699 | $ 1,655,472 | $ 1,531,311 | $ 1,416,463 | $ - |
| *- CAPEX #3 Depreciation* | $ 1,851,693 | $ 1,712,816 | $ 1,584,354 | $ 1,465,528 | $ 1,355,613 | $ 1,253,942 | $ 1,159,897 | $ 1,072,904 | $ 992,437 | $ 918,004 | $ 849,154 |
| Taxable Before Depletion | $ (290,116,118) | $ (297,944,318) | $ (306,384,507) | $ (315,115,262) | $ (324,142,498) | $ (333,472,518) | $ (343,112,008) | $ (353,068,034) | $ (363,348,040) | $ (373,751,608) | $ (383,408,809) |
| **Depletion** | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- 50% Limit* | $ (145,058,059) | $ (148,972,159) | $ (153,192,254) | $ (157,557,631) | $ (162,071,249) | $ (166,736,259) | $ (171,556,004) | $ (176,534,017) | $ (181,674,020) | $ (186,875,804) | $ (191,704,405) |
| *- Percent Depletion Coal* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Percent Depletion REE* | $ 15,473,529 | $ 15,937,735 | $ 16,415,867 | $ 16,908,343 | $ 17,415,593 | $ 17,938,061 | $ 18,476,203 | $ 19,030,489 | $ 19,601,403 | $ 20,189,446 | $ 20,795,129 |
| *- Percent Depletion Other* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Taxable Income** | **$ (290,116,118)** | **$ (297,944,318)** | **$ (306,384,507)** | **$ (315,115,262)** | **$ (324,142,498)** | **$ (333,472,518)** | **$ (343,112,008)** | **$ (353,068,034)** | **$ (363,348,040)** | **$ (373,751,608)** | **$ (383,408,809)** |
| *- Income Tax* | $ (75,430,191) | $ (77,465,523) | $ (79,659,972) | $ (81,929,968) | $ (84,277,050) | $ (86,702,855) | $ (89,209,122) | $ (91,797,689) | $ (94,470,491) | $ (97,175,418) | $ (99,686,290) |
| Net Profit/Loss | $ (214,685,928) | $ (220,478,795) | $ (226,724,535) | $ (233,185,294) | $ (239,865,449) | $ (246,769,664) | $ (253,902,886) | $ (261,270,345) | $ (268,877,550) | $ (276,576,190) | $ (283,722,519) |
| **Book Deductions** | **$ 5,128,861** | **$ 4,744,197** | **$ 4,388,382** | **$ 4,059,253** | **$ 3,754,809** | **$ 3,473,199** | **$ 3,212,709** | **$ 2,971,756** | **$ 2,748,874** | **$ 2,334,467** | **$ 849,154** |
| *+ CAPEX #1 Depreciation* | $ 420,041 | $ 388,538 | $ 359,398 | $ 332,443 | $ 307,510 | $ 284,446 | $ 263,113 | $ 243,379 | $ 225,126 | $ - | $ - |
| *+ CAPEX #2 Depreciation* | $ 2,857,128 | $ 2,642,843 | $ 2,444,630 | $ 2,261,283 | $ 2,091,687 | $ 1,934,810 | $ 1,789,699 | $ 1,655,472 | $ 1,531,311 | $ 1,416,463 | $ - |
| *+ CAPEX #3 Depreciation* | $ 1,851,693 | $ 1,712,816 | $ 1,584,354 | $ 1,465,528 | $ 1,355,613 | $ 1,253,942 | $ 1,159,897 | $ 1,072,904 | $ 992,437 | $ 918,004 | $ 849,154 |
| *+ Depletion* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| Cash Flow | $ (209,557,066) | $ (215,734,599) | $ (222,336,153) | $ (229,126,041) | $ (236,110,639) | $ (243,296,465) | $ (250,690,177) | $ (258,298,589) | $ (266,128,676) | $ (274,241,723) | $ (282,873,365) |
| **Captial Cost** | **$ 5,449,083** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |
| *- Fixed Capital Costs* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *+ Financing Cash* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #1 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #2 Principle Payments* | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| *- Loan #3 Principle Payments* | $ 5,449,083 | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - | $ - |
| **Net Cash Flow** | $ (215,006,149) | $ (215,734,599) | $ (222,336,153) | $ (229,126,041) | $ (236,110,639) | $ (243,296,465) | $ (250,690,177) | $ (258,298,589) | $ (266,128,676) | $ (274,241,723) | $ (282,873,365) |
| Discounted Cash Flow | $ (68,507,585) | $ (62,490,629) | $ (58,548,058) | $ (54,850,952) | $ (51,384,554) | $ (48,134,909) | $ (45,088,832) | $ (42,233,883) | $ (39,558,332) | $ (37,058,441) | $ (34,749,854) |
| **Cumulative DCF** | $ (1,123,907,336) | $ (1,186,397,965) | $ (1,244,946,023) | $ (1,299,796,975) | $ (1,351,181,530) | $ (1,399,316,438) | $ (1,444,405,270) | $ (1,486,639,154) | $ (1,526,197,486) | $ (1,563,255,926) | $ (1,598,005,780) |

## Economic Assumptions

| *Capital Structure* | | | *Tax Structure* | | | *Operating Structure* | | | *Inflation* | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Expenditure Period | 3 | years | Tax Rate | 26 | % | Operating Period | 20 | years | Price Escalation | 3 | % |
| *Y1 Allocaiton* | 10 | % | Coal Depletion Rate | 10 | % | Discount Rate | 10 | % | Cost Escalation | 3 | % |
| *Y2 Allocaiton* | 60 | % | REE Depletion Rate | 14 | % | | | | | | |
| *Y3 Allocation* | 30 | % | Other Depletion Rate | 10 | % | | | | | | |
| Capital Escalation | 3.6 | % | Depreciation Rate | 150 | % | | | | | | |
| % Financed | 50 | % | Depreciation Term | 20 | years | | | | | | |
| Loan Interest Rate | 6 | % | Coal Royalty Rate | 4 | % | | | | | | |
| Loan Repayment Period | 10 | years | Coal Severance Tax | 4.5 | % | | | | | | |
| | | | REE Royalty Rate | 0 | % | | | | | | |
| | | | Other Royalty Rate | 0 | % | | | | | | |

## Output Metrics

| *Cost Analysis* | | *Eeconomic Indicators* | | |
|---|---|---|---|---|
| Overnight Capital Cost | $ 264,059,303 | Payback Period | 0 | 25.5 |
| Constant Dollar OPEX | $ 277,175,172 | Internal Rate of Return | 0 | N/A |
| Constant Dollar REE Rev | $ 77,520,156 | Net Present Value | 0 | $ (1,598,005,780) |
| Constant Dollar Coal Rev | $ - | Total Severence Tax Paid | 0 | $ - |
| Constant Dollar Other Rev | $ - | Total State and Federal T | 0 | $ - |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-4 | By: ks

## Module Information

Module ID: PM1
Module Name: Feed Preparation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Sorter Product Rate | 500 | tph |
| Sorter Feed Rate | 500 | tph |
| Sorter Waste Rate | 0 | tph |
| Sorter Ejected Tons | 0 | tph |
| Loader Capacity | 500 | tph/10 yd Unit |
| VSI Efficiency | 1.20 | HP/TPH |
| Roll Crusher Efficiency | 0.75 | HP/TPH |
| Screen Opening | 1.00 | Inch |
| Diesel Consumption | 6.30 | gal/hr/loader |
| Sorter Leasing Cost | $0.25 | $/Ejected Ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | 1 | 100% | 500 | TPH | $ 1,423,859 | $ 1,423,859 | $ - | 10 yd bucket; 1 unit per 500 TPH |
| 2 | VSI Crusher | 1 | 100% | 600 | HP | $ 378,738 | $ 378,738 | $ - | |
| 3 | Roll Crusher | 1 | 100% | 375 | HP | $ 257,455 | $ 257,455 | $ - | |
| 4 | Vibrating Screen | 2 | 100% | 108 | ft^2 | $ 106,248 | $ 212,496 | $ - | |
| 5 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Feed Conveyor |
| 6 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | Product Conveyors |
| 7 | Conveyor | 1 | 100% | - | TPH | $ - | $ - | $ - | Takeaway Conveyor |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 2,463,659** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Front End Loader (10yd) | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | VSI Crusher | 600.00 | Effiicency: | 0.85 | | | $ 287.61 | $ 862.84 | $ 289,913 | $ 2.20 | $ 0.07 |
| 3 | Roll Crusher | 375.00 | | | | | $ 179.76 | $ 539.27 | $ 181,196 | $ 1.37 | $ 0.04 |
| 4 | Vibrating Screen | 28.35 | | | | | $ 13.59 | $ 40.77 | $ 13,697 | $ 0.10 | $ 0.00 |
| 5 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.11 | $ 0.00 |
| 6 | Conveyor | 31.00 | | | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.11 | $ 0.00 |
| 7 | Conveyor | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **1065** | | | | | **$ 510.68** | **$ 1,532.03** | **$ 514,764** | **$ 3.90** | **$ 0.13** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Diesel | 6.30 | gal/hr | $ 2.000 | / gal | $ 100.80 | $ 302.40 | $ 101,606 | $ 0.77 | $ 0.03 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 100.80** | **$ 302.40** | **$ 101,606** | **$ 0.77** | **$ 0.03** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | X-Ray Sorter | $0.25 | /Ejected Ton | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 611** | **$ 1,834** | **$ 616,370** | **$ 4.67** | **$ 0.15** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-4 | By: ks

## Module Information

Module ID: PM2
Module Name: Roasting
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 500 | tph |
| External Heat | 10.0% | % |
| Product Rate | 480 | tph |
| Roasting Temperature | 575 | deg C |
| Bin Storage Capacity | 20% | % of hourly prod. |
| Material Heat Capacity | 0.80 | kg/kg-K |
| Reaction Enthalpy | 1783.00 | kJ/kg |
| % of Feed Reacted | 50% | % |
| Scurbber Airflow | 20.00 | cfm/tph |
| Quencher Aiflow | 20.00 | cfm/tph |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Storage Bins | 2 | 100% | 100 | Tons | $ 115,387 | $ 230,773 | $ - | |
| 2 | Conveyor | 1 | 100% | 500 | TPH | $ 95,556 | $ 95,556 | $ - | |
| 3 | Roaster | 1 | 100% | 581 | MBTU/hr | $ 18,994,975 | $ 18,994,975 | $ - | |
| 4 | Gas Scrubber | 1 | 100% | 10,000 | cfm | $ 57,958 | $ 57,958 | $ - | |
| 5 | Spray Chamber Quencher | 1 | 100% | 10,000 | cfm | $ 206,510 | $ 206,510 | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 19,585,772** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Storage Bins | 10.00 | Power Cost: | $ 0.07 | $/kWhr | | $ 4.79 | $ 14.38 | $ 4,832 | $ 0.04 | $ 0.00 |
| 2 | Conveyor | 31.00 | Effiicency: | 0.85 | | | $ 14.86 | $ 44.58 | $ 14,979 | $ 0.11 | $ 0.00 |
| 3 | Roaster | 87.16 | | | | | $ 41.78 | $ 125.34 | $ 42,113 | $ 0.32 | $ 0.01 |
| 4 | Gas Scrubber | 50.00 | | | | | $ 23.97 | $ 71.90 | $ 24,159 | $ 0.18 | $ 0.01 |
| 5 | Spray Chamber Quencher | 20.00 | | | | | $ 9.59 | $ 28.76 | $ 9,664 | $ 0.07 | $ 0.00 |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **198** | | | | | **$ 94.99** | **$ 284.96** | **$ 95,747** | **$ 0.73** | **$ 0.02** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Natural Gas | 56.03 | MCF/hr | $ 2.860 | / MCF | $ 1,281.99 | $ 3,845.96 | $ 1,292,244 | $ 9.79 | $ 0.32 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 1,281.99** | **$ 3,845.96** | **$ 1,292,244** | **$ 9.79** | **$ 0.32** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | |
|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 1,377** | **$ 4,131** | **$ 1,387,991** | **$ 10.51** | **$ 0.34** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-4 | By: ks

## Module Information

Module ID: PM3
Module Name: Leaching
Include? Yes
Number of Parallel PMs: 13.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 480 | tph |
| Residue Rate | 355 | tph |
| Slurry Flow | 8880 | gpm |
| Leaching Time | 300 | min |
| Acid Consumption | 0.00 | kg/ton |
| Bioleaching Sol'n Consumption | 4656.888651 | L/ton |
| Mixing Power Intensity | 1.50 | HP/1000 gal |
| Leach Tanks H/D | 1.00 | ft/ft |
| Surface Heat Loss | 230 | BTU/ft^2/hr |
| Wall Heat Loss | 15 | BTU/ft^2/hr |
| Number of Belt Filters | 2 | -- |
| Thickener Capacity | 3.69 | ft^2/tph |
| Polymer Dose | 0.028 | g/L |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 52 | 100% | 51,231 | gal | $ 179,830 | $ 9,351,134 | $ - | |
| 2 | Tank Mixer | 52 | 100% | 77 | HP | $ 185,415 | $ 9,641,604 | $ - | |
| 3 | Process Pump | 26 | 100% | 683 | gpm | $ 78,089 | $ 2,030,323 | $ - | |
| 4 | Conveyor | 1 | 100% | 480 | TPH | $ 93,438 | $ 93,438 | $ - | |
| 5 | Belt Filter | 2 | 100% | 177 | TPH | $ 1,136,742 | $ 2,273,485 | $ - | |
| 6 | Solution Heater | 1 | 100% | 5 | MBTU/hr | $ 181,582 | $ 181,582 | $ - | |
| 7 | Thickener | 13 | 100% | 101 | ft^2 | $ 17,046 | $ 221,593 | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 23,793,160** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 3,996.00 | Effiicency: | 0.85 | | | $ 1,915.50 | $ 5,746.49 | $ 1,930,820 | $ 14.63 | $ 0.48 |
| 3 | Process Pump | 444.00 | | | | | $ 212.83 | $ 638.50 | $ 214,536 | $ 1.63 | $ 0.05 |
| 4 | Conveyor | 29.76 | | | | | $ 14.27 | $ 42.80 | $ 14,380 | $ 0.11 | $ 0.00 |
| 5 | Belt Filter | 85.98 | | | | | $ 41.22 | $ 123.65 | $ 41,547 | $ 0.31 | $ 0.01 |
| 6 | Solution Heater | 501.76 | | | | | $ 240.52 | $ 721.56 | $ 242,445 | $ 1.84 | $ 0.06 |
| 7 | Thickener | 0.83 | | | | | $ 0.40 | $ 1.20 | $ 402 | $ 0.00 | $ 0.00 |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **5058** | | | | | **$ 2,424.73** | **$ 7,274.20** | **$ 2,444,130** | **$ 18.52** | **$ 0.61** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Bioleaching Solution | 2,235,306.55 | L/hr | $ 0.008 | / L | $ 135,906.64 | $ 407,719.92 | $ 136,993,892 | $ 1,037.80 | $ 33.98 |
| 2 | H2SO4 | - | mt/hr | $ 220.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Natural Gas | 4.84 | MCF/hr | $ 2.860 | / MCF | $ 110.71 | $ 332.12 | $ 111,593 | $ 0.85 | $ 0.03 |
| 4 | Polymer | 56.47 | kg/hr | $ 2.250 | / kg | $ 1,016.50 | $ 3,049.50 | $ 1,024,633 | $ 7.76 | $ 0.25 |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 137,033.85** | **$ 411,101.54** | **$ 138,130,117** | **$ 1,046.40** | **$ 34.26** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| | **Total Operating Cost:** | **$ 139,459** | **$ 418,376** | **$ 140,574,247** | **$ 1,064.92** | **$ 34.86** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 Date: 3/18/2023
Case: WK-4 By: ks

## Module Information

Module ID: PM4
Module Name: Al Precipitation
Include? Yes
Number of Parallel PMs: 11.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 7995 | gpm |
| Feed Rate | 30264 | lpm |
| Precip Mass Flow | 34.59 | tph |
| Precip Conditioning Time | 72 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 484.29 | ft^2/tpd |
| NaOH Dose | 6.65 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes: _____

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 11 | 100% | 52,330 | gal | $ 121,326 | $ 1,334,588 | $ - | |
| 2 | Tank Mixer | 11 | 100% | 26 | HP | $ 100,983 | $ 1,110,815 | $ - | |
| 3 | Process Pump | 33 | 100% | 727 | gpm | $ 39,980 | $ 1,319,331 | $ - | |
| 4 | Filter Press | 1 | 100% | 163 | ft^3 | $ 357,626 | $ 357,626 | $ - | |
| 5 | Conveyor | 1 | 100% | 35 | TPH | $ 22,043 | $ 22,043 | $ - | |
| 6 | Thickener | 11 | 100% | 1,523 | ft^2 | $ 150,644 | $ 1,657,083 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 5,801,484** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 287.81 | Effiicency: | 0.85 | | | $ 137.96 | $ 413.89 | $ 139,069 | $ 1.05 | $ 0.03 |
| 3 | Process Pump | 599.61 | | | | | $ 287.43 | $ 862.28 | $ 289,726 | $ 2.19 | $ 0.07 |
| 4 | Filter Press | 27.36 | | | | | $ 13.11 | $ 39.34 | $ 13,220 | $ 0.10 | $ 0.00 |
| 5 | Conveyor | 2.14 | | | | | $ 1.03 | $ 3.08 | $ 1,036 | $ 0.01 | $ 0.00 |
| 6 | Thickener | 10.65 | | | | | $ 5.11 | $ 15.32 | $ 5,147 | $ 0.04 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **928** | | | | | **$ 444.64** | **$ 1,333.92** | **$ 448,198** | **$ 3.40** | **$ 0.11** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 12.08 | mt/hr | $ 125.000 | / mt | $ 12,081.25 | $ 36,243.76 | $ 12,177,902 | $ 92.25 | $ 3.02 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 12,081.25** | **$ 36,243.76** | **$ 12,177,902** | **$ 92.25** | **$ 3.02** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 12,526** **$ 37,578** **$ 12,626,100** **$ 95.65** **$ 3.13**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-4 | By: ks

## Module Information

Module ID: PM5
Module Name: REE Precipitation
Include? Yes
Number of Parallel PMs: 12.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 7954 | gpm |
| Feed Rate | 30110 | lpm |
| Precip Mass Flow | 18.62 | tph |
| Precip Conditioning Time | 72 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 60.00 | min |
| Thickener Capacity | 5915.84 | ft^2/tpd |
| NaOH Dose | 3.00 | g/L of Feed |
| CaCO3 Dose | 0.00 | g/L of Feed |
| Coal Calcite Dose | 0.00 | g/L of Feed |
| Solid Density | 3.40 | g/cc |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 12 | 100% | 47,725 | gal | $ 115,201 | $ 1,382,417 | $ - | |
| 2 | Tank Mixer | 12 | 100% | 24 | HP | $ 95,871 | $ 1,150,455 | $ - | |
| 3 | Process Pump | 36 | 100% | 663 | gpm | $ 38,600 | $ 1,389,587 | $ - | |
| 4 | Filter Press | 1 | 100% | 176 | ft^3 | $ 377,299 | $ 377,299 | $ - | |
| 5 | Conveyor | 1 | 100% | 19 | TPH | $ 15,689 | $ 15,689 | $ - | |
| 6 | Thickener | 12 | 100% | 9,180 | ft^2 | $ 636,651 | $ 7,639,807 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 11,955,253** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | Power Cost: | $ 0.07 | $/kWhr | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 286.35 | Effiicency: | 0.85 | | $ 137.26 | $ 411.79 | $ 138,362 | $ 1.05 | $ 0.03 |
| 3 | Process Pump | 596.57 | | | | $ 285.97 | $ 857.90 | $ 288,254 | $ 2.18 | $ 0.07 |
| 4 | Filter Press | 29.47 | | | | $ 14.13 | $ 42.38 | $ 14,240 | $ 0.11 | $ 0.00 |
| 5 | Conveyor | 1.15 | | | | $ 0.55 | $ 1.66 | $ 558 | $ 0.00 | $ 0.00 |
| 6 | Thickener | 70.06 | | | | $ 33.58 | $ 100.75 | $ 33,851 | $ 0.26 | $ 0.01 |
| 7 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **984** | | | | **$ 471.49** | **$ 1,414.48** | **$ 475,265** | **$ 3.60** | **$ 0.12** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 5.42 | mt/hr | $ 125.000 | / mt | $ 5,415.93 | $ 16,247.80 | $ 5,459,260 | $ 41.36 | $ 1.35 |
| 2 | CaCO3 | - | mt/hr | $ 80.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 3 | Coal Calcite | - | mt/hr | $ 0.500 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 5,415.93** | **$ 16,247.80** | **$ 5,459,260** | **$ 41.36** | **$ 1.35** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 5,887** **$ 17,662** **$ 5,934,525** **$ 44.96** **$ 1.47**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-4 | By: ks

## Module Information

Module ID: PM6
Module Name: RE-Redissolution
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Solid Feed Rate | 18.62 | tph |
| Slurry Feed Rate | 3333 | gpm |
| Leaching Time | 120 | min |
| Mixing Power Intensity | 1.5 | HP/1000 gal |
| Acid Consumption | 422.65 | kg/ton |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 99,988 | gal | $ 115,856 | $ 463,425 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 150 | HP | $ 270,359 | $ 1,081,435 | $ - | |
| 3 | Process Pump | 1 | 100% | 3,333 | gpm | $ 142,932 | $ 142,932 | $ - | |
| 4 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 5 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 6 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,687,793** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 599.93 | Effiicency: | 0.85 | | | $ 287.58 | $ 862.73 | $ 289,878 | $ 2.20 | $ 0.07 |
| 3 | Process Pump | 83.32 | | | | | $ 39.94 | $ 119.82 | $ 40,261 | $ 0.30 | $ 0.01 |
| 4 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **683** | | | | | **$ 327.52** | **$ 982.55** | **$ 330,138** | **$ 2.50** | **$ 0.08** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HCl | 7.87 | mt/hr | $ 270.000 | / mt | $ 16,998.83 | $ 50,996.48 | $ 17,134,818 | $ 129.80 | $ 4.25 |
| 2 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 16,998.83** | **$ 50,996.48** | **$ 17,134,818** | **$ 129.80** | **$ 4.25** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 17,326** **$ 51,979** **$ 17,464,957** **$ 132.31** **$ 4.33**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-4 | By: ks

## Module Information

Module ID: PM7
Module Name: REE Oxalate Precipitation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Feed Rate | 3333 | gpm |
| Feed Rate | 12616 | lpm |
| Precip Mass Flow | 0.05 | tph |
| Precip Conditioning Time | 20 | min |
| Mixing Power Intensity | 0.50 | HP/1000 gal |
| Filter Press Cycle Time | 30.00 | min |
| Thickener Capacity | 0.00 | ft^2/tpd |
| NaOH Dose | 2.35 | g/L of Feed |
| Oxalic Acid Dose | 7.63 | g/L of Feed |
| Solid Density | 3.40 | g/cc |
| REO Production | 0.03 | tph |
| Temperature Change | 1000 | deg C |
| Heat Capacity | 0.80 | kJ/kg-K |
| Enthalpy | 1000 | kJ/kg |
| % Reacted | 100% | % |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 4 | 100% | 16,665 | gal | $ 21,149 | $ 84,598 | $ - | |
| 2 | Tank Mixer | 4 | 100% | 8 | HP | $ 52,962 | $ 211,849 | $ - | |
| 3 | Process Pump | 3 | 100% | 3,333 | gpm | $ 142,932 | $ 428,797 | $ - | |
| 4 | Thickener | - | 0% | - | ft^2 | $ - | $ - | $ - | |
| 5 | Filter Press | 1 | 100% | 0.25 | ft^3 | $ 3,350 | $ 3,350 | $ - | |
| 6 | Roaster | 1 | 100% | 0.08 | MBTU/hr | $ 268,661 | $ 268,661 | $ - | |
| 7 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 8 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 9 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 10 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 997,256** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 33.33 | Effiicency: | 0.85 | | | $ 15.98 | $ 47.93 | $ 16,104 | $ 0.12 | $ 0.00 |
| 3 | Process Pump | 249.97 | | | | | $ 119.82 | $ 359.47 | $ 120,782 | $ 0.91 | $ 0.03 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 0.04 | | | | | $ 0.02 | $ 0.06 | $ 20 | $ 0.00 | $ 0.00 |
| 6 | Roaster | 0.01 | | | | | $ 0.01 | $ 0.02 | $ 6 | $ 0.00 | $ 0.00 |
| 7 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **283** | | | | | **$ 135.83** | **$ 407.48** | **$ 136,913** | **$ 1.04** | **$ 0.03** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | NaOH | 1.78 | mt/hr | $ 125.000 | / mt | $ 1,775.99 | $ 5,327.97 | $ 1,790,197 | $ 13.56 | $ 0.44 |
| 2 | Oxalic Acid | 5,778.19 | kg/hr | $ 0.600 | / kg | $ 27,735.30 | $ 83,205.90 | $ 27,957,184 | $ 211.79 | $ 6.93 |
| 3 | Natural Gas | 0.08 | MCF/hr | $ 2.860 | / MCF | $ 1.80 | $ 5.40 | $ 1,813 | $ 0.01 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 29,513.09** | **$ 88,539.27** | **$ 29,749,194** | **$ 225.36** | **$ 7.38** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Total Operating Cost:** | **$ 29,649** | **$ 88,947** | **$ 29,886,107** | **$ 226.40** | **$ 7.41** |

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-4 | By: ks

## Module Information

Module ID: PM8
Module Name: Sc SX Separation
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | |
|---|---|---|
| Mixing Time | 1.0 | min |
| SX Mixing Intensity | 2.5 | HP/1000 gal |
| Settling Time | 15.0 | min |
| Number of Loading Stages | 2.0 | -- |
| Number of Scrubbing Stages | 0.0 | -- |
| Number of Stripping Stages | 2.0 | -- |
| PLS Flow | 7995 | gpm |
| Raffinate Rate: | 7995 | gpm |
| Extraction Feed (O+A) | 15990 | gpm |
| Scrub Feed (O+A) | 220 | gpm |
| Strip Feed (O+A) | 440 | gpm |
| Scrup Aq Product | 0 | gpm |
| Strip Aq Product | 18 | gpm |
| Organic Advance | 220 | gpm |
| Product Rate | 0.00 | tph |

| | | |
|---|---|---|
| Stripping Acid Molarity | 2.00 | mol/L |
| Scrubbing Acid Molarity | 0.00 | mol/L |
| Neutralizaiton : Strip Ratio | 0.00 | mol Base/mol Acid |
| Stripping Acid Molar Mass | 98.00 | g/mol |
| Scrubbing Acid Molar Mass | 0.00 | g/mol |
| Neutralization Molar Mass | 0.00 | g/mol |
| Strip Acid Stock Conc | 0.37 | % |
| Scrub Acid Stock Conc | 0.93 | % |
| Neutralization Stock Solution | 0.50 | % |
| Diluent Density | 0.84 | g/cc |
| Extractant Density | 0.92 | g/cc |
| Extractant Concentration | 0.05 | % Volume |
| Organic Entrainment (loss) | 60 | mg/L |
| Total Organic Volume | 262869 | gal |
| Precip Mixing Intensity | 0.50 | HP/1000 gal |
| Precip Conditioning Time | 600.00 | min |
| Precip Thickener Capacity | 0.00 | ft^2/tph |
| Precip Solid Density | 3.40 | g/cc |
| Filter Press Cycle Time | 30.00 | min |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | 2 | 100% | 15,990 | gal | $ 93,424 | $ 186,849 | $ - | Extraction Mixing |
| 2 | Tank Mixer | 2 | 100% | 40 | HP | $ 128,250 | $ 256,501 | $ - | Extraction Mixing |
| 3 | Steel Tanks | 2 | 100% | 239,845 | gal | $ 428,444 | $ 856,888 | $ - | Extraction Settling |
| 4 | Process Pump | 2 | 100% | 7,995 | gpm | $ 199,552 | $ 399,105 | $ - | PLS and Raffinate |
| 5 | Steel Tanks | - | 100% | 220 | gal | $ 8,383 | $ - | $ - | Scrubbing Mixing |
| 6 | Tank Mixer | - | 100% | 1 | HP | $ 11,429 | $ - | $ - | Scrubbing Mixing |
| 7 | Steel Tanks | - | 100% | 3,297 | gal | $ 38,443 | $ - | $ - | Scrubbing Settling |
| 8 | Process Pump | 1 | 100% | - | gpm | $ - | $ - | $ - | Scrubber Product |
| 9 | Steel Tanks | 2 | 100% | 440 | gal | $ 12,379 | $ 24,758 | $ - | Stripping Mixing |
| 10 | Tank Mixer | 2 | 100% | 1 | HP | $ 16,896 | $ 33,792 | $ - | Stripping Mixing |
| 11 | Steel Tanks | 2 | 100% | 6,594 | gal | $ 56,770 | $ 113,540 | $ - | Stripping Settling |
| 12 | Process Pump | 1 | 100% | 18 | gpm | $ 19,317 | $ 19,317 | $ - | Stripping Product |
| 13 | Cyanex 272 | 1 | 125% | 57,216 | kg | $ 3,582,273 | $ - | $ 3,582,273 | Initial Fill |
| 14 | Kerosene Diluent | 1 | 125% | 992,579 | kg | $ 1,074,811 | $ - | $ 1,074,811 | Initial Fill |
| 15 | Polyethylene Tank | 4 | 100% | 2,630 | gal | $ 5,499 | $ 21,997 | $ - | Precipitation |
| 16 | Tank Mixer | 4 | 100% | 1.32 | HP | $ 18,696 | $ 74,782 | $ - | |
| 17 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | |
| 18 | Process Pump | 2 | 100% | 18 | gpm | $ 19,317 | $ 38,634 | $ - | |
| 19 | Filter Press | 1 | 100% | 0.01 | ft^3 | $ 272 | $ 272 | $ - | |
| 20 | Process Pump | 2 | 100% | 220 | gpm | $ 50,673 | $ 101,347 | $ - | Organic Recycle Pump |
| | **TOTAL:** | | | | | | **$ 2,127,781** | **$ 4,657,083** | |

## Power/Utilities

Power Cost: $ 0.07 $/kWhr
Effiicency: 0.85

| ID | Unit Operation | Total HP | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|
| 1 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 79.95 | $ 38.32 | $ 114.97 | $ 38,630 | $ 0.29 | $ 0.01 |
| 3 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 4 | Process Pump | 399.74 | $ 191.62 | $ 574.85 | $ 193,151 | $ 1.46 | $ 0.05 |
| 5 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 6 | Tank Mixer | - | $ - | $ - | $ - | $ - | $ - |
| 7 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 8 | Process Pump | - | $ - | $ - | $ - | $ - | $ - |
| 9 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 10 | Tank Mixer | 2.20 | $ 1.05 | $ 3.16 | $ 1,062 | $ 0.01 | $ 0.00 |
| 11 | Steel Tanks | - | $ - | $ - | $ - | $ - | $ - |
| 12 | Process Pump | 0.44 | $ 0.21 | $ 0.63 | $ 212 | $ 0.00 | $ 0.00 |
| 13 | Cyanex 272 | - | $ - | $ - | $ - | $ - | $ - |
| 14 | Kerosene Diluent | - | $ - | $ - | $ - | $ - | $ - |
| 15 | Polyethylene Tank | - | $ - | $ - | $ - | $ - | $ - |
| 16 | Tank Mixer | 5.26 | $ 2.52 | $ 7.56 | $ 2,542 | $ 0.02 | $ 0.00 |
| 17 | Thickener | - | $ - | $ - | $ - | $ - | $ - |
| 18 | Process Pump | 0.88 | $ 0.42 | $ 1.26 | $ 424 | $ 0.00 | $ 0.00 |
| 19 | Filter Press | 0.00 | $ 0.00 | $ 0.00 | $ 1 | $ 0.00 | $ 0.00 |
| 20 | Process Pump | 10.99 | $ 5.27 | $ 15.80 | $ 5,310 | $ 0.04 | $ 0.00 |
| | **TOTAL:** | **499** | **$ 239.42** | **$ 718.25** | **$ 241,331** | **$ 1.83** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Kerosene Diluent | 103.50 | kg/hr | $ 0.940 | / kg | $ 778.34 | $ 2,335.01 | $ 784,563 | $ 5.94 | $ 0.19 |
| 2 | D2EHPA | 5.45 | kg/hr | $ 5.000 | / kg | $ 217.90 | $ 653.70 | $ 219,642 | $ 1.66 | $ 0.05 |
| 3 | HCl | - | mt/hr | $ 270.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 4 | HCl | 0.84 | mt/hr | $ 270.000 | / mt | $ 1,812.88 | $ 5,438.65 | $ 1,827,387 | $ 13.84 | $ 0.45 |
| 5 | NaOH | - | mt/hr | $ 125.000 | / mt | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 2,809.12** | **$ 8,427.36** | **$ 2,831,592** | **$ 21.45** | **$ 0.70** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 3,049** **$ 9,146** **$ 3,072,923** **$ 23.28** **$ 0.76**

# PROCESS MODULE COST ANALYSIS WORKSHEET

Feedstock: WK No. 13 | Date: 3/18/2023
Case: WK-4 | By: ks

## Module Information

Module ID: PM11
Module Name: CM Recovery
Include? Yes
Number of Parallel PMs: 1.00

**Design Factors**

| | | | | | |
|---|---|---|---|---|---|
| Co Feed Rate | 3332.87 | gpm | Co Solid Density | 3.40 | g/cc |
| Co Feed Rate | 12616.27 | lpm | Mn Solid Density | 3.40 | g/cc |
| Mn Feed Rate | 3331.23 | gpm | HCL Dose | 21.59 | g/L of Feed |
| Mn Feed Rate | 12610.07 | lpm | | | |
| Co Precip Mass Flow | 1.39 | tph | | | |
| Mn Precip Mass Flow | 1.44 | tph | | | |
| Mixing Power Intensity | 1.50 | HP/1000 gal | | | |
| Co Precip Conditioning Time | 30.00 | min | | | |
| Mn Precip Conditioning Time | 30.00 | min | | | |
| Co Filter Press Cycle Time | 30.00 | min | | | |
| Mn Filter Press Cycle Time | 30.00 | min | | | |
| Co Thickener Capacity | 0.00 | ft^2/tph | | | |
| Mn Thickener Capacity | 0.00 | ft^2/tph | | | |
| Na2S Dose | 6.70 | g/L of Feed | | | |
| NaOH Dose | 4.08 | g/L of Feed | | | |

Other Notes:

## Capital Cost

| ID | Unit Operation | No. of Units | Design Capacity | Installed Design | Unit | Unit Cost | Total Equipment Cost | Total Reagent Cost | Notes/Description |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | 2 | 100% | 49,993 | gal | $ 60,003 | $ 120,006 | $ - | Co Conditioning |
| 2 | Tank Mixer | 2 | 100% | 75 | HP | $ 182,875 | $ 365,751 | $ - | Co Conditioning |
| 3 | Process Pump | 1 | 100% | 3,333 | gpm | $ 142,931 | $ 142,931 | $ - | |
| 4 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Co Thickener |
| 5 | Filter Press | 1 | 100% | 6.57 | ft^3 | $ 35,445 | $ 35,445 | $ - | Co Dewaterning |
| 6 | Polyethylene Tank | 2 | 100% | 49,968 | gal | $ 59,975 | $ 119,950 | $ - | Mn Conditioning |
| 7 | Tank Mixer | 2 | 100% | 75 | HP | $ 182,825 | $ 365,649 | $ - | Mn Conditioning |
| 8 | Process Pump | 1 | 100% | 3,331 | gpm | $ 142,905 | $ 142,905 | $ - | |
| 9 | Thickener | 1 | 100% | - | ft^2 | $ - | $ - | $ - | Mn Thickener |
| 10 | Filter Press | 1 | 100% | 7 | ft^3 | $ 36,356 | $ 36,356 | $ - | Mn Dewatering |
| 11 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 12 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 13 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 14 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 15 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 16 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 17 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 18 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 19 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| 20 | Reserved | - | 0% | - | | $ - | $ - | $ - | |
| | **TOTAL:** | | | | | | **$ 1,328,994** | **$ -** | |

## Power/Utilities

| ID | Unit Operation | Total HP | | | | | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Polyethylene Tank | - | Power Cost: | $ 0.07 | $/kWhr | | $ - | $ - | $ - | $ - | $ - |
| 2 | Tank Mixer | 149.98 | Effiicency: | 0.85 | | | $ 71.89 | $ 215.68 | $ 72,468 | $ 0.55 | $ 0.02 |
| 3 | Process Pump | 83.32 | | | | | $ 39.94 | $ 119.82 | $ 40,260 | $ 0.30 | $ 0.01 |
| 4 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Filter Press | 1.10 | | | | | $ 0.53 | $ 1.59 | $ 533 | $ 0.00 | $ 0.00 |
| 6 | Polyethylene Tank | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Tank Mixer | 149.91 | | | | | $ 71.86 | $ 215.57 | $ 72,432 | $ 0.55 | $ 0.02 |
| 8 | Process Pump | 83.28 | | | | | $ 39.92 | $ 119.76 | $ 40,240 | $ 0.30 | $ 0.01 |
| 9 | Thickener | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Filter Press | 1.14 | | | | | $ 0.55 | $ 1.64 | $ 552 | $ 0.00 | $ 0.00 |
| 11 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 12 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 13 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 14 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 15 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 16 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 17 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 18 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 19 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| 20 | Reserved | - | | | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | **469** | | | | | **$ 224.69** | **$ 674.07** | **$ 226,486** | **$ 1.72** | **$ 0.06** |

## Consumables/Reagents

| ID | Item | Consumption | Unit | Unit Cost | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | Na2S | 5.07 | mt/hr | $ 360.000 | / mt | $ 14,606.73 | $ 43,820.18 | $ 14,723,582 | $ 111.54 | $ 3.65 |
| 2 | NaOH | 3.09 | mt/hr | $ 125.000 | / mt | $ 3,085.60 | $ 9,256.81 | $ 3,110,287 | $ 23.56 | $ 0.77 |
| 3 | HCl | 0.00 | mt/hr | $ 270.000 | / mt | $ 3.90 | $ 11.71 | $ 3,934 | $ 0.03 | $ 0.00 |
| 4 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | - | | $ - | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | | | **$ 17,696.23** | **$ 53,088.70** | **$ 17,837,803** | **$ 135.13** | **$ 4.42** |

## Lease/Other

| ID | Item | Price | Unit | Shift Cost | Daily Cost | Annual Cost | Per kg Cost | Per Ton Cost |
|---|---|---|---|---|---|---|---|---|
| 1 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 2 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 3 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 4 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 5 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 6 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 7 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 8 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 9 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| 10 | Reserved | | | $ - | $ - | $ - | $ - | $ - |
| | **TOTAL:** | | | **$ -** | **$ -** | **$ -** | **$ -** | **$ -** |

**Total Operating Cost:** **$ 17,921** **$ 53,763** **$ 18,064,289** **$ 136.85** **$ 4.48**