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Management of Dry Flue Gas Desulfurization By-Products in Underground Mines

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FOREWORD

The United States currently produces about 80 million tons of coal combustion by-products (CCB's) annually, almost all from coal-burning electric power plants. This includes fly ash, bottom ash, and wet and dry scrubber sludge. The production of fluidized bed combustion by-products (FBC's) is not included in this estimate. In spite of significant research and development efforts over the past two decades to find uses for CCB's, the current utilization of fly ash and bottom ash in the United States is only about 25% and 40%, respectively. Illinois falls far behind the national average, utilizing only about 10% of the fly ash produced in the state. The utilization of FBC by-products and flue gas desulfurization (FGD) by-products in the United States is only 2% to 3%. Transportation cost has been cited as a strong deterrent to more utilization of CCB's and FGD by-products. Therefore, large volume beneficial use of these materials must be emphasized to assist the coal and electric utility industries.

Large volume management of CCB's and FGD by-products in underground mines to control subsidence and acid mine drainage has significant possibilities, particularly in Illinois where high sulfur coals at shallow mining depths are exploited and the protection of prime agriculture lands and groundwater resources is crucial. Furthermore, the hydrogeological characteristics surrounding the Illinois coal seams being actively mined are favorable for large volume management of CCB's and FGD by-products.

This cooperative research agreement between the U.S. Department of Energy and Southern Illinois University at Carbondale, whose foremost purpose is development of technologies to manage CCB's and FGD by-products in underground coal mines, is highly important to the state of Illinois. Successful development and demonstration of materials handling and underground placement technologies will help maintain the Illinois coal industry's competitiveness with western coal industries, particularly with Powder River basin coal. Furthermore, mixtures of these coal combustion by-products can be developed which will provide subsidence control. Based on these results, the SIUC investigators and industrial cooperators are planning to develop mining systems with backfill which will permit higher productivity and lower costs. These developments should be significant for the entire coal industry in the United States.

The authors and cooperating organizations are quite enthusiastic about the cooperative agreement and the results that should be obtained under it, and because of the potential of the results to significantly enhance the Illinois coal and electric power industries state and industry representatives are closely watching the developments.

Yoginder P. Chugh
Program Director

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CHAPTER 1

INTRODUCTION

1.0 INTRODUCTION

1.1 Overview

On September 30, 1993, the U.S. Department of Energy - MORGANTOWN Energy Technology Center (DOE-METC) and Southern Illinois University at Carbondale (SIUC) entered into a cooperative research agreement entitled "Management of Dry Flue Gas Desulfurization By-Products in Underground Mines" (DE-FC21-93MC-30252). Under the agreement Southern Illinois University at Carbondale will develop and demonstrate technologies for the handling, transport, and placement in abandoned underground coal mines of dry flue gas desulfurization (FGD) by-products, such as fly ash, scrubber sludge, fluidized bed combustion (FBC) by-products, and will assess the environmental impact of such underground placement. The term "Management of FGD By-Products" implies the placement of by-products material underground for disposal and/or beneficial use.

The overall program set forth in the cooperative agreement is for a period of four (4) years, beginning September 30, 1993. It is divided into three (3) phases as follows: Phase I -- 30 Months, beginning September 30, 1993; Phase II -- 6 Months, beginning April 1, 1996; and Phase III – twelve Months, beginning January 1, 1997. However, two no-cost extensions were granted by the Department of Energy, and thus Phase I of the program ended March 31, 1996 and Phase II was extended until April 30, 1997.

A final Technical Progress Report for Phase I of the program was submitted to the Department of Energy in June, 1996. This report represents the Final Technical Progress Report for Phase II of the overall program.

1.2 Background And Statement Of Work

The Final Technical Progress Report for Phase I of the program contained a comprehensive discussion of the problems concerning the management of coal combustion by-products. Such discussion will not be repeated here. It is sufficient to say that as the management of CCB's near power plants becomes more costly, and as land for such disposal becomes more scarce the underground management of the CCB's becomes more and more of a favorable option.

Phase I of the overall program was primarily concerned with the physical and chemical characterization of CCB's and mixtures of various CCB's, and with the identification of technologies for the handling, transportation, and underground placement of these materials. Also, during Phase I a thorough geologic and hydrologic study of the underground placement demonstration area was conducted, and long-term leaching tests were undertaken to assist in predicting how the material will behave when placed underground.

Phase II of the program, while continuing some of the work begun in Phase I, is primarily concerned with developing and testing the hardware for the actual underground placement demonstrations. Two separate technologies have been identified and hardware procured for full-scale demonstrations: (1), hydraulic placement, where CCB's will be placed underground as a "paste" containing about 70-75 percent solids; and (2) pneumatic placement, where CCB's will be placed underground as a relatively dry material using compressed air.

There are sound geologic reasons for these technologies. In most coal mines in the Illinois basin, the floor strata are composed of claystones, which, when wet, become soft and weak. This leads to pillar settling, which in turn leads to surface subsidence with damage to surface lands and possibly to surface structures. The technologies identified here use little water, thus limiting the weakening of the mine floor and alleviating surface subsidence. Further, by filling the mine voids, the available space for movement is reduced, which again should help alleviate surface subsidence. These materials upon placement should set up as a solid material with very low permeability. This should reduce leaching potential of CCBs and associated groundwater contamination potential. Thus the two technologies appear to be environmentally sound as well as offering economic advantages.

1.3 Phase II Objectives

The principal objective of Phase II of the program was to construct or acquire the necessary hardware for both the hydraulic and pneumatic underground placements, to erect such hardware at the demonstration site at the Peabody Coal Company Mine No. 10 near Pawnee, Illinois, and to test and "de-bug" the equipment and develop the equipment operational parameters. The required equipment and components to construct the underground placement systems shall be procured and assembled into a functioning system. A surface demonstration shall be conducted on both the pneumatic and hydraulic placement systems prior to conducting the underground demonstrations. The surface demonstration shall be designed to simulate a full-scale underground placement methods by injecting the fill materials into a large-scale simulated mine opening constructed from plywood. Cross sections shall be constructed in the surface structure to determine the capacity of the fill techniques tested to fill these areas. During the course of Phase II tasks it was determined that surface demonstration of technologies in simulated openings may not provide a good test environment. Therefore, it was decided to perform eqipment performance testing in an underground panel adjacent to Phase III final demonstration panels. For the pneumatic placement, the entire hardware package was designed in-house and fabricated at the University's Carterville facility. After it was completely fabricated, it was transported via flat-bed trailer trucks to the demonstration site.

Preliminary tests at the demonstration site, using newly drilled boreholes revealed some flaws in the operation of both the hydraulic and pneumatic hardware. For example, a new electric motor to drive the hydraulic mixing plant had to be obtained when the original motor failed (and subsequently was shown to be beyond repair). Likewise, the hydraulic motors on the pneumatic hardware proved to be under-powered, and had to be supplemented with a power pack. It was to discover things like this, of course, that was the purpose of the "de-bugging" tests.

By the end of Phase II, however, most of the "de-bugging" was complete, and both the hydraulic and pneumatic placement systems had been tested and shown to be in good working order. Each system had placed several hundred tons of CCB's under-ground, and preparations were underway for the full-scale Phase III demonstrations.

CHAPTER 2

PRELIMINARY PLANNING

2.0 PRELIMINARY PLANNING

2.1 Introduction

As has been previously indicated, the primary objective of Phase II was to assemble, test, and "de-bug" the equipment necessary for the Phase III demonstrations of the hydraulic and pneumatic underground placement technologies. Originally it was planned to conduct the preliminary equipment testing and "de-bugging" on the surface, using covered trenches. However, it became evident that surface testing would net be feasible. The trenches would be unable to withstand the pressures generated in both the hydraulic and pneumatic placement tests. Therefore, with the approval of the U. S. Department of Energy, it was decided to conduct the preliminary testing by using boreholes drilled into the underground mine voids.

2.2 Equipment Acquisition and Assembly

Early in Phase II a search was instituted to locate the equipment necessary to mix the fly ash, scrubber sludge, and water necessary for the hydraulic placement. A mixing plant was located and purchased from Montgomery County, Illinois where it had been used for several years to mix CCB material from the Cofeen Power Plant for use as road repair material. Also, a heavy duty concrete pump was located, which basically completed the equipment necessary for the hydraulic placement. Discussions with officials of the Springfield City, Water, Light and Power Company (CWLP) confirmed a supply of CCB material for both the preliminary and the full-scale Phase III demonstration.

Searches showed that equipment necessary for the pneumatic underground placement was unavailable; therefore, it was necessary to design and fabricate the equipment "from scratch". Early on preliminary drawings of the equipment were made, and shops throughout Southern Illinois were contacted to determine their capabilities. Shop drawings were prepared and, with the assistance of the Purchasing Department of SIUC, contractors were selected to fabricate the various component parts of the pneumatic equipment. The components were delivered to the SIUC facility at Carterville, Illinois, where they were assembled into the complete pneumatic placement equipment. In March, 1997, the pneumatic equipment was transported to the demonstration site.

2.3 Preliminary Planning Results

The planning for the new boreholes for the Phase II tests was undertaken during the first quarter of Phase II. Specifications for the boreholes were developed jointly by SIUC program team and Peabody Coal Company, and Peabody was responsible for having the wells drilled. Actual drilling was delayed for about a month while Peabody obtained bids for several drilling contractors. However, this delay did not affect the overall Phase II testing program, as work continued on the development of the hydraulic and pneumatic underground placement equipment. Likewise, the search for mixing equipment for the hydraulic underground placement resulted in significant savings over developing and fabricating the equipment either within the University or by outside contractors. Planning in detail for the pneumatic underground placement equipment made it much easier to select competent contractors for the fabrication of components and for the assembly of the equipment at Carterville. As originally planned, the equipment was fabricated to be transported on two flat-bed over-the-highway trailers, and this mode of transportation was used to move the equipment from Carterville to the demonstration site.

In summary, the preliminary planning for the Phase II equipment tests and the Phase III full-scale demonstrations resulted in a more efficient and more effective overall operation.

CHAPTER 3

ENVIRONMENTAL STUDIES

3.0 ENVIROMENTAL STUDIES

3.1 Introduction

Dry FGD by-products are to be injected into underground mine workings as a means of controlling subsidence and providing a large volume disposal alternative to surface landfilling. To ensure that such practices do not harm the environment, a team of researchers is investigating the demonstration site hydrogeology and monitoring for groundwater quality. The team reporting in this section is determining the nature of the leachates produced by the residues both in the short term and over time.

Based on interaction between the environmental and physical properties characterization teams, two mixes have been selected for placement. A mix selected for pneumatic placement will consist of 80% FBC fly ash and 20% bottom ash treated with 30% moisture during injection. A paste backfill mix consisting of 55% force oxidized scrubber sludge, and 45% F-type fly ash, with a small amount of lime and sufficient water to make a paste with 70-75 percent solids.

Work completed to date and used as a basis for analysis includes the following tests:

1) TCLP tests on all mix components and proposed mixes.
2) ASTM column tests on mix components and mixes.
3) SLP tests on preliminary mixes.
4) SEM and EDX determinations of component mineralogy and composition.
5) XRF determination of oxide phases
6) Acid Digestion and ICP analysis of composition
7) CCE and paste pH determinations on all mixes and mix components.
8) Initial results of the rapid aging tests.

Results of this work has been previously reported, particularly in the Phase I final report.

3.2 Material Characteristics Influencing Environmental Risk

The potential for flue gas desulfurization byproducts to introduce contaminants into the groundwater depends on several factors relating to the material itself. These factors include

1- The bulk composition of the material to be placed

Only elements present in the material can be leached into the groundwater and the total quantity of a contaminant that can ultimately be released into the environment cannot exceed the quantity available in the FGD materials.

2- The mineralogical structure of the material

Mineralogical structure of FGD byproducts changes over time with weathering and break-down. Not all minerals are subject to break-down and trace element release in all environments. The mineralogical structure not only influences what may ultimately be released but the rate and time period over which the release is likely to take place. In addition to site specific hydrogeological factors, the extent of groundwater contamination plumes is determined by leachate volume, leachate concentration, and persistence over time, all of which are highly influenced by material characteristics.

3- Permeability of the material

Permeability of the FGD byproduct mixes will also change with time as various minerals form, dissolve, and swell. The permeability or hydraulic conductivity influence the volume of leachate that may be released into the environment for a given hydraulic gradient.

4- Amenability to alkaline or acid side leaching conditions

Many of trace contaminants of greatest concern are more mobile at extremes of pH, particularly on the acid side. Whether or not the natural pH of material leachate is alkaline or acid can influence what elements could be leached out. Similarly the acid neutralizing capacity of FGD byproducts may influence the material response to an acidic environment.

Fly ash contains several heavy trace metals including zinc, copper and nickel. As with iron, fairly high concentrations must be present in the water to be of much concern. Chromium is present and is considered a toxic metal, but unless it is present in the hexavalent form, fairly high concentrations are acceptable in the water. Vanadium is not even part of normal class I and class II groundwater standards. Most other trace metals are most conspicuous by their absence.

The most remarkable thing about the synthetic gypsum produced by the scrubber is it's remarkable purity. Even comparisons of the calcium and sulfur stiochiometries indicate almost complete reaction of calcium with sulfate. The power plant uses a washing procedure to eliminate salts that could interfere with wallboard manufacture, and the composition analysis indicates that the process has been effective in eliminating almost all of the salts. The small amounts of alumino-silicate present probably came from the reagent limestone, which itself appears to be rather pure. The alumino silicates appear to have digested well, although the boiled down open beaker solution may have become saturated with calcium.

One question that may arise about the trace element composition of a material as seemingly pure as the force oxidized scrubber sludge is "how can elements not found in the composition be found in the leachates produced by shake or column tests"? It must be remembered that non-detection in a composition test indicates that the concentration was below the detection limit, not that the element was totally absent. In acid digestion, there must be far most acid and liquid than solid because the extract solutions should not become saturated even for major elements. This necessity can cause dilution of trace elements to where the ICP can no longer detect them. In the digestions used in this study, the dilution ranged between 250 and 500 times, and most trace elements must be present in the solid in the range of around 5 ppm or more to be detected. By contrast, shake tests use a 20 to 1 liquid solid ratio, and column tests may have 100 times more solid than liquid. If the trace elements can be mobilized easily, most of the element can be taken into the initial solutions with very little dilution and can be detected and measured. It must be remembered that once the trace metals are depleted by the initial solutions they are gone. The ASTM column tests illustrate the remarkable falls in concentration as trace elements rapidly become depleted.

3.3 Hydraulic Conductivity

In assessing environmental impacts from leachate emanating from the FGD residues fill an important consideration is the relative amount of leachate to be released. A leachate can have very hazardous characteristics and, if released in a very small quantity into a much higher volume groundwater system, may have no impact because of dilution.

Pressure and leachate volumes taken in both the ASTM column and Rapid Age test procedures in this program allow estimates of hydraulic conductivity to be made by the constant head method. In assessing hydraulic conductivity it should be understood that FGD materials are often soluble or composed of high temperature phases that may not be in equilibrium with a low temperature environment below the water table. The tendency for pores to open or close and minerals to reform means that initial hydraulic conductivity may not correspond well to long term hydraulic conductivity. ASTM columns were run only 16 days and thus unless reactions proceed very fast the indications given by this data may be representative of only initial hydraulic conductivity or short trends. All of the FGD materials in this program are fine grained and when tightly packed give low hydraulic conductivities. Estimated initial hydraulic conductivities are given in Table 3.1.

Table 3.1
Initial Hydraulic Conductivities of FGD Materials
and Additive Fly Ashes

Material	Hydraulic Conductivity (cm/sec)
Force Oxidized Scrubber Sludge	4.5 X 10^-5
Venturi Scrubber Sludge	4 X 10^-5
PCC Fly Ash	6.5 X 10^-6
FBC Fly Ash	2 X 10^-5
FBC Spent Bed	1.5 X 10^-5

One of the factors that influences initial permeability is the degree of compaction. Not all the materials listed in the table above were packed in the same way. The official ASTM procedure calls for the columns to be packed to the optimum moisture and density as determined by the Proctor method. Proctor tests were performed and PCC fly ash and Force Oxidized Scrubber Sludge were both packed at the optimum moisture and density as specified by ASTM. The FBC fly ash and spent-bed ash were not amenable to the ASTM procedure. Initial hydration reactions for FBC fly ash consume approximately 28% by weight of water as a result of incorporation in hydrated lime and vaporization from the heat of reaction. Unless the FBC ash is prehydrated, a Proctor Test cannot be run because of rapid changes in moisture. The same is true of FBC spent-bed ash. When packed into a column FBC fly ash cements rapidly. ASTM columns are specified not to operate above 40 psi, this will not force one pore volume per day through the column as required by the test if the FBC fly ash is packed. FBC fly ash and spent-bed ash both expand significantly on initial contact with water causing them to crack (sometimes explosively) any closed vessel in which they are placed. As a result of these problems FBC fly ash and spent-bed ash were both pre-hydrated for several hours to reduce expansion problems and avoid very high temperatures. Afterwards the material was poured into the column in 2 inch lifts and packed around the edges to avoid channeling. The venturi scrubber sludge was also made of lime, however, this lime had been prehydrated and had far less sulfate so the heating and expansiveness problems found for FBC fly ash did not have to be dealt with. The venturi scrubber sludge was poured into the ASTM column with 46% as received moisture and packed around the edges only, again using 2 inch lifts. In examining the above hydraulic conductivity figures it should be understood that PCC fly-ash and Force Oxidized Scrubber sludge hydraulic conductivities are on a packed material basis, while the other hydraulic conductivities represent loose poured values. The tendency for FGD byproducts to give very low hydraulic conductivity even without compactive effort is clear.

Although the ASTM test duration is short, the procedure can give an indication of trends in hydraulic conductivity. Both venturi scrubber sludge and force oxidized scrubber sludge are soluble enough that pore space is opened even in short term leaching and the hydraulic conductivity of both materials more than doubles in 16 days to the low 10^-4 cm/sec range. PCC fly ash by itself is relatively unreactive in the short term and shows no change in hydraulic conductivity. FBC fly ash rapidly experiences expansive sulfo-pozzolanic reactions that cause hydraulic conductivity to fall a full order of magnitude to the low 10^-6 cm/sec range. FBC spent-bed ash permeability increased to the low 10^-5 cm/sec range at first but by the end of the test had returned to the original value. Based on composition, one would expect spent bed to follow the same trend as FBC fly ash except for some response to the initially coarser size.

In estimating hydraulic conductivity of the pneumatic and hydraulic mixes to be placed in the field the question of how to estimate compaction as well as reactions between mix components should be determined. It was decided that in Rapid Age and ASTM column tests of the mixes that no compactive effort would be assumed for the mixes, giving the worst case scenario for hydraulic conductivity. The cement chemistry for FBC fly ash and spent bed both rely on sulfo-pozzolanic reactions and formation of expansive minerals such as ettringite. Not surprisingly the ASTM column on the pneumatic mix gave roughly the same behavior as the individual materials starting in the 10^-5 cm/sec range and falling an order of magnitude to 10^-6 cm/sec.

As would be anticipated when materials are simply poured into place, there is considerable non-uniformity in compaction and initial hydraulic conductivity in replicate experiments ranged from 1.24 X 10^-4 cm/sec to 1.28 X 10^-5 cm/sec. There are also some indications that the water flow history may influence hydraulic conductivity. High water flows may be able to remove and dissolve enough mineral to keep pores open longer against expansive and swelling reactions. One Rapid Age test sealed itself after just 7 days. As leachate volumes fell, the material became more resistant to future leachate flows. The two other columns continued for up to 2 months. The fall in permeability to the low 10^-6 cm/sec range is clear, and although the data is not shown here, attempts were made to keep these columns operating by setting the pressure to the top of the design limit. In previous work with FBC fly ash it has been found that columns will try to seal themselves and that once sealed the column cannot be revived. The higher the flow of water, the better the column seems to resist sealing. The Rapid Age procedure involves several pore volumes of water every day instead of one (1) pore volume as in the ASTM column procedure. Hydraulic conductivity seems to fall less rapidly in the case of the Rapid Age test.

The hydraulic mix differs from the pneumatic mix in that mix components tend to interact with each other in ways that none of the individual materials would react by themselves. Particularly the venturi scrubber sludge will attack the conventional cement pozzolans in the PCC fly ash. Only ASTM data is prepared at this time on the hydraulic mix, but initial hydraulic conductivity is around 6 X 10^-5 cm/sec. Column tests on hydraulic conductivity must be interpreted loosely as even under well controlled conditions measured values will show some variation. The hydraulic conductivity of the hydraulic mix appeared to fall clearly with each measurement ending with a value of 2.5 X 10^-5 cm/sec. The change in value is borderline as to being interpreted as a trend, although the implication that the material is sealing is there.

In considering how these hydraulic conductivities might impact the ability of these mixes to release leachate the hydro-geology of the site must be considered. At the Peabody #10 mine the hydraulic conductivity coal and surrounding rocks are in the 10^-8 to 10^-9 cm/sec range. FBC by-products have been tested in triaxial cells and found to reach these low hydraulic conductivity values when placed at the Proctor optimum density. It seems rather unlikely that paste flowing loosely through an opening or material pushed into place by surges of air will reach these densities. At any rate, for this project both mix materials will be more permeable than the surrounding rock layers and will be a preferred path of water flow through solid. Because the openings will not be filled, however, there will be voids above the fill where hydraulic conductivity will be infinite by comparison. Early indications at the site are also that the hydraulic gradients at the mine are extremely small. This would mean that total leachate volumes will be small, though the potential for dilution may be related to flows in the void space above the material more than to flows in the surrounding rock.

Any contaminant plumes developing in the surrounding rock seem likely to move very slowly. The mined-out panels to be backfilled have been sealed for 40 years and are still dry. If water flows are anywhere near this slow in the future, the influence of water flow rate on the hydraulic conductivity of the pneumatic mix may become important. The pneumatic mix may develop a sealed layer around an un-reacted core of material. If a tightly sealed layer develops the pneumatic mix may become as tight as the surrounding formations yielding leachate at the same rate as the surrounding formations. Any contaminants released would be quite vulnerable to dilution under these conditions.

Data from other research institutes on long term atmospheric weathering of FBC fly ash suggests the possibility that over a long enough time span the permeability will again increase. This being the case, then leaching of the pneumatic mix would proceed like a shrinking core moving through a single very large particle. In this scenario, the rate of release of anything into the environment would be extremely slow and no high concentrations of anything would result.

As attempts are made to generalize underground disposal options to other sites, the possibility that a hydrologic system may have much high hydraulic conductivities and be coupled to the meteoric water system must be considered. In these cases the proposed mixes are developing hydraulic conductivities on the order of those used in liners that protect valuable drinking water from potential contaminants. The small water flows possible at the projected hydraulic conductivities would be easily diluted in such a situation.

Based on the hydraulic conductivities found to date it may be summarized that it seems unlikely that the proposed mixes could release a large enough leachate volume into the environment to adversely impact most any water supply. The biggest uncertainty from this stand-point may be the very long term hydraulic conductivities. The Rapid Age tests are continuing with the intent to develop this information.

3.4 Leachate and Permitting Considerations Based on ASTM Columns

ASTM column tests have been partially completed for a number of the individual materials to be used in the mixes. The ASTM columns should furnish insights not available from any of the shake tests for several reasons.

1- The ASTM column test uses a nitrogen atmosphere and simulates leaching in an oxygen depleted environment below the water table. According to measurements taken in the bore-holes drilled into the test panels at the Peabody #10 mine site, the oxygen in the test panels is almost completely depleted. The mine strata is already below several low grade aquifer layers, though the mine itself is dry.

2- The ASTM column uses a more realistic solid to liquid ratio. With the mine itself dry and only enough water added to the materials to promote paste flow or set-off hydration reactions it is clear that shake tests using 20 parts water to every part solid are unrealistic. The ASTM column uses about 8 parts solid to 1 part liquid. Shake tests use a high ratio of water to solid to avoid common ion effects. With the water in the nearby strata at 3 to 4% solids it is clear that common ion effects and even ion exchange reactions will dominate groundwater reactions in the Peabody #10 mine.

3- The high ratio of solids to liquids in the ASTM column tend to promote higher concentrations of elements in the leachate. While most elements can be detected in 10s of parts per billion with ICP, concentrations must be over 10 times this high before good quantification is possible according to the American Public Health Association. ASTM columns generally produce higher concentrations of trace elements in the leachate, improving the chances that a potentially leaching element will be detected and quantified within accuracy limits commonly expressed for ICP and AA techniques.

4- The ASTM column test produces leachate samples over time. One of the faults of shake tests is that they contact fresh material with solution, measure element concentrations, and assume that the concentrations seen are representative of the leaching of the bulk material. This is not the case with FGD byproducts and associated combustion residues. Many of the most noted toxic elements are volatilized at low temperature and are fumed out over the surface of particles exposed to the combustion exhaust gasses. Shake tests on fresh material leach unrepresentative surfaces and once the surface trace elements are leached away they will never be available in the bulk material in such concentrations again.

At this point ASTM column tests are only complete for the FBC fly ash and spent bed used in the pneumatic mixes, and the force oxidized scrubber sludge used in the hydraulic mix. Most of the observations have more to do with what was not found than any problems found.

1- Most heavy trace metals were either not detected in the ASTM column leachates, or were just barely detected. With the concentration boost expected in ASTM columns this absence of trace heavy metals is a good sign that the materials pose no danger to the groundwater. Elements not found in concentrations high enough for good quantification or not found at all include

a- FBC Fly Ash, --- Ag, Be, Cd, Co, Cr, Cu, Hg, Mg, Ni, Pb, V.
b- FBC Spent Bed, --- Ag, Be, Co, Cr, Cu, Hg, Mg, Ni, Pb, Sb, V, Zn.
c- Force Oxidized FGD, --- Ag, As, Be, Cd, Co, Cr, Cu, Ni, Pb.

2- Among the elements that were detected, most were present in concentrations that, at their peak, would not produce groundwater violations even if no dilution or attenuation occurred. These elements include

a- FBC Fly Ash, --- Al, B, Ba, Fe, Mn, Mo, Si, V, Zn.
b- FBC Spent Bed, --- Al, Ba, Fe, K, Mn, Mo, Na, Si.
c- Force Oxidized FGD, --- Al, Fe, K, Mg, Mn, Mo, Si, V, Zn.

3- The only elements that leached in high concentrations were those typical of dissolved solids such as Ca, K, Na. Conductivity measurements indicate dissolved solids declined with time and never approached levels typical of the brines in the area.

4- Of the elements that were detected in concentrations high enough to produce class II groundwater violations, most occurred only as spike values or declined within less than 16 days to comply with class II standards. This means that leaching of these elements is unlikely to be persistent enough to cause the water in the nearest water bearing layers to violate class II standards. These elements included

a- FBC Fly Ash, --- As, Sb
b- FBC Spent Bed, --- As, B, Cd
c- Force Oxidized FGD, --- B, Sb, Se

The only element that seemed to leach with some persistence at concentrations in excess of class II standards was selenium from the FBC material. The concentrations found for selenium were below the levels needed for good ICP quantification, and the hydride system purchased for mercury analysis on this project, together with the EDL system also purchased on this project, are being readied for more precise measurements. For such deep water units selenium should not be difficult to obtain a variance for even if it proves a problem. The ICP data is good enough to be certain that dilution or attenuation of less than 1 order of magnitude should bring water bearing units into the area into compliance unless selenium is already in violation for natural causes.

3.5 Conclusions

The analyses of the coal combustion by-products to be placed underground at the Peabody Mine #10 demonstration site show that the underground placement presents no dangers to potable water acquifers. The permeability of the placement materials is such that water flow through the materials is very limited. Further, the formations surrounding the abandoned mined areas have very low permeability, and contain water with high dissolved solids, which make it unfit for either drinking water or for agricultural use. Further, it would appear that even where potable aquifers are involved, the leachate from the CCBs is such that it poses no danger to water usage.

CHAPTER 4

GEOTECHNICAL STUDIES

4.0 GEOTECHNICAL STUDIES

4.1 Introduction

Geotechnical studies in Phase II of the project concentrated on measurements of surface and subsurface movements over the proposed injection panels. Laboratory studies of long-term swelling strain measurements of pneumatic mixes was also conducted. During the Phase II demonstration, freshly prepared grouts for hydraulic injection were tested for slump and water content. Grout samples were also collected for strength testing in the laboratory.

4.2 Surface Subsidence Measurements

Sixty-two (62) surface subsidence monuments were installed over the two (2) injection panels to monitor surface movements prior to underground placements. Figure 4.1 shows locations of the survey monuments numbered one to sixty two. The grid consists of subsidence-monument-lines in the transverse and longitudinal directions over the injection panels. Monument numbers 32 to 37 and 45 to 57 over the pneumatic injection panel constitute longitudinal (PL line) and transverse directions (PT line), respectively. Over the hydraulic injection panel, monument numbers 1 to 6 (D line) and 13 to 25 (A line) constitute longitudinal and transverse directions, respectively. In addition, four (4) subsidence-monument-lines were installed at 45° angles to the longitudinal directions of the two panels. Over the pneumatic injection panel, monument numbers 48 to 62 (PB line) and 42 to 51 (PA line) show two monument lines at approximately 45^0 to the longitudinal directions. Over the hydraulic injection panel, the angled lines encompass monument numbers 8 to 20 (C line) and 15 to 30 (B line). This grid provides enough surface movement data to develop contours of vertical surface movements over the panels and around the injection boreholes before and after backfilling.

The surface monuments were spaced at approximately 25-ft intervals in the longitudinal and transverse directions. The subsidence-monument-lines extend at least 100 ft beyond the panel edges.

A subsidence monitoring point consists of a 7/8 inch diameter 7-ft-long roof bolt. The monitoring point is installed by auguring a hole (four inches diameter) to a depth of three (3) feet, inserting the 7-ft-long roof-bolt (a rebar) through three (3) feet of foam insulation and two-inch-diameter PVC pipe, then hammering the bolt between 3.0 and 3.5 ft into the ground. As the frost heave zone in this area is eighteen (18) inches deep from the surface, the subsidence monuments are free from the effects of ground freezing.

The level surveying technique is being used to measure the elevations of the subsidence monuments. An autoset level with an optical micrometer is being used for subsidence measurements. The accuracy of the autoset level is 0.000328 ft.

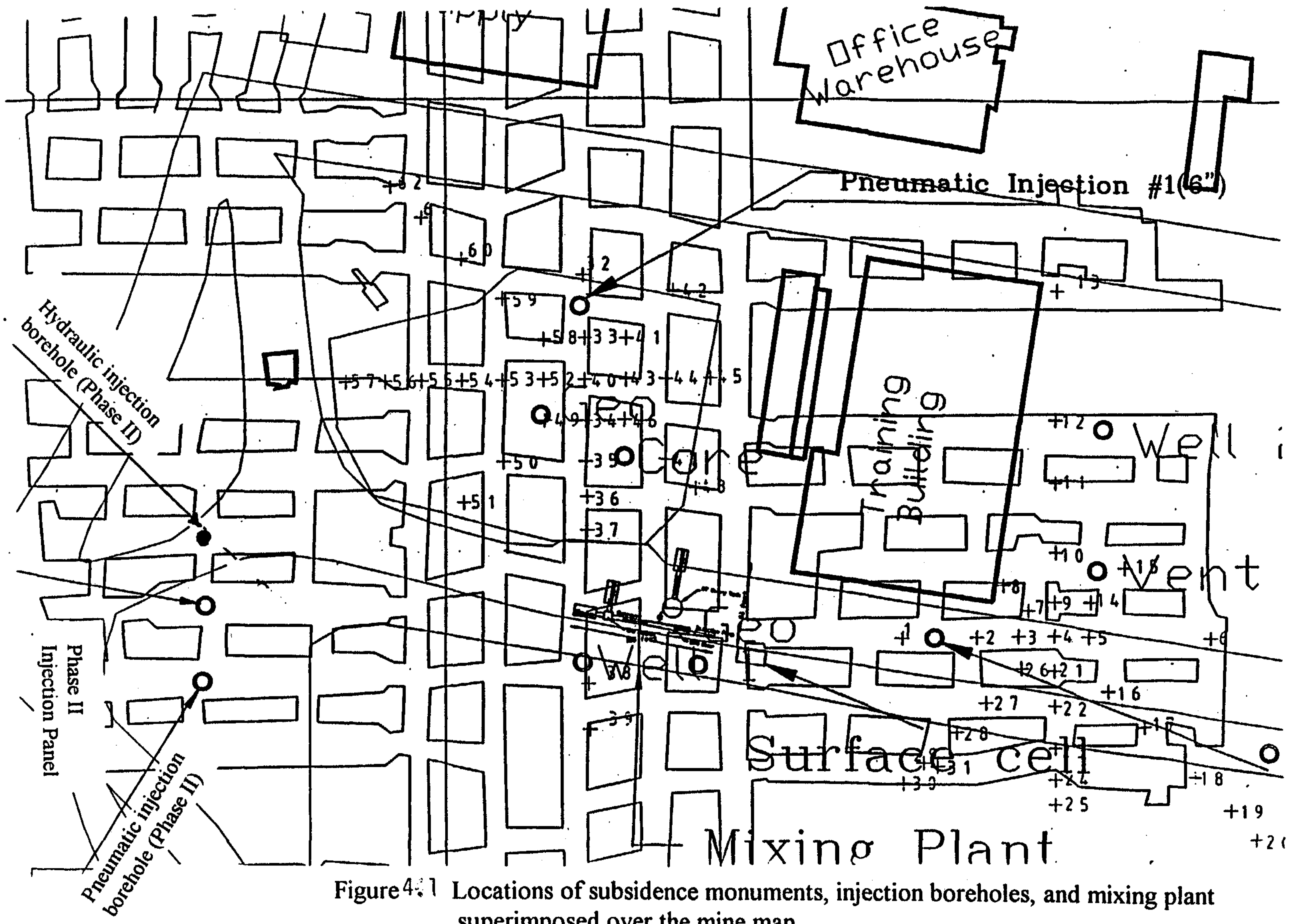

Figure 4.1 Locations of subsidence monuments, injection boreholes, and mixing plant superimposed over the mine map

Two (2) geotechnical holes were drilled to install settlement sensors in the boreholes above the coal seam (up to 50 ft above the roof-line). The settlement probe system consists of the probe with a cable reel, an integral readout device and a corrugated plastic casing with settlement sensors. Fifty (50) feet of corrugated plastic casing with sensors (metal rings) placed every ten (10) feet were installed at the bottom of each geotechnical borehole. Each corrugated casing was attached to the bottom of a four-inch-in-diameter PVC casing installed in the borehole. The casing is built by coupling ten-foot sections of PVC pipe. The annulus between the borehole was and the PVC casing is filled with silica sand.

The settlement probe detects the position of metal rings (in the corrugated pipe) in the borehole and provides a buzzing signal on the readout device when one is approached. The position of the probe is then slowly adjusted for the maximum signal which can be seen on a meter. Once the meter is peaked with the probe, the depth of that particular ring can be directly read off the cable marks and the scale provided on the pipe. The vertical locations of the rings in the corrugated casing will change when vertical movements of the subsurface strata occur. The relative change in the locations of the rings will indicate the amount of vertical subsurface movements. The accuracy of the system is 0.0008 ft.

Baseline measurements of surface subsidence monuments were taken in March, 1996. March, 1996 and April, 1997, five sets of subsidence data were collected. The first two sets of data, measured within a span of 30 days, were used to establish the baseline relative to which the movements of the monuments were calculated. Figures 4.2a through and 4.2d show subsidence of the monuments along the four subsidence lines over the pneumatic injection panel. Subsidence over an elapsed time of 148 days show movements within the error band ($\pm$ 3 mm including measurement errors). Underground roof-to-floor convergence measurements in Phase I of the program indicated approximately 0.5 inches of convergence per year. Thus translates to 0.17 inches (or 4.2 mm) of surface movements per year. The surface movement data over a period of six-month is consistent with the underground measured roof-to-floor convergence data.

Subsidence over an elapsed time of 397 days does not show any definitive trend and many monuments registered surface heave. The maximum subsidence observed was 4.8 mm at monument #54. The average of all the negative movements was -2.73 mm and 57% monuments show downward movements.

Figures 4.3a through 4.3d show surface movements over the hydraulic injection panel. Movements for an elapsed time of 148 days show approximately 1 mm of downward movements. For an elapsed time of 397 days, the downward movement of monuments show less scattering than the movements over the pneumatic panel. This may be due to the fact that the hydraulic injection panel is a rooming area in contrast to the submains of pneumatic injection panel. Over the entire area, downward surface movements of approximately 4 to 5 mm were observed. The average of all the monuments showing downward movements was -3.54 mm and 69% monuments showed downward movements over the hydraulic injection panel.

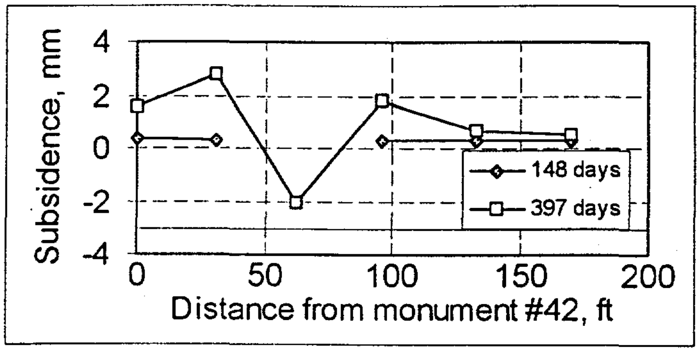

Figure 4.2a Measured subsidence over pneumatic injection panel (PA line)

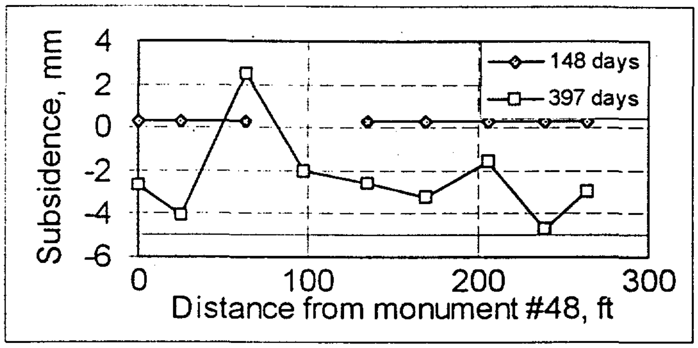

Figure 4.2b Measured subsidence over pneumatic injection panel (PB line)

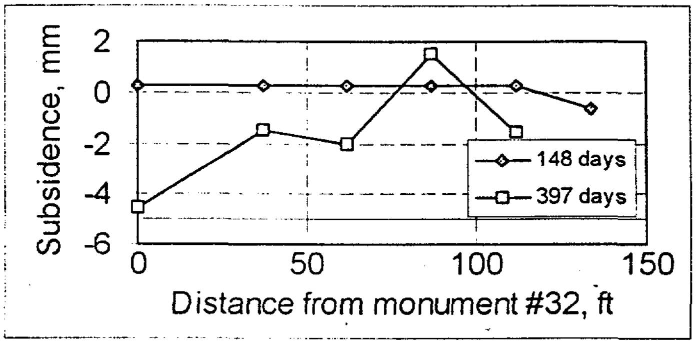

Figure 4.2c Measured subsidence over pneumatic injection panel (PL line)

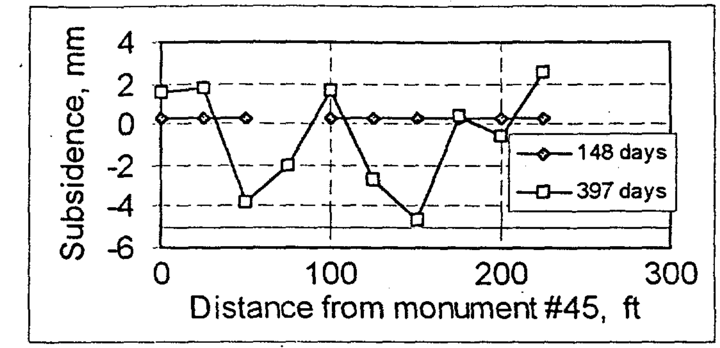

Figure 4.2d Measured subsidence over pneumatic injection panel (PT line)

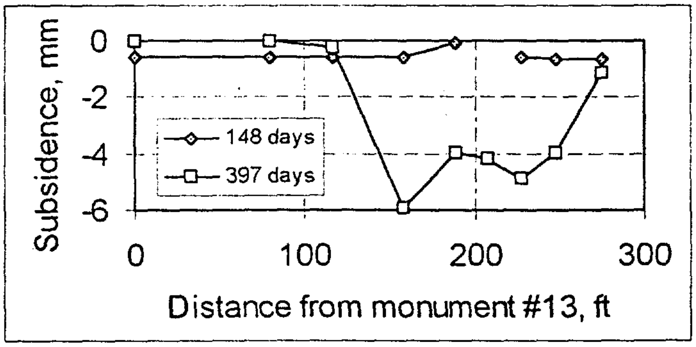

Figure 4.3a Measured subsidence over hydraulic injection panel (A line)

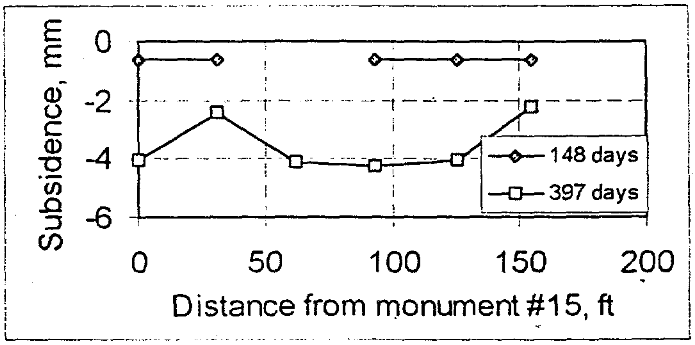

Figure 4.3b Measured subsidence over hydraulic injection panel (B line)

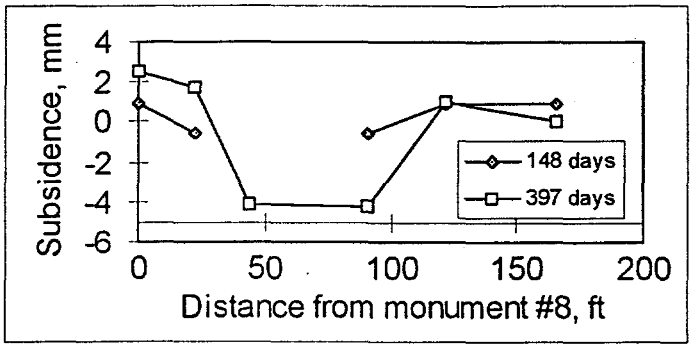

Figure 4.3c Measured subsidence over hydraulic injection panel (C line)

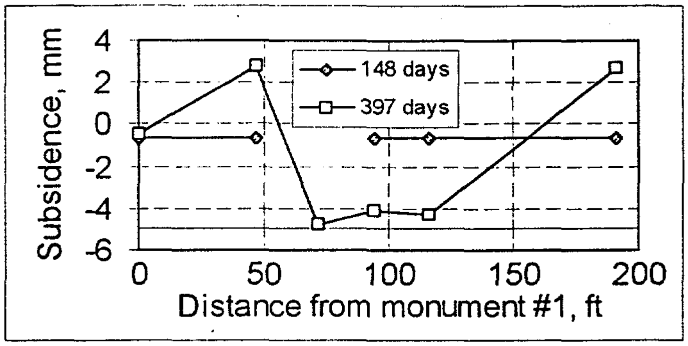

Figure 4.3d Measured subsidence over hydraulic injection panel (D line)

For baseline measurement, the sonic probe was introduced in the two subsurface measurement boreholes. The probe in the boreholes over hydraulic and pneumatic panels could go only 282 ft and 248 ft, respectively. This indicates that the geotechnical boreholes collapsed after the installation of the settlement sensors. No readings could be taken from these two boreholes because the sensors are located at a depth between 290 ft and 340 ft below the collapsed region. However, the elevation of the top of the steel casing of the geotechnical boreholes are being measured by the autoset level

4.3 Long-term Swelling Strain Measurements

Three samples of pneumatic mix with 80% FBC fly ash and 20% spent bed ash were monitored for long-term free swelling strain. Figure 4.4 shows the schematic of the measurement apparatus. The plastic container was filled with water after placing the sample in the container. The swelling of the sample was measured using a dial gage (shown in Figure 4.4). Figure 4.5 shows the average swelling strain of the three samples. Average swelling strain after 36 days is approximately 1%. Because the samples were made using preconditioned FBC fly ash, the observed long-term swelling strain was low.

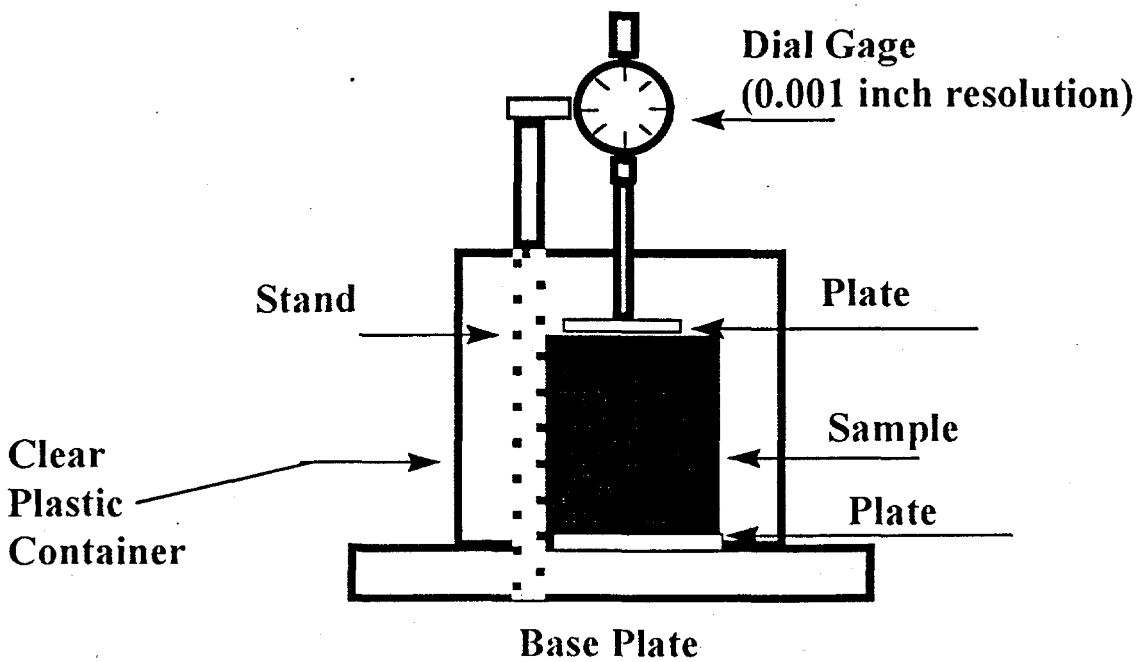

Figure 4.4 Schematic of experimental setup for swelling strain measurement

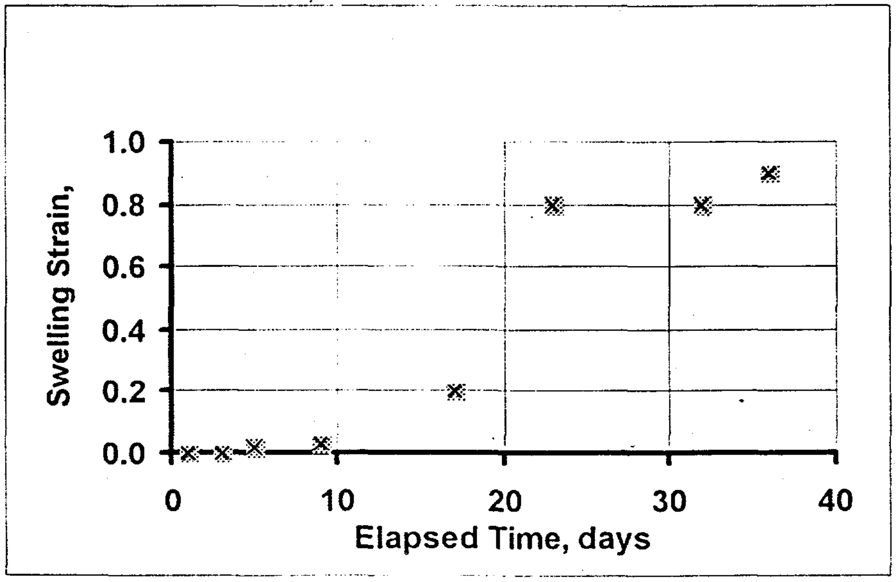

Figure 4.5 Long-term swelling strain of pneumatic mix.

CHAPTER 5

MATERIAL HANDLING AND SYSTEM ECONOMICS

5.0 MATERIALS HANDLING AND SYSTEM ECONOMICS

5.1 Introduction

The activities of Phase II in materials handling and systems economics are mostly continuations of the activities completed in Phase I. In this chapter, first, the objectives of the materials handling and system economics are stated. Next, the activities completed in Phase I are summarized to provide a link between Phase I and Phase II activities. Within this context, the transportation and handling technologies that were found environmentally acceptable in Phase I are briefly presented to clarify the activities for improved analyses of these technologies in Phase II. Finally, the activities of Phase II are described in detail.

The term "materials handling" in this research project includes loading, transporting, unloading, and temporary storage of the dry coal combustion by-products for the purpose of placing them in abandoned areas of the underground coal mines in Illinois. Materials handling systems have been analyzed in four consecutive modules: 1) storage, handling and loading of the residues at the plant site, 2) transportation from the plant to the mine site, 3) unloading , handling and storage at the mine site, and 4) transportation from the mine site to the injection site.

5.2 Objectives

The objectives of the materials handling research are defined as follows: 1) Identify the systems that are technically, economically, and environmentally feasible in handling and transporting the coal combustion by-products (CCBs) from the power plant to the underground placement site, 2) Demonstrate the operation of one or two of the identified systems.

The objectives of the system economics research are defined as follows: 1) Develop a generalized "Engineering Design and Economic Evaluation" model that can be used in evaluating various types of materials handling and underground placement systems, 2) Conduct economic analyses of the selected materials handling systems along with the underground placement system using the developed model, and 3) Conduct a case study for the hydraulic and pneumatic underground placement demonstration which is planned as a part of the overall research project

In accordance with the above stated objectives, several transportation and handling (T&H) technologies for CCBs were investigated from technical, environmental and economic points of view. Five technologies were found promising: (1) Pneumatic Trucks (PT), (2) Pressure Differential Rail Cars(PD-car), (3) Collapsible Intermodal Containers (CIC^{TM}), (4) Cylindrical Intermodal Tanks (CIT), and (5) Coal Hopper Cars with Automatic Retractable Tarping (CHC). Among these technologies, engineering design and cost models were developed for the first four technologies. The fifth technology, CHC, was not amenable for such developments due to lack of cost and engineering data at this time. The developed software were used in the evaluation of a number of hypothetical cases as well as the case study.

In addition to the quarterly and annual reports, one of the tasks of this segment of the research project was to submit a topical report on "Materials Handling and Systems Economics". This report has been finalized and will be submitted to USDOE in June, 1997. Another deliverable is a user's manual for the software developed for the selected technologies. This manual will be submitted to USDOE in August 1997.

Another objective of this research project was to train a graduate student in the area of "coal combustion by-product handling and transportation". Accordingly, Mr. Samual Gwamaka was trained in the Department of Mining Engineering from 1994 to 1996. He completed a master's thesis entitled "Development of Simulation and Cost Models to Evaluate Coal Combustion Residue Transportation and Handling Systems", in January, 1996 (Gwamaka, 1996) and was awarded a Master of Science degree in Mining Engineering.

5.3 Summary of Activities During Phase I

In Phase I of the cooperative agreement the following tasks were completed in the Materials Handling and Systems Economics research:

1. Investigation of existing transportation and handling systems:

A number of T&H systems that are being practiced in surface and underground disposal of CCBs were investigated and their descriptions were given in the Phase I final report (Chugh et al., 1996). These systems can be listed as follows:

- Pneumatic Trucks (PT)
- Pressure Differential Rail Cars (PD-car)
- Open Hopper Coal Cars
- Tarped Rear-Dump Trucks
- Bottom-Dump Container Trucks

2. Exploration of new and adaptable transportation and handling systems:

A number of T&H systems were explored for possible application in CCB transportation and handling. They, too, were described in the Phase I final report (Chugh et al., 1996). These systems can be listed as follows:

- Collapsible Intermodal Containers (CIC)
- Cylindrical Intermodal Tanks (CIT)
- Coal Hopper Cars with Automatic Retractable Tarping (CHC)
- Steel Intermodal Containers
- Covered Hopper Cars - Grain Cars

3. Development and demonstration of Collapsible Intermodal Containers (CIC™):

The CIC technology was developed by SEEC, Inc. as a part of the overall DOE-SIUC cooperative agreement. A field demonstration of CIC technology was held at the Illinois Power Company's Baldwin Power Plant on November 17, 1994. A final topical report entitled "The Development and Testing of Collapsible Intermodal Containers for the Handling and Transport of Coal Combustion Residues" was submitted to USDOE in 1995 (Carpenter and Thomasson, 1995).

4. Investigation of surface disposal practices:

In addition to a literature survey, a number of surface disposal operations were visited to gain insight and compile information for comparison with underground disposal. The observations during these visits and the findings of the literature survey were used in the development of operating scenarios for the environmentally acceptable transportation technologies.

5. Development of operating scenarios for the environmentally acceptable transportation and handling technologies:

Among the ten existing and futuristic T&H technologies listed above, the following five were found to be environmentally promising:

1. Pneumatic Trucks (PT)
2. Pressure Differential Rail Cars (PD-car)
3. Collapsible Intermodal Containers (CIC)
4. Cylindrical Intermodal Tanks (CIT)
5. Coal Hopper Cars with Automatic Retractable Tarping (CHC)

Among these five technologies, reliable engineering and cost data were available for the first three technologies during Phase I. Therefore, operating scenarios were developed for these three technologies. The operating scenarios define operation activities such as CCB loading at the mine site, transportation between the plant and the mine site, and unloading and handling at the mine site.

6. Development of engineering design, cost, and economic evaluation models for the environmentally acceptable technologies.

The operating scenarios developed for each T&H technology constituted the basis for the development of the engineering and cost models. The engineering design models were developed for the determination of factors such as number of transport units, silo capacity, loading and unloading rates, placement system capacity, and number of shifts. The cost models were developed for the determination of the operating and capital costs in each T&H technology. The economic evaluation model was common to all T&H technologies. It was developed based on the "After-Tax Cost" method where the ultimate product is cost-per-ton of CCB transported and delivered to the final destination.

7. Development of an interactive software:

In activity 6 above, the engineering design and cost models were developed using EXCEL spreadsheets, whereas the economic evaluation model was developed using Fortran programming language. These models were gathered under one roof using DELPHI, a rapid application development tool, to make the software as practical as possible. The final product was an interactive software with Windows application. In this application, a template was built for each environmentally acceptable T&H technology. When the user selects one of the technologies, the corresponding template is activated and the input data are presented in various dialog boxes. The source of a particular data entry can be retrieved by clicking on that entry. If the entry is not satisfactory, the user can overwrite it by typing in the desired number. In other words, default values are provided for every single entry in the template.

8. Engineering and economic analyses of the environmentally acceptable technologies using the developed software:

Distance and tonnage are two critical parameters in the selection of an appropriate technology for the transportation and handling of CCBs from a plant to a mine site. In order to determine the distance and tonnage ranges in which a T&H technology is favorable over the others, several hypothetical cases were systematically generated and evaluated using the developed software. Specifically, nine tonnage-distance combinations were generated for each technology; where the tonnage was set at 50,000, 100,000, and 200,000 tons per year, and the distance at 30, 100, and 200 miles.

9. Completion of a master's thesis and publications and presentations of four technical papers summarizing the research findings.

A graduate student was trained in the area of CCBs transportation and handling. He completed a master's thesis in January, 1996 (Gwamaka, 1996) and was awarded a Master of Science degree in Mining Engineering. Also, three technical papers summarizing the research results were published and presented (Sevim and Gwamaka, 1995; Sevim, Lei and Gwamaka, 1995; Sevim, Gwamaka and Lei, 1996).

10. Reporting

The activities listed above were reported in Phase I final report under Materials Handling and Systems Economics (Chugh et al., 1996). In addition, a topical report covering all the completed tasks of Materials Handling research in Phase I and Phase II has been finalized and will be submitted to USDOE in June, 1997.

5.4 Environmentally Acceptable Technologies of Phase I

In Phase I, after an extensive investigation of the existing and adaptable technologies, five T&H technologies were found environmentally acceptable. Operating scenarios and engineering and economic evaluation models were developed for three of these technologies, namely; PT, PD-car, and CIC. In Phase II, CIT was added to the list. The fifth technology, CHC is still in the conceptualization phase, and therefore could not be developed fully. In the following, brief descriptions of the three T&H technologies developed in Phase I will be given. The CIT technology, which was developed in Phase II, will be given later within the context of the activities of Phase II.

5.4.1 Pneumatic Trucks (PT).

Pneumatic trucks, also referred to as bulk tank trucks, are widely used in transporting low density dry flowable powder and granular materials as well as high density materials such as cement, limestone and fly ash. The most common method of loading the material is gravity feeding from a silo by a collapsible spout which engages to the gate on top of the tank. The tank is air tight when the lids of the gates are closed. During offloading, the material flows through the piping below the tank due to pressure difference created by the blower.

The operating scenario developed for the PT transportation is schematically shown in Figure 5.1. The trucks are loaded from the fly ash bin of the plant and they deliver the material directly to the underground placement point at the mine site. There, the pressure necessary for offloading the fly ash into the injection hopper is supplied by either a stationary blower or a by a blower mounted on the truck. These trucks are approximately 20-25 tons in capacity and can offload in about 20-25 minutes (Freitag et al., 1991). The preliminary testing at the demonstration site at Peabody No. 10 mine, however, indicate that the unloading can be accomplished in 30 to 40 minutes.

5.4.2 Pressure Differential Rail Cars (PD-car)

These are special type of rail cars used to handle powdered materials. They are operated under the principle of pressure differences between the car and the container to which the product is discharged. Normally PD cars are complemented with pneumatic trucks at rail terminals to deliver the material to the final destination. When a PD car is pressurized to about 5 psi or more, the outlet valves are opened to form a steady flow of material into the truck until all the material in the compartment is cleared.

The operating scenario developed for the PD-car transportation is schematically shown in Figure 5.2. As seen, one set of cars is being filled at the power plant while the other set is being emptied at the mine. When all the cars at the plant are filled, either a "local train" or a "unit coal train" will take them to the mine. Similarly, the empty PD cars will be delivered to the plant for another round. At the mine, the product in the PD cars will be transloaded into a silo with the aid of a stationary blower. Delivery from the silo to the underground placement site can be done either by pneumatic trucks, or by regular dump trucks if the silo is equipped with a pugmill to wet the by-product to prevent fugitive dust.

5.4.3 Collapsible Intermodal Containers (CIC™)

These containers are made of rubber coated aramid and nylon fabric with polyester webbing. They are patented by SEEC™ Inc., one of the research partners in the DOE-SIUC cooperative agreement. The CICs are collapsible storage bins that are portable and intermodal - designed to ride inside coal cars, barges and trucks. Those CICs made to transport fly ash by riding in coal cars have a height of 120 inches, diameter of 110 inches and a 19-inch filling port. For ash of 60 lb/ft^3 bulk density, the CIC capacity is about 20 tons. These containers are extremely durable and provide fully encapsulated transport, eliminating fugitive dust problems (Carpenter and Thomasson, 1995).

The operating scenario developed for the CIC transportation is schematically shown in Figure 5.3. The coal train arrives at the plant and offloads coal into an under-track bin. Next, the CICs which have already been filled with fly ash and staged along the rail are lifted, one at a time, by an overhead crane and placed into the bays of the empty coal cars. Two specially designed lifting brackets mounted on both sides of the CIC facilitate lifting and placement by the crane into the bay of the car. Three or four CICs occupy a car, each taking one of the bays of a typical coal car. The overhead crane is on rubber tires and travels along the rail track looking inside the cars. When all the CICs are loaded, the train leaves for the mine.

At the mine site, an overhead crane lifts the CICs, one at a time, and places them on the concrete pad along the rail track. When all the CICs are offloaded, the train pulls under the silo for coal loading. After filling all the hopper cars, the train leaves for the power plant. The CICs are then loaded on tote trailer(s) by the same crane and transported to the injection site. There, the ash is offloaded into the hopper of the underground placement system by the use of a vacuum system designed specifically for the CICs. The empty CIC can be transported back to the rail site on the same trailer. At the rail site, the empty bag is lifted with a small fork lift, carried into a baghouse where the air trapped in the CIC is extracted. The collapsed bag is then retrieved by the forklift and hung like a vest onto the rail guides of a covered trailer. After collecting 25-40 empty CICs, the trailer is transported back to the plant.

At the plant, the tractor leaves the filled trailer, takes the empty trailer and drives back to the mine. The empty CICs are retrieved from the trailer with the help of a small forklift and placed, one at a time, on a specially designed trailer and pulled under the fly ash silo by a tractor. There, it is filled by gravity similar to filling a pneumatic truck. Then, the CIC is transported back to the rail site, where the trailer pulls under the overhead crane, and it is lifted and staged along the track and kept there until the coal train comes back from the mine.

5.5 Activities During Phase II

The duration of the Phase II of the cooperative agreement was from April 1, 1996 to April 30, 1997. During this period, the following tasks were completed under the Materials Handling and Systems Economics research:

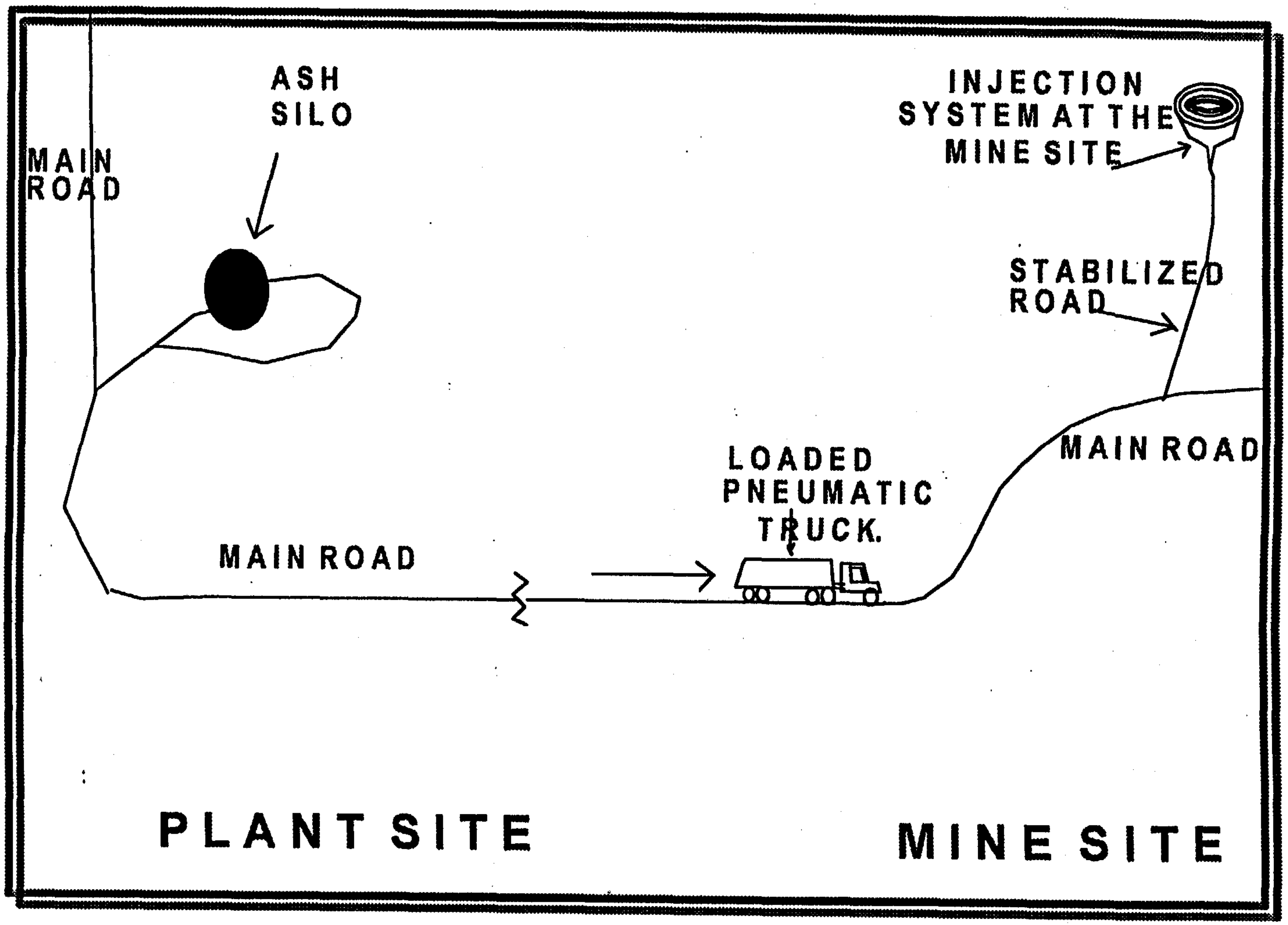

Figure 5.1 Dry FGD By-Products Transported by Pneumatic Trucks

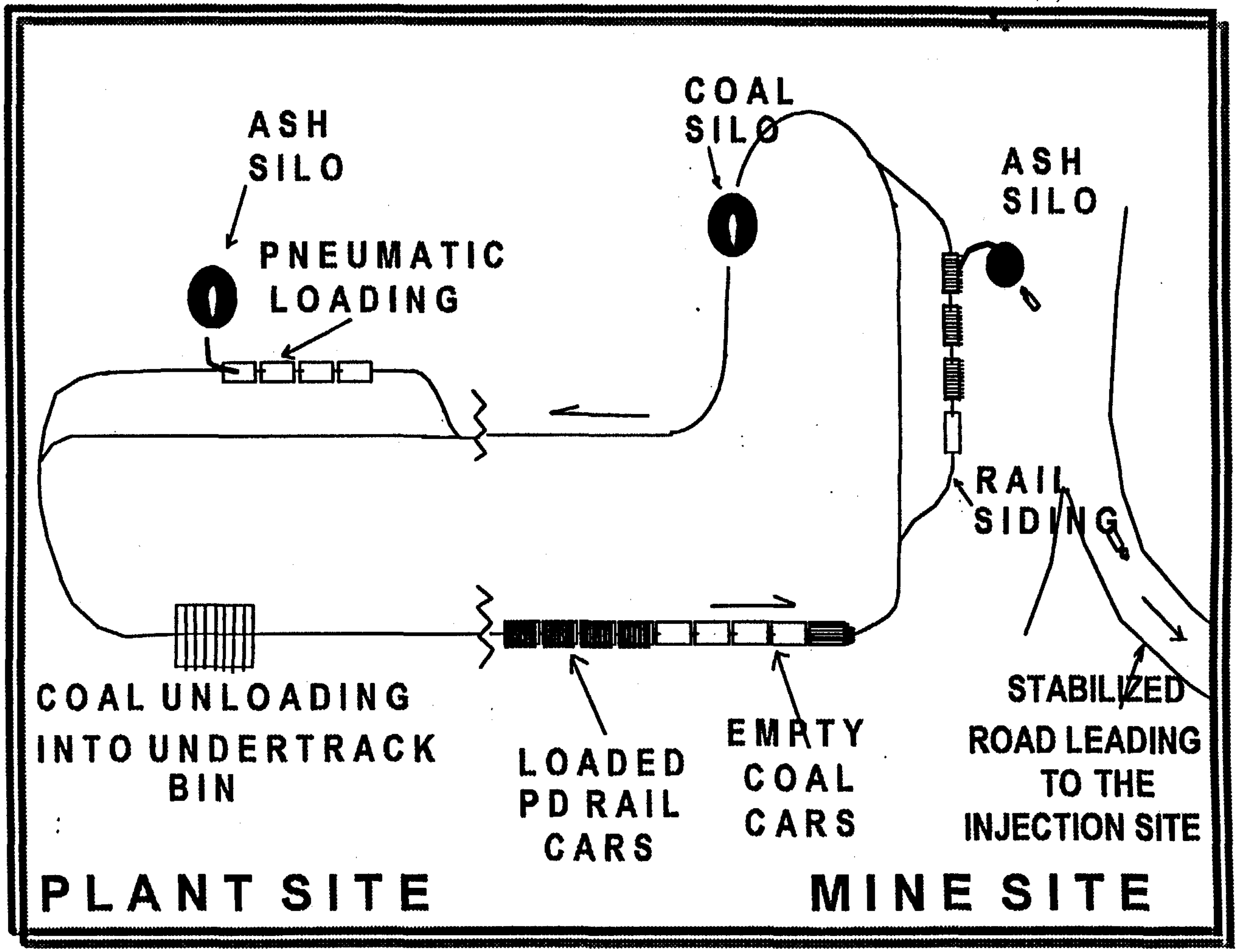

Figure 5.2 Dry FGD By-Products Transported by Pressure Differential Rail Cars

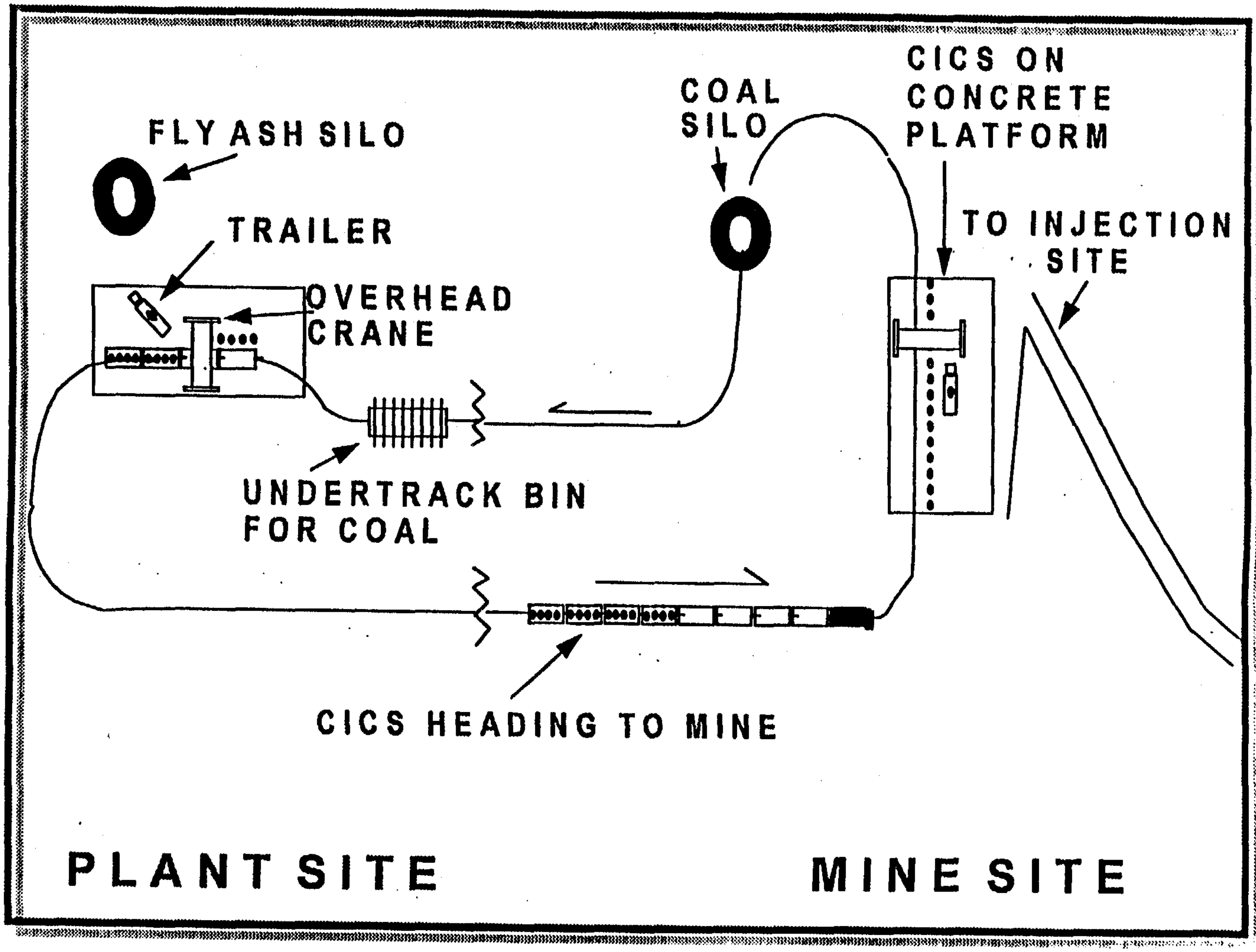

Figure 5.3 Dry FGD By-Products Transported by Collapsible Intermodal Containers

1. Development of engineering design spreadsheets for the environmentally acceptable transportation and handling technologies:

One of the activities in Phase II was the incorporation of the pneumatic and hydraulic underground placement modules into the software developed in Phase I. During this incorporation, it was observed that the engineering design computations of the selected technologies were getting more and more voluminous. Therefore, it was decided that the engineering design computations be separated from the cost computations in the templates developed for each technology in order to keep the use of the templates as practical as possible. For this purpose, a spreadsheet was developed using Microsoft EXCEL software for each T&H technologies. In these spreadsheets, operating schedules composed of plant schedule, transportation schedule between the plant and the mine, and mine schedule, are devised; and critical system parameters such as silo capacities, number of transportation units, loading rate, transloading rate, and injection rate are calculated for a given tonnage-distance combination. The data generated in these spreadsheets are then fed into the cost computation models.

The spreadsheet developed for CCB transportation using pneumatic trucks is shown in Table 5.1. It is noted that this spreadsheet can actually be used for any type of truck transportation; tarped or open rear-dump trucks, bottom-dump container trucks, trailer mounted containers, etc. In Table 5.1, the first two blocks of data, printed in italic, are user defined. The first block - line 1 to 3 - defines the operating schedule, whereas the second block - line 5 to 11 - defines values of operation parameters. The third block, printed in bold letters, exhibits the values of the design parameters calculated using the input data from the first two blocks.

For example, for a tonnage-distance combination of 100,000 tons and 100 miles (second column), the cycle time of a truck is found to be 133.3 minutes by adding loading time (line 5), travel time loaded (line 13), unloading time (line 8), and travel time empty (line 14). Another important parameter is the "maximum number of trucks without queuing". This is the maximum number of trucks in a fleet that would operate without waiting in line at either end of the operation. It is obtained by dividing the cycle time to the greater of the unloading time or loading time (133.3/20 in this example). The "required number of trucks" (line 20) to deliver the targeted tonnage, on the other hand, is calculated by dividing the "required number of trips per day" (line 18) to "maximum number of trips per truck per shift" (row 19). If the "required number of trucks" is smaller than the "maximum number of trucks without queuing," the scenario receives a "yes" (line 23) for its feasibility. Otherwise, the unloading time is decreased (line 8), or equivalently, the injection rate is increased (line 21) and the new scenario is re-tested for feasibility. It is noted that the upper limit for increased injection rate is assumed to be 2 tpm. Any scenario that requires more than 2 tpm to become feasible should be rejected.

The plant silo capacity is shown in lines 24 and 25. Assuming two idle shifts on Saturday and no work on Sunday, total fly ash accumulation hours at the plant adds up to 40 hours. This translates into an accumulation of 457 tons when the annual production is 100,000 tons. If the plant does not have a silo to accommodate this accumulation, either the operating schedule has to be changed or extra storage capacity has to be built in the system.

An important point in the determination of the truck fleet size should be mentioned here. The "required number of trucks" calculated in the engineering design spreadsheet will in most cases fall short of transporting the annual production due to the fact that the availability of a truck will always be less than 100 %. For instance, in the 100,000 tons - 100 miles combination, the required number of trucks was calculated to be 10.21 as seen in Table 5.1. This number was rounded to 10 and entered in the cost computation spreadsheet where it was subjected to "availability" concept. It is assumed that each truck in the fleet will have a 90 % availability.

Table 5.1 Engineering design computations in CCB transportation by PT technology

1	*Annual Production (tons)*	*100000*	*100000*	*100000*
2	*Distance (miles)*	*30*	*100*	*200*
3	*Shifts per day*	*1*	*1*	*1*
4				
5	*Loading time (min)*	*10*	*10*	*10*
6	*Average speed loaded (mph)*	*45*	*45*	*45*
7	*Average speed empty (mph)*	*45*	*45*	*45*
8	*Unloading time (min)*	*29*	*29*	*29*
9	*Duration of a shift (min)*	*480*	*480*	*480*
10	*Work days per year*	*312*	*312*	*312*
11	*Truck payload (tons)*	*20*	*20*	*20*
12				
13	**Travel time loaded (min)**	**40.0**	**133.3**	**266.7**
14	**Travel time empty (min)**	**40.0**	**133.3**	**266.7**
15	**Cycle time (min)**	**119.0**	**305.7**	**572.3**
16	**Maximum number of trucks without queuing**	**4.10**	**10.54**	**19.74**
17	**Required number of trips per year**	**5000**	**5000**	**5000**
18	**Required number of trips per day**	**16.03**	**16.03**	**16.03**
19	**Maximum number of trips per truck per shift**	**4.03**	**1.57**	**0.84**
20	**Required number of trucks**	**3.97**	**10.21**	**19.11**
21	**Injection rate (tpm)**	**0.69**	**0.69**	**0.69**
22	**Injection system capacity (tph)**	**41**	**41**	**41**
23	**Feasible scenario?**	**Yes**	**Yes**	**Yes**
24	**Max. ash accumulation time (hrs)**	**40**	**40**	**40**
25	**Max. ash accumulation (tons)**	**457**	**457**	**457**

With ten trucks, the availability of the fleet will be 0.9^{10}, or simply 0.348 (approximately 35%). This is an unacceptable level of availability. Therefore, it is resolved in this design that the fleet availability must be at least 85%. The number of trucks that would provide such an availability is then calculated using the Binoial probability distribution. This number was found to be 12 for the above combination. The Binomial probability distribution and the computation of the number of trucks for the above case are given in the Appendix.

The spreadsheet developed for CCB transportation using PD cars is shown in Table 5.2. This spreadsheet is divided in two sections; plant site and disposal site. The first block in plant site (lines 1 to 7) corresponds to user defined data, whereas the second block (lines 9 to 17), marked in bold letters, corresponds to calculated values of the design parameters for the operations at the utility plant site. An important value in the user defined block is "the number of trips per week" (line 4) which is a determining factor of the number of PD-car needed in the system (line 11). Obviously, the more the trips per week the less the number of PD cars, and consequently, the less the capital investment in expensive PD cars (approximately $80,000 a piece). However, the number of trips that can be scheduled depends on the location of the plant and the mine, as well as the flexibility of the rail company operating within the area.

Another important value is the silo capacity at the plant (line 16). As seen on line 15, for 100 miles transportation distance, the train cycle time is estimated to be 28.4 hours. During this time period, the CCB produced in the plant has to be stored. At a continuous flow of 0.191 tpm (line 9), a silo of 326 tons (line 16) is needed. Since an existing silo of only 100 tons is reported by the user (line 5), a warning is given on line 17 to "increase" the silo capacity. If for any reason this extra capacity can not be provided, then the operating schedule must be changed.

The first block of the second section is again for the user defined data, whereas the second block is for the calculated values of the design parameters pertaining to disposal site. Here the calculated transloading rate from the PD-car into the silo (line 32) is based on 480 minutes of continuous operation (line 23) in a shift. Similarly, the injection rate of 0.89 tpm (line 34) is based on 360 minutes of continuous operation (line 24). If these times are not realistic, they should be changed. Finally, in the last line of this block it is seen that three trucks will be sufficient to transport 100,000 tons annually from the silo to the injection point. In another operating scenario, however, silo may be totally omitted at the mine site, and the pneumatic trucks can be used between the PD cars and the under-ground placement system where the PD cars themselves will play the role of a silo. It is noted that, if desired, this spreadsheet can also be used for rail cars other than the PD cars.

Table 5.2 Engineering design computations in CCB transportation by PD-car technology

	PLANT SITE:			
1	*Distance between the plant and the mine (miles)*	*30*	*100*	*200*
2	*Annual residue production (tons)*	*100000*	*100000*	*100000*
3	*PD car capacity (tons)*	*100*	*100*	*100*
4	*Number of trips per week*	*1*	*1*	*1*
5	*Existing silo capacity (tons)*	*100*	*100*	*100*
6	*Average speed of train (mph)*	*45*	*45*	*45*
7	*Time between delivery of filled cars and pick-up of the empties (hours)*	*24*	*24*	*24*
8				
9	**Continuous flow rate from plant into silo (tpm)**	**0.191**	**0.191**	**0.191**
10	**Weekly residue production (tons)**	**1923**	**1923**	**1923**
11	**Required number of PD cars per trip**	**19.23**	**19.23**	**19.23**
12	**Final number of PD cars per trip**	**19**	**19**	**19**
13	**Tonnage transported per trip (tons)**	**1923**	**1923**	**1923**
14	**Round trip time (hours)**	**1.333**	**4.444**	**8.889**
15	**Total time for train arrival at plant (hours)**	**25.333**	**28.444**	**32.889**
16	**Minimum required silo capacity at plant (tons)**	**290**	**326**	**376**
17	**Need to adjust the silo capacity?**	increase	increase	increase
18				
19	**DISPOSAL SITE:**			
20	*Number of working weeks per year*	*52*	*52*	*52*
21	*Working days per week*	*6*	*6*	*6*
22	*Shifts per day (100 tph inj. rate is max, so 100*6*6*52=187200t)*	*1*	*1*	*1*
23	*Transloading time per shift (minutes)*	*480*	*480*	*480*
24	*Injection time per shift (minutes)*	*360*	*360*	*360*
25	*Truck capacity (tons)*	*20*	*20*	*20*
26	*Truck travel time empty (minutes)*	*10*	*10*	*10*
27	*Truck travel time loaded (minutes)*	*10*	*10*	*10*
28	*Truck loading time from silo (minutes)*	*10*	*10*	*10*
29	*Truck unloading time into injection hopper (minutes)*	*20*	*20*	*20*
30				
31	**Truck cycle time (minutes)**	**50**	**50**	**50**
32	**Transloading rate from train to silo (tpm)**	**0.668**	**0.668**	**0.668**
33	**Silo capacity (one shift injection, tons)**	**321**	**321**	**321**
34	**Injection rate (tpm)**	**0.890**	**0.890**	**0.890**
35	**Injection capacity (tph)**	**54**	**54**	**54**
36	**Number of trucks in the fleet**	**2.226**	**2.226**	**2.226**
37	**Final number of trucks in the fleet**	**3**	**3**	**3**

The spreadsheet developed for CCB transportation using CIT is shown in Table 5.3. The content of this table is very much similar to that of the PD-car table, except that there is an extra operation of container transfer and container loading at the plant site, which requires the determination of the right number of tote trailers (line 25). If, however, the facility provides direct loading of the containers from the silo while they are on the flat-bed rail car, this extra operation can be avoided. As in the first two spreadsheets discussed above, this spreadsheet too, can accommodate any type of intermodal container system. The operating scenario developed for the CIT technology will be given in item 2 below.

Finally, the spreadsheet developed for the CIC system is shown in Table 5.4. This spreadsheet is quite similar to CIT spreadsheet with the exception of a truck operation to transport the empty CICs from the mine to the power plant. The required number of truck trips shown on line 27 is obtained by dividing the number of containers to be transported on a weekly basis by the number of empty CIC that can be loaded into a truck trailer. For example, for the combination of 100 miles - 100,000 tons, the required number of truck trips per week is 3 (line 27). Furthermore, the time available for a round trip is 19 hours (line 28), and one round trip takes 3.64 hours (line 29). Since the duration of a round trip is smaller than the time available for a round trip, only one tractor can do the job (line 30). It is noted that, if the operating scenario is such that the empties are to be returned in coal cars rather than in trailers, the above mentioned section can be eliminated, in which case the design computations will be almost identical to those in CIT technology.

Table 5.3 Engineering design computations in CCB transportation by CIT technology

	PLANT SITE:			
1	*Distance between the plant and the mine (miles)*	*30*	*100*	*200*
2	*Annual by-product production (tons)*	*100000*	*100000*	*100000*
3	*Container capacity (tons)*	*20*	*20*	*20*
4	*Number of trips per week*	*1*	*1*	*1*
5	*Existing silo capacity (tons)*	*100*	*100*	*100*
6	*Average speed of train (mph)*	*45*	*45*	*45*
7	*Time between delivery of filled cars and pick-up of the empties (hours)*	*24*	*24*	*24*
8	*Number of work weeks per year*	*52*	*52*	*52*
9	*Number of work days per week*	*7*	*7*	*7*
10	*Number of shifts per day*	*1*	*1*	*1*
11	*Duration of a shift (minutes)*	*480*	*480*	*480*
12	*Cycle time for filling one container (minutes)*	*30*	*30*	*30*
13				
14	**Continuous flow rate from plant into silo (tpm)**	**0.191**	**0.191**	**0.191**
15	**Weekly residue production (tons)**	**1923**	**1923**	**1923**
16	**Required number of containers per trip**	**96.15**	**96.15**	**96.15**
17	**Final number of containers per trip**	**96**	**96**	**96**
18	**Tonnage transported per trip (tons)**	**1923**	**1923**	**1923**
19	**Round trip time (hours)**	**1.333**	**4.444**	**8.889**
20	**Total time for train arrival at plant (hours)**	**25.333**	**28.444**	**32.889**
21	**Minimum required silo capacity at plant (tons)**	**290**	**326**	**376**
22	**Need to adjust the silo capacity?**	increase	increase	increase
23	**Storage and staging area (twice the area of containers - sq. ft)**	**38400**	**38400**	**38400**
24	**Total number of containers filled per shift per trailer**	**16**	**16**	**16**
25	**Number of tote trailers**	**1**	**1**	**1**
26				
27	**DISPOSAL SITE:**			
28	*Number of work weeks per year*	*52*	*52*	*52*
29	*Working days per week*	*6*	*6*	*6*
30	*Injection time per shift (minutes)*	*360*	*360*	*360*
31	*Shifts per day (100 tph inj. rate is max, so 100*6*6*52=187200t)*	*1*	*1*	*1*
32	*Tote trailer travel time empty (minutes)*	*10*	*10*	*10*
33	*Tote trailer travel time loaded (minutes)*	*10*	*10*	*10*
34	*Trailer loading time (minutes)*	*5*	*5*	*5*
35	*Container unloading time into injection hopper (minutes)*	*5*	*5*	*5*
36				
37	**Tote trailer cycle time (minutes)**	**30**	**30**	**30**
38	**Total number of containers transported per shift per trailer**	**12**	**12**	**12**
39	**Number of tote trailers**	**2**	**2**	**2**
40	**Storage and staging area (twice the area of all containers - sq. ft)**	**76800**	**76800**	**76800**
41	**Injection rate (tpm)**	**0.890**	**0.890**	**0.890**
42	**Injection system capacity (tph)**	**53**	**53**	**53**

Table 5.4 Engineering design computations in CCB transportation by CIC technology

	PLANT SITE:			
1	*Distance between the plant and the mine (miles)*	*30*	*100*	200
2	*Annual by-product production (tons)*	*100000*	*100000*	100000
3	*Container capacity (tons)*	*20*	*20*	20
4	*Number of train trips per week*	*1*	*1*	1
5	*Existing silo capacity (tons)*	*100*	*100*	100
6	*Average speed of train (mph)*	*45*	*45*	45
7	*Average speed of truck to transport empty CICs*	*55*	*55*	55
8	*Maximum ash accumulation time (hours)*	*40*	*40*	40
9	*Number of work weeks per year*	*52*	*52*	52
10	*Number of work days per week*	*7*	*7*	7
11	*Number of shifts per day*	*1*	*1*	1
12	*Duration of a shift (minutes)*	*480*	*480*	480
13	*Cycle time for filling one container (minutes)*	*33*	*33*	33
14	*Number of empty CICs delivered per truck trip*	*30*	*30*	30
15				
16	**Continuous flow rate from plant into silo (tpm)**	**0.191**	**0.191**	**0.191**
17	**Weekly residue production (tons)**	**1923**	**1923**	**1923**
18	**Required number of containers per trip**	**96.15**	**96.15**	**96.15**
19	**Final number of containers per trip**	**96**	**96**	**96**
20	**Total number of containers**	**144**	**144**	**144**
21	**Tonnage transported per trip (tons)**	**1923**	**1923**	**1923**
22	**Silo capacity (tons)**	**458**	**458**	**458**
23	**Need to adjust the silo capacity?**	increase	increase	increase
24	**Storage and staging area (3 times the area of containers - sq. ft)**	**22608**	**22608**	**22608**
25	**Total number of containers filled per shift per trailer**	**15**	**15**	**15**
26	**Number of tote trailers needed**	**1**	**1**	**1**
27	**Required number of truck trips per week**	**3**	**3**	**3**
28	**Time available for a round trip (hours)**	**19**	**19**	**19**
29	**Duration of a round trip (hours)**	**1.09090**	**3.63636**	**7.27272**
30	**Number of tractors needed**	**1**	**1**	**1**
31	**Number of trailers needed**	**2**	**2**	**2**
32				
33	**DISPOSAL SITE:**			
34	*Number of working weeks per year*	*52*	*52*	*52*
35	*Working days per week*	*6*	*6*	*6*
36	*Injection time per shift (minutes)*	*360*	*360*	*360*
37	*Shifts per day (100 tph inj. rate is max, so 100*6*6*52=187200t)*	*1*	*1*	*1*
38				
39	**Tote trailer cycle time (minutes)**	**38**	**38**	**38**
40	**Total number of containers transported per shift per trailer**	**9**	**9**	**9**
41	**Number of tote trailers needed**	**2**	**2**	**2**
42	**Storage and staging area (3 times the area of containers - sq. ft)**	**22608**	**22608**	**22608**
43	**Injection rate (tpm)**	**0.890**	**0.890**	**0.890**
44	**Injection system capacity (tph)**	**53**	**53**	**53**

2. Development of a template for the Cylindrical Intermodal Containers (CIT):

In Phase I of the project, templates were developed for Pneumatic Trucks, Pressure Differential Rail Cars, and Collapsible Intermodal Containers technologies as parts of the interactive engineering and economic evaluation software. The intermodal containers were left out due to insufficient information in costing and engineering design. In the beginning of Phase II, more information were obtained for CIT system, and therefore, an operating scenario and a template were developed for this system.

These tanks are made of either steel or aluminum and have a volume of approximately 6400 gallons. They are currently being used in transporting liquids and liquefied gases. The capacity of a tank will be approximately 20 to 25 tons assuming an average density of 60 lbs/ft^3 for coal combustion by-products. Since they are cylindrical, the bridging and sticking problems that occur in rectangular containers when handling powdered material like fly ash, or damp material like scrubber sludge, can be eliminated. These tanks may be mounted in steel frames to facilitate handling as shown in Figure 5.4.

The CIT transportation scenario is shown schematically in Figure 5.5. At the plant, an empty tank will be placed on a trailer with the aid of a piggy packer (a specialized crane) and shuttled to by-product storage silo where it will be filled like a pneumatic truck. Figure 4.6 shows a piggy packer in action, placing a container on a trailer. At the rail siding, the piggy packer will lift the filled tank and stage it along the railroad on a concrete pad. When the train arrives, the piggy packer will lift these tanks again one by one and place them on flat bed rail cars. The length of these rail cars are suitable to handle 3 or 4 of these tanks on one car.

At the mine site, the tanks will be lifted again by a piggy packer and placed on the concrete pad. After the unloading is completed, the same packer will lift the tanks one by one and place them on a trailer to be taken to the injection site. The unloading of the residue into the injection hopper will be done by elevating the head of the tank with the aid of a hydraulic jack mounted on the trailer. The gate of the tank will then be opened and the content transferred into the injection system hopper. If emptying the tank through a flop gate creates unacceptable levels of fugitive dust, they can be designed to be emptied like a pneumatic truck with the aid of a portable blower attached to the hopper of the underground placement system. The empty tanks will be staged at the rail siding and will be waiting for the train to pick them up. After delivering the empty tanks to the plant another cycle will restart.

The template developed for CIT systems is general enough to accommodate any type of intermodal containers or tanks except Collapsible Intermodal Containers for which a separate template was developed in Phase I. The spreadsheet leading to the development of the CIT template is shown in Table 5.5. The case shown in this spreadsheet simulates transportation and handling of 50,000 tons of CCB from a power plant to a mine located 30 miles away. The lines printed in italic represent user defined entries, whereas those in bold letters represent parameters whose values are internally calculated.

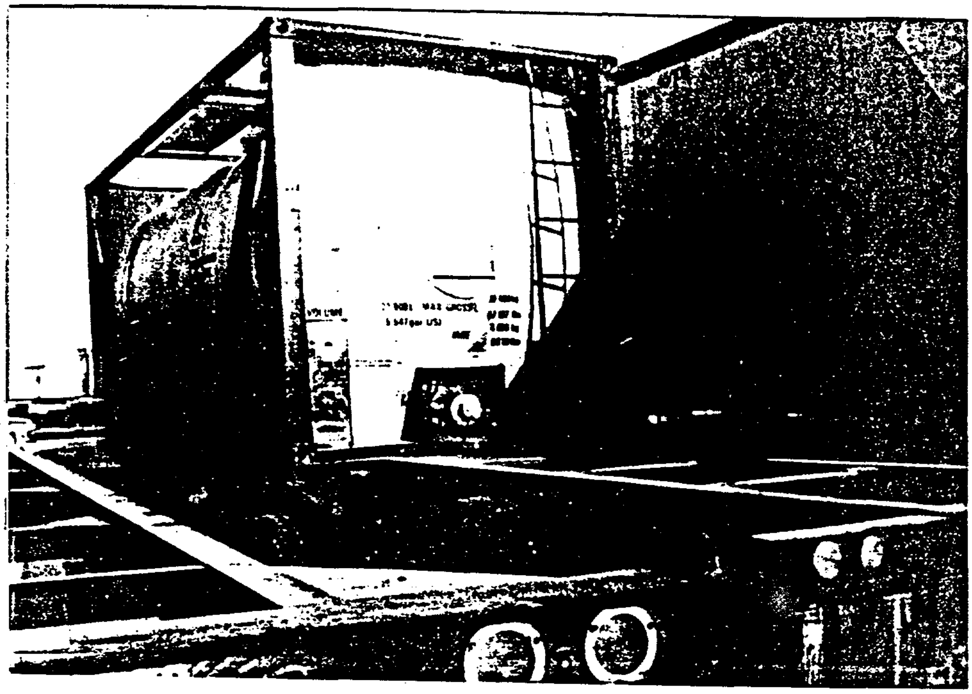

Figure 5.4 A 20 Ton Cylindrical Intermodal Tank Mounted in a Steel Frame and Placed on a Trailer Chassis

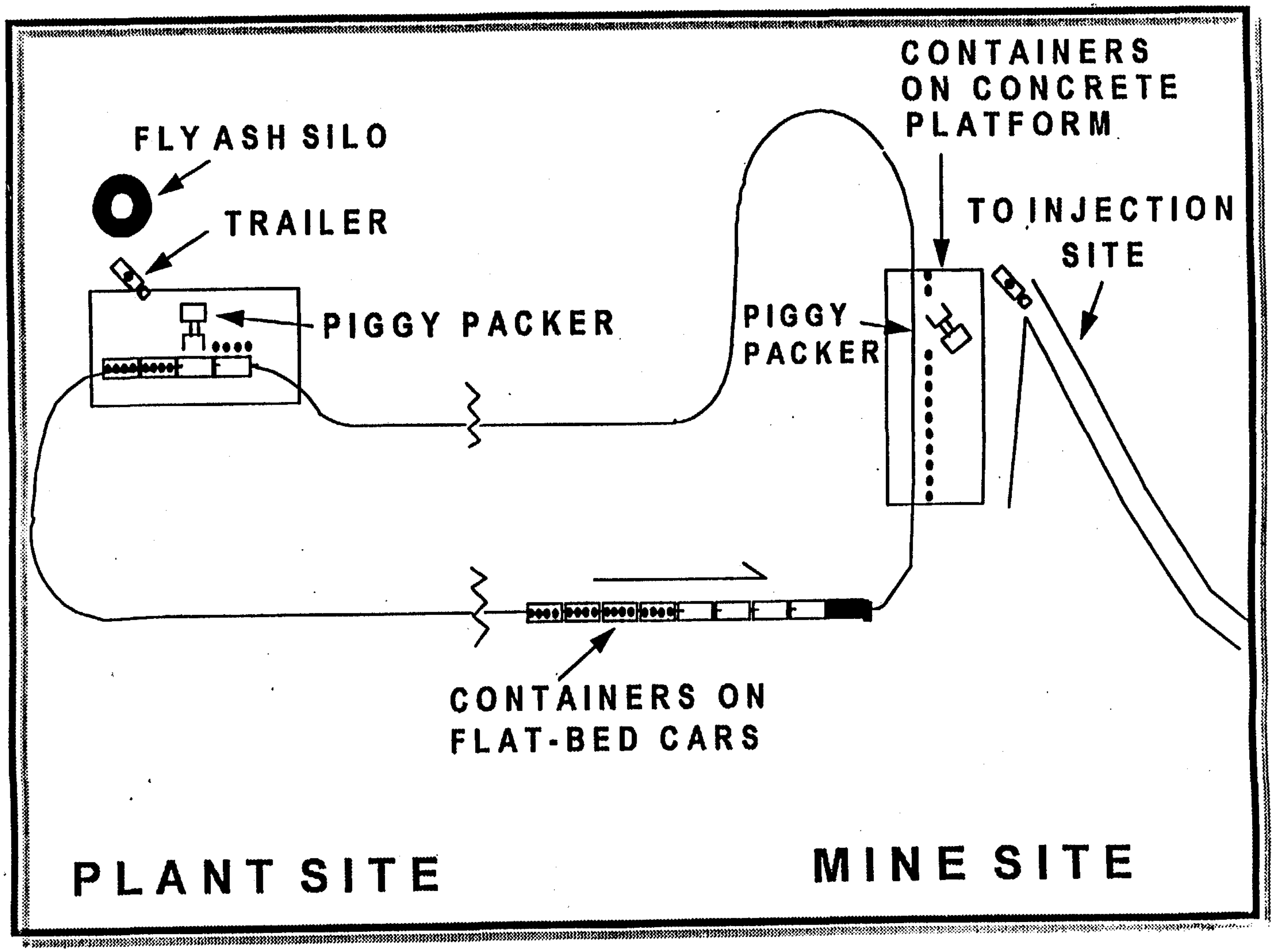

Figure 5.5 Dry FGD By-Products Transported by Cylindrical Intermodal Tanks

Table 5.5. Spreadsheet for Cylindrical Intermodal Tanks leading to the development of its template

GENERAL PARAMETERS	
Distance between power plant and mine (miles)	*30*
Annual amount of by-products to be handled (tons)	*50000*
Project life	*10*
Container capacity (tons)	*20*
Number of containers loaded on a flat-bed car	*4*
Length of a flat-bed car (yds)	*20*
Average depth of the mine (feet)	*300*
CAPITAL COST ITEMS:	
Cost of a container	*4000*
Cost of a tote trailer	*60000*
Cost of a flat-bed rail car	*25000*
Cost of a piggy packer	*250000*
Cost of storage and staging area (15" concrete) ($/sq. ft)	*4.5*
Cost of rail siding ($/yard)	*570*
OPERATING COST ITEMS:	
Railroad charge (from chart in user's manual - $/ton/mile)	*0.04*
Cost of road construction ($/sq. yd)	*5*
Fuel consumption (gal/hour/unit)	*2.5*
Fuel cost ($/gallon)	*1.1*
Hourly wage for equipment operators (including 35 % fringe)	*25*
Insurance cost (% of capital cost - in decimals)	*0.01*
Maintenance cost (% of capital cost - in decimals)	*0.02*
Overhead cost (% of total operating cost - in decimals)	*0.1*
Average cost of injection holes ($/foot)	*20*
Leasing cost of a container ($/unit/yr)	*0*
Leasing cost of a tote trailer ($/unit/yr)	*0*
Leasing cost of a piggy packer ($/unit/yr)	*0*
Leasing cost of a flat-bed rail car ($/unit/yr)	*0*
POWER PLANT:	
Do we need silo (1 for yes, 0 for no)?	*0*
Silo capacity (from engr. design sheet - tons)	*100*
Containers required per trip (from engr. design sheet)	*48*
Number of container sets	*2*
Total number of containers required	**96**
Number of flat-bed cars required	**12**
Length of rail siding (yds)	**300**
Number of working weeks per year (from engr. design sheet)	*52*
Number of working days per week (from engr. design sheet)	*7*
Duration of a shift (from engr. design sheet - hours)	*8*
Number of shifts per day	*1*
Number of tote trailers at the plant (from engr. design sheet)	*1*
Number of piggy packers	*1*

Total number of tote trailer operators	**1**
Total number of piggy packer operators	**1**
Area occupied by one container (sq. ft)	*200*
Storage and staging area (15" concrete) (sq. ft)	**19200**
PLANT CAPITAL COSTS:	
Silo	**0**
Storage and staging area	**86400**
Piggy packer	**250000**
Tote trailers	**60000**
Flat-bed rail cars	**300000**
Rail siding	**171000**
Containers	**384000**
Total plant capital cost	**1251400**
PLANT OPERATING COSTS:	
Leasing cost of containers	**0**
Leasing cost of tote trailers	**0**
Leasing cost of piggy packers	**0**
Leasing cost of flat-bed rail cars	**0**
Railroad charge	**60000**
Insurance	**12514**
Operator wages	**145600**
Fuel	**16016**
Maintenance	**25028**
Subtotal	**259158**
Overhead cost	**25915.8**
Total plant operating cost	**285073.8**
MINE SITE:	
Do we need silo (1 for yes, 0 for no)?	*0*
Silo capacity (tons)	*0*
Number of silo operator per shift	*0*
Area of new road to be constructed (sq. yd)	*7000*
Storage and staging area (15" concrete - sq. ft)	**38400**
Length of rail siding (yds)	*300*
Number of working weeks per year	*52*
Number of working days per week	*6*
Duration of a shift (from engr. design sheet - hours)	*8*
Number of shifts per day (from engr. design sheet)	*1*
Number of tote trailers (from engr. design sheet)	*1*
Number of piggy packers	*1*
Total number of tote trailer operators	*1*
Total number of piggy packer operators	*1*
Total number of silo operators	*0*
Expected amount of by-product injected through one hole (tons)	*10000*
Number of injection holes to be drilled per year	*5*
Do we have hydraulic injection system (1 for yes, 0 for no)?	*0*
Do we have pneumatic injection system (1 for yes, 0 for no)?	*1*

Injection system capacity (from engr. design sheet)	*30*
Operating cost of hydraulic injection system ($/ton)	**0**
Operating cost of pneumatic injection system ($/ton)	**2.08**
Injection system operating cost ($/ton)	**2.08**
MINE CAPITAL COSTS:	
Storage and staging area	**172800**
Rail siding	**171000**
Silo	**0**
Piggy packer	**250000**
Tote trailers	**60000**
Hydraulic injection system	**0**
Pneumatic injection system	**113333.0**
Total mine capital cost	**767133.0**
MINE OPERATING COSTS:	
Leasing cost of piggy packer(s)	**0**
Leasing cost of tote trailer(s)	**0**
Road construction	**35000**
Insurance	**7671.330**
Operators wage	**124800**
Fuel	**13728**
Maintenance	**15342.66**
Cost of injection holes	**30000**
Injection system operating cost	**104168**
Subtotal	**330709**
Overhead cost	**33070.9**
Total mine operating cost	**363779.9**
TOTAL CAPITAL COST	**2018533.0**
TOTAL OPERATING COST	**648854**
Operating cost per ton	**12.98**

Figure 5.6 A Piggy Packer Placing a Container on a Trailer

The software that was envisaged for this research project was completed after the addition of the CIT module and the development of the engineering design spreadsheets. Figure 5.7 shows the software developed for all four T&H technologies. As seen, the evaluation of any of the selected T&H technologies is done in three steps:

1. Engineering design computations,
2. Capital and operating cost computations,
3. Economic evaluation.

The economic evaluation software is common to all technologies. The windows application was developed for the second and third steps covering the cost computations and economic evaluation.

3. Revision of the software to accommodate underground placement systems and other modifications:

In the beginning of Phase II, capital and operating costs for both pneumatic and hydraulic underground placement systems became available due to progress in placement system research. Although the cost estimations were relatively crude, they were satisfactory to warrant incorporation of the underground placement modules into the templates of the software. It should be noted that, the addition of these modules, as well as the separation of the engineering design computations mentioned under activity 1 above, necessitated substantial restructuring of each template.

The capital and operating costs of the pneumatic and hydraulic underground placement systems were expressed as a function of the system capacity. These functions were obtained by finding the best fit to the cost estimations obtained from the researchers in charge of developing the placement systems. Specifically, the following equations were obtained:

- For the capital cost of the pneumatic placement system:

$$y = 113{,}333 - 200\,x + 6.67\,x^2 \quad \text{.......................... Eq. (4.1)}$$

where:

y is the capital cost in \$
x is the system capacity in tph.

- For example, an underground placement system with 50 tph capacity will cost:

$$y = 113{,}333 - 200\,(50) + 6.67\,(50)^2 = \$120{,}000$$

- For the operating cost of the pneumatic placement system:

$$y = 2.0833 + 0.0025\,x - 0.0000833\,x^2 \quad \text{.................. Eq. (4.2)}$$

where:

y is the operating cost in \$/ton
x is the system capacity in tph

- For example, the operation of the underground placement system with 50 tph capacity will cost:

$$y = 2.0833 + 0.0025\ (50) - 0.000083\ (50)^2 = \$2.0/\text{ton}$$

- For the capital cost of the hydraulic placement system:

$$y = 16{,}667 + 5500\ x - 16.67\ x^2 \quad \text{.............................} \quad \text{Eq. (4.3)}$$

where x and y are defined as in Eq. (4.1)

- For the operating cost of the hydraulic placement system:

$$y = 1.5833 + 0.0025\ x - 0.0000833\ x^2 \quad \text{....................} \quad \text{Eq. (4.4)}$$

where x and y are defined as in Eq. (4.2)

4. Evaluation of hypothetical cases using the modified software:

The nine hypothetical cases formed by the combinations of three production rates and three distances, as given in the 8^{th} activity of Phase I, had to be re-evaluated using the modified software in order to determine the favorable ranges of each technology in terms of distance and tonnage. As an improvement to Phase I evaluations, where only an overall transportation cost was determined for each case, in Phase II evaluations this cost is broken down in its two components: 1) the cost of transportation and handling from the power plant to the mine site (primary transportation and handling), and 2) the cost of transportation and handling from the mine site to the underground placement site (secondary transportation and handling). Furthermore, the underground placement cost is also determined for each tonnage considered in the cases.

The results of the runs for 50,000 annual production for all four technologies are shown in Table 5.6. It is noted that there is no secondary transportation and handling cost in PT technology since the trucks can drive directly from the power plant to the underground placement site. The distance of transportation from the mine site to the underground placement site is assumed to be within 1 mile for all cases.

The placement cost is shown as a dip note to Table 5.6. This cost is the same for all transportation technologies since the capacity of the system is a function of only the tonnage. The placement cost is calculated based on the assumption that 10,000 tons of CCB can be injected through a borehole. A better estimate of this tonnage will be available after the placement demonstration at the Peabody No. 10 mine as planned in the DOE-SIUC cooperative agreement.

Also, it is important to note that the capital and operating cost estimates of the underground placement system are the first time estimates, and the researches are currently working on the improvement of the system as well as on the cost estimates. Therefore, a warning is in order in the interpretation of the placement cost given in this chapter. The evaluations of the systems will be repeated in Phase III using the updated cost and engineering data.

Table 5.6 Cost of transporting 50,000 tons of CCB annually from power plant to mine and underground placement sites using four different technologies ($/ton)

Operation	Distance (miles)	TECHNOLOGIES			
		PD-car	PT	CIC	CIT
primary transportation and handling	30 100 200	5.84 7.41 9.77	4.11 9.34 16.92	9.88 10.28 11.05	6.15 7.72 10.09
secondary transportation and handling	1	3.16	-	2.09	3.36
total transportation and handling	30 100 200	9.00 10.57 12.93	4.11 9.34 16.92	11.97 12.37 13.14	9.51 11.08 13.45

** Underground placement cost: $2.48/ton

For 50,000 tons, the total costs of product delivery to the underground placement system for varying distances of transportation are shown in Figure 5.8. As seen, the PT technology gave lower costs than the PD-car and CIT technologies up to approximately 70 miles, and lower than the CIC technology up to 110 miles. The PD-car and the CIT technologies indicated almost the same cost for all distances, and they remained lower than the costs of CIC technology for all distances, though the difference was significantly reduced at 200 miles. The steeper slope of the PT cost line in Figures 5.8 is due to longer cycle time of trucks as distance increases, which imposes additional units for the fleet in order to handle the same 50,000 tons annual by-product.

To show the steps involved in obtaining the "cost-per-ton" figures exhibited in Table 5.6, the outputs of the runs made for Pneumatic Truck system for transporting 50,000 tons to a mine located 100 miles away from the power plant are given as an example. The results corresponding to the engineering design computations, the cost and scheduling computations, and the economic evaluation are shown in tables 4.7, 4.8, and 4.9, respectively. The $11.82/ton after-tax cost reported at the bottom of Table 4.9 corresponds to the delivery of CCB from the power plant to the underground placement site.

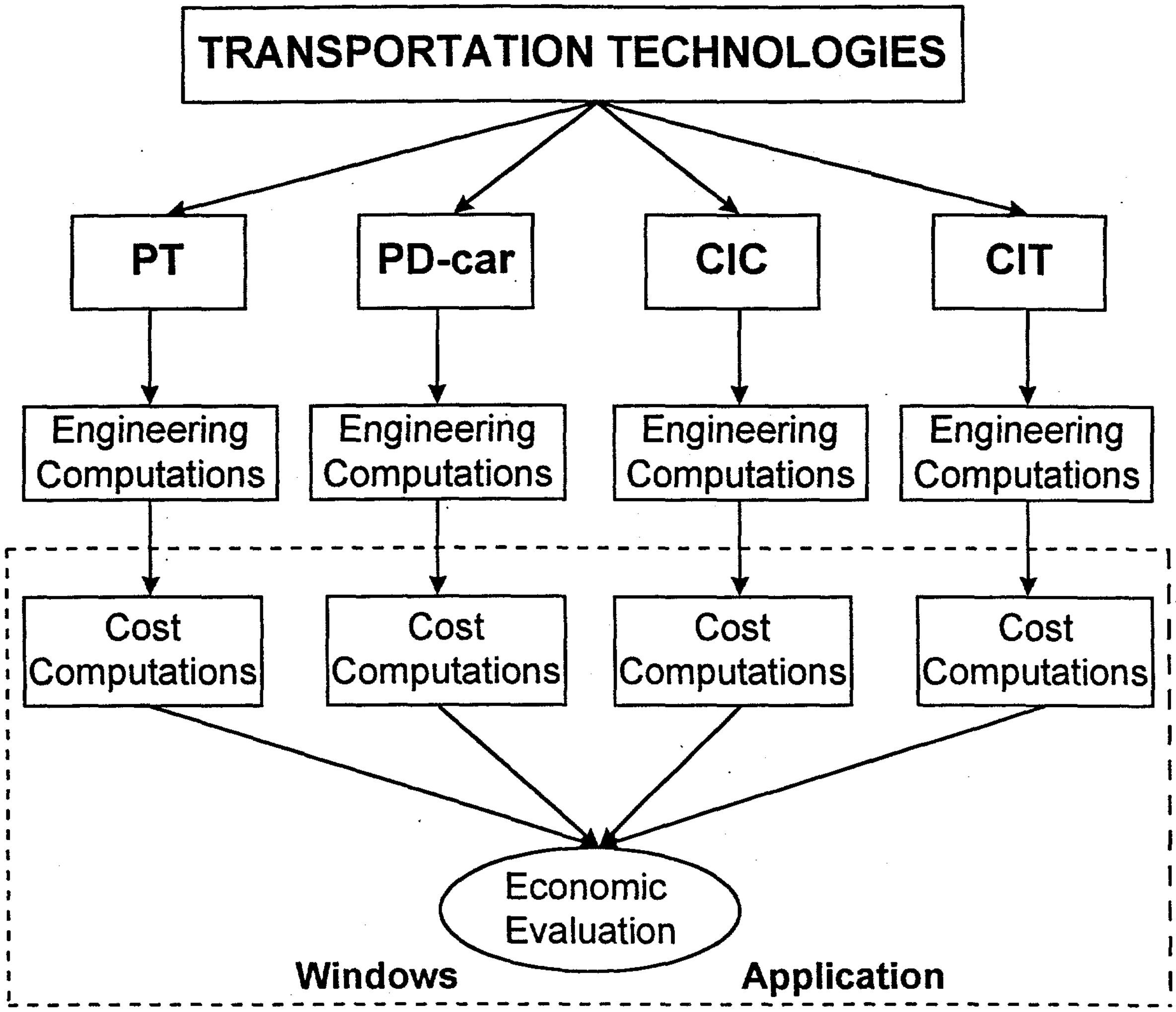

Figure 5.7 Software Developed for the Selected Transportation Technologies

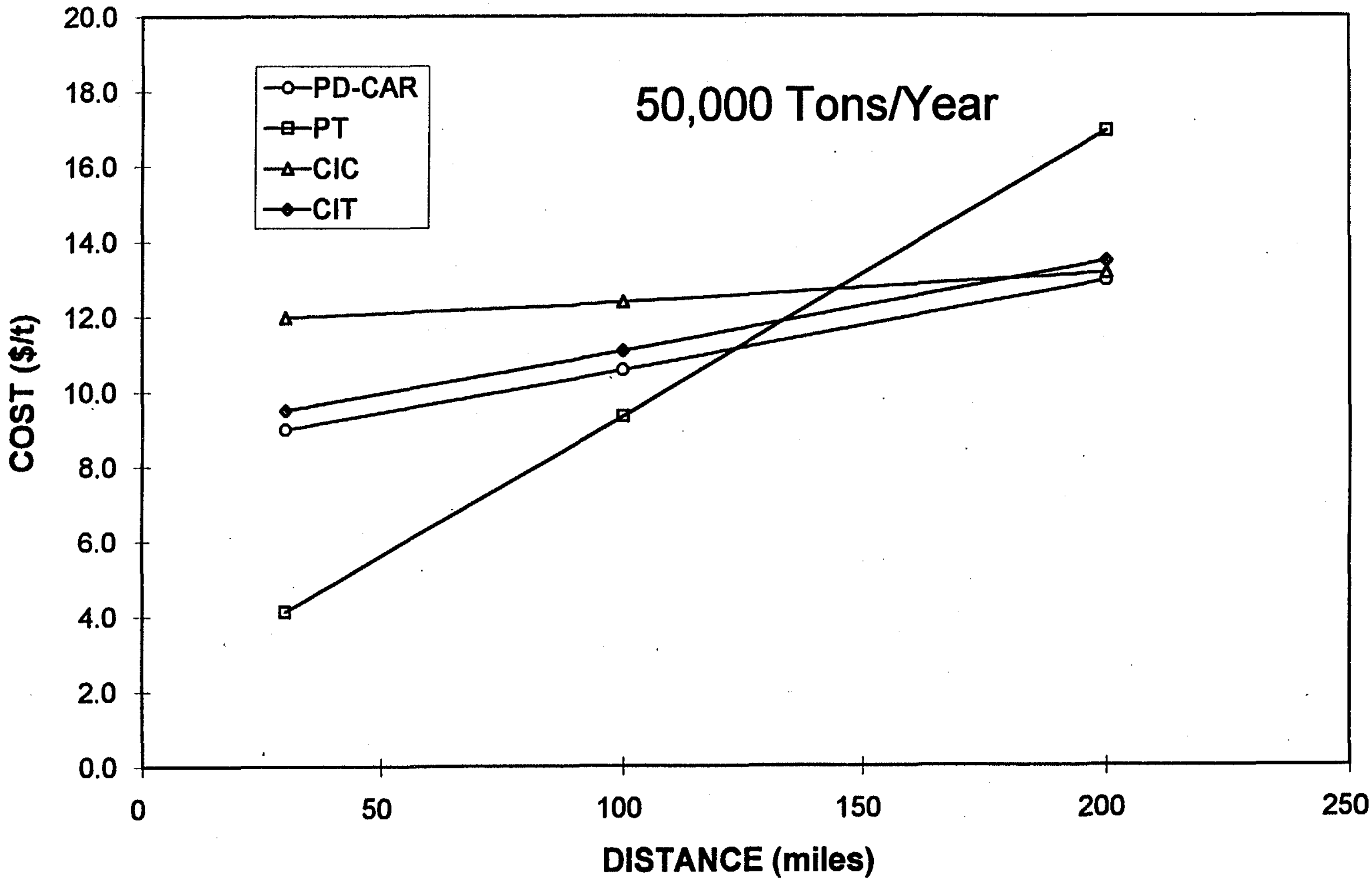

Figure 5.8 Cost of Transporting 50,000 tons of CCB Annually from Power Plants to Underground Placement Site Using Four Different Technologies

Table 5.7 Engineering design computations in CCB transportation by pneumatic trucks (50,000 tons per year)

1	*Annual Production (tons)*	*50000*
2	*Distance (miles)*	*100*
3	*Shifts per day*	*1*
4		
5	*Loading time (min)*	*10*
6	*Average speed loaded (mph)*	*50*
7	*Average speed empty (mph)*	*50*
8	*Unloading time (min)*	*25*
9	*Duration of a shift (min)*	*480*
10	*Work days per year*	*312*
11	*Truck payload (tons)*	*20*
12		
13	**Travel time loaded (min)**	**120.0**
14	**Travel time empty (min)**	**120.0**
15	**Cycle time (min)**	**275.0**
16	**Maximum number of trucks without queuing**	**11.00**
17	**Required number of trips per year**	**2500**
18	**Required number of trips per day**	**8.01**
19	**Maximum number of trips per truck per shift**	**1.75**
20	**Required number of trucks**	**4.59**
21	**Injection rate (tpm)**	**0.80**
22	**Injection system capacity (tph)**	**48**
23	**Feasible scenario?**	**Yes**
24	**Max. ash accumulation time (hrs)**	**40**
25	**Max. ash accumulation (tons)**	**228**

Table 5.8 Capital and Operating Cost Computations in CCB Transportation by Pneumatic Trucks (50,000 Tons Annually, 100 miles, Including Pneumatic Placement System)

GENERAL PARAMETERS:	
Distance between power plant and mine (miles)	= 100
Annual amount of by-product to be handled (tons)	= 50000
Project life (years)	= 10
Truck capacity (tons)	= 20
Availability of a truck (in decimal)	= 0.9
Number of front tires	= 10
Number of rear tires	= 8
Front tire replacement frequency (miles)	= 75000
Rear tire replacement frequency (miles)	= 150000
Average depth of the mine (feet)	= 300
CAPITAL COST ITEMS:	
Cost of a tractor	= 70000
Cost of a tank trailer	= 45000
Cost of a blower	= 5000
Total cost of a truck (tractor-trailer-blower)	= 120000
OPERATING COST ITEMS:	
Truck leasing cost ($/year)	= 0
Fuel consumption (miles per gallon)	= 5
Fuel cost ($/gallon)	= 1.1
Insurance cost per truck	= 7500
Licence cost per truck	= 2222
Annual highway tax per truck	= 550
Tire cost (each)	= 325
Maintenance cost ($/mile)	= 0.14
Wage for truck drivers ($/hr)	= 20
Wage for silo operator ($/hr)	= 20
Overhead cost (% of other operating costs - in decimal)	= 0.1
Cost of road construction ($/sq. yd)	= 5
Average cost of injection hole ($/ft)	= 20
POWER PLANT:	
Do we need silo (1 for yes, 0 for no)?	= 0
Silo capacity (from prescheduling sheet - tons)	= 867
Truck cycle time (from prescheduling sheet - minutes)	= 275
Number of working days per year (from prescheduling sheet)	= 312
Number of shifts per day (from prescheduling sheet)	= 1
Duration of a shift (from prescheduling sheet - hours)	= 8
Number of trips per day (from prescheduling sheet)	= 8.01
Number of trucks required (from prescheduling sheet)	= 4.59
Number of trucks required (rounded)	= 5
Number of trucks to have at least 80% fleet availability	= 6
Actual fleet availability	= 0.89
Number of drivers required	= 5
Number of silo operators per shift	= 1
Total number of silo operators	= 1
PLANT CAPITAL COSTS:	
Trucks	= 720000
Silo	= 0
Total plant capital cost	= 720000

Table 5.8 Continued

PLANT OPERATING COSTS:

Leasing cost for pneumatic trucks	= 0
Fuel	= 109961.28
Insurance	= 45000
Licence for tractor-trailer	= 13332
Highway tax	= 3300
Front tire replacement	= 21659
Rear tire replacement	= 8663
Maintenance	= 69975.36
Driver wages	= 249600
Silo operators wages	= 49920
Subtotal	= 571410.64
Overhead cost	= 57141.06
Total plant operating cost	= 628551.7

MINE SITE:

Do we have silo (1 for yes, 0 for no)	= 0
Silo capacity (tons)	= 0
Area of new road to be constructed (sq. yd/year)	= 7000
Expected amount of by-product injected through one hole (tons)	= 10000
Number of injection holes to be drilled per year	= 5
Injection system capacity (from prescheduling sheet, tph)	= 50
Do we have hydraulic injection system (1 for yes, 0 for no)?	= 0
Do we have pneumatic injection system (1 for yes, 0 for no)?	= 1
Operating cost of hydraulic injection system ($/ton)	= 0
Operating cost of pneumatic injection system ($/ton)	= 2
Injection system operating cost ($/ton)	= 2

MINE CAPITAL COSTS:

Hydraulic injection system	= 0
Pneumatic injection system	= 119999.75
Silo	= 0
Total mine capital cost	= 119999.75

MINE OPERATING COSTS:

Road construction cost	= 35000
Injection system operating cost	= 100000
Cost of injection holes	= 30000
Subtotal	= 165000
Overhead cost	= 16500
Total mine operating cost	= 181500

TOTAL SYSTEM CAPITAL COST	= 839999.75
TOTAL SYSTEM OPERATING COST	= 810051.7

Operating cost per ton	= 16.2

Table 5.9 Economic evaluation of CCB transportation by pneumatic trucks (50,000 tons per year, 100 miles, including pneumatic placement system)

```
DISTANCE (MILES) ------------------------------ = 100
PRODUCTION (TONS/YEAR) ------------------------ = 50000
MINIMUM REQUIRED RATE OF RETURN (%) ------------ = 12
EFFECTIVE TAX RATE (%) ------------------------ = 40
```

Year	0	1	2	3	4	5
Revenue.........	0	0	0	0	0	0
-Oper. Cost.....	0	-810	-810	-810	-810	-810
-Depreciation...	0	-168	-269	-161	-97	-97
Taxable Income..	0	-978	-1079	-971	-907	-907
-Tax...........	0	391	432	389	363	363
Net Income......	0	-587	-647	-583	-544	-544
+Depreciation...	0	168	269	161	97	97
-Capital Cost...	-840	0	0	0	0	0
Net Cash Flow...	-840	-419	-378	-421	-447	-447

Year	6	7	8	9	10
Revenue.........	0	0	0	0	0
-Oper. Cost.....	-810	-810	-810	-810	-810
-Depreciation...	-48	0	0	0	0
Taxable Income..	-858	-810	-810	-810	-810
-Tax...........	343	324	324	324	324
Net Income......	-515	-486	-486	-486	-486
+Depreciation...	48	0	0	0	0
-Capital Cost...	0	0	0	0	0
Net Cash Flow...	-467	-486	-486	-486	-486

CAPITAL GAIN (OR LOSS) COMPUTATION ($1,000)

Salvage Value..................................	0
Book Value.....................................	0
Capital Gain (or Loss).........................	0
Tax Liability..................................	0
After-Tax Capital Gain (or Loss)...............	0
Book Value.....................................	0
After-Tax Cash Flow Due to Cap. Gain (or Loss)..	0

```
AFTER-TAX NET PRESENT VALUE ($1,000) ........=      -3338
AFTER-TAX ANNUAL EQUIVALENT COST ($1,000) ...=       -591
AFTER-TAX COST PER TON ($) .................=      -11.82
BEFORE-TAX PRICE TO BE CHARGED ($/TON) ......=      19.69
```

The results of the runs for 100,000 tons of annual production for all four technologies are given in Table 5.10. The total costs of product delivery to the underground placement system for varying distances of transportation are shown in Figure 5.9. As seen, the PT technology gave lower costs than the CIT technology up to a distance of approximately 45 miles, and those of the PD-car and CIC technologies up to a distance of 65 miles. The cost line of the CIT technology remained parallel and slightly below the PD-car. The advantage of these two latter technologies over the CIC technology disappeared after approximately 100 miles. This phenomenon is explained by the fact that the CIC technology is benefiting from the backhaul rail charge which is assumed to be substantially lower than the fronthaul charge. If this assumption is not correct the advantage of the CIC technology in distances longer than 120 miles will not hold.

The results of the runs for 200,000 tons annual production are shown in Table 5.11. The total costs of product delivery to the underground placement system are shown in Table 5.11. The total costs of product delivery to the underground placement system for varying distances of transportation are shown in Figure 5.10. As seen, the advantage of the PT technology over the others disappeared at 200,000 tons annual production. The difference among the other three technologies were insignificant up to approximately 140 miles, after which the CIC became better, again due to assumed low backhaul cost.

When the cost lines are compared simultaneously in Figures 5.8, 5.9, and 5.10, it is seen that significant cost reductions occur in PD-car, CIT, and CIC technologies as the annual tonnage increases. This, however, is not true for the PT technology, which remains almost unchanged for all production rates. This is an indication that the economies of scale is favoring all three technologies, except the PT technology. Also, in all figures, the slopes of the cost lines remain relatively flat, except the ones for the PT technology. Therefore, we can conclude from these observations that the PT technology is sensitive to distance but insensitive to tonnage, whereas the opposite is true for the other three technologies.

Table 5.10 Cost of transporting 100,000 tons of CCB annually from power plant to mine and underground placement sites using four different technologies ($/ton)

Operation	Distance (miles)	TECHNOLOGIES			
		PD-car	PT	CIC	CIT
primary transportation and handling	30	5.55	3.57	6.70	4.61
	100	7.12	9.01	7.09	6.17
	200	9.48	16.38	7.68	8.54
secondary transportation and handling	1	2.16	-	1.55	2.47
total transportation and handling	30	7.71	3.57	8.25	7.08
	100	9.28	9.01	8.64	8.64
	200	11.64	16.38	9.23	11.01

** Underground placement cost: $2.33/ton

Table 5.11 Cost of transporting 200,000 tons of CCB annually from power plant to mine and underground placement sites using four different technologies ($/ton)

Operation	Distance (miles)	TECHNOLOGIES			
		PD-car	PT	CIC	CIT
primary transportation and handling	30	3.20	3.03	4.00	2.96
	100	4.66	7.71	4.37	4.42
	200	6.87	14.09	4.92	6.63
secondary transportation and handling	1	1.80	-	1.29	1.99
total transportation and handling	30	5.00	3.03	5.29	4.95
	100	6.46	7.71	5.66	6.41
	200	8.67	14.09	6.21	8.62

** Underground placement cost: $2.25/ton

5. Evaluation of a case study with the modified software:

Under the cooperative research program, Peabody No. 10 mine in Pawnee, Illinois, is the site for underground placement demonstration. To enhance this demonstration, it is decided to use the developed software to conduct an engineering design and economic analysis specifically for this site.

Peabody No. 10 mine, located approximately 20 miles south of Springfield, Illinois, was opened in the 1950s and operated until August, 1994. Room-and-pillar mining method had been employed utilizing continuous miners to mine Herrin (No. 6) coal seam in this mine. The mine is about 350 feet deep with a seam thickness varying from 6 ft to 8 ft.

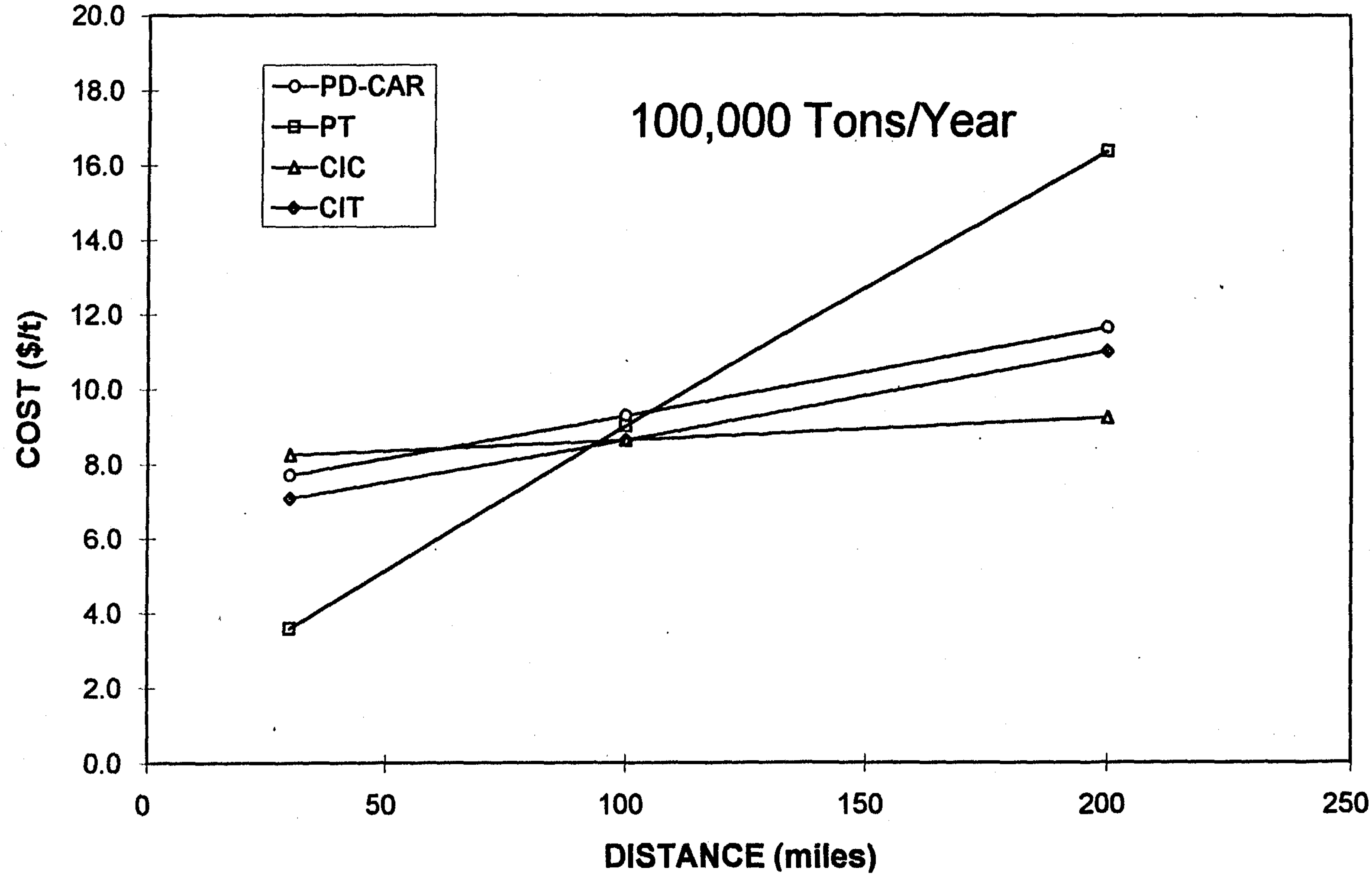

Figure 5.9 Cost of Transporting 100,000 tons of CCB Annually from Power Plants to Underground Placement Site Using Four Different Technologies

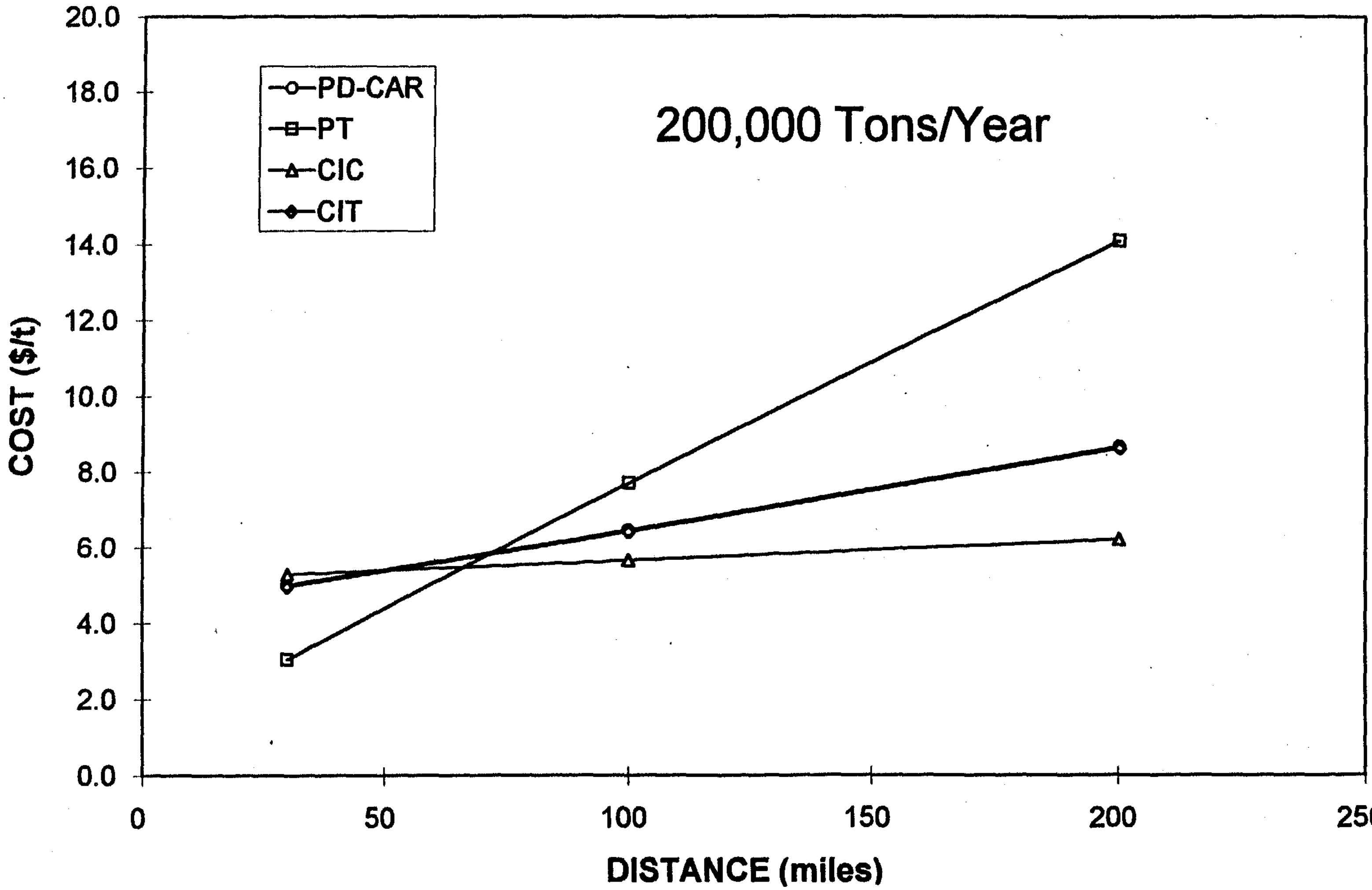

Figure 5.10 Cost of Transporting 200,000 tons of CCB Annually from Power Plants to Underground Placement Site Using Four Different Technologies

First, hydraulic placement study was conducted. It is assumed that scrubber sludge was brought from the Dallman plant by rear dump trucks, and fly ash by pneumatic trucks. First, scrubber sludge was dumped into the hopper of the hydraulic injection system. Then, fly ash and lime are added at predetermined percentages, and mixed thoroughly with sludge to form high density paste before injecting it into the mine.

The outcomes of the economic evaluation is shown in Table 5.12. The initial investment and all ten years net cash flows are shown in this table. As seen, these net cash flows are all negative because this is not a revenue generating project. The capital gain (or loss) computation which takes place at the end of the project life is shown at the bottom of this table. The last four lines summarize the project economics. The after-tax cost is found to be \$5.98 per ton. It is noted that this cost will be absorbed by either the power plant or the coal company if the transportation, handling, and placement operation is handled by either of them or in cooperation. On the other hand, if a contractor is hired to undertake the project, the price that the contractor would charge would be \$9.96 per ton, as seen in the last line, assuming that the contractor's minimum required rate of return is 12 %.

A separate run was made using the developed model to isolate the cost of transportation from that of the underground placement. The outcomes of the economic evaluation excluding placement system is shown in Table 5.13. As seen at the bottom of this table, the after-tax cost per ton of by-product delivered from the Dallman plant to Peabody No. 10 mine is \$3.29. The placement cost is, therefore, the difference between \$5.98 and \$3.29, which amounts to \$2.69 per ton.

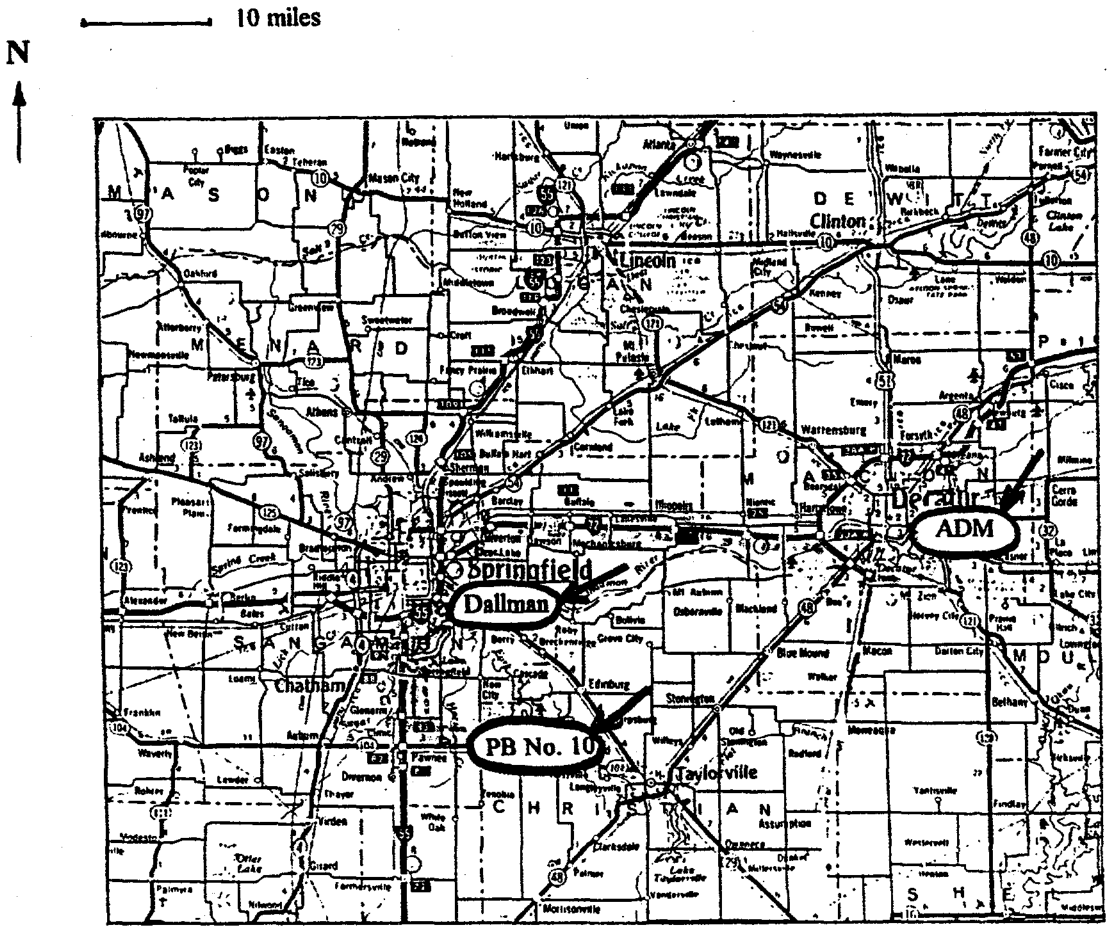

Figure 5.11 Location of Peabody No. 10 Mine with Respect to ADM and Dallman Pla

Table 5.12 Economic evaluation of CCB transportation from CWLP-Dallman plant to Peabody No. 10 mine (including hydraulic placement system)

```
DISTANCE (MILES) ------------------------------ = 25
PRODUCTION (TONS/YEAR) ------------------------ = 100000
MINIMUM REQUIRED RATE OF RETURN (%) ------------ = 12
EFFECTIVE TAX RATE (%) ------------------------ = 40
```

Year	0	1	2	3	4	5
Revenue.........	0	0	0	0	0	0
-Oper. Cost.....	0	-844	-844	-844	-844	-844
-Depreciation...	0	-150	-240	-144	-86	-86
Taxable Income..	0	-994	-1084	-988	-930	-930
-Tax............	0	398	434	395	372	372
Net Income......	0	-596	-650	-593	-558	-558
+Depreciation...	0	150	240	144	86	86
-Capital Cost...	-750	0	0	0	0	0
Net Cash Flow...	-750	-446	-410	-449	-472	-472

Year	6	7	8	9	10
Revenue.........	0	0	0	0	0
-Oper. Cost.....	-844	-844	-844	-844	-844
-Depreciation...	-43	0	0	0	0
Taxable Income..	-887	-844	-844	-844	-844
-Tax............	355	338	338	338	338
Net Income......	-532	-506	-506	-506	-506
+Depreciation...	43	0	0	0	0
-Capital Cost...	0	0	0	0	0
Net Cash Flow...	-489	-506	-506	-506	-506

CAPITAL GAIN (OR LOSS) COMPUTATION ($1,000)

Salvage Value..................................	68
Book Value.....................................	0
Capital Gain (or Loss).........................	68
Tax Liability..................................	-27
After-Tax Capital Gain (or Loss)...............	41
Book Value.....................................	0
After-Tax Cash Flow Due to Cap. Gain (or Loss)..	41

```
AFTER-TAX NET PRESENT VALUE ($1,000) ........=      -3377
AFTER-TAX ANNUAL EQUIVALENT COST ($1,000) ...=       -598
AFTER-TAX COST PER TON ($) .................=       -5.98
BEFORE-TAX PRICE TO BE CHARGED ($/TON) ......=       9.96
```

Table 5.13 Economic evaluation of CCB transportation from CWLP-Dallman plant to Peabody No. 10 mine (excluding hydraulic placement system)

DISTANCE (MILES) ------------------------------ = 25
PRODUCTION (TONS/YEAR) ------------------------ = 100000
MINIMUM REQUIRED RATE OF RETURN (%) ----------- = 12
EFFECTIVE TAX RATE (%) ------------------------ = 40

Year	0	1	2	3	4	5
Revenue.........	0	0	0	0	0	0
-Oper. Cost.....	0	-447	-447	-447	-447	-447
-Depreciation...	0	-100	-160	-96	-58	-58
Taxable Income..	0	-547	-607	-543	-505	-505
-Tax............	0	219	243	217	202	202
Net Income......	0	-328	-364	-326	-303	-303
+Depreciation...	0	100	160	96	58	58
-Capital Cost...	-500	0	0	0	0	0
Net Cash Flow...	-500	-228	-204	-230	-245	-245

Year	6	7	8	9	10
Revenue.........	0	0	0	0	0
-Oper. Cost.....	-447	-447	-447	-447	-447
-Depreciation...	-29	0	0	0	0
Taxable Income..	-476	-447	-447	-447	-447
-Tax............	190	179	179	179	179
Net Income......	-285	-268	-268	-268	-268
+Depreciation...	29	0	0	0	0
-Capital Cost...	0	0	0	0	0
Net Cash Flow...	-257	-268	-268	-268	-268

CAPITAL GAIN (OR LOSS) COMPUTATION ($1,000)

Salvage Value..................................	45
Book Value.....................................	0
Capital Gain (or Loss).........................	45
Tax Liability..................................	-18
After-Tax Capital Gain (or Loss)...............	27
Book Value.....................................	0
After-Tax Cash Flow Due to Cap. Gain (or Loss)..	27

AFTER-TAX NET PRESENT VALUE ($1,000) ....= -1859
AFTER-TAX ANNUAL EQUIVALENT COST ($1,000) ...= -329
AFTER-TAX COST PER TON ($) ....= -3.29
BEFORE-TAX PRICE TO BE CHARGED ($/TON) ......= 5.48

To reveal the effect of economies of scale on the cost of by-product transportation and underground placement in this case study, a sensitivity analysis was conducted on the annual production. Besides the 100,000-ton per year base case, two more runs were conducted for 50,000 and 150,000-ton per year, respectively. Furthermore, to isolate the transportation cost from that of the injection, "only transportation" scenarios were run for both production rates as it was done for 100,000 tons base case. The costs are summarized in Table 5.14. The numbers in parentheses reflect the "before-tax price to be charged" should the operation be undertaken by a contractor. The cost-per-ton for each production rate is plotted in Figure 5.12. As seen, the cost decreases as the production rate increases. The injection cost at any production rate can be read as the difference between the two curves.

Table 5.14 Cost of CCB transportation from Dallman plant to Peabody No. 10 mine

Tonnage (tons/year)	Cost with placement ($/ton)	Cost without placement ($/ton)
50,000	6.58 (10.96) **	3.81 (6.35)
100,000	5.98 (9.96)	3.29 (5.48)
150,000	5.03 (8.38)	2.77 (4.62)

** Numbers in parentheses represent "price to be charged per ton of CCB"

The above procedures of system evaluation were also followed for pneumatic placement case. It is assumed that the FBC fly ash and spent-bed ash will be delivered from the ADM plant to Peabody No. 10 in pneumatic trucks of approximately 20-ton capacity. The content of a truck will be directly transloaded in to the hopper of the pneumatic injection system and subsequently pumped into the old mine workings.

Again, as in the Dallman case, a sensitivity analysis was conducted on the annual production by varying the base case of 100,000 tons by ±50,000 tons. Also, to isolate the transportation cost from that of the injection, "only transportation" scenarios were run for both production rates as it was done for 100,000 tons base case. The costs are summarized in Table 5.15 and plotted in Figure 5.13. The slight upward trend when production increases from 100,000 tons to 150,000 tons is due to the increase in the truck fleet size from 8 to 13 trucks. It is noted that the fleet size in Dallman case went up from 5 to 6 only due to transportation distance of only 25 miles.

Table 5.15 Cost of CCB transportation from ADM plant to Peabody No. 10 mine

Tonnage (tons/year)	Cost with placement ($/ton)	Cost without placement ($/ton)
50,000	9.08 (15.13) **	6.52 (10.86)
100,000	8.13 (13.55)	5.77 (9.62)
150,000	8.31 (13.85)	6.02 (10.03)

** Numbers in parentheses represent "price to be charged per ton of CCB"

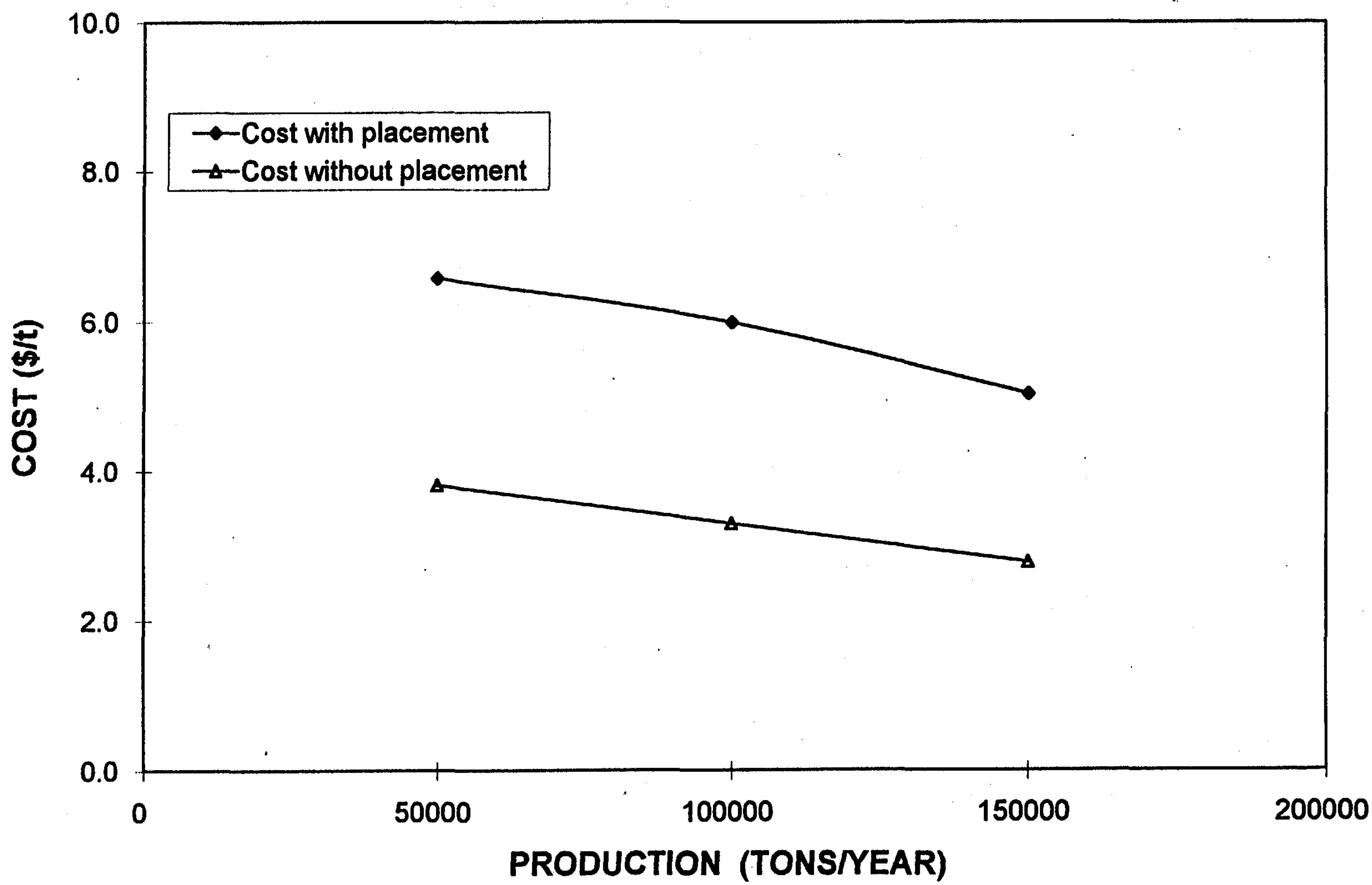

Figure 5.12 Cost of By-Products Transportation from Dallman Plant to Peabody No. 10 Mine

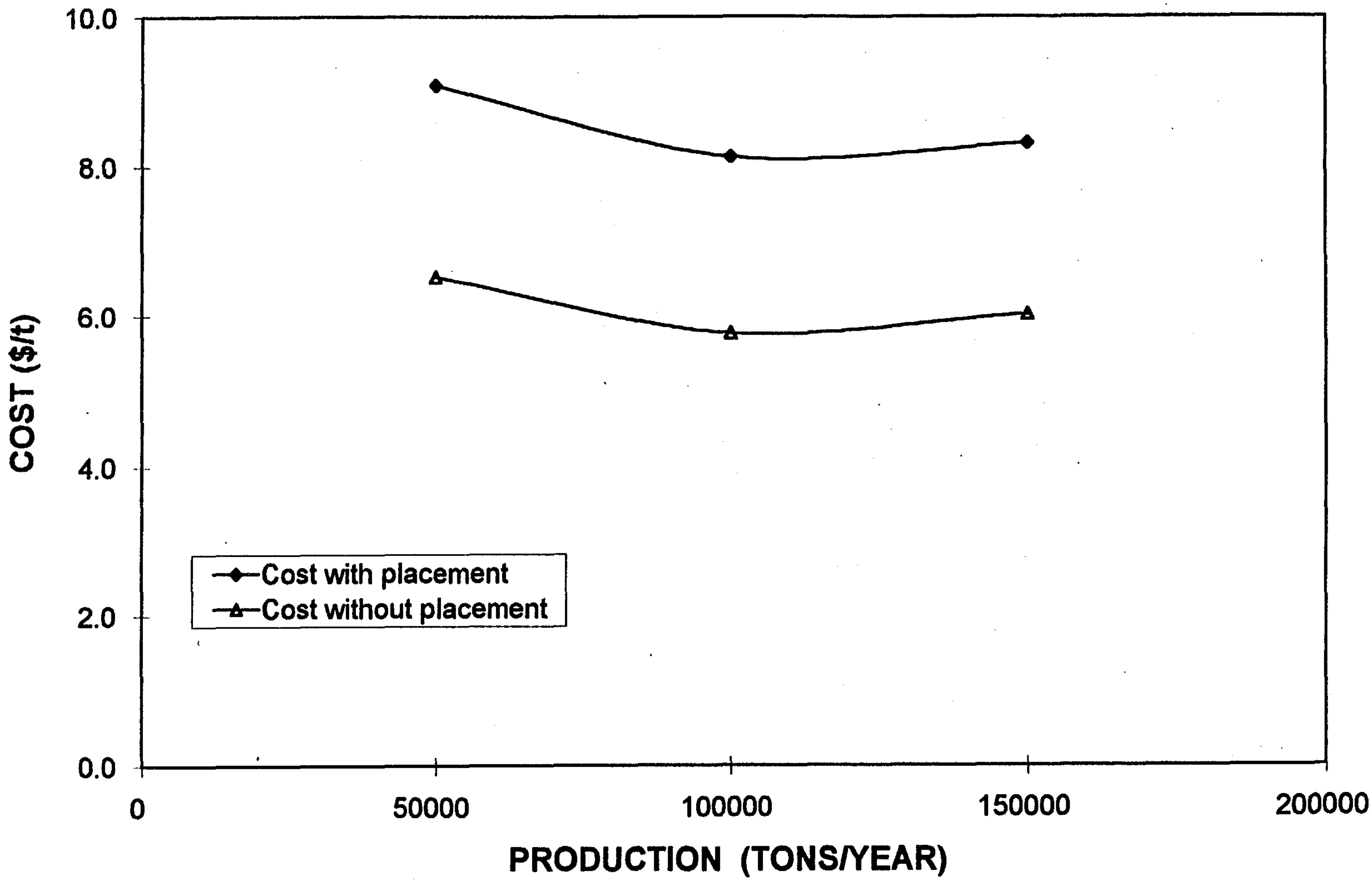

Figure 5.13 Cost of By-Products Transportation from ADM Plant to Peabody No. 10 Mine

6. Publication and presentation of two technical papers summarizing the research findings:

During Phase II, two technical papers summarizing the research results were published and presented (Sevim, 1996; Sevim 1997).

7. Completion of a Topical Report:

One of the deliverables of the Materials Handling and System Economics research was a "topical report" covering all the tasks accomplished during Phase I and Phase II of the cooperative agreement. This report has been finalized and will be submitted to USDOE in June, 1997.

5.6 Planned Activities for Phase III

In Phase III, the following activities are planned for materials handling and system economics research:

1. Work with the researchers in charge of the underground placement demonstration to update the capital and operating cost estimates of the pneumatic and hydraulic placement systems of various capacities.

2. Finalize the developed software using the feedback from the underground placement research group.

3. Re-evaluate the case study using the latest version of the software and the data from the underground placement research group.

4. Finalize the user's manual for the developed software.

CHAPTER 6

UNDERGROUND PLACEMENT OF COAL COMBUSTION BY-PRODUCTS MIXTURES

6.0 UNDERGROUND PLACEMENT

6.1 Hydraulic Placement

The objective of the hydraulic placement portion of the program during Phase II was to design, select and obtain, erect, and test the hydraulic placement equipment, in order to be prepared for the major underground placement demonstration in Phase III. A mixing plant was purchased from Montgomery County, Illinois, where it had been used for several years to mix CCB material for use in road repair. The mixing plant, including a large pug mill, was dismantled from its location at the Coffeen Power Plant, transported to the Peabody Coal Company No. 10 demonstration site, and re-erected. A high-capacity, high-pressure concrete pump was rented to supplement the mixing plant. A mixture of fly ash and scrubber sludge, obtained from the City Water, Light, and Power Company in Springfield, Illinois will be the basis of the hydraulic placement. A small amount of lime (about 2.0%) will be added to the mix. The hydraulic mixture, as it is placed underground, will be composed of from 70 to 75 percent solids.

6.2 Design and Discussion of Hydraulic Paste Backfill Placement System

Phase I studies followed a systematic approach to answer the following questions:

1. What mixtures of fly ash, scrubber sludge, and water will form stable, flowable pastes?
2. How should the components be mixed?
3. How will the viscosity of the paste be controlled?
4. How will the paste mixing plant be designed using readily available, conventional equipment from the concrete/aggregate and mining industries?
5. What are the design criteria for the demonstration plant to be built at Peabody No. 10 mine?
6. What are the estimated setup and operating costs for paste injection?

A paste must be differentiated from a stable slurry. A stable slurry has a larger water content than a paste. Larger particles in a stable slurry do not settle perceptibly in a 15 minute period. At the end of the 15 minute settling test, there is no visual classification of particles and there is no harder layer at the bottom of the container. Stable slurries have very low critical velocities when pumping so the flow can be stopped for short periods without plugging the pipeline.

Pastes made from scrubber sludge and from 35% to 50% fly ash form pumpable mixes at 74% to 77% solids. The pastes developed from scrubber sludge were unstable and settled rapidly. Fly ash must be added to provide a stable paste. Pastes with less than 40% fly ash show signs of instability because hard or denser layers were observed after 15 minutes of quiescence. Pastes with 50% fly ash were stable. Even though pastes with less than 40% fly ash showed signs of instability in 15 minutes, the pastes could be remixed quickly indicating that shearing would re-liquefy the pastes.

After sitting for 24 hours, all pastes showed signs of hardening to materials with significant shear strength (certainly not pumpable). The addition of 1% lime clearly accelerates the hardening process. These observations led to the development and characterization of hydraulic paste backfill mixes.

Since it appears that scrubber sludge must first be mixed with water in order to avoid agglomerates, vacuum filtration may not be necessary at the power plant, if the power plant and mine are located close to each other. Furthermore, the production ratios of fly ash and scrubber sludge must also be considered to ensure a stable paste. The development of strength in pastes should be controlled to provide adequate strength over a long time period, and not early strength. Early strength provided by lime addition would only limit the horizontal placement distance from the borehole injection point.

6.2.1 Conceptual Design of Mixing Plant in Phase I.

Based on Phase I studies, the overall plant layout as shown in Figure 6.1 was developed. Dewatered scrubber sludge and fly ash will be transported to the injection site. The scrubber sludge will be mixed with water to form a homogeneous slurry which will be pumped in a circulation loop containing an agitated tank for storage capacity and mixing. Fly ash will be stored dry in a silo. Slurry and fly ash will be measured into a high intensity mixer operated in a batch mixing mode. High slump paste produced by the mixer will be pumped underground through an injection well into abandoned mine panels. The paste mixture will slowly develop sufficient strength to help control subsidence.

Scrubber sludge will be dewatered at the power plant to a yellowish-brown filter cake containing about 15% free water. The filter cake would be hauled in covered trucks to the injection site to a surge stockpile adjacent to the paste mixing plant. It is necessary to first produce a slurry with the filter cake to avoid agglomeration. The filter cake would be fed to an agitated tank using a belt feeder. Water will also be added to the tank to dilute the slurry to about 61% solids. The slurry loop passes close to the mixer where a valve can be opened to allow the slurry to enter the mixer.

Concurrent with the addition of slurry to the mixing chamber, fly ash will be augured into the mixer. The amount of fly ash will be determined by operating the screw conveyor for a calculated time period based on calibration of the delivery rate of the screw. After mixing for a specified time period, normally 30 seconds, the mixer power will be observed by a power meter. Should the power draw be too high, a small amount of additional slurry will be added. Should the power draw be too low, a small amount of fly ash will be added. The power draw measurement will thus assure that the viscosity of the mix will be correct. When a batch is mixed and the viscosity is correct, the mixer will dump into the pump hopper if the pump hopper is low.

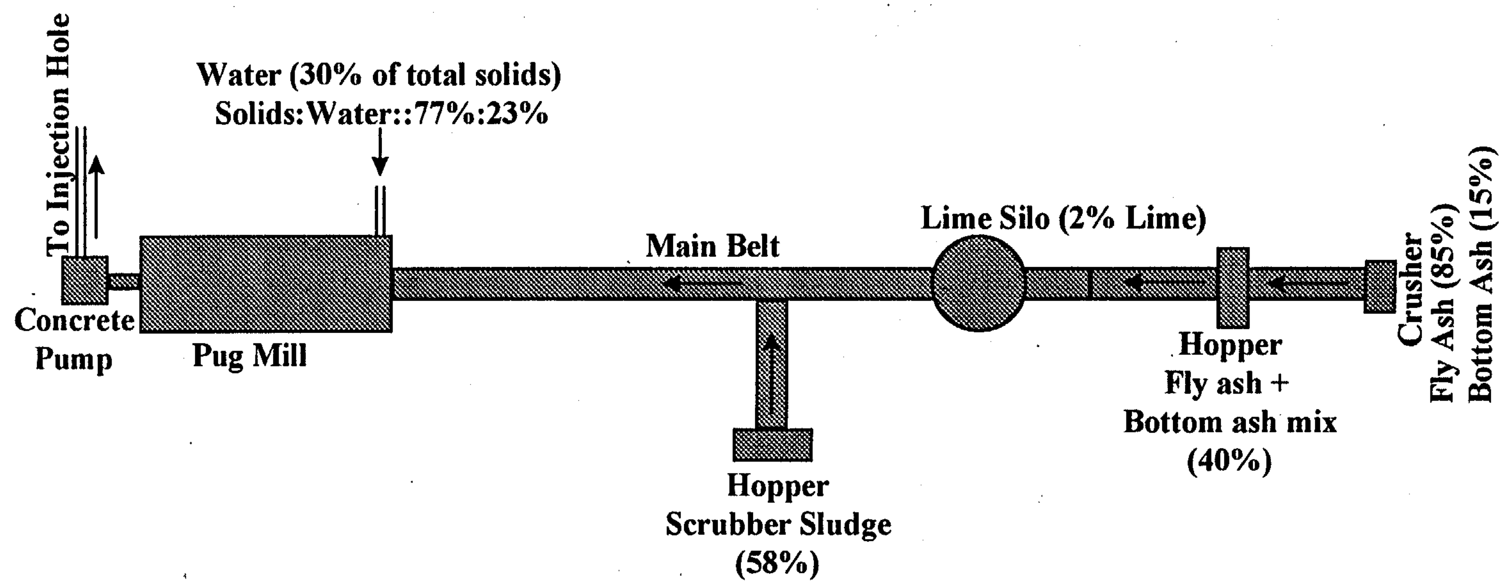

Figure 6.1 Schematic of Mixing Plant

A more detailed analysis of the plant in Figure 6.1 indicated that slurrying scrubber sludge and associated pumping system would significantly increase the capital, and operating costs. It was estimated that this may amount to about $1.50/ton of material pumped. Therefore, alternative ways of handling and mixing scrubber sludge were considered in Phase II. At the end of Phase II, the investigators had developed ways to feed scrubber sludge directly on the belt. The pug-mill successfully mixed fly ash and scrubber sludge to provide a homogenous mix for pumping.

The City Water, Light Power plant (CWLP) does not segregate dry fly ash. Fly ash and bottom ash are mixed with water and pumped to the ponds. The CWLP planned to dredge the ponded material and supply it to the investigators for underground placement. The dredged material had considerable coarse material which must be crushed in the hammer mill prior to mixing with scrubber sludge. A non-homogeneous mixture would result in varying slump paste which would create pumping problems.

6.3 Preparation For Hydraulic Placement of CCB-based Grouts and Quality Control

Two storage pads for CCBs (fly ash, scrubber sludge) storage were prepared on the ground using FBC fly ash and gravel. Approximately 400 tons of ponded fly ash/bottom ash mixture and wet scrubber sludge were brought to the site. Samples of ponded fly ash/bottom ash mixture and scrubber sludge were taken for as-received moisture content determination. Average of six samples of two materials taken from different locations of the piles indicated 18% moisture (solids to water ratio) in fly ash/bottom ash mixture and 21% moisture in the scrubber sludge.

One bucket of each grout component (fly ash/bottom ash mixture and scrubber sludge) was also collected for preparing few grout samples to study the paste characteristics and bleeding. A small high speed kitchen mixer was used and a few samples were prepared using 2% lime. It was found that a 1:1 ratio of fly ash/bottom ash mixture and scrubber sludge by dry weight produced a stable paste with 34% water (of dry solid weights). Hence, it was decided to make the pumpable paste with 49% fly ash/bottom ash mixture, 49% scrubber sludge, and 2% lime.

Assuming a paste of 74% solids, the 50 ton per hour paste backfill rate translates to feed rates of 37 tons per hour of dry solids and 13 tons per hour of water. While attempting to calibrate the scrubber sludge feed rate, it was found that scrubber sludge bridged across the hopper walls and would not pass through the small opening of the feed control door of the hopper. Also, the variable speed control motor of the lime did not function well to supply a smooth lime feed to the main belt. Hence, for the demonstration purpose, it was decided to blend as-received fly ash/bottom ash mixture and scrubber sludge, in equal proportions by volume, on the surface using a front-end loader and not to use lime in the paste.

Equal volumes of fly ash mixture and scrubber sludge in the paste translated to approximately 55% fly ash mixture and 45% scrubber sludge by dry weights because of different as-received moisture content in the two constituents and their different densities. Depending on the compaction, density of fly ash mixture varied from 75 to 92 pcf and the scrubber sludge density varied from 55 to 86 pcf. The moisture content of the blend was approximately 20% by dry solid weights. Feed rate of dry solids of 37 tons per hour was converted to the feed rate of as-received blend of fly ash mixture and scrubber sludge. Assuming 20% moisture in the blend, feed rate was fixed at 44.4 tons of as-received blend of two materials.

Main belt was calibrated with a feed rate of 45 tons per hour. However, the feed rate varied from 27 to 45 tons per hour. This proved difficult in controlling the slump of the mix. Initially it was decided to start with 15 gallons per minute of water feed that amounted to approximately 77% solids in the paste. Approximately 20 tons of paste (6-inch slump) was pumped to a 10 ft by 10 ft pit on the surface (shown as surface cell in Figure 6.1) to visually inspect the grout characteristics. Freshly prepared grouts were found to be satisfactory and it was decided to pump through the injection bore hole.

Since the end of April, 1997 (end of Phase II), the investigators have made significant modifications to the mixing plant to provide a more uniform quality control of the paste backfill mix. Data collected during the operation of the plant during June 1 - June 15 1997, indicate that values can be controlled within ± 0.2 inches. These data will be included in the Phase III Quarterly report due July, 1997.

6.4 Underground Placement of CCBs-based Grout

Six-inch pipelines were laid on the surface for a distance of 480 ft from the pump to the injection borehole. Location of the hydraulic injection borehole for the phase II underground injection panel is shown in Figure 6.1. Long pumping distance on the surface coupled with leakage on pipeline joints and fluctuating solid feed rate to the pug mill plugged the surface pipelines twice during the first hour of pumping. To prevent plugging of the surface pipelines, water feed rate was increased to 25 gallons per minute. Pumping was done approximately for four hours and slump measurements were taken four times. Grouts of approximately 10-inch slump pumped through the borehole with no problem. Freshly prepared mixture samples were taken for water content measurements. Ten 3-inch by 6-inch cylindrical samples of freshly prepared grouts were also prepared for strength testing in the laboratory. As there was no lime in the mix, the samples were not expected to exhibit adequate strength upon curing. To study the strength characteristics of the same grout containing lime, approximately 35 lbs of samples were drawn from the pug mill and 2% lime and 2% additional water were added to the grout and mixed thoroughly using a steel rod. Six 3-inch by 6-inch cylindrical samples were prepared using the grout with lime.

All the cylindrical samples were taken to the laboratory and approximately after 40 days, they were demolded. As expected, six samples of grouts without any lime showed an average compressive strength of 22 psi. However, average of three samples of grouts with lime showed a compressive strength of 217 psi. Hence, it was concluded that the addition of 2% lime to the grout would be adequate to achieve strength comparable to that obtained in the laboratory with 5% of lime waste.

Since June 1, 1997, investigators have successfully pumped 250-tons of paste backfill with relatively uniform slump of 9.0 ± 0.2 inches (based on seven samples) without any breakdowns. An average pumping rate of about 40 tons/hour with two 10 cu-yd concrete trucks feeding the concrete pump was achieved. The above slump values represents about 74% solids content. In the above demonstrations, scrubber sludge and fly ash/bottom ash mixture were mixed on the surface and loaded into the mixing plant hopper feeding the hammer mill. On June 18, 1997, pumping rate of 90 tons per hour was achieved without any breakdowns.

For Phase III demonstration, it is proposed to put scrubber sludge in one hopper and fly ash/bottom ash mixture into another hopper. The later will feed into the hammer mill.

The investigators are also developing several new concepts for simpler storage, handling, and mixing CCBs prior to underground placement. The goals are to develop 1) mobile equipment which can be moved from hole to hole, and 2) minimize the use of hoppers since the materials are wet and do not flow readily from the hopper to the belt. It is hoped that the above developments would drastically reduce hydraulic backfill placement cost.

6.5 Pneumatic Placement

The objective of the pneumatic injection portion of the program during Phase II was to design, construct, and test the pneumatic injection equipment, in order to be prepared for the major injection demonstration in Phase III. Essentially this objective was accomplished during Phase II.

6.5.1 Equipment Design

In the first part of Phase II, efforts were directed in designing the injection equipment, in developing specifications, in preparing shop drawings, in searching out qualified contractors and suppliers, in working with the University procurement officials, and in consulting with contractors and suppliers. It should be noted that, to the maximum extent possible, suppliers and contractors from Southern Illinois were used. It was decided that the pneumatic injection equipment would be assembled at the University's Carterville facility, where a high-bay building and a 25 ton overhead crane was available. Space was cleared and designated as the pneumatic equipment work area. As suppliers and contractors filled various orders the material was delivered to the Carterville facility. The injection equipment was designed to be placed on two skid bases, so that it can be transported over the highway on flat-bed trailer trucks.

6.5.2 Assembly of Major Components

As indicated above, the assembly of the major components of the pneumatic injection equipment was carried out at the University's Carterville facility. The final assembly was completed on February 28, 1997, including the pneumatic and hydraulic components, hoses, valves, etc. Testing of the pipelines and valve sequencing and operation of the feed screws and drive motors was completed utilizing the compressed air supply available at the facility. The completely assembled équipment was finished in a primer gray paint.

6.5.3 Delivery and Site Assembly

On March 3, 1997, both skid assemblies were loaded on two flat-bed trailers and transported from the Carterville facility to the Peabody Coal Company Mine #10 demonstration site near Pawnee, Illinois, a distance of about 175 miles. At the demonstration site the skids were unloaded and placed in close proximity to the test borehole. A diesel-engine powered air compressor, with a capacity of 1,300 scfm of air at 125 psig, was rented for the duration of the test, and interconnecting hoses were installed between the skids and the compressor. A water supply to the holding tank was established by a direct line from the on-site reclamation contractors' pumps.

6.5.4 Preliminary Testing of Pneumatic Equipment

The Archer Daniel Midland Company (ADM) agreed to transport the fly ash to the Peabody Mine No. 10 demonstration site in tanker trucks, and the pneumatic injection equipment was modified to receive the fly ash from the tanker discharge lines.

This modification included installing a 1000 cfm air filter unit to eliminate dust being discharged to the atmosphere with the pneumatic tanker exhaust air and two pipeline hook-ups to allow two tanker units to deliver fly ash at the same time. The infeed hopper, which was previously open at the top had to be enclosed and baffle plates installed to assist with the air-material separation.

The average load of fly ash transported by a self discharging pneumatic transfer truck is 55,360 # (gross weight, 80,000 #, tank weight 24,640#). As the proposed delivery rate of the fly ash into the mine is 50 tons per hour (on a dry basis, i.e. Previous to mixing with 12% water), each tanker has to discharge it's load in 33 minutes allowing 3 minutes for the hose line hook-up. The unloading rate must be set for 0.92 tons per minute, which is within the capability of the tanker pneumatic discharge equipment.

At the top of the 6" cased borehole a pipe tee was installed, with a blind flange opposite the delivery pipe from the injection equipment. This is a simple technique to eliminate wear as opposed to using an elbow at the top of the borehole. To elevate the backfill material to the infeed hopper from the delivery dump trucks (or alternatively, ground storage) a screw conveyor was leased from a local supply company. An infeed bin with the capacity to receive the fly ash from a front end loader was arranged at the loading end.

During the second week of March the pneumatic injection equipment was tested in a dry run mode, i.e., without the fly ash injection material. Some minor modifications and adjustments were made to the equipment to improve the operation of various valves, etc. Fly ash injection material was run through the equipment during the third week of March, but with less than satisfactory results. Several problems with the fly ash surfaced, including quantities of clay and rock (probably picked up in the bucket of the front end loader) which choked the bar screen on the infeed hopper. It was also apparent that the feed screws were under powered. Modifications were made to the hydraulic drive motors so that the screws are driven through a four-to-one reduction rather than by direct (one-to-one) drive. Further modifications included replacing the air powered hydraulic pumps with a diesel powered package.

During the first week of April a test run on the injection equipment was initiated using fly ash freshly delivered from dump trucks. The equipment operated satisfactory, however the airborne dust created during the loading of the screw conveyor hopper was not acceptable. Conditioning the fly ash with water using powered sprays proved time consuming so arrangements were made to precondition the fly ash in the hydraulic mixing plant and transporting the dampened material to the pneumatic injection plant with a front end loader. A spray system was installed on the mixing plant conveyor belts to eliminate the fugitive dust at the transfer stations prior to delivery to the pug mill. However, there remained the problem of dust generation when dumping from the trucks onto the ground and subsequent handling with a front end loader to the infeed hopper of the mixing plant, as this is undertaken prior to conditioning with water.

Therefore, consideration for delivery of the fly ash to the demonstration site in self-discharging pneumatic tanker trucks was given. The pre-conditioning of the fly ash with 12% of water by weight would then have to take place within the pneumatic injection equipment. To ensure the temperature rise in the fly ash when water is added is within acceptable limits, a series of laboratory tests were undertaken. These tests showed that the temperature rise, when the water was added to the fly ash, created no hazard to operating personnel, to the pneumatic injection equipment, or to the underground environment.

A major disadvantage of removing the fly ash from tankers is the re-arrangement of the operating mode of the pneumatic injection equipment. When initially set-up, the infeed (trucks, front-end loader, etc) would be intermittent and the feed tanks would cycle to produce a constant stream of material to the borehole via the conveying line. As the fly ash would have been preconditioned, the additional water (15%) for setting up the material deposited in the mine would be added at a constant rate. However, with changing the equipment to receive tankers, the infeed of material must be constant, leaving the feed tanks to cycle only when empty. This resulted in an disrupted flow of material in the conveyor lines. Although the flow of water related to the average material rate, the instantaneous flow of water was in excess during the tank change over period. This excess water overwetted the material in the vicinity of the borehole discharge into the mine, prematurely setting it up, and resulting in a borehole plug.

The borehole was reamed out and the backfilling continued at the designed rate of 50 tons per hour, with the water injection being judiciously controlled to match the instantaneous rate of material flow. Consideration is being given to injecting the water at the bottom of the borehole. This will be done by lowering a small diameter pipeline within the 6" conveying line and spraying the water through apertures to mix with the fly ash as it emerges into the mine entry.

6.2.5 Mine Model Studies

In order to more fully understand the pneumatic injection process, a physical model of a mine entry was constructed in the Mining Engineering Department laboratory, to scale the model to accurately depict the flow conditions in the underground entries and cross-cut. The scale was based on an air velocity of five feet per minute (compressor output divided by the typical mine cross section). This laboratory air compressor produces about three scrm of air at l00 psi which when related to a velocity of five feet per minute, resulted in a model cross section of 4 inches high and 13 inches wide. The sections were formed from steel plates with an open top glass sheets were at the "root" with a reinforced frame and clamps to prevent leakage of the conveying and blast air. Air for the blast was stored in the compressor reservoir and released instantaneously through a ball valve. The fly ash was conveyed to the model from a pressurized tank, with a variable speed screw feeder to control the discharge rate. The flow of the material along the length of the model was observed through the glass roof. The model is about 20 feet in length, which is scaled to simulate some 300 feet of under-ground mine entry. Of particular interest is the observation of the channeling in the deposited material at the roof level of the simulated roadway. Also, the effect of the "blast" air in heaving the material down the entry and into the cross-cut was observed. The results to date indicate that the deposited fly ash can be transported a distance of 30 feet in the model. An attempt is being made to correlate data from model studies with observations in the field. Laboratory studies will form the thesis of Mr. Indranil De.

CHAPTER 7

ENVIRONMENTAL ASSESSMENT AND GEOTECHNICAL STABILITY AND SUBSIDENCE IMPACTS

7.0 ENVIRONMENTAL ASSESSMENT AND GEOTECHNICAL STABILITY AND SUBSIDENCE IMPACTS

7. 1 Introduction

The disposal of coal combustion residues near the surface in landfills threatens groundwater resources in many geologic environments. Hydraulic or pneumatic injection of these residues into deep underground mines offers distinct environmental advantages. First, many underground mines remain dry years after they are sealed (Cartwright and Hunt, 1981). The absence or slow recovery of saturated conditions would minimize leachate production from the residues. Second, deep mines are located generally below potable groundwater resources where the units bounding the mine contain native brine of little economic value. Third, the coal combustion residues have chemical characteristics which may actually ameliorate an environmental problem associated with coal extraction; acid mine drainage. Fourth, the residues harden like cement when wetted which could help prevent subsidence associated with underground mining.

At first, it may seem that disposal into abandoned mines should not create any environmental impacts that do not already exist as a consequence of mining. In effect, the residues are returned to their original environment. Coal combustion residues, however, are more concentrated with respect to some chemical constituents and combustion has altered their physical and chemical properties. Leachate generated by the residues is distinctly different than that generated by the coal and its bounding strata.

This report describes work completed during Phase II of the project on characterizing the possible environmental impacts of disposing coal combustion residues in deep underground abandoned coal mines. Specific study objectives were developed to answer two questions; 1) Do the increased hydraulic pressures associated with injection induce hydraulic gradients capable of driving native brine or leachate from the residues into units containing potable resources? and 2) As saturated conditions return, can the natural hydrologic properties of the bounding strata contain the leachate? Although coal combustion residues have been placed underground as part of small trial studies, little is known on the environmental effects of this practice. Industry and regulatory agency acceptance of refuse injection into deep underground mines hinge on a detailed demonstration project with a monitoring plan specifically designed to evaluate environmental impacts.

The Illinois State Geological Survey (ISGS) provided technical assistance on this multi-faceted project to Southern Illinois University at Carbondale (SIUC), particularly in the areas of residue characterization and environmental assessment. Specific tasks described in this report where the ISGS took the lead role included groundwater modeling, isotope geochemistry of groundwater, and chemical analysis of water samples.

7.2 Goals and Specific Objectives

The purpose of the environmental assessment portion of the project is to determine if the placement of coal combustion residues by hydraulic or pneumatic injection into an abandoned underground mine panel would impact surface and groundwater resources. Specific objectives are to:

1. Develop a conceptual model for the stratigraphy in the vicinity of the target panels, specifically locating any units which may function as aquifers;

2. Determine the flow system in the vicinity of the mine panels and the hydrologic properties of the bounding strata, including hydraulic conductivity;

3. Assess the impact, if any, of injecting coal combustion residues into the underground mine panels on groundwater quality;

4. Assess the environmental impact, if any, of temporary surface impoundments or spills, and;

5. Assess the impact, if any, of injecting the residues into underground mine panels on the extraction of economic resources.

Ultimately, we intend to generalize the findings from the demonstration project for other cases of underground placement of coal combustion residues. During Phase II of the project, additional work was completed on the first four objectives.

7.3 Literature Review

Numerous previous studies have investigated the environmental impact of coal combustion residue disposal sites on surface water and groundwater quality (Villaume and others, 1983; Le Seur Spencer and Drake, 1987; Cherkauer, 1980; Hardy, 1981; Simsiman and others, 1987; Sakata, 1987; Theis and others, 1978; Beaver and others, 1987; Fruchter and others, 1988; Gerber, 1981; Le Seur, 1985; Hall, 1977; Rai and others, 1989; Rehage and Holcombe, 1990; Libicki, 1978; U.S. Waterways Experiment Station, 1979). Distinct contaminant plumes have been detected in the groundwater, downgradient from slurry ponds or landfills. In some cases contaminants exceeded drinking water standards. Disposal in ponds poses a greater risk to the environment. The residues are saturated or nearly saturated, leading to greater leachate production. In addition, leachate migration under saturated conditions generally exceeds those under unsaturated conditions associated with most landfills. Dry disposal has replaced slurry disposal in recent years because of its environmental advantages. Adriano and others (1980), Ferraiolo and others (1990), Theis and Marley (1979), and Theis and Gardner (1990) provided general reviews on residue disposal methods and environmental impacts.

Complex chemical reactions between the coal combustion residues, groundwater, and natural geologic materials make predicting the impact of disposal on a site difficult. A coal combustion residue is unique; each coal source and burning method yields residues with distinct chemical and physical properties. The leachate derived from the residue varies considerably (Theis and Marley, 1979). Natural factors such as ion exchange between the residue and soil constituents, adsorption, and dilution can attenuate contaminants and these factors vary from site to site. To minimize the environmental impact, several control strategies have been applied, including natural and artificial liners, chemical stabilization, and selective mixing of coal combustion residues (Theis and Marley, 1979).

Pneumatic and hydraulic injection are the two disposal methods under consideration. In certain geologic settings, these methods pose little risk to the environment. For example, the strata bounding some mines have low permeability and the mines remain dry years after they are sealed (Cartwright and Hunt, 1981). Many mines are located below potable groundwater resources where the units bounding the mine contain native brine of little economic value. As long as no preferred pathways to potable groundwater supplies exist, such as faults, even if leachate escapes from the mine it would not create an environmental impact in the biosphere.

Davis and Walton (1982) reviewed factors impacting groundwater from deep coal mine drainage. Esling and Jones (1990, 1992) reviewed existing analytic models and developed a new model that describe the hydrologic effects of injecting refuse into underground mines.

7.4 Technical Discussion

With the exception of one task, all work proposed for Phase II was completed. Researchers involved in environmental assessment:

1. Refined the conceptual model of site hydrology and geology;

2. Regularly monitored ambient water quality and hydraulic head in the strata immediately bounding the target panels;

3. Sampled and analyzed water samples for isotope geochemistry; and

4. Made progress toward developing flow and contaminant transport models of the site.

This report presents results completed in each of the above areas. In addition, one task scheduled for the second phase, collection of falling head permeability test data, was not completed. One pressure transducer designed to provide data on atmospheric pressure performed erratically. The unit was eventually replaced by the manufacturer with a new one at the end of the second phase. Reliable readings on atmospheric pressure are necessary in order to perform falling head permeability tests at the site. These tests will be completed early in the third phase of the project.

7.4.1 Conceptual Model (Site Characterization)

A working conceptual model on the geology and hydrology of the study was completed and described in the Phase I report. This model was based on a detailed analysis of the computerized well logs on file with Peabody Coal Company, an examination of the well logs on file at the Illinois State Geological Survey, a review of the literature on the geology and hydrology of the area around the mine, geophysical logs of boreholes, the description of the continuous core collected in the study area, and background water quality samples. In addition, individuals working in the area of environmental assessment visited the mine prior to its closure in August, 1994 to examine surface workings and underground conditions in one of the target areas.

The ultimate goal of developing a conceptual model is to locate any artificial or natural pathways capable of transmitting contaminants originating from the coal combustion residues to potable groundwater supplies. Possible pathways include permeable units, fractures, fault zones, and artificial pathways such as improperly sealed boreholes. The working conceptual model was used to design the groundwater monitoring program and to determine potential impacts from surface activities such as temporary storage of the residues. The conceptual model is a work in progress. It is refined as additional hydrologic and geotechnical data become available and will be adjusted to reflect information provided by groundwater numeric models. Ultimately, the conceptual model will include the following components:

Hydrostratigraphy
Groundwater flow system
Hydrologic boundaries
Hydraulic conductivity of the bounding strata
Fluid sources and sinks
Potential interactions between the flow system near the target panel and other local flow systems

SIUC and the ISGS personnel are working together on the conceptual model of the site. Data on ambient water quality of the strata bounding the mine panels collected during Phase 2 have supported the preliminary conceptual model of the site.

Site Description

The study area is part of the Peabody Number 10 Mine, with large underground works located in portions of the South Fork and Cotton Hill Townships, T 13 N, R 4 W and T 14 N, R 4 W in Christian and Sangamon Counties, Illinois. Portions of Sections 10 and 11, T 13 N, R 4 W in Christian County, herein referred to as the study area, contain extensive surface works, including disposal areas, as indicated on the Pawnee 7.5 Minute Quadrangle. The underground mine panels targeted for disposal are located in these sections. Pawnee is the nearest town, situated about 2 miles west of the mine. The town of Kincaid is located about four miles east. Springfield is about 20 miles northwest of the study area. The room and pillar method was used to extract Herrin (No. 6) Coal from an average depth of 350 feet. The mine ceased operation in August, 1994.

Geology

The surficial deposits of the study area average about 40 feet thick and consist of about 10 feet of loess over unconsolidated glacial diamicton with minor lenses of sand and gravel (Glasford Formation). The unconsolidated surficial deposits are underlain by Pennsylvanian Age shale, limestone, clay, coal and minor sandstone units of the Bond, Modesto, and Carbondale

Formations (Willman and others, 1975; Bergstrom and others, 1976). The stratigraphy of the Pennsylvanian bedrock is complex. Most Pennsylvanian lithologies are not continuous over the study area and the sandstones and gray shales exhibit extreme changes in thickness laterally.

The Herrin (No. 6) Coal, about 7 feet thick, was mined at depths ranging from 325-375 feet. In most of the underground works, the immediate roof rock is the Anna Shale which is overlain by the Brereton Limestone. The sandstone above the Herrin Coal is the Anvil Rock Sandstone (Figure 6.1). Nelson (1987) believed the Brereton Shale and Anvil Rock Sandstone were relatively continuous in the study area. Outside of the panels targeted for disposal operations, the Anvil Rock Sandstone may thicken. In these areas, the Herrin Coal is absent. The Herrin Coal is underlain by underclay and siltstone.

The study area is northwest of the Walshville Channel, a broad former course of a large river that flowed through Illinois at the time the peat deposits that would become the Herrin Coal were deposited. The Herrin Coal is absent in areas of the Walshville Channel. The study area is also about 3 miles north of a geologic disturbance known as the Hill, a 150 to 200 foot wide trough where the Herrin Coal is 20 to 30 feet lower than the surrounding coal and up to twice as thick (Nelson, 1983). The Hill begins in section 27 and trends roughly north-south for at least 5000 feet, although its southern end has not been located.

The Phase I report presented maps of the depth to the top of major units, the elevation at the top of major units (structure contour) and thickness of the Herrin Coal (isopach map). The maps were modified to correct errors in the original well log file, but did not reflect data collected from the borehole or other boreholes drilled at the site in connection with this project. More recent data suggests that the major sandstone units are continuous across the study area. The maps will be adjusted during the third phase of the project to reflect new data.

Hydrology

The study area includes portions of the Horse Creek and Clear Creek drainage basins, both tributaries to the South Fork of the Sangamon River. Sanchris Lake, a 2,700 acre body of water constructed in 1964 to supply the nearby coal-fired Commonwealth Edison power plant, is located immediately adjacent to the mine processing and waste disposal facilities. The quality of lake water is good and suitable for most uses. No adverse effects of existing surface works are expected to impact any surface water resources. No community public groundwater supplies exist within a one mile radius of the study area.

Some private groundwater wells (27) fall within a one half mile radius of the target panels, but to date, no adverse impact of mining has been reported. These private wells only produce if they encounter sand and gravel lenses in the diamicton or a thin sandstone unit or fractured limestone in the shallow bedrock. Selkregg and Kempton (1958) reported some wells in Christian County producing from shallow Pennsylvanian sandstones. Five groundwater monitoring wells are screened at the base of the surficial deposits in the study area, above the contact with the bedrock. They are monitored quarterly for groundwater quality and hydraulic head by Peabody Coal Company.

The lithologies above and immediately below the mine are characterized by low hydraulic conductivity and at present, the target panels are dry. In fact, hand specimens of the coal as well as the roof and floor rock do not appear to be saturated. Discussions with mine engineers suggest only two sources of underground water; leakage from higher more permeable horizons along the seal of the air shafts and wet zones associated with roof falls. Fractures in the roof rock may transmit some water and could explain the moist material where the roof has collapsed. These areas, however do not appear to yield sufficient groundwater flow to flood closed mine works. We saw no evidence of standing water in any of the mine works visited on June 10, 1994. Many of the rooms have debris on the floor from roof fall. This unconsolidated debris would have a higher intrinsic permeability than the surrounding rock. Excess water from slurry injection would be able to move through the debris more readily than through the combustion residues or the bounding strata.

The areas targeted for disposal are at depths between 325-375 feet, well below potable groundwater resources. Selkregg and Kempton (1958) reported that groundwater generally is too mineralized at depths in excess of 200 to 250 ft in Sangamon and Christian Counties to serve as a domestic supply. The rock between the maximum depth of potable water and the mine is mainly shale, characterized by low hydraulic conductivity. Even though potable water may occur in the shallow Pennsylvanian sandstone, most groundwater wells are finished in the sand and gravel horizons within the surficial deposits.

The handling of materials at the surface, prior to injection, should not threaten groundwater or surface water quality. The target panels are located below an upland area of low relief where the surficial deposits were classified as generally suitable for waste disposal (Bergstrom and others, 1976). Current surface operations have not impacted the water quantity or quality of Sanchris Lake and no one has reported any adverse effects of mining or current disposal operations on groundwater quantity and quality. It is hard to see how the surface operations of the project could impact groundwater or surface water resources in a way that existing mine operations have not. Any temporary storage facility for the coal combustion residues on the surface would have less of an impact that the existing, larger storage facilities. In addition, each injection site affects only a small surface area. Some care is necessary to ensure that water contacting refuse at the surface does not discharge directly into local drainage.

The rock units in the study area can be classified into seven hydrostratigraphic units (Figure 7.1); the unconsolidated surficial deposits, a thick sequence of low-permeable interbedded shale and limestone, the Trivoli Sandstone, another unit of low-permeable interbedded shale and limestone, the Anvil Rock Sandstone, fractured interbedded shale and limestone, and the Herrin Coal. The hydrostratigraphic units cross formation boundaries: the upper low-permeable interbedded shale and limestone, dominated by shale, includes parts of the Bond and Modesto Formations; the Trivoli Sandstone is a member of the Modesto Formation; the lower low-permeable sequence of interbedded shales and sandstones includes members of the Modesto and Carbondale Formations; the Anvil Rock Sandstone, and fractured interbedded shale and limestone, and the Herrin Coal are all part of the Carbondale Formation. The uppermost hydrostratigraphic unit, the unconsolidated surficial deposits, include relatively continuous sand and gravel units that contain the principal groundwater resources of the study area.

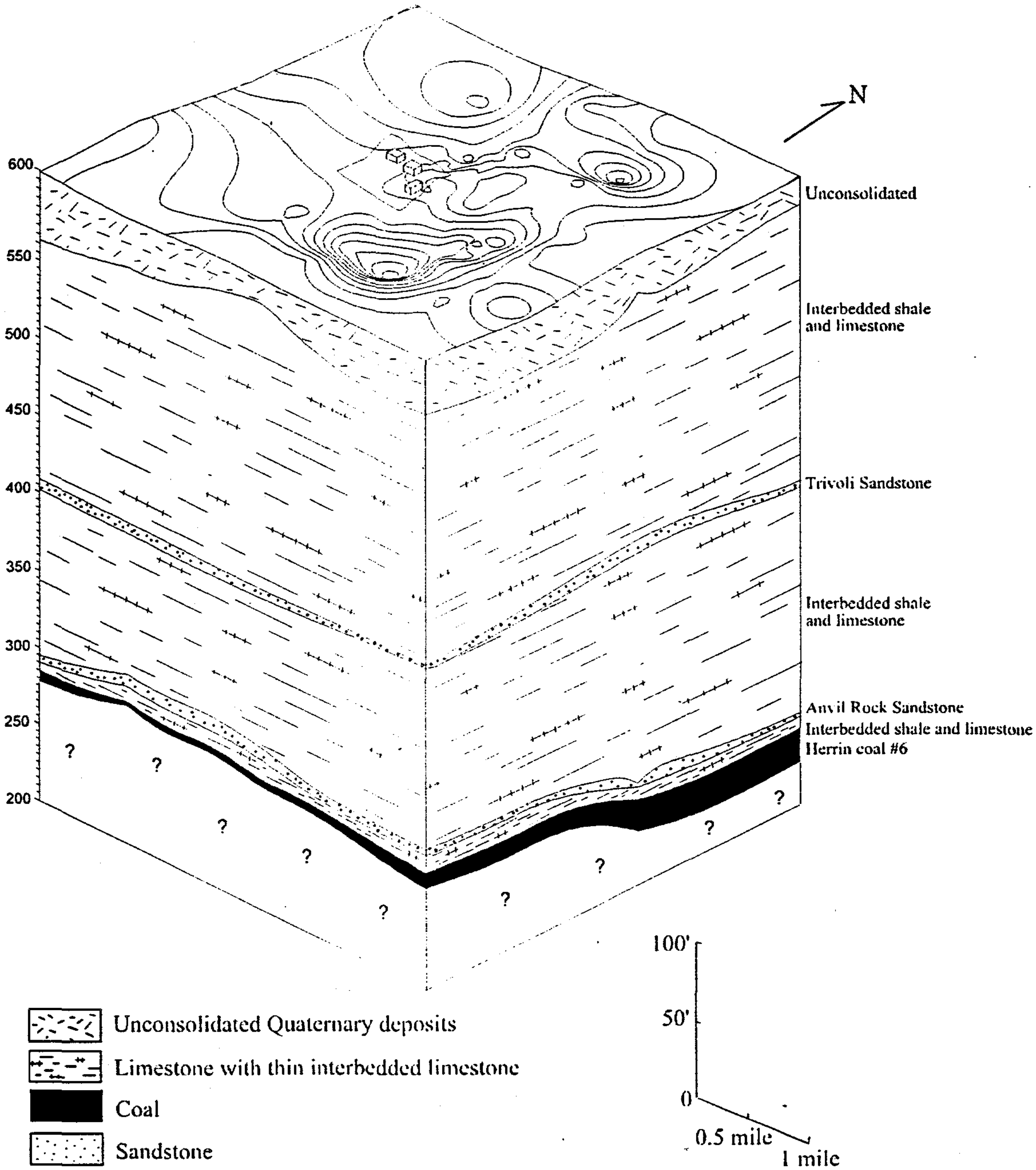

Figure 7.1. Pre-mining surface topography and underlying geology / hydrostratigraphic units. Study area is shown on the surface with three main buildings. Contour interval is 4 feet.

Shallow wells installed for this project indicate groundwater flow in the fractured and weathered bedrock immediately below the surficial unconsolidated deposits to the southwest, likely discharging to a stream located off of the mine site (Figure 7.2). A preliminary map of hydraulic head (Figure 7.2) in the Trivoli Sandstone suggests flow to the east, with one data point indicating an anomalous high head. Data on average heads measured in wells finished in the Anvil Rock Sandstone suggest flow to the north (Figure 7.3). Groundwater potential, however, is a function of the water density between the points of measurement. If this relationship is unknown, the direction of groundwater flow cannot be determined. At this stage of the project, any interpretation of groundwater flow directions in the lower strata containing dense groundwater, including the Herrin Coal, should be made with extreme caution.. The lower head at Well 3 in the Herrin Coal may represent a cone of depression associated with pumping (Figure 7.3). Well 3 regularly yields groundwater samples whereas Well 6 does not. Figures 7.4-7.6 show the water level elevations in the shallow bedrock, the Trivoli Sandstone, and the lower monitored units including the Anvil Rock Sandstone and the Herrin Coal through time. In general, heads in the different monitored units appear relatively stable through time

7.4.2 Groundwater Quality

The original plan was to sample the five monitoring wells with screens in stratigraphic units immediately adjacent to the target panels, including the Herrin Coal and the units immediately above the Herrin Coal. Only two of these wells (3 and 4), however, regularly yield groundwater samples. Two wells (1 and 2) never yielded a groundwater sample and well 5 yielded samples early, but stopped producing samples prior to Phase II. Well 3 was damaged by heavy equipment and did not produce samples from August through December, 1996. Wells 4 and 5 have screens in the Herrin Coal, whereas Well 3 has a screen in the fractured interval immediately above the coal.

In general, water quality monitoring followed accepted U.S. Environmental Protection Agency (USEPA) procedures for detection monitoring (USEPA, 1979; 1984a; 1984b; 1986). Samples were collected approximately every five weeks, a slight departure from the original monthly sampling plan, but a necessary change in order to stay within the budget. A detailed record was maintained of each sample trip. Each trip included a blank containing deionized water that serves a singular purpose; to document how preservation chemicals, sample container, or handling procedures affect sample analyses. Each well is equipped with a dedicated positive displacement bladder pump (QED type). The wells were purged with micro purging technology in order to obtain representative samples (Puls and Barcelona, 1989; Robin and Gillham, 1987; Kearl and others, 1992; Barcelona and others, 1985). When compared to the alternative, purging multiple well volumes with a purge pump and packer, micro purging significantly reduces the cost of sampling. The dedicated pumps reduce sampling time and eliminate any possibility of cross-contamination between wells.

Upon arrival at a well head, the depth to water is determined (Tables 7.1 and 7.2) three times with a tape, and recorded to the nearest 0.01 ft. According to the original plan, the wells were to be purged until stable readings of dissolved oxygen, pH, temperature and specific conductance were obtained from a flow-through cell. If the well yields were low enough so that the well was pumped dry before parameters stabilize, the field personnel were to wait for the

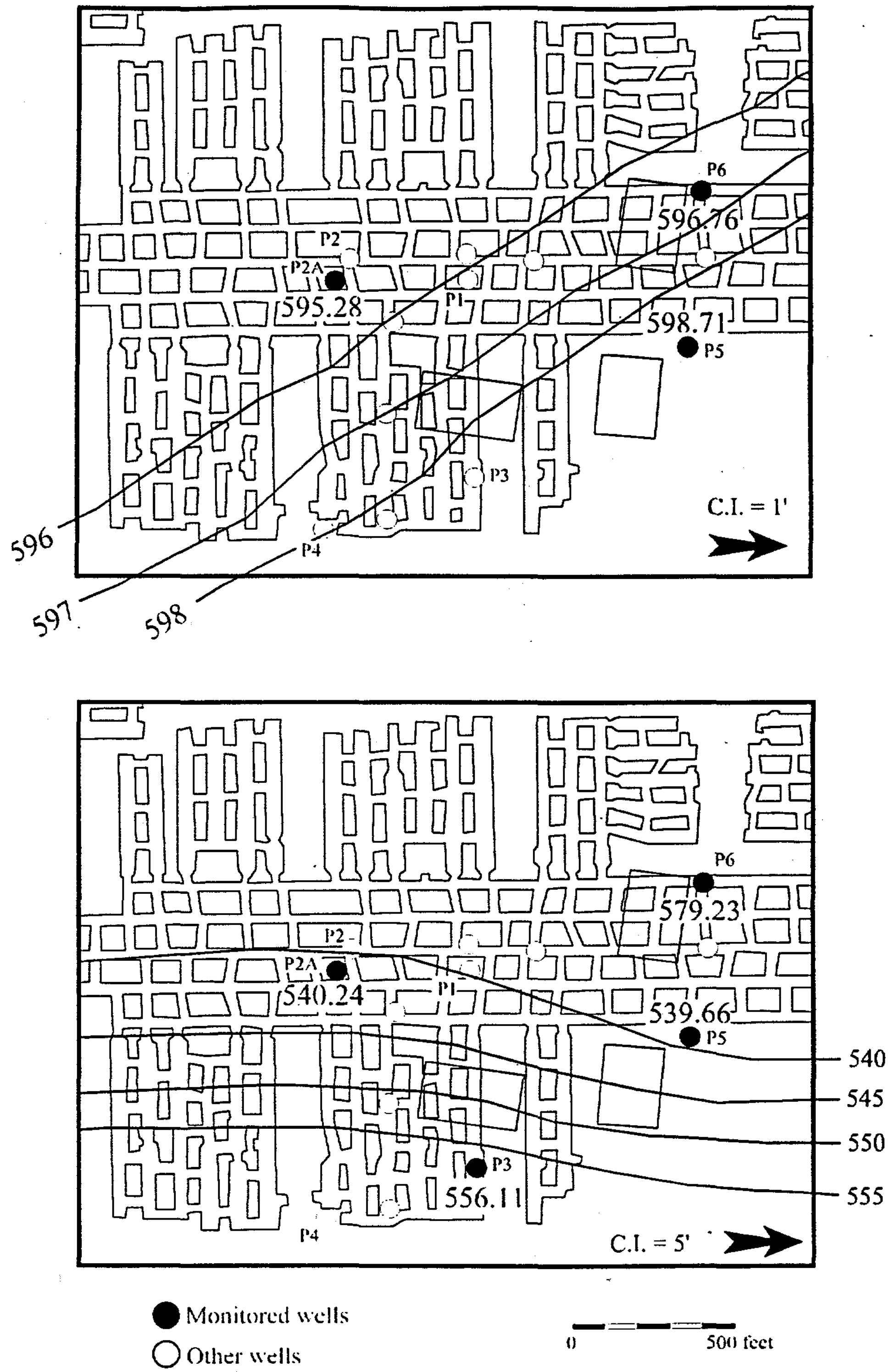

Figure 7.2. Water levels (feet) in the shallow wells (top figure) February 13, 1997, and wells within the Trivoli Sandstone (bottom figure) July 18, 1996.

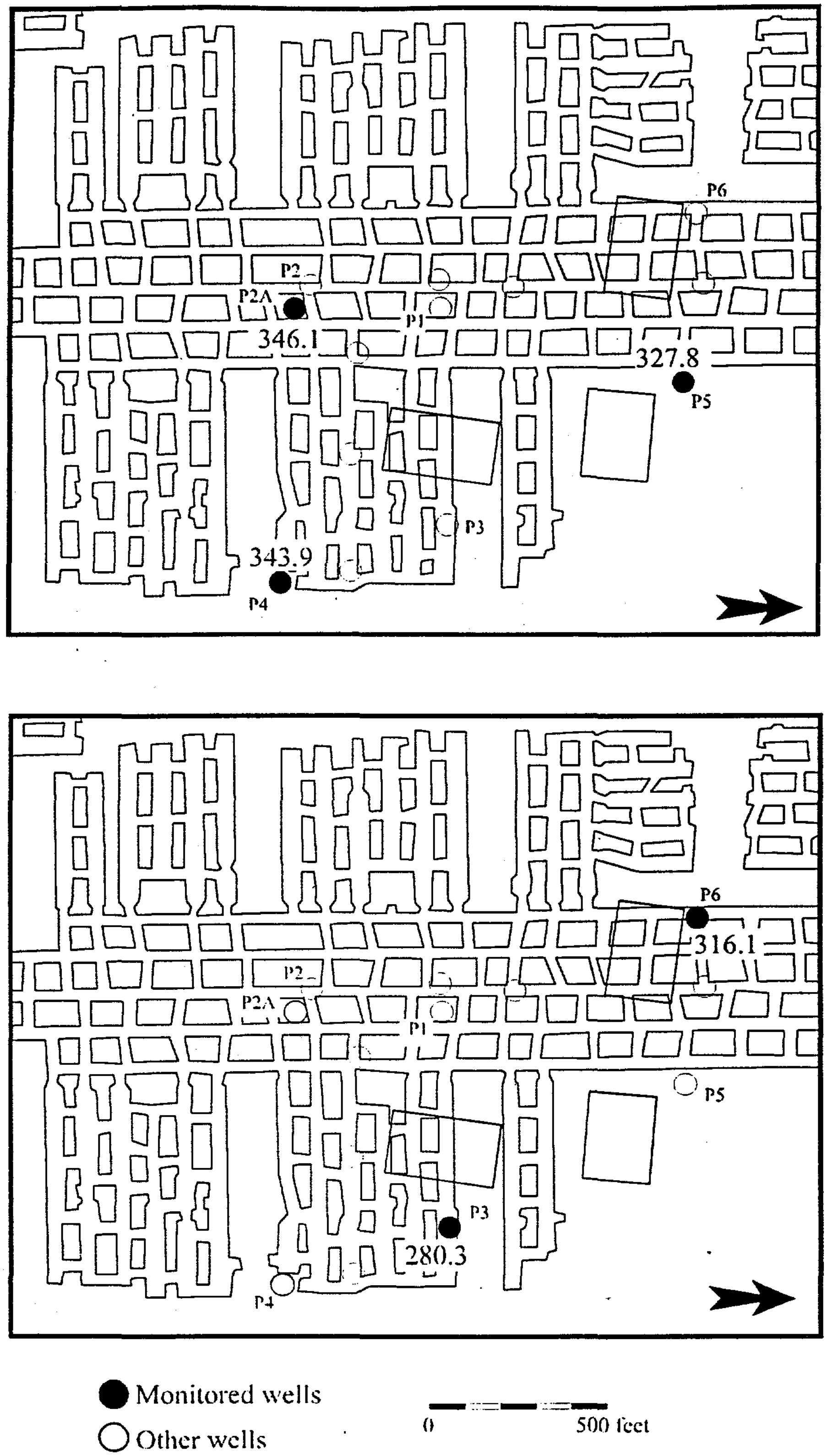

Figure 7.3. Water levels (feet), averaged for 1997, in the Anvil Rock Sandstone wells (top figure), and wells within the Herrin #6 coal (bottom figure).

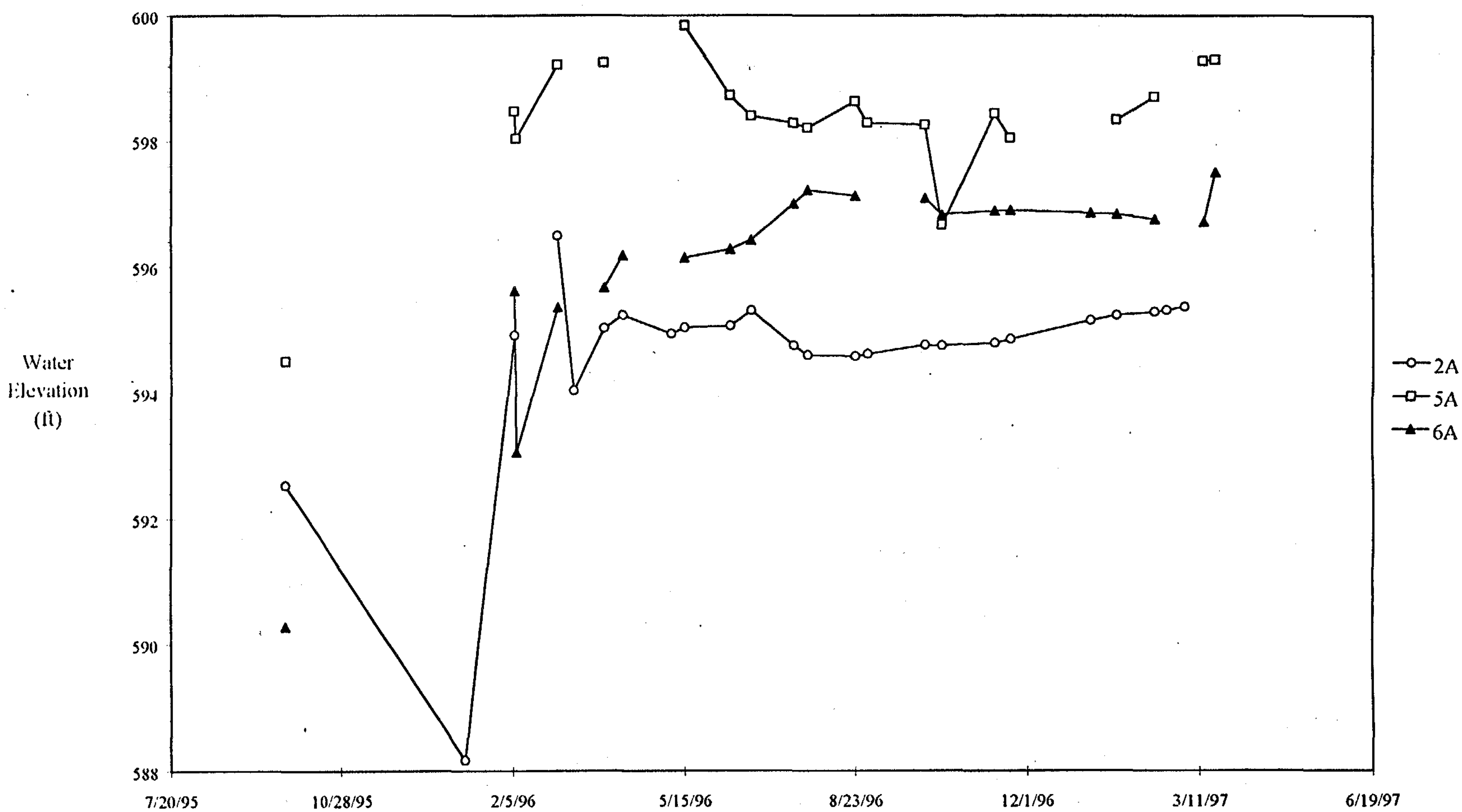

Figure 7.4. Water elevation in shallow wells.

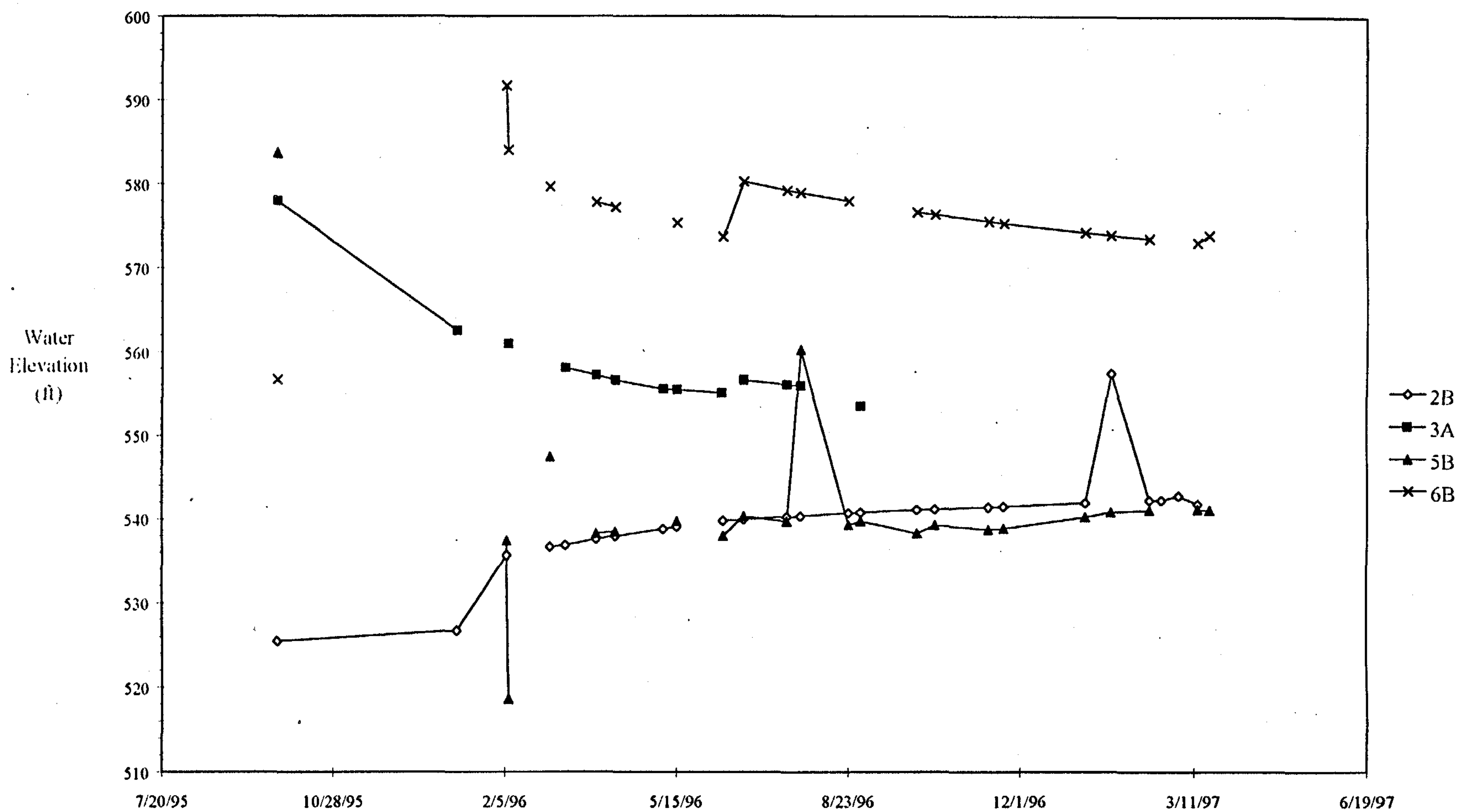

Figure 7.5. Water elevations in Trivoli Sandstone.

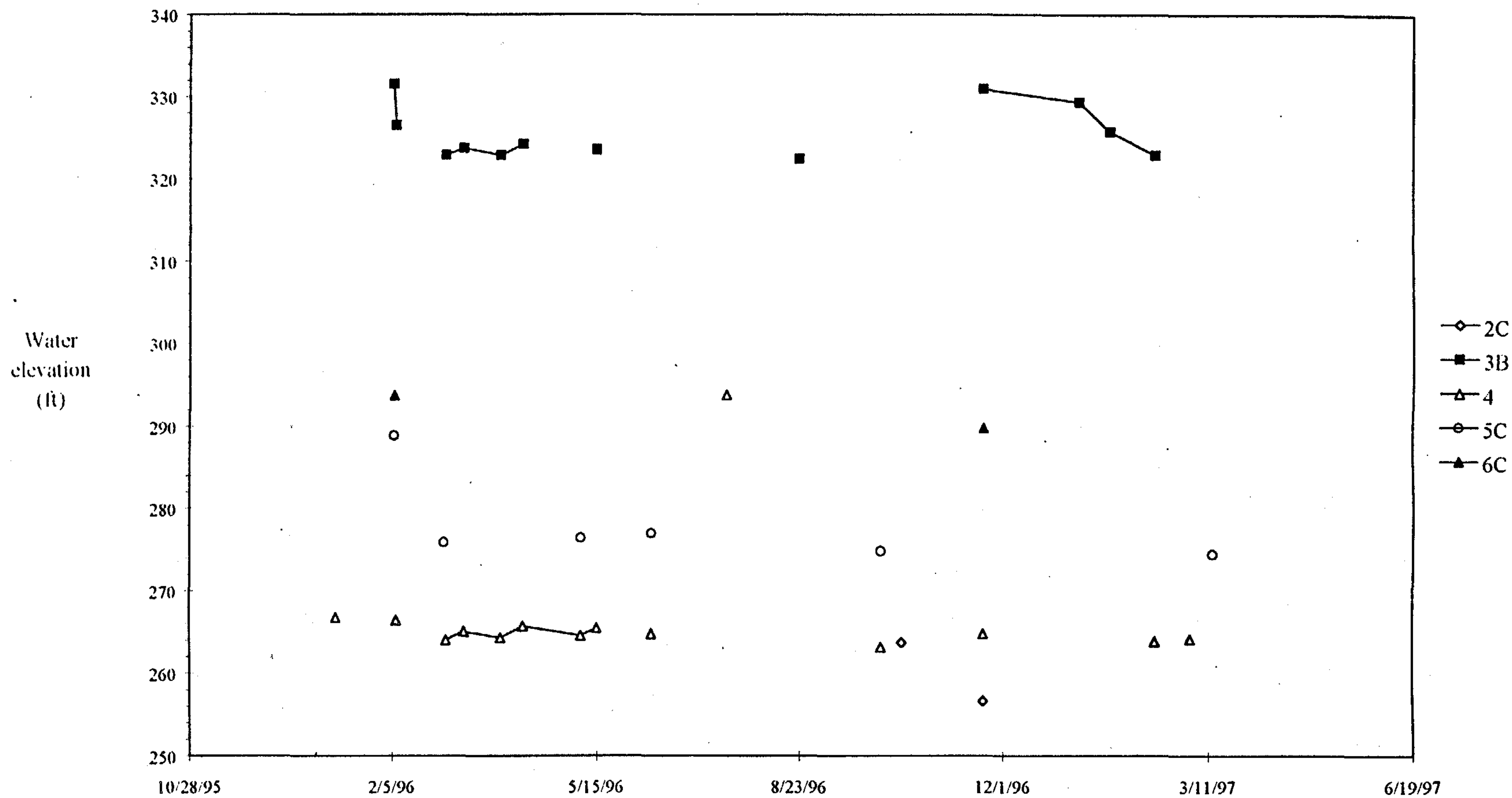

Figure 7.6. Water elevation for wells in Herrin #6 coal (open symbols), and Anvil Rock Sandstone (filled symbols).

Table 7.1: Screened interval and depth to water measurement in monitoring wells 2A, 3, and 4. Deepest wells in each nest are the sampled wells.

Well nest	2A	2A	2A	3	3	4
Screened interval (feet)	75-80	213.25-218.25	340-345	201-206	282.5-287.5	343.5-348.5
9/25/95*	16.87	83.7	237.45	27.5	13.17	257.73
1/8/96	21.22	82.46	NW	42.92	23.67	-
2/6/96	14.48	73.52	NW	-	-	341.43
2/7/96	-	-	-	44.51	274.21	-
3/1/96	-	-	-	-	279.18	341.77
3/2/96	12.9	72.5	NW	-	-	-
3/11/96	15.35	72.27	NW	47.36	282.8	344.16
3/29/96	14.37	71.54	NW	48.25	281.95	343.13
4/9/96	14.16	71.26	NW	48.83	282.81	343.91
5/7/96	14.46	70.33	NW	49.87	281.49	342.48
5/15/96	14.36	70.07	NW	49.97	NW	343.61
6/10/96	-	-	-	50.36	282.14	342.69
6/11/96	14.33	69.36	NW	-	-	-
6/23/96	14.09	69.25	NW	48.81	NW	343.44
7/18/96	14.65	68.90	NA	49.33	NA	NA
7/26/96	14.80	68.80	NW	49.51	NW	314.33
8/23/96	14.82	68.42	NA	**	**	NA
8/30/96	14.78	68.36	NA	51.95*	283.28*	NA
10/2/96	14.64	68.02	NW	**	**	NW
10/12/96	14.65	67.99	NW	**	**	344.99
11/12/96	14.605	67.755	345.72	**	**	NA
11/21/96	14.54	67.61	NW	**	**	NA
1/7/96	14.245	67.2	352.73	50.38	274.775	343.39
1/22/97	14.165	51.67	NW	50.43	276.37	NA
2/13/97	14.11	66.88	NW	50.48	279.97	NA
2/20/97	14.085	66.82	NW	50.49	282.795	344.29
3/2/97	14.03	66.34	NW	-	-	-
3/13/97		67.33	NW	50.55	NW	344.02
3/20/97	-	-	-	50.44	NA	NA

Table 7.2: Screened interval and depth to water measurement in monitoring wells 5 and 6. Deepest wells in each nest are the sampled wells.

Well nest	5	5	5	6	6	6
Screened interval (feet)	73-78	215-220	337-342	71-76	216.5-221.5	311-316
9/25/95*	8.1	18.9	66.7	16.18	48.85	260.4
1/8/96	-	-	-	-	-	-
2/6/96	4.12	65.23	N/A	10.84	13.77	NW
2/7/96	4.56	83.94	313.44	13.4	21.47	312.77
3/1/96	-	-	-	-	-	-
3/2/96	3.38	55.13	326.4	11.1	25.8	NW
3/11/96	N/A	N/A	N/A	N/A	N/A	NW
3/29/96	3.34	64.37	NW	10.79	27.67	NW
4/9/96	35.2	64.17	NW	10.28	28.29	NW
5/7/96	N/A	N/A	N/A	N/A	N/A	N/A
5/15/96	2.75	62.89	325.84	10.31	30.16	NW
6/10/96	-	-	-	-	-	-
6/11/96	3.86	64.66	325.24	10.18	31.82	NW
6/23/96	4.19	62.31	325.26	10.03	25.21	NW
7/18/96	4.31	62.93	NA	9.45	26.28	NA
7/26/96	4.38	42.33	NW	9.24	26.55	NW
8/23/96	3.96	63.36	NA	9.33	27.59	NA
8/30/96	4.30	62.87	NA	NA	NA	NA
10/2/96	4.34	64.33	NW	9.36	28.87	NW
10/12/96	5.93	63.32	327.44	9.62	29.12	NW
11/12/96	4.15	63.89	NA	9.56	29.975	316.2??
11/21/96	4.545	63.75	NA	9.555	30.23	NW
1/7/96	4.35	62.36	NW	9.60	31.27	316.68
1/22/97	4.245	61.75	NW	9.61	31.575	NW
2/13/97	3.88	61.51	NW	9.7	32.00	NW
2/20/97	-	-	-	-	-	-
3/2/97	-	-	-	-	-	-
3/13/97	3.31	61.4	NW	9.745	32.435	NW
3/20/97	3.285	61.47	327.84	8.95	31.615	NW

N/A Indicates broken tape measure.

NW Indicates no measurable water level.

* Measuring tape not fully labeled so there is a greater, and unknown error with these readings.

\- No measurement taken on this date.

water levels to recover and then collect the groundwater sample. Recovery was so slow, however, that the wells took several days to produce a sample. The current plan is to purge the wells until dry, then return to sample within five to ten days. Table 7.3 summarizes the parameters determined in the field with the flow-through cell.

Samples were collected through an in-line filter (0.45 micron) attached to the dedicated pumping system into sample bottles to overflowing. One 250 ml sample and one 500 ml sample were collected and preserved following USEPA (1979; 1984a; 1984b) guidelines (Table 7.4). The original plan called for a greater volume of sample, but the small yields produced by the wells forced a reduction in total sample volume. All proposed analyses were able to be completed on the reduced sample volume. The samples were placed in an ice bath immediately after collection and kept cold for transport to the ISGS within 36 hours, along with the chain of custody/analysis request forms.

Tables 7.1 and 7.2 summarize depth to water in each of the monitoring wells on the site since sampling began in August, 1996. Tables 7.6-7.10 present the water quality data collected with the flow-through cell and Tables 7.11-7.18 summarize water quality data determined in the laboratory since sampling began in February, 1996. Results from the first sampling event do not have reliable anion data. Samples collected for anion analysis were mistakenly acidified in the field. The total dissolved solids (sum of anion and cation concentrations) have increased through time, reflecting a gradual change in chemistry to conditions that existed prior to well development (Figures 7.7 and 7.8).

The purpose of well development is to remove fines from the well casing and screen to improve the hydraulic connection between the well and monitored geologic unit. Wells were developed by injecting water under pressure through a 1 inch plastic pipe. Water was acquired at Pawnee from the city supply and hauled to the site. This method of well development suffers from one drawback; clean water remains in the casing after the pump is shut down. Some of this water will seep into the monitored interval, altering groundwater chemistry. All wells at the site yielded relatively clear water after development. The wells will have to undergo many purges before a representative sample of background water quality is collected.

Background water quality collected during Phase II indicates nonpotable groundwater in the strata immediately above and within the coal. In general, water with total dissolved solids in excess of 3,500 mg/l, as found in the coal and its bounding formations, is considered beyond treatment. Through time, the total dissolved solids have increased to values near 10,000 mg/l (Figure 7.7 and 7.8). These values were expected in this area at depths in excess of 200 ft. In fact, Meents and others (1952) reported total dissolved solids from a well in Christian County finished in Pennsylvanian Rocks at a depth of only 176 ft of over 44,000 mg/l. Drever (1988) presented a classification of groundwater, calling all nonpotable water between 1000 and 20,000 mg/l brackish. The groundwater collected from the study area under this classification scheme is brackish, whereas the water collected by Meents would be classified as saline or brine.

The dominant cation in the groundwater samples collected from the study area is sodium; the dominant anion is chloride. Other major constituents include calcium, magnesium, potassium and sulfate (Figures 7.7-7.8). The groundwater contains boron in concentrations less than 5 mg/l.

Table 7.3: Field parameters analyzed in the proposed study along with tentative analysis method.

Parameter	Method	Preservation
Water Temperature	Flow Through Cell - Single Probe (QED Model FC1000) (0.1 degrees Celsius)	Field Determined Unfiltered
Specific Conductance	Flow Through Cell - Single Probe (QED Model FC1000) (0.1% of range)	Field Determined Unfiltered
pH	Flow Through Cell - Single Probe (QED Model FC1000) (pH units 0.01)	Field Determined Unfiltered
Dissolved Oxygen	Flow Through Cell - Single Probe (QED Model FC1000) (0.1 ppm)	Field Determined Unfiltered
Hydraulic Head	Electric Tape (within 0.01 feet)	NA

Table 7.4: Cations and anions for laboratory analysis, analysis method, and sample preservation method.

Parameter	Method	Preservation
Cations: Al, B, Ba Be, Ca, Cd, Co, Cr, Cu, Fe K, La, Mg, Mn, Na, Ni, Pb Sc, Si, Sr, Ti, Zn, Zr	ICP	Cooled to 4 Degrees C Filtered, Acidified to pH less than 2 with Nitric Acid
Cations: Se, As	ICP-MS	Cooled to 4 Degrees C Filtered, Acidified to pH less than 2 with Nitric Acid
Cations: Hg	Hg-AAS	Cooled to 4 Degrees C Filtered, Acidified to pH less than 2 with Nitric Acid
Anions: Cl, F, NO_3, SO_4	IC	Cooled to 4 Degrees C Filtered
Alkalinity: HCO_3, CO_3	Titration	Cooled to 4 Degrees C Filtered
Acidity	Titration	Cooled to 4 Degrees C Filtered

Table 7.5: Depth to water measurement, samples collected, and flow-through-cell parameters for deep wells in each well nest, February 6 to March 11, 1996.

Date	Well #	DTW (feet)	Time pumped (min)	Cation sample	Anion, Acidity, Alkalinity sample	Temperature (C)	ORP (mV)	Conductance (mhos)	pH	O2 (ppm)
2/6/96	2A	NW	38							
2/7/96	3	274.21	14	250	500, 250	17.5	27	11.3	12.87	2.3
2/6/96	4	341.43	20	250	IW (125)	13.3	169	2.51	8.21	3.1
2/7/96	4	ND	150	250	500					
2/6/96	5	NW	14	250	500, 250	15.7	56	17.2	12.88	4.3
2/6/96	6	NW	40							
3/2/96	2A	NW	20							
3/1/96	3	279.18	25	NW	IW (150)	11.7	-96	10	7.82	3.7
3/2/96	3	ND	65	IW (75)	NW					
3/1/96	4	341.77	118	NW	IW (25)	9.7	-78	5.03	7.24	2.3
3/2/96	4	ND	105	250	IW (75)					
3/2/96	5	NW	40							
3/2/96	6	NW	45							
3/11/96	2A	NW	18							
3/11/96	3	282.8	110	175	500					
3/11/96	3	ND	71	175	NS					
3/11/96	4	344.16	10	250	500	8.4	19	4.99	8.47	7.1
3/11/96	5	ND	20	IW	IW (25)					
3/11/96	6	NW	15							

Table 7.6: Depth to water measurement, samples collected, and flow-through-cell parameters for deep wells in each well nest, March 29 to May 7, 1996.

Date	Well #	DTW (feet)	Time pumped (min)	Cation sample	Anion Acidity Alkalinity sample	Temperature (C)	ORP (mV)	Conductance (mhos)	pH	O2 (ppm)
3/29/96	2A	NW	13							
3/29/96	3	281.95	57			11.3	-52	12.6	7.24	7.3
3/29/96	4	343.13	82			9.6	-44	7.25	6.96	3.4
3/29/96	5	NW	20							
3/29/96	6	NW	*							
4/9/96	2A	NW	10							
4/9/96	3	282.81	118	230	500					
4/9/96	4	343.91	26	250	500					
4/9/96	5	NW	18	IW	IW (20)					
4/9/96	6	NW	11							
5/15/96	2A	NW	11							
5/15/96	3	NW	103	IW (50-75)	500					
5/15/96	4	343.61	7	250	500	20.2	40	12.1	8.14	3.3
5/15/96	5	325.84	15							
5/15/96	6	NW	*							
5/7/96	2A	NW	10							
5/7/96	3	281.49	28			16.3	-58	15	11.7	4.6
5/7/96	4	342.48	98			16.0	-29	9.23	8.11	4.1
5/7/96	5	NW	7							
5/7/96	6	NW	*							

Table 7.7: Depth to water measurement, samples collected, and flow-through-cell parameters for deep wells in each well nest, July 10 to July 26, 1996.

Date	Well #	DTW (feet)	Time pumped (min)	Cation sample	Anion Acidity Alkalinity sample	Temperature (C)	ORP (mV)	Conductance (mhos)	pH	O2 (ppm)
6/11/96	2A	NW	10							
6/10/96	3	282.14	59			19.1	-61	13 uS	12.14	4.9
6/10/96	4	342.69	93			16.3	-44	16 uS	8.18	3.7
6/11/96	5	325.24	46			18.2	-108	28.9	12.258	4.5
6/10/96	6	NW	10							
6/23/96	2A	NW	10							
6/23/96	3	NW	112	IW (100)	IW (450)					
6/23/96	4	343.44	10	250	500	23.8	-75	15.8	7.87	3.4
6/23/96	5	325.26	23	250	500					
6/23/96	6	NW	10							
7/18/96	2A	NA	15							
7/18/96	3	NA	53			23.4	18	15.4	11.62	5.8
7/18/96	4	NA	64			19.2	42	16.4	8.02	8.4
7/18/96	5	NA	15							
7/18/96	6	NA	13							
7/26/96	2A	NW	10							
7/26/96	3	NW	60	NS	IW-(250)					
7/26/96	4	314.33	12	250	500	33.9	45	16.1	8.19	2.7
7/26/96	5	NW	8							
7/26/96	6	NW	20							

Table 7.8: Depth to water measurement, samples collected, and flow-through-cell parameters for deep wells in each well nest, August 30 to November 21, 1996.

8/30/96	2A	NA	15							
8/30/96	3	283.28	0**	**	**					
8/30/96	4	NA	26	250	500					
8/30/96	5	NA	10							
8/30/96	6	NA	NA	NA	NA					
10/2/96	2A	NW	10							
10/2/96	3	**	**							
10/2/96	4	255.75	46			15.8	21	17.7 uS	8.31	6.2
10/2/96	5	NW	13							
10/2/96	6	NW	10							
10/12/96	2A	NW	12							
10/12/96	3	**	**	**	**					
10/12/96	4	344.99	77	250	500					
10/12/96	5	327.44	8							
10/12/96	6	NW	10							
11/12/96	2A	345.72	10							
11/12/96	3	NA	**							
11/12/96	4	NA	15							
11/12/96	5	NA	12							
11/12/96	6	316.2 ? .	10							
11/21/96	2A	NW	11							
11/21/96	3	NA	**	**	**					
11/21/96	4	NA	95	500	250					
11/21/96	5	NA	10							
11/21/96	6	NW	*							

Table 6.9: Depth to water measurement, samples collected, and flow-through-cell parameters for deep wells in each well nest, January 7 to February 13, 1997

Date	Well #	DTW (feet)	Time pumped (min)	Cation sample	Anion Acidity Alkalinity sample	Temperature (C)	ORP (mV)	Conductance (mhos)	pH	O2 (ppm)
1/7/97	2A	352.73	10							
1/7/97	3	274.78	20			9.4	51	13.3	6.33	6.9
1/7/97	4	343.39	35			8.0	76	14.6	7.46	6.4
1/7/97	5	NW	10							
1/7/97	6	316.68	10							
1/22/97	2A	NW	10							
1/22/97	3	276.37	10	500	250	11.1	27	13.5	10.02	3.3
1/22/97	4	NA	20	500	250	11.0	39	16.7	7.4	4.1
1/22/97	5	NW	10							
1/22/97	6	NW	0							
2/13/97	2A	NW	8							
2/13/97	3	279.97	31			8.9	-125	15.4	7.01	7.3
2/13/97	4	NA	57			6.4	-165	17.3	6.94	2.8
2/13/97	5	NW	8							
2/13/97	6	NW	7							

Table 7.10: Depth to water measurement, samples collected, and flow-through-cell parameters for deep wells in each well nest, February 20 to March 20, 1997

Date	Well #	DTW (feet)	Time pumped (min)	Cation sample	Anion Acidity Alkalinity sample	Temperature (C)	ORP (mV)	Conductance (mhos)	pH	O2 (ppm)
2/20/97	2A	NW	0							
2/20/97	3	282795	82	500	250					
2/20/97	4	344.29	20	500	250					
2/20/97	5	ND	0							
2/20/97	6	ND	0							
3/12/97	2A	NW	0							
3/12/97	3	344.02	23			11.0	-9	20.8	7.72	5.7
3/12/97	4	NA	0							
3/12/97	5	NW	0							
3/12/97	6	NW	0							
3/20/97	2A	ND	0							
3/20/97	3	NA	155	350	175					
3/20/97	4	NA	26	500	250					
3/20/97	5	327.84	4							
3/20/97	6	NW	0							

IW Water yielded but insufficient for full sample. Amount (ml) collected in parentheses.
NA Electric tape measure not functioning
ND No depth to water measurement taken.
NS No water available for sampling.
NW No measurable water in well.
* Could not access well with sampling equipment.
** Well damaged and unpumpable

Table 7.11: Laboratory analysis of cation constituents, February and March sampling, 1996. All units are mg/l, P unless noted.

Lab #	Date	Well #	Al	As	B	Ba	Be *	Ca	Cd	CO	CR	CU
W01003	2/6/96	4	<0.02	<0.1	0.42	0.14	<2	39.7	<0.01	<0.01	<0.01	<0.01
W01005	2/6/96	5	0.29	<0.1	0.42	1.73	<2	115	<0.01	<0.01	<0.01	<0.01
W01006	2/7/96	3	0.11	<0.1	0.54	0.37	<2	10.1	<0.01	<0.01	<0.01	<0.01
W01004	2/7/96	4	<0.02	<0.1	0.4	0.12	<2	38.8	<0.01	<0.01	<0.01	<0.01
W01007	2/7/96	blank	<0.02	<0.1	<0.02	<0.02	<2	<0.01	<0.01	<0.01	<0.01	<0.01
W01063	3/10/96	3	0.1	<0.1	0.77	0.19	<2	14.8	<0.01	<0.01	<0.01	<0.01
W01064	3/10/96	4	0.03	<0.1	0.51	0.14	<2	67.8	<0.01	<0.01	<0.01	<0.01
W01065	3/10/96	blank	<0.01	<0.1	<0.02	<0.01	<2	<0.01	<0.01	<0.01	<0.01	<0.01
Lab #	Date	Well #	FE	K	LA *	LI	MG	MN	MO	NA	NI	PB
W01003	2/6/96	4	0.02	7	<2	0.03	12	0.09	0.03	377	<0.03	<0.05
W01005	2/6/96	5	0.01	224	<2	1.51	0.01	<0.01	0.12	1605	<0.03	<0.05
W01006	2/7/96	3	0.01	418	<2	1.22	0.02	<0.01	0.13	1027	<0.03	<0.05
W01004	2/7/96	4	0.01	8	<2	0.03	12	0.08	0.02	374	<0.03	<0.05
W01007	2/7/96	blank	<0.01	<1	<2	<0.01	<0.01	<0.01	<0.02	<0.1	<0.03	<0.05
W01063	3/10/96	3	0.01	324	<2	1.02	0.01	<0.01	0.11	1680	<0.03	<0.08
W01064	3/10/96	4	0.04	12	<2	0.05	20.2	<0.01	<0.02	705	<0.03	<0.08
W01065	3/10/96	blank	<0.01	<1	<2	<0.01	<0.01	<0.01	<0.02	0.6	<0.03	<0.08
Lab #	Date	Well #	SB	SC *	SE	SI	SR	TI	TL	V	ZN	ZR
W01003	2/6/96	4	<0.2	<3	<0.1	1.79	0.62	<0.01	<0.6	<0.01	0.01	<0.01
W01005	2/6/96	5	<0.2	<3	<0.1	1.77	8.68	<0.01	<0.6	<0.01	0.3	<0.01
W01006	2/7/96	3	<0.2	<3	<0.1	1.79	1.54	<0.01	<0.6	<0.01	0.01	<0.01
W01004	2/7/96	4	<0.2	<3	<0.1	1.82	0.6	<0.01	<0.6	<0.01	0.02	<0.01
W01007	2/7/96	blank	<0.2	<3	<0.1	<0.01	<0.01	<0.01	<0.6	<0.01	<0.01	<0.01
W01063	3/10/96	3	<0.1	<3	<0.1	2.11	1.86	<0.01	<0.3	<0.01	0.01	<0.01
W01064	3/10/96	4	<0.1	<3	<0.1	2.13	1.31	<0.01	<0.3	<0.01	0.01	<0.01
W01065	3/10/96	blank	<0.1	<3	<0.1	<0.01	<0.01	<0.01	<0.3	<0.01	<0.01	<0.01

Table 7.12: Laboratory analysis of anion constituents, February and March sampling, 1996. . All units are mg/l, P unless noted.

Lab #	Date	Well #	HG *	CL	F	NO3 **	SO4 **	Total K ***	Pal K ***	Acid ***	TDS
W01003	2/6/96	4	<0.05	ND	ND	ND	ND	93.5	ND	4	NDT
W01005	2/6/96	5	<0.05	1064	ND	0.09	81.9	2532	2418	ND	NDT
W01006	2/7/96	3	<0.05	705	ND	0.19	250	901.3	823.1	ND	NDT
W01004	2/7/96	4	<0.05	478	ND	0.1	214	127.6	ND	3	NDT
W01007	2/7/96	blank	<0.05	ND	ND	ND	ND	ND	ND	ND	NDT
W01063	3/10/96	3	<0.05	1668	ND	0.07	549	1033	953	ND	NDT
W01064	3/10/96	4	<0.05	1113	ND	0.08	291	150.9	ND	5	NDT
W01065	3/10/96	blank	0.07	2.14	ND	<0.01	0.31	1	ND	2	NDT

Remarks:

Total K Alkalinity to pH 4.5 as mg $CaCO_3$/L
Pal K Alkalinity to pH 8.3 as mg $CaCO_3$/L
Acid Acidity to pH 8.3 as mg $CaCO_3$/L

Analytic Method Codes:

* µg/L, Inductively coupled plasma
** mg/L, Ion chromatography
*** mg/L, Titrimetric
ND No detection
NDT Not determined

Table 7.13: Laboratory analysis of cation constituents, April, May, and June sampling, 1996. All units are mg/l, P unless noted.

Lab #	Date	Well #	Al	As	B	Ba	Be *	Ca	Cd	CO	CR	CU
W01122	4/9/96	3	0.1	<0.1	0.77	0.14	<1	20.2	<0.01	<0.01	<0.1	<0.01
W01123	4/9/96	4	0.04	<0.1	0.58	0.14	<1	97.6	<0.01	<0.01	<0.1	<0.01
W01229	5/15/96	3	0.04	<0.1	0.78	0.13	<1	19.4	<0.1	<0.1	<0.1	0.01
W01228	5/15/96	4	<.02	<0.1	0.66	0.15	1	118	<0.1	<0.1	<0.1	0.01
W01230	5/15/96	blank	<.02	<0.1	<0.01	<0.01	<1	0.02	<0.1	<0.1	<0.1	<0.01
W01363	6/23/96	3	0.05	<0.1	0.78	0.13	1	23	<0.1	<0.1	<0.1	<0.01
W01362	6/23/96	4	0.04	<0.1	0.82	0.16	2	168	<0.1	<0.1	<0.1	0.01
W01364	6/23/96	5	0.24	<0.1	0.27	4.78	2	328	<0.1	<0.1	<0.1	<0.01
W01365	6/23/96	blank	<.02	<0.1	<0.02	<0.01	<1	0.06	<0.1	<0.1	<0.1	<0.01
Lab #	Date	Well #	FE	K	LA *	LI	MG	MN	MO	NA	NI	PB
W01122	4/9/96	3	0.03	255	<2	0.88	0.03	<0.01	0.12	1780	<0.03	<0.05
W01123	4/9/96	4	0.09	12	<2	0.06	30	0.03	<0.02	1030	<0.03	<0.05
W01229	5/15/96	3	<0.01	229	<2	0.81	0.02	<0.01	0.12	1830	<0.3	<0.5
W01228	5/15/96	4	0.07	14	<2	0.07	36.7	0.03	<0.02	1260	<0.3	<0.5
W01230	5/15/96	blank	<0.01	<1	<2	<0.01	<0.01	<0.01	<0.02	1.1	<0.3	<0.5
W01363	6/23/96	3	0.04	185	<2	0.66	0.12	<0.01	0.1	2280	<0.02	<0.08
W01362	6/23/96	4	0.15	19	<2	0.09	52.7	0.04	<0.02	2040	<0.02	<0.08
W01364	6/23/96	5	0.02	159	<2	1.07	0.01	<0.01	0.15	4100	<0.2	<0.8
W01365	6/23/96	blank	0.02	<1	<2	<0.01	<0.01	<0.01	<0.02	0.36	<0.2	<0.8
Lab #	Date	Well #	SB	SC *	SE	SI	SR	TI	TL	V	ZN	ZR
W01122	4/9/96	3	<0.1	<3	<0.1	1.81	1.9	<0.01	<0.3	<0.01	<0.01	<0.01
W01123	4/9/96	4	<0.1	<3	<0.1	2.46	2.1	<0.01	<0.3	<0.01	0.01	<0.01
W01229	5/15/96	3	<0.1	<3	<0.2	1.68	1.97	<0.1	<0.3	<0.1	<0.01	<0.1
W01228	5/15/96	4	<0.1	<3	<0.2	2.7	2.79	<0.1	0.4	<0.1	0.01	<0.1
W01230	5/15/96	blank	<0.1	<3	<0.2	<0.01	<0.01	<0.1	<0.3	<0.1	<0.01	<0.1
W01363	6/23/96	3	<0.2	<3	<0.2	1.59	2.25	<0.1	<0.3	<0.1	<0.01	<0.1
W01362	6/23/96	4	<0.2	<3	<0.2	3.12	4.01	<0.1	<0.3	<0.1	0.02	<0.1
W01364	6/23/96	5	<0.2	<3	<0.2	0.68	12.75	<0.1	<0.3	<0.1	0.02	<0.1
W01365	6/23/96	blank	<0.2	<3	<0.2	<0.01	0.01	<0.1	<0.3	<0.1	0.03	<0.1

Table 7.14: Laboratory analysis of anion constituents, April, May, and June sampling, 1996. All units are mg/l, P unless noted.

Lab #	Date	Well #	HG *	CL	F	NO3 **	SO4 **	Total K ***	Pal K ***	Acid ***	TDS
W01122	4/9/96	3	0.31	2059	ND	0.04	684	833.6	789.4	ND	NDT
W01123	4/9/96	4	0.34	1527	ND	0.06	383	162.2	ND	7	NDT
W01229	5/15/96	3	0.06	2614	ND	<0.01	682	629.9	597.7	ND	NDT
W01228	5/15/96	4	0.09	2062	ND	<0.01	327	171.3	ND	9	NDT
W01230	5/15/96	blank	0.09	<0.01	ND	<0.01	<0.01	1.8	ND	ND	NDT
W01363	6/23/96	3	0.08	2994	ND	0.39	745	382.6	367.5	ND	NDT
W01362	6/23/96	4	<0.05	3385	ND	0.44	401	205.5	ND	13	NDT
W01364	6/23/96	5	<0.05	6045	ND	<0.01	26.8	2804	2760	ND	NDT
W01365	6/23/96	blank	<0.05	0.29	ND	<0.01	<0.01	0.9	ND	1	NDT

Remarks:

Total K	Alkalinity to pH 4.5 as mg $CaCO_3$/L
Pal K	Alkalinity to pH 8.3 as mg $CaCO_3$/L
Acid	Acidity to pH 8.3 as mg $CaCO_3$/L

Analytic Method Codes:

*	µg/L, Inductively coupled plasma
**	mg/L, Ion chromatography
***	mg/L, Titrimetric
ND	No detection
NDT	Not determined

Table 7.15: Laboratory analysis of cation constituents, July through November sampling, 1996. All units are mg/l, P unless noted.

Lab #	Date	Well #	Al	As	B	Ba	Be *	Ca	Cd	CO	CR	CU
W01497	7/26/96	3	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND
W01496	7/26/96	4	<0.02	<0.1	0.86	0.14	1	176	<0.01	<0.01	<0.01	<0.01
W01498	7/26/96	blank	<0.02	<0.1	<0.01	<0.01	<1	0.07	<0.01	<0.01	<0.01	<0.01
W01585	8/30/96	4	0.04	<0.1	1.00	0.15	3	200	<0.01	<0.01	<0.01	<0.01
W01586	8/30/96	blank	<0.01	<0.1	<0.02	<0.01	<1	0.01	<0.01	<0.01	<0.01	<0.01
W01771	10/12/96	4	0.06	<0.1	0.89	0.12	<1	166	<0.01	<0.01	<0.01	0.01
W01772	10/12/96	blank	<0.03	<0.1		<0.01	<1	0.01	<0.01	<0.01	<0.01	<0.01
W01867	11/21/96	4	<0.03	<0.1	0.99	0.12	<1	186	<0.01	<0.01	<0.01	0.01
W01868	11/21/96	blank	<0.03	<0.1	<0.01	<0/01	<1	0.03	<0.01	<0.01	<0.01	<0.01

Lab #	Date	Well #	FE	K	LA *	LI	MG	MN	MO	NA	NI	PB
W01497	7/26/96	3	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND
W01496	7/26/96	4	0.11	20	<2	0.09	56.9	0.07	<0.02	2290	<0.03	<0.04
W01498	7/26/96	blank	<0.01	<1	<2	<0.01	0.02	<0.01	<0.02	1	<0.03	<0.04
W01585	8/30/96	4	0.10	22	<2	0.09	63.5	0.10	<0.02	2770	0.05	<0.05
W01586	8/30/96	blank	<0.01	<1	<2	<0.03	0.01	<0.01	<0.02	1	<0.03	<0.05
W01771	10/12/96	4	<0.01	22	<2	0.10	53.9	0.10	<0.02	2510	0.05	<0.05
W01772	10/12/96	blank	<0.01	3	<2	<0.01	0.01	<0.01	<0.02	0.40	<0.03	<0.05
W01867	11/21/96	4	<0.01	21	<2	0.12	57.5	0.15	<0.02	2790	<0.03	<0.05
W01868	11/21/96	blank	<0.01	<1	<2	<0.01	0.01	<0.01	<0.02	<0.03	<0.03	<0.05

Lab #	Date	Well #	SB	SC *	SE	SI	SR	TI	TL	V	ZN	ZR
W01497	7/26/96	3	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND
W01496	7/26/96	4	<0.2	<3	<0.2	3.16	4.40	<0.01	<0.2	<0.01	0.01	<0.01
W01498	7/26/96	blank	<0.2	<3	<0.2	<0.01	<0.01	<0.01	<0.2	<0.01	<0.01	<0.01
W01585	8/30/96	4	<0.2	<7	<0.2	3.52	5.11	<0.01	<0.3	<0.01	0.02	<0.01
W01586	8/30/96	blank	<0.2	<7	<0.2	<0.01	<0.01	<0.01	<0.3	<0.01	<0.01	<0.01
W01771	10/12/96	4	<0.2	<3	<0.2	3.21	4.44	<0.01	<0.3	<0.01	0.04	<0.01
W01772	10/12/96	blank	<0.2	<3	<0.2	<0.01	<0.01	<0.01	<0.3	<0.01	<0.01	<0.01
W01867	11/21/96	4	<0.2	<3	<0.2	3.4	4.79	<0.01	<0.3	<0.01	0.02	<0.01
W01868	11/21/96	blank	<0.2	<3	<0.2	<0.01	<0.01	<0.01	<0.3	<0.01	<0.01	<0.01

Table 7.16: Laboratory analysis of anion constituents, July through November sampling, 1996. All units are mg/l, P unless noted.

Lab #	Date	Well #	HG *	CL	F	NO3 **	SO4 **	Total K ***	Pal K ***	Acid ***	TDS
W01497	7/26/96	3	ND	3259	ND	<0.01	759	210.5	205.8	6	NDT
W01496	7/26/96	4	.16	4007	ND	<0.01	417	226.9	ND	7	NDT
W01498	7/26/96	blank	<0.05	<0.01	ND	<0.01	0.31	2	ND	ND	NDT
W01585	8/30/96	4	<0.05	4242	ND	<0.01	454	239.2	ND	8	NDT
W01586	8/30/96	blank	<0.05	<0.01	ND	<0.01	<0.01	<2	ND	ND	NDT
W01771	10/12/96	4	<0.05	4490	ND	<0.01	434	247.6	ND	19	8204
W01772	10/12/96	blank	<0.05	<0.01	ND	<0.01	<0.01	<1	ND	ND	<10
W01867	11/21/96	4	<0.05	5203	ND	<0.01	449	262.0	ND	46	8584
W01868	11/21/96	blank	<0.05	<0.01	ND	<0.01	0.30	<1	ND	ND	<10

Remarks:

Total K	Alkalinity to pH 4.5 as mg $CaCO_3/L$
Pal K	Alkalinity to pH 8.3 as mg $CaCO_3/L$
Acid	Acidity to pH 8.3 as mg $CaCO_3/L$

Analytic Method Codes:

*	µg/L, Inductively coupled plasma
**	mg/L, Ion chromatography
***	mg/L, Titrimetric
ND	No detection
NDT	Not determined

Table 7.17: Laboratory analysis of cation constituents, January through March, 1997. All units are mg/l, P unless noted.

Lab #	Date	Well #	Al	As	B	Ba	Be *	Ca	Cd	CO	CR	CU
2059	1/22/97	3	<0.04	<0.1	1.72	0.05	<1	60.7	<0.01	<0.01	<0.01	<0.01
2060	1/22/97	4	0.07	<0.1	2.08	0.14	<1	200	<0.01	<0.01	<0.01	<0.01
2153	2/20/97	3	<0.02	<0.1	0.78	0.05	<1	35.8	<0.01	<0.01	<0.01	<0.01
2154	2/20/97	4	<0.02	<0.1	1.07	0.12	<1	177	<0.01	<0.01	<0.01	<0.01
2299	3/20/97	3	-	-	-	-	-	-	-	-	-	-
2300	3/20/97	4	-	-	-	-	-	-	-	-	-	-
2301	3/20/97	Blank	-	-	-	-	-	-	-	-	-	-
Lab #	Date	Well #	FE	K	LA *	LI	MG	MN	MO	NA	NI	PB
2059	1/22/97	3	<0.1	50	<2	.29	30.1	<0.01	0.04	3460	<0.03	<0.05
2060	1/22/97	4	<0.1	25	<2	.14	65.3	0.15	<0.02	3390	<0.03	<0.05
2153	2/20/97	3	<0.02	63	<2	.29	20.5	<0.01	0.04	3360	<0.03	<0.05
2154	2/20/97	4	<0.02	24	<2	.12	58.9	0.14	<0.02	3470	<0.03	<0.05
2299	3/20/97	3	-	-	-	-	-	-	-	-	-	-
2300	3/20/97	4	-	-	-	-	-	-	-	-	-	-
2301	3/20/97	Blank	-	-	-	-	-	-	-	-	-	-
Lab #	Date	Well #	SB	SC *	SE	SI	SR	TI	TL	V	ZN	ZR
2059	1/22/97	3	<0.2	<3	<0.2	0.74	3.77	<0.01	<0.3	<0.01	<0.01	<0.01
2060	1/22/97	4	<0.2	<3	<0.2	3.91	5.50	<0.01	<0.3	<0.01	1.43	<0.01
2153	2/20/97	3	<0.2	<3	<0.2	0.69	3.25	<0.01	<0.3	<0.01	<0.01	<0.01
2154	2/20/97	4	<0.2	<3	<0.2	3.65	5.09	<0.01	<0.3	<0.01	0.02	<0.01
2299	3/20/97	3	-	-	-	-	-	-	-	-	-	-
2300	3/20/97	4	-	-	-	-	-	-	-	-	-	-
2301	3/20/97	Blank	-	-	-	-	-	-	-	-	-	-

Table 7.18: Laboratory analysis of anion constituents, January through March 1997. All units are mg/l, P unless noted.

Lab #	Date	Well #	HG *	CL	F	NO3 **	SO4 **	Total K ***	Pal K ***	Acid ***	TDS
2059	1/22/97	3	<0.05	4002	ND	<0.01	847	109.6	108.4	ND	7470
2060	1/22/97	4	<0.05	5466	ND	<0.01	404	265.7	ND	52	8860
2153	2/20/97	3	0.05	5042	ND	<0.01	845	55.9	51.5	ND	8720
2154	2/20/97	4	<0.05	5947	ND	<0.01	413	281.0	ND	64	9600
2299	3/20/97	3	-	5326	ND	<0.01	832	52.5	35.2	ND	8946
2300	3/20/97	4	-	6456	ND	<0.01	425	284.2	ND	36.1	10080
2301	3/30/97	Blank	-	<0.01	ND	<0.01	<0.01	<1	ND	<1	<10

Remarks:

Total K	Alkalinity to pH 4.5 as mg $CaCO_3$/L
Pal K	Alkalinity to pH 8.3 as mg $CaCO_3$/L
Acid	Acidity to pH 8.3 as mg $CaCO_3$/L

Analytic Method Codes:

*	µg/L, Inductively coupled plasma
**	mg/L, Ion chromatography
***	mg/L, Titrimetric
-	Analysis not yet completed
ND	No detection
NDT	Not determined

Table 7.19: Tritium data collected from deep wells at the study site.

Sample	Well Depth (ft)	Tritium Concentration (TU)
Well 3B	285	4.11±0.24
Well 4A	345	5.19±0.25
Surface Water Supply		8.02±0.29

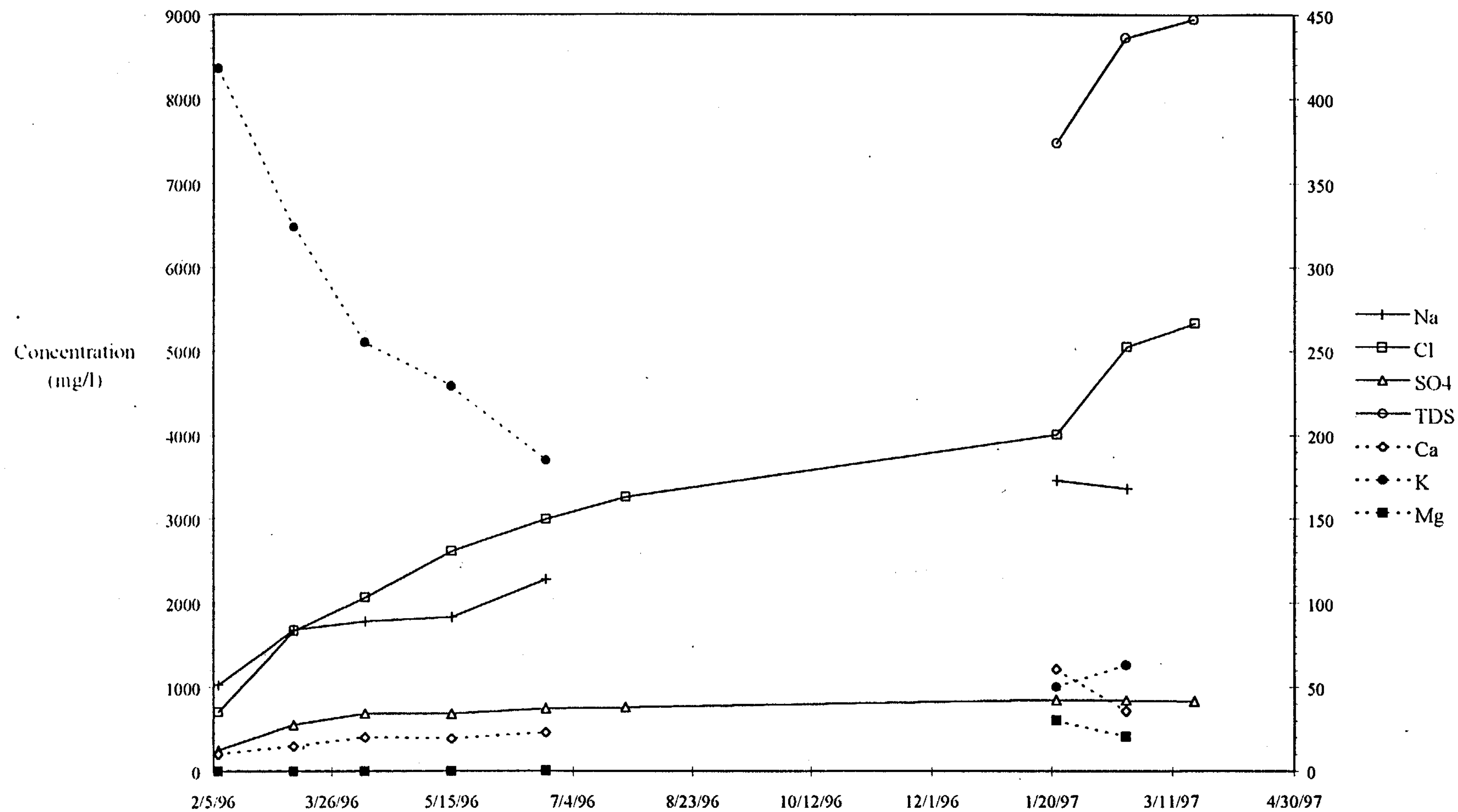

Figure 7.7. Major anions and cations of well 3. Solid lines refer to the left axes while dashed lines refer to the right axes.

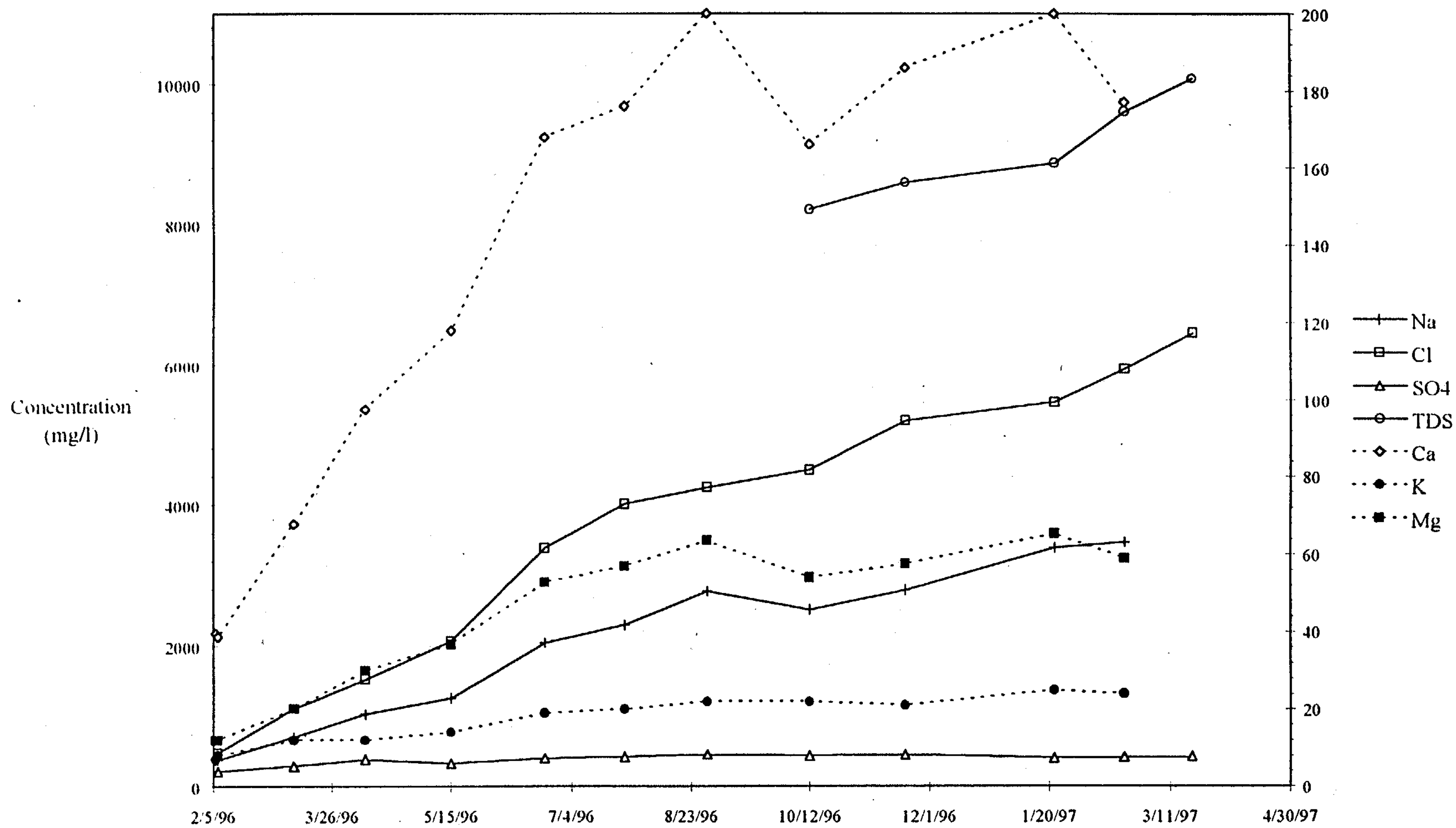

Figure 7.8. Major anions and cations of well 4. Solid lines refer to the left axes while dashed lines refer to the right axes.

Leachate generated by the coal combustion residues will contain boron in much greater concentrations, and this will be a key parameter for detecting leachate migration from the target panels. Figure 7.9 shows the dominant cations and Figure 7.10 the dominant anions through time for the two wells that consistently yielded samples (3 and 4). Note that Well 4 has total dissolved solids consistently greater than that of Well 3.

The monitoring wells were installed around the target panels in a pattern that should detect leachate generated by the coal combustion residues. No background wells are necessary because each well will have a record of groundwater quality prior to injection. A statistically significant detection of leachate generated by the coal combustion residues in each well will be based on an intrawell comparison between the water quality prior to and after injection. Since the present groundwater quality data indicates a serial correlation, some procedure to create temporally stationary data will have to be applied prior to any statistical tests.

7.4.3 Isotope Studies

Figures 7.7 and 7.8 suggest that the wells have not completely recovered from well development. The initial purpose of the isotope studies is to determine if tritium is present in the sampled water. Brackish water from deep in the Illinois Basin is below the zone of meteoric circulation and should not contain measurable quantities of tritium (an isotope of hydrogen). The presence of tritium would suggest contamination of the wells from some near surface supply of water, such as that used during well development. Table 7.19 shows the results of the analysis for tritium from samples collected from both Wells 3 and 4 in March, 1997. These data strongly suggest that the sampled wells still show the influence of well development. Additional groundwater samples will be collected during Phase 3 and analyzed for tritium to confirm these results and to test wells finished in other geologic strata. If funds allow, other isotopes will also be determined that could provide insight into the origin of the groundwater and background data for the detection of a leachate plume associated with the injection of coal combustion residues into the mine.

7.4.4 Groundwater Flow and Transport Models

The purpose of modeling is to determine if the disposal of coal combustion residues in an abandoned underground mine panel at the project site may impact potable groundwater resources. Specific modeling objectives as proposed included 1) develop a model of the steady state groundwater flow system in the strata bounding the target panel; 2) determine weakness in the available data through the calibration and sensitivity analysis of the flow model; and 3) apply models of contaminant transport from the target panels to worse case scenarios, based on the steady state flow system generated through flow modeling efforts.

A model calibrated to site conditions requires a significant data base, something that may not be available in this study. As discussed earlier, groundwater potential cannot be determined unless the density of the groundwater is known between points of measurement. The functional relationship between density and groundwater potential requires more than the two data points now available from the wells yielding groundwater samples. Nevertheless, generic modeling can

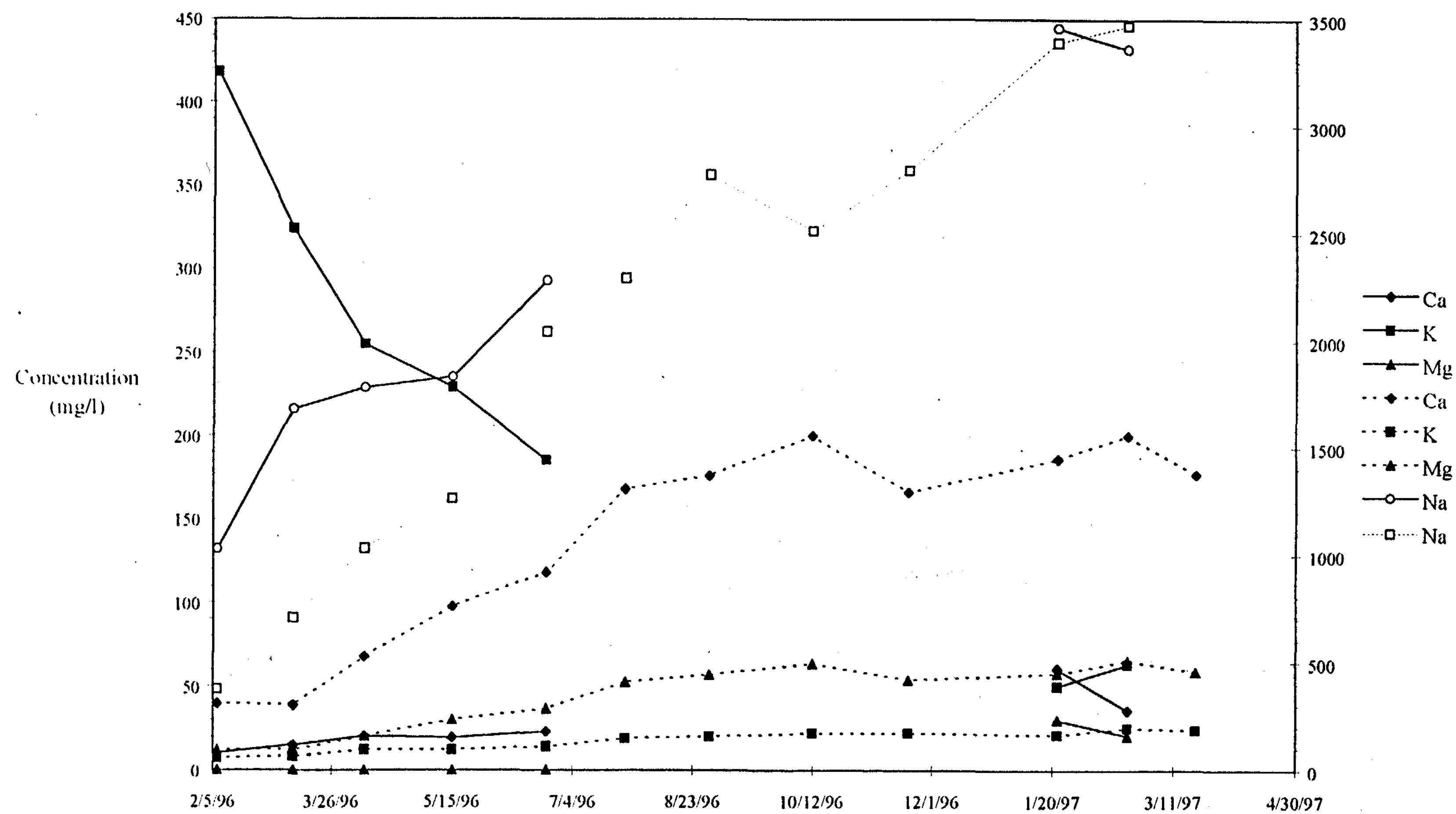

Figure 7.9. A comparison of calcium, potasium, magnesium, and sodium between well 3 (solid line) and well 4 (dashed line). The right axes is for sodium.

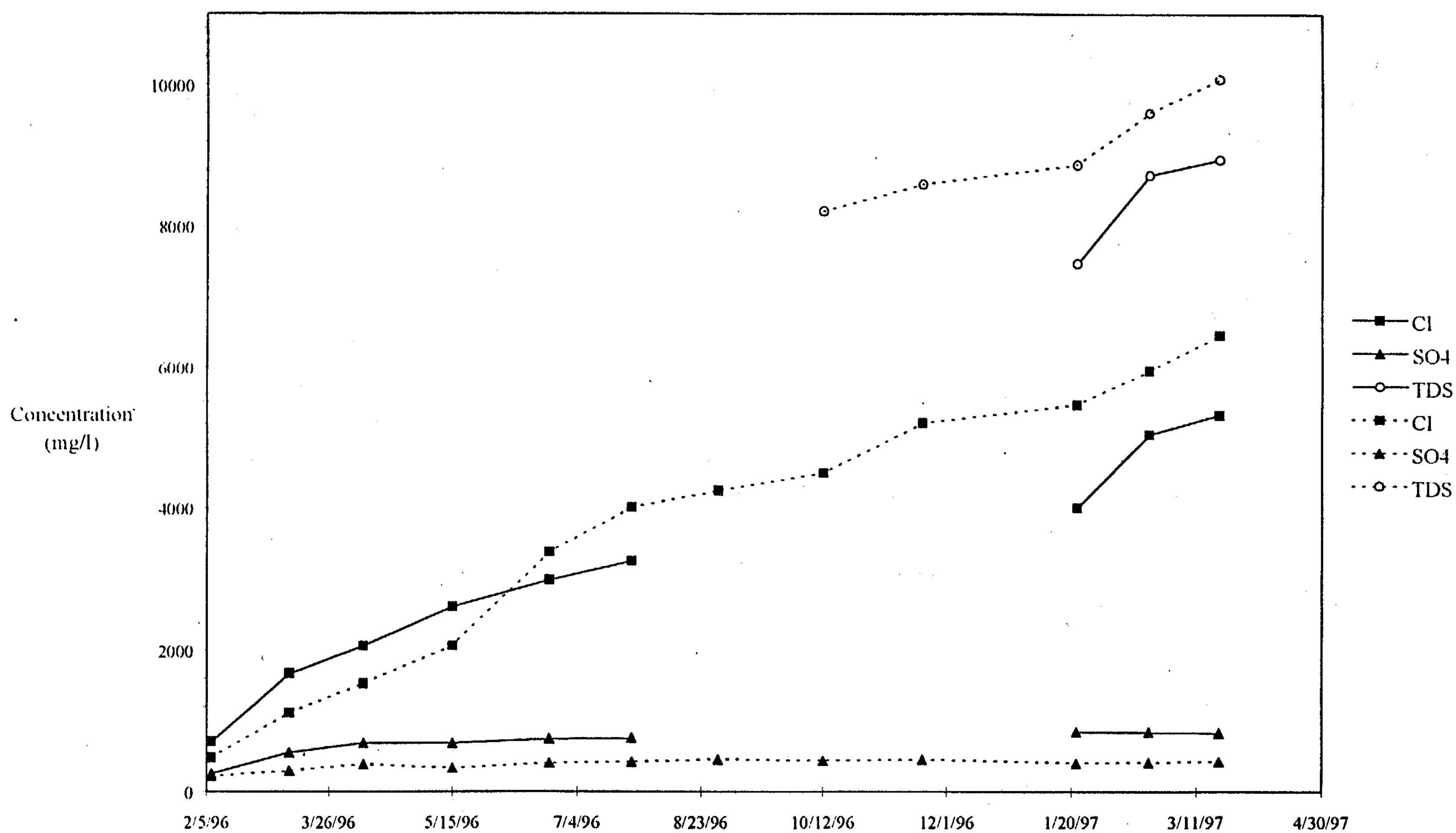

Figure 7.10. A comparison of chloride, sulfate, and total dissolved solids between well 3 (solid line) and well 4 (dashed line).

provide insight into the significance of different hydrogeologic features on groundwater flow at the mine. Current modeling efforts focus on generic modeling.

A widely used flow model (MODFLOW) developed by the United States Geological Survey (McDonald and Harbaugh, 1988) was used to develop a generic model of groundwater flow under two scenarios; a mine with rock of uniform hydraulic conductivity and a mine within the same rock, but with a sandstone layer at a depth of about 220 ft (to simulate the Trivoli Sandstone). These models can demonstrate the conditions that could lead to a gradient reversal, i.e., when groundwater could flow from the mine into the surrounding rock. All generic modeling results are currently under review.

As originally proposed, the environmental assessment will include simulations of contaminant transport away from the coal combustion residues with analytic and numeric algorithms. An analytic model of contaminant transport may be sufficient because of the relatively simple boundary conditions associated with the mine panel. Any model of contaminant transport must consider the results of recent column studies with combustion residues which indicate relatively high initial concentrations of contaminants, rapidly decreasing with time (Paul and Esling, 1993). This research also indicated that laboratory time is compressed. A scale-up based on three different column sizes suggested a first flush period of eleven months for the residues studied. Information from column studies such as that by Paul and Esling (1993) could provide relationships between the groundwater chemistry and the residue chemistry that could provide more realistic simulations of contaminant transport.

Calibration of the transport models is not possible if no contaminants reach one of the monitoring wells in less than one year of monitoring, a very real possibility. Because of this, the purpose of the contaminant transport modeling, like the flow modeling, is to investigate different scenarios for contaminant migration away from the target panels based on a range for each input parameter, including nominal, minimum, and maximum values. Further transport modeling must wait for the complete analysis of the generic flow models.

7.5 Summary and Conclusions

The purpose of environmental assessment is to determine the impact, if any, of injecting coal combustion residues underground on groundwater and surface water resources. During Phase 2, researchers refined the conceptual model of site hydrology and geology; monitored ambient water quality and hydraulic head in the strata immediately bounding the target panels; sampled and analyzed water samples for isotope geochemistry; and continued work on flow models of the site.

The rock units in the study area can be classified into seven hydrostratigraphic units based on hydrologic properties. The uppermost hydrostratigraphic unit, the unconsolidated surficial deposits, include relatively continuous sand and gravel units that contain the principal groundwater resources beneath the study area. In general, heads in the different monitored units appear relatively stable through time. Background water quality collected during Phase II indicates nonpotable groundwater in the strata immediately above and within the coal. Through time, the total dissolved solids have increased to values near 10,000 mg/l (brackish). The dominant cation is sodium; the dominant anion is chloride. Other major constituents include calcium, magnesium, potassium and sulfate. Tritium data and inorganic geochemistry suggest that the wells have not completely recovered from well development. Generic flow models that can provide insight into site conditions were developed for two scenarios; a mine with rock of uniform hydraulic conductivity and a mine within the same rock, but with a sandstone layer at a depth of about 220 ft. Transport modeling must wait for the complete analysis of the generic flow models, currently under review. Phase II work supports the earlier conclusion that the mine site has a favorable geologic and hydrologic setting for the disposal of coal combustion residues.

All tasks proposed for Phase II, with the exception of falling head permeability tests, were completed. Researchers working on environmental assessment refined the conceptual model of site hydrology and geology, regularly monitored groundwater quality and hydraulic head, completed initial isotope geochemistry work, and made progress on developing numeric models of site conditions. The environmental assessment test plan was modified. Sampling protocol was adapted to actual site conditions; only two deep wells regularly yielded samples. Researchers shifted their efforts to the isotope geochemistry in the deeper wells and the installation of sampling equipment for collecting geochemical samples from the shallower monitored units. Isotope geochemistry can provide data that indicates when the impact of well development no longer effects groundwater chemistry, as well as the age and origin of the groundwater.

The rock units in the study area can be classified into seven hydrostratigraphic units based on hydrologic properties. The uppermost hydrostratigraphic unit, the unconsolidated surficial deposits, include relatively continuous sand and gravel units that contain the principal groundwater resources of the study area. In general, heads in the different monitored units appear relatively stable through time. Background water quality indicates nonpotable groundwater in the strata immediately above and within the coal. Through time, the total dissolved solids have increased to values near 10,000 mg/l (brackish). The dominant cation is sodium; the dominant anion is chloride. Other major constituents include calcium, magnesium, potassium and sulfate. Tritium data and inorganic geochemistry suggest that the wells have not completely recovered from well development.

Generic flow models that can provide insight into site conditions were developed for two scenarios; a mine with rock of uniform hydraulic conductivity and a mine within the same rock, but with a sandstone layer at a depth of about 220 ft. Transport modeling must wait for the complete analysis of the generic flow models, currently under review. Phase II work supports the earlier conclusion that the mine site has a favorable geologic and hydrologic setting for the disposal of coal combustion residues. The target panels avoid complex geologic features in the region. Bounding strata have a low intrinsic permeability and at present the target panels are dry. The mine is located at depths between 325 and 375 feet below the land surface. The nearest potable groundwater supplies occur at depths less than 200 feet.

CHAPTER 8

DEVELOPMENT OF AN ALTERNATE TECHNOLOGY FOR HANDLING AND TRANSPORTATION OF FGD BY-PRODUCT

8.0 DEVELOPMENT OF AN ALTERNATE TECHNOLOGY FOR HANDLING AND TRANSPORTATION OF FGD BY-PRODUCTS

8.1 Introduction

In October, 1995, the Department of Energy granted additional funding under the cooperative research agreement to develop and demonstrate an alternate technology for the handling and transportation of FGD by-products. A study indicated that "Steel Intermodal Containers" (SIC) should offer an environmentally and economically sound technology for FGD By-Products transportation. Such intermodal containers are about 8 ft. wide, 8 ft. high, and 20 ft. in length, and can be readily transported by rail flat-car, by over the highway flatbed trucks, and by barges or other water transportation. Further, many such used containers are readily available at relatively low prices. It was recognized that the containers would have to be modified to meet the special problems associated with the handling and transportation of FGD by-products.

8.2 Initial Work

The researchers had proposed to modify the SIC so that it could 1) end-dump the material by filling the truck chassis, and 2) permit vacuuming the material out of the container at a rate of about 50 tons/hour. The former was to be achieved by modifying the current door design, while the latter was to be implemented by mounting a pressure-differential system similar to a pneumatic truck. Contacts with commercial pneumatic equipment designers, and SIC manufacturers indicated that vacuuming fly ash material from a flat-bed container may be extremely difficult and no commercial technology was available to achieve it.

SIUC investigators performed physical model studies to attempt innovative concepts to vaccum the fly ash materials from the container. Air jets to move the material to the vacuuming port, having inclined sides for the container with a vacuuming drain at the bottom, a sliding vacuum hose at the bottom was attempted in the laboratory in a wooden container about 2 ft. x 2 ft. x 5 ft. Each method demonstrated limited success with no more than 50-60% material removed. Considering significant costs associated with modifying the SIC container and limited success, the proposed idea of vacuuming fly ash from a flat container was shelved. Alternative cheap methods were sought which would meet the goal of developing an intermodal transportation technology, and permit loading and unloading of CCBs with fugitive dust control.

8.3 Work During Phase II

During Phase II, however, considerable progress was made in the development of the SIC for use in handling and transporting FGD by-products. Specifically, the following was achieved:

1. A material handling system concept using SICs, a Wilson Manufacturing conveyor, flat-bed rail car, and flat-bed truck chassis, was developed. The concept was developed in cooperation with Wilson Manufacturing Company of Cecilia, KY. and meets the approval of, all cooperators. The system is designed to handle fly ash and dry FGD by-products at the rate of 100 tons per hour, and incorporates a fugitive dust control system at the unloading point.

2. An engineering economic analysis of the system was performed and compared to the following existing or developing technologies.

   - Collapsible Intermodal Containers (CIC)
   - Covered Hopper Rail Cars
   - Open Rail Coal Cars with Tarping

For the analysis, the following assumptions were made, and are shown on Figure 8.1.

- Distance by rail from power plant to mine .... 100 miles
- Distance by road from mine to demand center .. 20 miles
- Tons to be handled per year .... 100,000 tons

The scenarios analyzed for each technology are discussed below.

Collapsible Intermodal Container (SEEC, Inc. System)

Four (4) Collapsible Intermodal Containers (CIC), each containing 20 tons of CCBs, are transported in an open coal car from the power plant to the mine. At the mine the CICs are unloaded from the coal car, placed on flat bed trucks, and transported to the demand center. After being emptied at the demand center the CICs are returned to the power plant. Analyses were conducted for back-haul railroad rates at 100% and 50% of the front-haul railroad rates.

Covered Hopper Cars

Each covered hopper car transports 100 tons of CCBs from the power plant to the mine. A Wilson Conveyor System is utilized to unload the CCB material from the hopper car to 20 ton SICs located on a flat-bed truck chassis. The SICs are then transported to the demand center and emptied using the developed material handling system.

Open Coal Cars with Tarping

After unloading a coal car at the power plant, the open coal car enters an enclosed area (about 150 ft. x 40 ft.) where about 100 tons of CCBs are loaded into the car. The top of the coal car is tarped with a disposable plastic material. The enclosed area is dust controlled. The loaded car is transported to the mine where the CCB material is unloaded using a Wilson Conveyor System into SICs located on a flat-bed truck chassis. The tarp is removed and the coal car is ready for coal loading. The SICs are transported by truck to the demand center and emptied using the developed material handling system.

Steel Intermodal Container (SIC)

In this developed system, four 20 ton SICs ride on a flat-bed rail car from the power plant to the mine. At the mine each SIC is unloaded and placed on a flat-bed truck chassis and transported to the demand center. At the demand center it is emptied using the developed material handling system. After being emptied, each SIC is returned to the mine site, loaded on the flat-bed rail car, and returned to the power plant.

8.4 Results of Engineering Economic Analysis

The computer software developed by Sevim (1996) under this cooperative research agreement was utilized for all analyses. Capital cost requirements and after-tax cash costs were calculated for each scenario. The results are summarized in Table 8.1 and are briefly described below:

- CIC and Coal Cars with Tarping systems are more expensive than Covered Hopper Cars and the proposed SIC system.
- The Covered Hopper Car system and the proposed SIC system have similar costs, however, the proposed SIC system cost is slightly lower.

The proposed SIC system has additional advantages over the Covered Hopper Car System which are not considered in the engineering economic analyses results shown.

- The SIC system provides for backhaul of coal or other materials from the mine to the power plant.
- Additional infrastructure is required at both the power plant and the mine site for covered hopper car system. For spot market sales the covered hopper car concept may be far more expensive than the proposed SIC system.

Based on these analyses it has been determined by all cooperators that the proposed SIC system is the least costly and utilizes existing, and proven, technology for its implementation.

During the final quarter of Phase II, Mr. Scott Renninger of the U. S. Department of Energy (COR for this program) and Dr. Y. P. Chugh (Program Director, SIUC) visited the Wilson Manufacturing Company in Cecilia, Kentucky, to discuss the proposed SIC system. Based on these discussions Mr. Renninger approved modifying the existing SIC and proceeding with a demonstration of the SIC materials handling and transportation concept.

8.5 Future Plans

Arrangements are being made with Wilson Manufacturing Company to perform all design and fabrication work on the SIC and to assist with the demonstration of the developed system. The Illinois Central Railroad staff is arranging for the delivery of the purchased SIC from New Orleans, Louisiana to Wilson's facility in Cecilia, Kentucky. It is expected that the modification of the SIC will be completed and a demonstration can take place in the early fall of 1997.

Also, a topical report will be prepared and submitted to the Department of Energy on the developed SIC system.

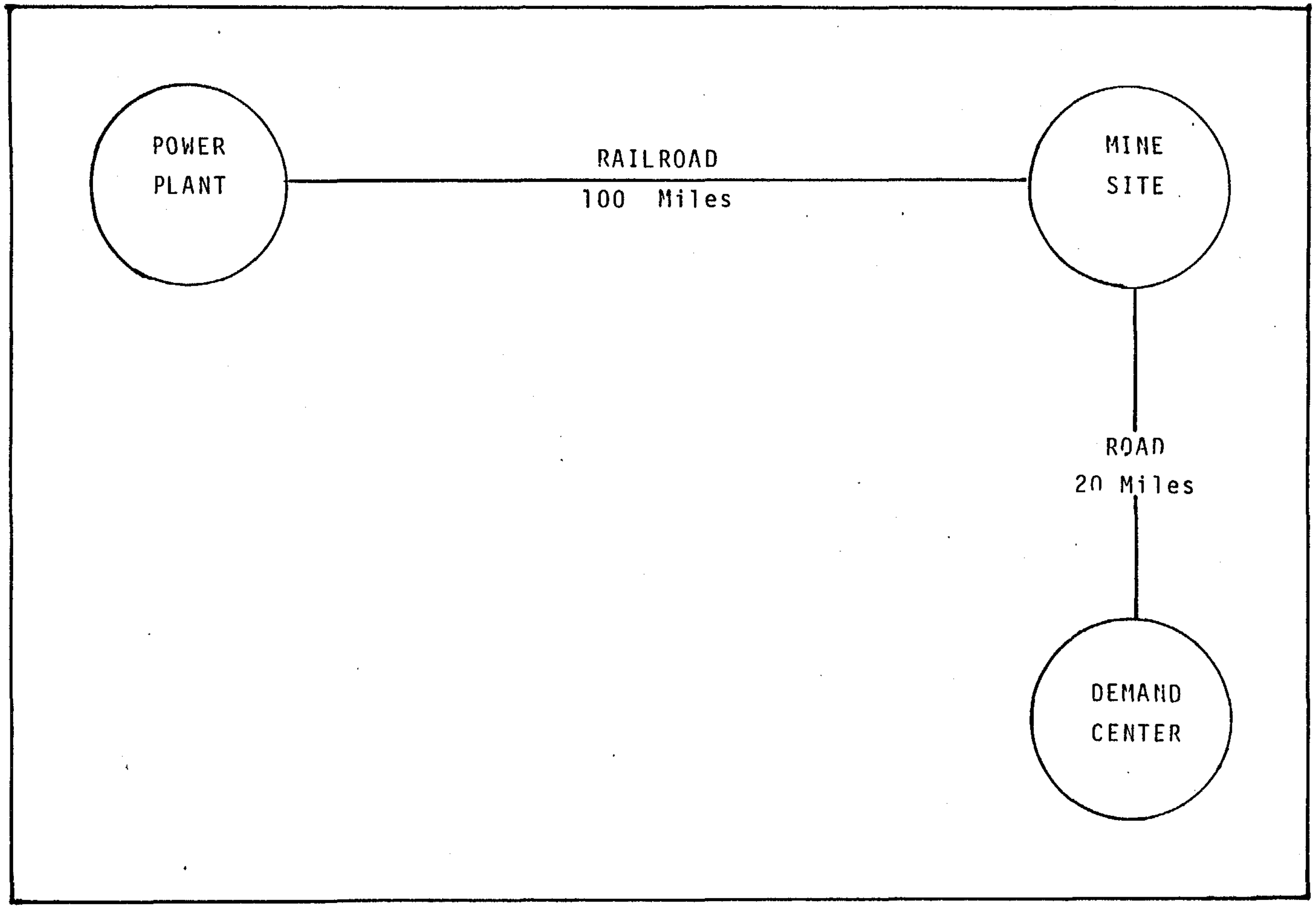

Figure 8.1 General Configuration of Power Plant, Mine Site, and Demand Center for the Engineering Economic Analyses

Table 8.1 Summary of Comparison Costs for Different Technologies

No.	Technology	Initial Capital Cost ($\$ \times 10^6$)	After-tax Cash Cost ($ 1 ton)	Remarks
1.	Collapsible Intermodal Container (SEEC, Inc.)	1.72	8.74	Back-Haul cost 100% of front-haul cost
			7.37	Back-Haul cost 50% of front-haul cost
2.	Open Coal Cars With Tarping	0.485	6.67	Back-Haul cost 100% of front-haul cost
3.	Covered Hopper Cars	0.515	6.19	One Person at mine site (100% Time)
4.	Steel Intermodal Containers (proposed system)	0.656	5.99	

CHAPTER 9

SUMMARY AND CONCLUSIONS

9.0 SUMMARY

9.1 Introduction

This report presents results of Phase II studies of the research cooperative agreement between the U.S. Department of Energy and Southern Illinois University at Carbondale entitled "Management of Dry Flue Gas Desulfurization By-Products in Underground Mines" (DE-FC21-93MC-30252). The term 'management' implies placement of by-products underground for disposal and/or beneficial use.

The principal objective of Phase II of the program was to construct or acquire the necessary hardware for both the hydraulic and pneumatic underground placements, to erect such hardware at the demonstration site, Peabody Coal Company Mine No. 10 near Pawnee, Illinois, and to test and "de-bug" the equipment and develop the equipment operational parameters.

Preliminary tests at the demonstration site revealed some flaws in the operation of both the hydraulic and pneumatic hardware. By the end of phase II, however, most of the "de-bugging" was complete, and both the hydraulic and pneumatic placement systems had been tested and shown to be in good working order. Several hundred tons of CCB's were placed under-ground, and preparations were underway for the full-scale Phase III demonstrations.

9.2 Summary Results

9.2.1 Environmental Studies

The analyses of the coal combustion by-products to be placed underground at the Peabody Mine # 10 demonstrates that the underground placement presents no dangers to potable water aquifer. The permeability of the placement material is low. It was decided that in Rapid Age and ASTM column tests of the mixes no compactive efforts would be assumed for the mixes such that hydraulic conductivity under the worst scenario can be evaluated.

Based on the hydraulic conductivity results it may be summarized that it seems unlikely that the proposed mixes could release a large enough volume into the environment to adversely impact local water supply. The Rapid Age tests will be continued to determine the trends of very long term hydraulic conductivities.

9.2.2 Geotechnical Studies

In Phase II of the project efforts were concentrated on measurements of surface and subsurface movements over the proposed injection panels. Also, the long-term swelling strain measurements of pneumatic mixes was conducted in the laboratory. Underground roof-to-floor convergence measurements in Phase I of the program were approximately 0.5 inches of convergence per year or when translated is 0.17 inches of surface movements per year. The results show that the surface movement data over a period of six-month is consistent with the underground measured roof-to-floor convergence data.

However, surface movements over the hydraulic injection panel for an elapsed time of 148 days show approximately 1 mm of downward movements. Over the entire area, downward surface movements of approximately 4 to 5 mm were observed. The average of all the monuments showing downward movements was -3.54 mm and 69% monuments showed downward movements over the hydraulic injection panel.

Three samples of pneumatic mix with 80% FBC fly ash and 20% spent bed ash were monitored for long-term free swelling strain. Average swelling strain of the three samples after 36 days is approximately 1%, which is low because the samples were made using preconditioned FBC fly ash.

9.2.3 Materials Handling And System Economics

The activities of Phase II in materials handling and systems economics are mostly continuation of the activities completed in Phase I. The software required for this research project was completed after the addition of the CIT module and the development of the engineering design spreadsheets. The fifth technology, CHC, could not be developed due to lack of availability of cost and engineering data during the Phase II period.

For 50,000 tons of by-products per year, PT technology gave lower costs of product delivery to the underground placement system than the PD-car and CIT technologies up to approximately 70 miles. It was lower than the CIC technology up to 110 miles. Even at the significantly reduced distances, the PD-car and the CIT technologies remained lower than the costs of CIC technology.

The results of the runs for 200,000 tons annual production shows that the advantage of the PT technology over the others disappear at this annual production. The difference among the other three technologies were insignificant up to approximately 140 miles, after which CIC became the better. Thus significant cost reductions occur in PD-car, CIT, and CIC technologies as the annual tonnage increases. In conclusion, PT technology is sensitive to distance but insensitive to tonnage, whereas the opposite is true for the other three technologies.

9.2.4 Underground Placement

The equipment necessary for the hydraulic placement of the CCBs were purchased from Montgomery County, Illinois and were transported to the Peabody Coal Company No.10 demonstration site. A high-capacity, high-pressure concrete pump was rented to provide the necessary power to the mix plant. A mixture of fly ash and scrubber sludge was obtained from the City, Water, Light, and Power Company in Springfield, Illinois for the hydraulic placement purposes.

It was observed that pastes from scrubber sludge and from 35% to 50% fly ash form pumpable mixes at 74% to 77% solids. Paste with 50% fly ash were stable. After sitting for 24 hours, all pastes showed signs of hardening to materials with significant shear strength. The addition of 1% lime accelerates the hardening process. It was decided to make the pumpable paste with 49% fly ash/bottom ash mixture, 49% scrubber sludge, and 2% lime. Since the end of

April, 1997, the investigators have made significant modifications to the mixing plant to provide a more uniform quality control of paste backfill mix. Since June 1, 1997, 250-tons of paste backfill with relatively uniform slump of 9.0+-0.2 inches without any breakdowns.

For the pneumatic injection placement of the CCBs, it was decided that the pneumatic injection equipment would be assembled at the University's Carterville facility. The completely assembled equipment was transported to a distance of about 175 miles. Based on the problems incurred during the second and third week of March tests, modifications were made to the hydraulic drive motors, so that the screws are driven through four-to-one reduction rather than by direct, and replacement of air powered hydraulic pumps with a diesel powered package.

During the first week of April a test run on the injection equipment was initiated using fly ash freshly delivered from dump trucks. The equipment operated satisfactorily, however the airborne dust created during the loading of the screw conveyor hopper was not acceptable. A spray system was installed on the mixing plant conveyor belts to eliminate the fugitive dust at the transfer stations prior to delivery to the pug mill.

In order to fully understand the pneumatic injection process, a physical model of a mine entry was constructed in the Mining Engineering Department Laboratory. The results to date indicate that the deposited fly ash can be transported a distance of 30 feet in the model. An attempt is being made to correlate data from model studies with observations in the field.

9.2.5 Environmental assessment And Geotechnical Stability And Subsidence Impacts

The purpose of environmental assessment is to determine the impact of injecting CCBs underground on underground and surface water resources. To achieve the results, researchers redefined the conceptual model of site hydrology and geology; monitored ambient water quality and hydraulic head in the strata immediately bounding the target panels; sampled and analyzed water samples for isotope geochemistry; and continued work on flow models of the site.

Generic flow models that can provide insight into site conditions were developed for two scenarios; a mine with rock of uniform hydraulic conductivity and a mine within the same rock, but with a sandstone layer at a depth of about 220 ft. Phase II work supports the earlier conclusion that the mine site has a favorable geologic and hydrologic setting for the disposal of CCBs.

9.2.6 Development of an Alternative technology for Handling And Transporting Of FGD By-Products

A study indicated that "Steel Intermodal Containers" (SIC) should offer an environmentally and economically sound technology for the handling and transportation of FGD by-products. During Phase II, a material handling system concept using SICs, a Wilson Manufacturing conveyor, flat-bed rail car, and flat-bed truck chassis, was developed. The concept was developed in cooperation with Wilson Manufacturing Company of Cecilia, KY, and meets the approval of, all cooperators. The system is designed to handle fly ash and dry FGD by-products at the rate of 100 tons per hour, and incorporates a fugitive dust control system at the unloading point. It is expected that the modification of the SIC will be completed and a demonstration can take place in the early fall of 1997.

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APPENDIX

USE OF BINOMIAL PROBABILITY DISTRIBUTION IN FLEET SIZE DETERMINATION

The probability distribution for a binomial random variable is given by:

$$p(y)=\binom{n}{y} p^{y} q^{n-y} \qquad (y=0, 1, 2, \ldots, n)$$

where:

p = Probability of a truck availability
q = Probability of a truck unavailability, (1-p)
n = Number of trucks in the fleet
y = Number of trucks working

$$\binom{n}{y}=\frac{n!}{y!(n-y)!}$$

Let us assume that in a truck fleet operation the overall fleet availability is desired to be 85 % (or 0.85). Engineering computations indicate that 10 trucks will be sufficient to do the job if each truck is 100 % available. What happens if truck availability is 90 %? The overall fleet availability is calculated as:

$$p\,(y=10)=\binom{10}{10} 0.9^{10}\, 0.1^{0} = 0.348$$

Clearly, 10 trucks are not sufficient to provide an overall availability of 85 %. Therefore, we must try a fleet of 11 trucks, and calculate the probability that at least 10 of them are in service at any time.

$$p\,(y \geq 10)=p(y=10)+p\,(y=11)=\binom{11}{10} 0.9^{10}\, 0.1^{1} + \binom{11}{11} 0.9^{11}\, 0.1^{0} = 0.697$$

As seen, adding one more truck to the fleet could not provide the desired 85 % availability. Next step is to increase the size by one more truck and recalculate the probability that at least 10 trucks are in service at any time.

$$p\,(y \geq 10)=p(y=10)+p\,(y=11)+p(y=12)=\binom{12}{10} 0.9^{10}\, 0.1^{2} + \binom{12}{11} 0.9^{11}\, 0.1^{1} + \binom{12}{12} 0.9^{12}\, 0.1^{0}=0.89$$

Since 0.89 is greater than 0.85 we conclude that 12 trucks were needed to ensure an 85 % overall fleet availability.