UNDERGROUND PLACEMENT OF COAL PROCESSING WASTE AND COAL COMBUSTION BY-PRODUCTS BASED PASTE BACKFILL FOR ENHANCED MINING ECONOMICS

FINAL REPORT

by

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Prepared for

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Project Contract No.: DE-FC26-99FT40553

June. 2002

ACKNOWLEDGEMENTS

On behalf of the project team, the principal investigator expresses his sincere gratitude to the following individuals and organizations.

- U.S. Department of Energy and National Energy Technology Laboratory for their financial and technical support. Particular thanks are due to Dr. Peter Botros and Mr. Scott Renninger, Project managers for the project.
- Illinois Clean Coal Institute, Office of Coal Development DCCA for financial support, and technical direction.
- Freeman United Coal Company and Crown III Operations staff for their unequivocal support. Particular thanks are due to Mr. Mike Caldwell and Mr. Bill Giles.
- West Virginia University for providing the *Groutnet* program for grout flow simulation.
- Project steering committee members for their time, encouragement, and their direction.
- Office of Surface Mining for providing the borehole camera and staff for borehole surveys.
- Cement-Tech Corporation of Iowa for donating high-speed auger system for the study.
- Members of the project team who worked diligently to complete the project under very challenging circumstances.

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ABSTRACT

This project has successfully demonstrated that the extraction ratio in a room-and-pillar panel at an Illinois mine can be increased from the current value of approximately 56% to about 64%, with backfilling done from the surface upon completion of all mining activities. This was achieved without significant ground control problems due to the increased extraction ratio. The mined-out areas were backfilled from the surface with gob, coal combustion by-products (CCBs), and fine coal processing waste (FCPW)-based paste backfill containing 65%-70% solids to minimize short-term and long-term surface deformations risk. This concept has the potential to increase mine productivity, reduce mining costs, manage large volumes of CCBs beneficially, and improve the miner's health, safety, and environment.

Two injection holes were drilled over the demonstration panel to inject the paste backfill. Backfilling was started on August 11, 1999 through the first borehole. About 9,293 tons of paste backfill were injected through this borehole with a maximum flow distance of 300-ft underground. On September 27, 2000, backfilling operation was resumed through the second borehole with a mixture of F ash and FBC ash. A high-speed auger mixer (new technology) was used to mix solids with water. About 6,000 tons of paste backfill were injected underground through this hole. Underground backfilling using the "Groutnet" flow model was simulated. Studies indicate that grout flow over 300-foot distance is possible. Approximately 13,000 tons of grout may be pumped through a single hole.

The effect of backfilling on the stability of the mine workings was analyzed using SIUPANEL.3D computer program and further verified using finite element analysis techniques. Stiffness of the backfill mix is most critical for enhancing the stability of mine workings. Mine openings do not have to be completely backfilled to enhance their stability. Backfill height of about 50% of the seam height is adequate to minimize surface deformations.

Freeman United Coal Company performed engineering economic evaluation studies for commercialization. They found that the costs for underground management at the Crown III mine would be slightly higher than surface management at this time.

The developed technologies have commercial potential but each site must be analyzed on its merit. The Company maintains significant interest in commercializing the technology.

EXECUTIVE SUMMARY

In order to maintain a healthy Illinois high-sulfur coal industry, production costs must be reduced and economically viable management technologies for coal combustion by-products (CCBs), fine coal processing waste (FCPW) and coarse coal processing waste (gob) must be developed. Over the past decade considerable research has been done in Illinois on high volume, low-value disposal/utilization technologies (disposal in surface mines, reclamation, disposal in abandoned underground mine workings). However, little or no work has been done to beneficially use these by-products in large volumes to enhance the economics of mining coal and power generation.

About 70% of the underground mined coal in Illinois is extracted using a room-and-pillar mining method that permits extraction of about 50% of the coal. The remaining coal is left behind in the form of support pillars to control surface and subsurface movements. Typically, power plants in Illinois, in rural settings, are presently spending about $10/ton to dispose of CCBs on-site in ponds. Nationwide, this cost is about is about $20/ton. This cost is expected to grow rapidly in light of new requirements for landfill sites. If coal companies could negotiate coal contracts with electric utility companies, which will reduce their CCBs management costs and cover the cost of underground backfilling and transportation, the hypothesis for partial extraction mining with backfilling is economically feasible. Implementation of this technology will result in strengthening the high sulfur Illinois coal industry and keeping the coal industry jobs in Illinois while providing a secure source of coal supply to power plants from their backyards.

Toward the above goal, this project has successfully demonstrated that: 1) the extraction ratio in a room-and-pillar geometry at the demonstration mine can be increased from current values of about 56% to about 64%, and 2) the mined-out areas can be backfilled from the surface with FCPW-, gob-, and CCBs- based backfills containing 65%-70% solids that will minimize short-term and long-term surface movement and acid-mine drainage potential, and 3) grout may be expected to flow 300 feet or more depending upon sheer stress of the grout. All demonstration studies were performed at Crown III mine near Springfield, Illinois.

Crown III mine of Freeman United coal company is currently mining 600 ft wide panels with 11 entries on 60 ft centers with 20 ft wide entries, and extraction ratio of 50% to 55%. Coal is extracted from the No. 6 coal seam at a depth of 300 to 350 ft. The panels vary in length from 3,000 ft to 5,000 ft. Seam height is seven feet and the weak floor strata are 2 to 4.5 ft weak claystone. The dip of the coal seam is about 1.6% in the southeast direction.

The mining company developed a small panel (hereafter called the backfilling panel) with eight entries and 80 ft by 60 ft pillar sizes (center-to-center). The entry width in the backfilling panels was 20 ft. Secondary mining was done in this panel to increase the extraction ratio to 64% from about 55%. In November 1997, three rows of pillars in the backfilling panel were notched to a depth of 20 ft by two cuts of 18 ft wide in each pillar.

In and around the demonstration panel, roof-to-floor convergence and surface subsidence data were collected periodically. Measurements taken on March 23, 1999, in the backfilling panel indicated about 1.8 inches of convergence at the center of the panel. Roof falls were observed at a few intersections and as a result some of the measuring stations were destroyed. Due to this reason and also due to safety concerns, no underground measurements were taken after this date. However, surface deformations of monuments along several subsidence grids were measured periodically. It was found that, on the average, surface deformations of about 1.16 inch occurred during the last one year.

Two steel-cased injection holes (6-inch inside diameter) were utilized to inject paste backfill in the panel. A concrete mixing plant was built to mix crushed gob, FBC fly ash, and F-type fly ash with water. Several preliminary mixes were developed using gob and FBC fly ash and F-ash. Their engineering properties were developed and documented. Two mixes were selected for underground demonstration purposes, one having 25% gob, and the other having 40% gob in the mix.

In order to demonstrate the flow characteristics of selected mixes a trench was dug on the surface with two perpendicular crosscuts. The trench was about 100 ft long, 9 ft wide and 6-10 ft deep. On August 9, 1999, the mix with 40% gob was pumped into this trench to observe the flow behavior. The mix flowed in all directions after discharge with little separation of water and solid components. It was also found that the mix flowed under water without much separation.

Due to the labor strike at Crown III mine, no progress was made in the field demonstration of underground backfilling until February 1999. Underground observations in March and borehole camera survey by the Office of Surface Mining (OSM) on July 7, 1999 showed that both the injection holes were open for backfilling. In the first (primary) hole, the camera was lowered to the mine floor level and the distances of coal pillars from the borehole were measured. There was a roof fall underneath the second (alternate) injection hole area. However, it was found that the entries in three directions were open from that borehole.

After two days of preparation, Phase I underground placement was started at 7:00 a.m. on August 11, 1999 through the first (primary) hole and the operation ended on September 8, 1999. About 8,159 tons of mix were pumped underground through the primary hole (5,873 ton of solid and 2,286 ton of water). The daily average backfilling rate of mix was 627 tons (452 ton of solid and 175 ton of water). The average water to solids ratio was about 40%. With this ratio, 11-inch slump for the mix was achieved. The average hourly pumping rate of the mix was 117.1 tons/hour (83.5 ton/hour of solid and 33.6 ton/hour of water).

On August 24, 1999, an underground visit of the backfilling panel revealed that the mix had flowed a considerable distance (about 120 ft), as expected. It was found that the flow pattern was sheet-like and uniform in all directions. The gradient of the backfilled material underground was 1 ft from the roof in all directions, 30-ft from the point of discharge. The backfilling operation was continued after that period. Mining Company staff visited the backfilled area again and found that the backfill had flowed about 300 ft from the primary borehole. During the early part of October the backfilling panel was sealed off under instructions from MSHA and another underground visit was not possible.

During the Phase I backfilling operation, cylindrical samples (3-inch diameter and 6-inch long) were prepared for testing for compressive strength, elastic modulus, slake durability, swelling strain and hydraulic conductivity of the cured backfill. In order to perform a sensitivity analysis of these results, five new mixes similar to the field mix were prepared in the laboratory by slightly varying the proportion of each mix component. For each mix, three (3) cylindrical samples were prepared to obtain average results. It was found that the average strength and elastic modulus after 28-day curing were 190 psi and 17,960 psi, respectively. Similar values after 90-day curing were 334 psi and 40,445 psi, respectively. After about 540 days of curing, the values are about 550 psi, and 56,000 psi, respectively.

Slake durability index (second cycle) for field and laboratory samples ranged from 75 to 89%, and 79 to 92%, respectively. Swelling strain for field and laboratory samples ranged from 6 to 10%, and 7 to 15%, respectively. Samples for testing hydraulic conductivity could not be prepared at Crown III mine site but were made in the laboratory. For pressure head between 30 to 50 psi, hydraulic conductivity varied from 0.01 to 0.06 inch/day. Thus, the backfilled material underground has very low permeability.

On October 13, 1999 backfilling operation resumed through the second (alternate) borehole. A concrete pump was used to pump the mix from the concrete plant site to this hole, a distance of about 250 ft. After four days of operation, 1,134 ton of solid and water (773 ton solid and 361 ton of water) was backfilled underground. Altogether using both boreholes, 9,293 ton of material was injected underground.

On September 18, 2000, a borehole camera survey was conducted again in the second borehole in cooperation with OSM to observe the underground conditions in the vicinity of the borehole. It was found that entries in the south, west and east directions were open. No new roof falls had occurred in this intersection. Thus, Phase II backfilling operation (through second hole) resumed on September 27, 2000. This time, a high-speed auger mixer was used to mix solids and water and then inject them underground as a paste backfill. This mix was composed of F fly ash and FBC fly ash. These two ashes were premixed in 1:2 ratio (F to FBC ash) by weight and dumped into a hopper using a front-end-loader. Water was added at the rear end of the auger mixer and grout mix came out from the front end (borehole side). As of October 30, 2000, about 6,000 tons of grout were injected through the second hole with a water to powder ratio of 0.47. It is estimated that the injected grout filled over 140,000 cft of underground voids. The injected grout filled about 900~1,000 ft of mine voids assuming average entry width and opening height of 20 ft and 7 ft, respectively. During this operation several 3x6 inch cylindrical samples were prepared for obtaining the compressive strength and elastic modulus of the injected grout.

The ASTM shake test was performed for the field backfill mixes. It was found that the mixes were environmentally benign. The pH of the leachate was 11.23, with Ca concentration of 669 ppm, and 1540 mg/l of dissolved solids. The concentration of most of the heavy trace elements was below the Class I ground water (GW) standard.

Strength and elastic modulus data from laboratory and field samples were analyzed using linear regression models. It was found that the ratio of the proportion between FBC ash and water content is the most important parameter for determining 7-day and 28-day cured strength and elastic modulus of mixes. The ratio between F ash and FBC ash also plays an important role in estimating 7-day compressive strength. These relationships were verified with the samples (similar to field samples) prepared in the laboratory. This analysis provides a mathematical foundation for forecasting strength and elastic modulus of samples composed of FBC ash, F-ash, gob and water.

The effect of backfilling on the stability of the mine workings was analyzed using SIUPANEL.3D computer program and further verified using finite element analysis techniques. Stiffness of the backfill mix is most critical for enhancing the stability of mine workings. Mine openings do not have to be completely backfilled to enhance their stability. Backfill height of about 50% of the seam height is adequate to minimize surface deformations.

The cooperating mining company performed engineering economic evaluation studies for commercial implementation of the concepts demonstrated. The company also sought input from an independent consultant regarding use of this technology at Crown III mine. The results indicate that underground management cost is comparable, but slightly higher, to that for surface management techniques currently practiced at the mine. The decision to implement underground management concepts must however be made on a site-specific basis. Underground management minimizes the capital cost for disposal ponds and closure costs for the ponds. However, the processing plant must be modified for alternate handling of refuse. Underground management also requires capital for mixing solids with water and the transportation of paste backfill to the borehole. Overall, economic evaluation studies indicate that this alternate underground management technology has potential for commercial implementation but it favors new mines with long life.

An economic evaluation of mine backfilling in this and earlier studies indicates that the amount of grout injected through each borehole is an important variable. This is particularly true where prime agricultural lands are involved, since the land acquisition and reclamation costs are very high. Therefore, a thorough understanding of grout flow in underground partial extraction mine workings is extremely important. Stiles (1999) of West Virginia University developed an approximate mathematical model of grout flow in room-and-pillar mine workings. Therefore, grout flow simulation was undertaken to develop a better understanding of grout flow in room-and-pillar coal mine workings typically encountered in Illinois. The overall goal of this task was to develop a better understanding of grout flow phenomenon in flat and slightly pitching coal seams and relative importance of variables, such as grout yield stress, grout hardening, and slope of the coal seam. Overall, the simulation results compared favorably with the experience in the field.

In summary, the project was successful in achieving its objectives. Underground placement of paste backfill mixes technology has been developed and demonstrated successfully for industry use.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

I. INTRODUCTION AND BACKGROUND

The history of mining is replete with many instances of backfilling to make a safe underground mining environment, control subsidence and increase coal extraction. Table 1 summarizes some of these studies conducted in the USA and abroad. Conclusions derived from earlier studies reported in Table 1 are summarized below.

i) In the USA, most backfilling has been done in abandoned coal mines to control surface subsidence. In addition, underground backfilling with mine tailings was done in deep silver and gold mines to enhance recovery and profitability.

ii) Underground backfilling was done mostly using low solid content slurry. Paste backfilling is an emerging technology, which offers higher economic and environmental advantages than the slurry backfill system (Brackebusch, 1994).

iii) Instances of systematic mix development to engineer a paste of appropriate structural and environmental characteristics are few and far between. Most systematic mix development procedures were directed towards developing flowable fills except a paste development study by the PI (Chugh et al., 1996c).

iv) Instances of integrated environmental studies in conjunction with backfilling are very few.

v) Engineered mixes were never developed to alter leaching behavior of individual components.

vi) Integrated approach to manage FCPW, gob, and CCBs to decrease disposal costs and enhance recovery has never been attempted.

The PI has addressed some of the above issues in developing and demonstrating the feasibility of a paste backfill system using various coal related by-products. For instance, in the project contracted by the DOE, environmentally benign and structurally appropriate grouts have been developed with 70% to 75% solids and pumped in an abandoned underground mine using a concrete pump (Chugh et al., 1996b). In the project funded by the ICCI, leaching behavior of individual grout components (FCPW and CCBs) has been altered to produce an environmentally benign paste with 60% to 65% solids and no visible bleed off water. The paste which had ASTM slump of 4 to 6 inches was pumped on the surface over a distance of approximately 450 ft through six-inch diameter steel pipes (Chugh, et al., 1996a).

The system developed under the U.S. Department of Energy grant (100 tons/hour) has been successfully demonstrated on the surface and for backfilling an abandoned underground mining panel. To the best of the authors' knowledge, paste backfilling in an active underground coal mine has not been demonstrated in the USA. The coal companies contacted believe that paste backfilling has the potential to enhance mining economics. However, backfilling capabilities and effectiveness must be demonstrated in an active mine before coal companies can seek permits for mining plans with backfill.

Table 1. Studies related to filling underground voids.

Authors and Sources	**Backfilling Reason**	**Comments**
Carlson (1975)	Subsidence control	Model studies.
Maser et al. (1975)	Subsidence control	Fly ash-cement mine sealant.
Whaite and Allen (1975)	Subsidence control	Slurry backfill.
Galvin and Wagner (1982)	Enhance extraction	South African coal mines to increase extraction by 8%-12%. Fly ash only.
Petulanas (1988)	Subsidence control	High volume use of fly ash in underground void filling. Low solid density mixes.
Palarski (1993)	Enhance extraction	In Polish coal mines. Fly ash, tailings, rocks.
Hollinderbaumer and Kramer (1994)	Subsidence and ground control	Integrated approach in German longwall mines to dispose of incinerator ash.
Meiers, et al. (1995)	Subsidence control	Fly ash-scrubber sludge mix.
Gray et al. (1995)	Acid Mine Drainage control	Disposal of FBC.
Chugh (1996c)	Acid Mine Drainage control	Fly ash, scrubber sludge-based pastes pumped into an abandoned Maryland mine.
Chugh et al (1996b)	Subsidence Control	Fly ash and scrubber sludge pastes pumped into an abandoned mine panel in Illinois.

The thrust of the current project was to develop and demonstrate the feasibility of pumping backfill material in an active mine to enhance mining economics by recovering more coal and increasing productivity.

II. GOALS AND OBJECTIVES

The goal of this project was to demonstrate that coal processing waste and CCBs-based paste backfill can be managed underground, which can increase extraction ratio, decrease production cost, and enhance the environment. The more specific objectives of the project are to:

1. Demonstrate that environmentally benign pumpable paste backfill mixtures containing 55% to 60% solids can be developed using FCPW (fine coal processing waste or coal slurry), gob, and CCBs (coal combustion by-products).

2. Demonstrate that the reduction of pillar sizes is possible without affecting the surface if the panel is subsequently backfilled.

3. Demonstrate that gob- and CCB-based high-density paste backfill can flow at least 300 feet from the injection borehole.

4. Study flow characteristics of paste backfill in entries and crosscuts during the pumping process and evaluate the extent to which entries and crosscuts are fully backfilled away from the injection point.

5. Study shrinkage, durability, and strength-deformation properties of the pumped backfill as curing progresses.

6. Study impacts of backfilling on surface movements.

Perform cost studies for paste backfill placement in an active mine.

III. EXPERIMENTAL PROCEDURES, RESULTS AND DISCUSSION

- **Demonstration Mine Characteristics**

The backfilling demonstration was conducted at Crown III mine of Freeman United Coal Company near Farmersville, IL. Crown III mine is currently mining 600 ft wide panels with 11 entries on 60 ft centers with 20 ft wide entries. Coal is extracted from No. 6 coal seam at a depth of 300 to 350 ft. The panels vary in length from 3,000 ft to 5,000 ft. Seam height is seven feet. The floor is 2-4.5 ft thick weak claystone and is dipping approximately 1.6% in the southeast direction of the panel.

- **Characteristics of the Backfilling Demonstration Area**

Figure 1 shows the panel under study. Mining was done in the regular panel which was 600 ft wide. After mining had progressed 2860 ft, bad roof conditions due to local geologic anomalies were encountered. To improve the roof conditions, pillar dimensions were changed from 40x40 ft to 60x40 ft (as shown in Figure 1). For the purpose of the demonstration, the mining company developed the backfilling panel (shown in Figure 1) with eight entries and 60 ft by 40 ft pillar sizes. The entry width in both the regular and the backfilling panels was 20 ft. In the backfilling panel, secondary mining was done to increase the extraction ratio to 65% from 50-55%. Three rows of pillars in the backfilling panel were extracted to a depth of 20 ft by two cuts of 18 ft wide in each pillar.

- **Surface and Underground Geotechnical Studies**

Rock mechanics studies involved underground visits prior to and after secondary mining in the backfilling panel, plate loading tests in a typical panel of regular geometry to estimate floor bearing capacity, installation of underground convergence points and surface movement monitoring stations, data collection from the monitoring stations, floor and pillar safety factor analyses for different pillar geometries and numerical modeling for predicting surface movements and pillar and floor stability. Four plate loading tests and assessment of floor safety factors and surface vertical movements by SIUPANEL.3D model were also performed.

Underground Visit: During a visit to the backfilling panel in December 1997, the mining areas were found to be stable throughout. Localized roof falls due to geologic anomalies were observed in one pillar extraction cut. Rib sloughing in the backfilling panel was no different than in the areas where no pillar extraction was done (in the regular panel). Roof bolts in general did not indicate signs of significant loading and all intersections were found stable. No floor heave was observed in any area. However, the last underground visit on March 23, 1999 showed different conditions of the roof and the floor. However, roof falls in a few intersections are considered normal at this mine for the geologic conditions present even without the higher extraction implemented in the demonstration area. Roof falls occurred in a few intersections with high volume of rock debris on the floor. A few measuring stations were destroyed due to roof falls. Rib failures were also visible in some coal pillars.

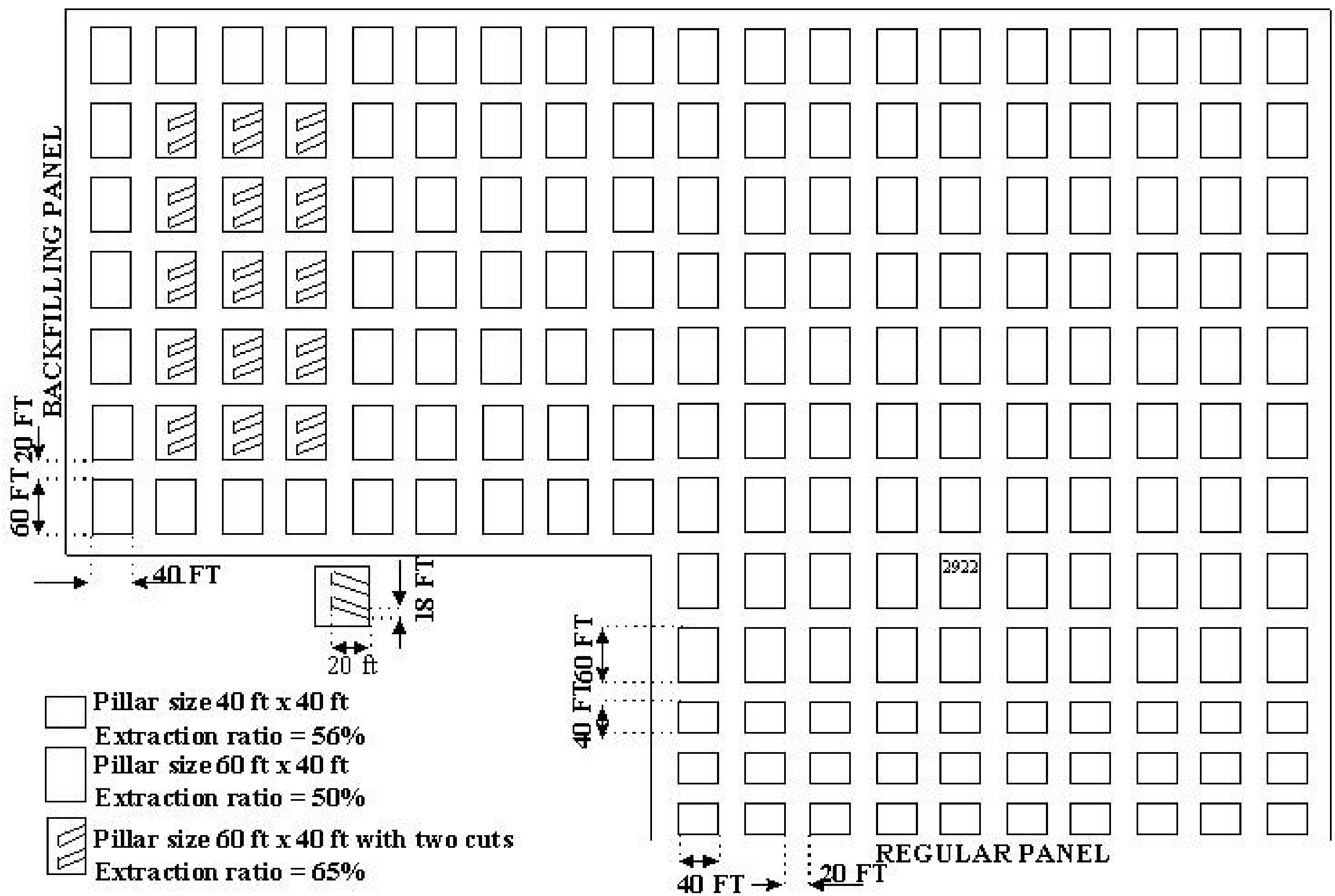


Figure 1 Panel geometry in the proposed backfilling areas

Surface Vertical Movements, Underground Convergence: The surface vertical movements monitoring network and the underground convergence stations are shown in Figure 2. Surface movement stations along line A-A', B-B' and C-C' were installed at variable intervals. A surface movement station consisted of a 7-ft long frost-free design roof bolt of 7/8-inch in diameter. The roof bolts were inserted into the ground to a depth of 5.5 to 6.0 ft. The top three feet of the bolt in the ground was surrounded by closed-cell foam insulation and a PVC pipe. An auto-set level was used to record the levels of the bolt heads.

Surface deformation was measured along X-X' line until March 2, 1998 (Figure 2). However, these deformation points were lost due to equipment movements, digging of the surface trench and dumping of coal combustion by-products. So, new points were established along A-A', B-B' and C-C' lines. Elevations of these points were measured in August 1998, March 1999 and again on August 12, 1999. It was found that average movement of the backfilling area was little over 1.16 inch for the last year and occurred uniformly downward over the entire area. Figure 3 shows the surface deformation along A-A', B-B' and C-C' lines. The maximum deformation of 1.8 inch was recorded around the primary borehole area. However, no differential ground settlement occurred over the backfilling area.

Underground convergence stations numbered G1 through G7 and H1 through H7 were monitored periodically to measure underground movements. A convergence station consisted of a roof bolt head and a square-head bolt, vertically beneath the roof bolt, anchored into the floor. It is designed to measure the roof-to-floor convergence using a convergence rod. Underground roof-to-floor convergence monitoring was carried out until March 23, 1999. Some of the convergence stations were damaged due to roof falls and reaching other measuring stations became unsafe. As a result, the underground monitoring program was abandoned.

Underground observation of the study area in March 1999 revealed that roof falls had occurred in a few intersections. In some cases, it was impossible to measure the height of debris fallen on the floor. Wherever possible roof-to-floor convergence data was collected from the underground measuring points to evaluate condition of entries in the study area. In some areas, about 1.8 inches of roof-to-floor movements were recorded. Figures 4 and 5 indicate the roof-to-floor convergence along GG' and HH' lines, respectively measured over the two-year period. Some underground areas were inaccessible due to roof falls. In general, the backfilling area was unsafe for traveling.

Figure 2. Location of underground convergence stations and surface subsidence monuments.

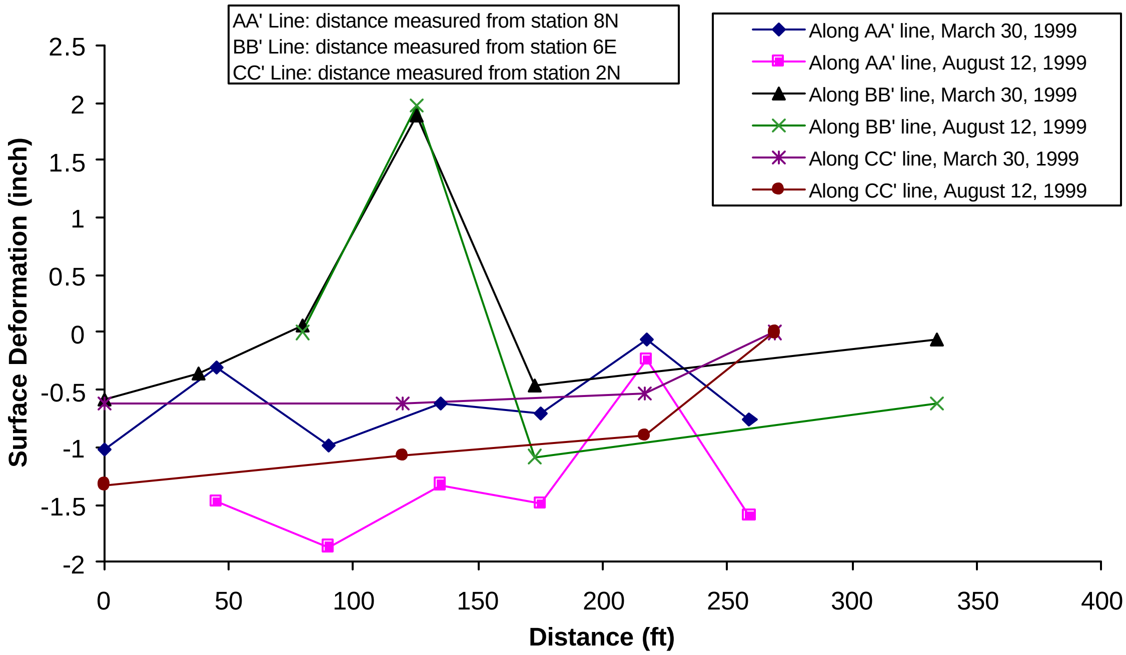


Figure 3. Surface deformation over the backfilling panel.

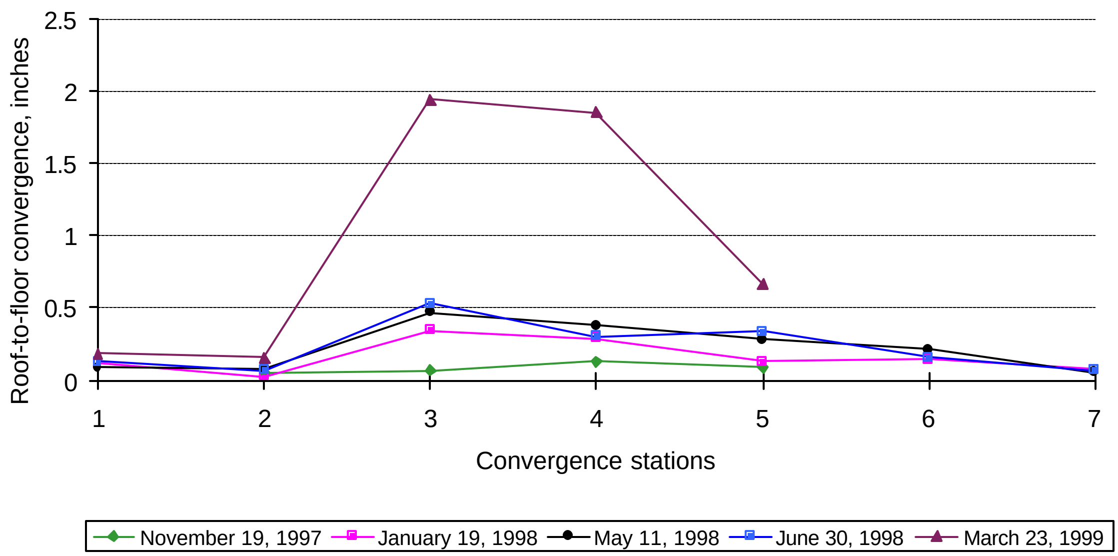


Figure 4. Roof-to floor convergence along GG' line.

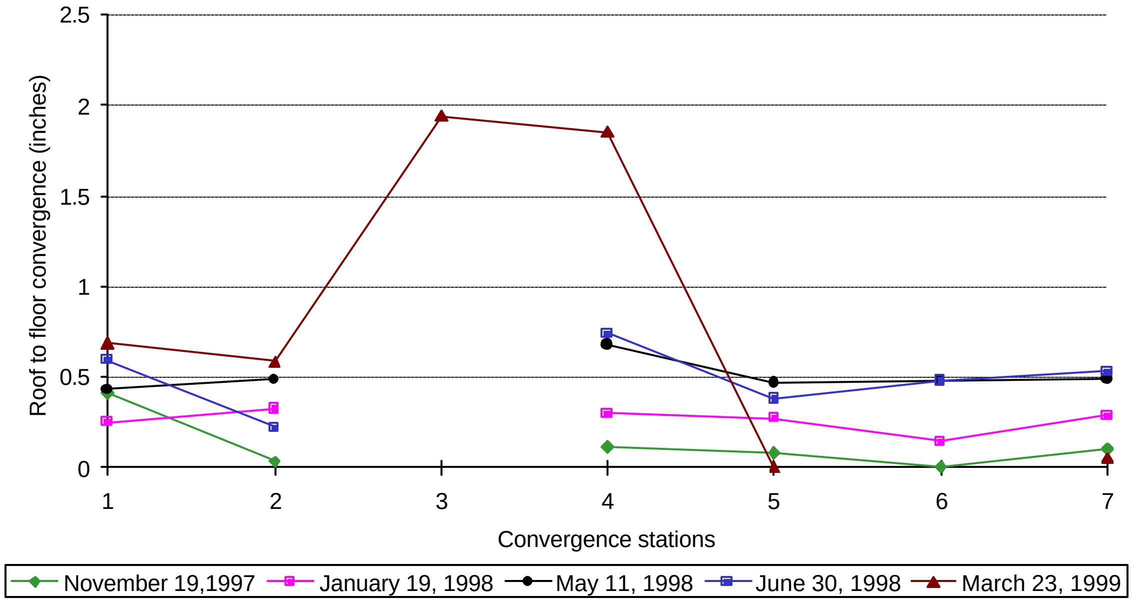


Figure 5. Roof-to-floor convergence along HH' line.

- **Mix Development and Selection of Backfilling Mixes**

One of the objectives of this project was to develop a backfill material with sufficient strength and stiffness that can enhance underground mine stability, and minimize additional ground movements. Compressive strength and elastic modulus of the backfill material are considered most important engineering properties. Therefore, laboratory studies were conducted to identify the most significant variables that dominate the outcome of these two engineering properties. For Illinois Basin mines, hydraulic conductivity, swelling strain and durability properties are also considered important since a weak claystone layer exists in the floor. The backfill material should behave as an impermeable layer with the maximum durability index.

Crown III mine supplied gob (coarse coal refuse of coal processing rejects) and FBC fly ash. As the sizes of gob varied between -4 inches to +28 mesh, the as-received gob was crushed to sizes less than 0.25 inches. Particle size distribution for gob is given in Figure 6. F-type fly ash was obtained from the Coffeen power plant. The moisture content of the as-received materials, particle sizes and calcium carbonate equivalent (CCE) values of the raw materials (gob, FBC and F-type fly ash) were determined. Mix development and selection of final mixes were reported earlier (Chugh et al, 1998). However, in the field demonstration, gob was crushed to a maximum size of 0.75 inch. Thus, gob particle size distribution in the field was slightly different from that of laboratory tests (Figure 6).

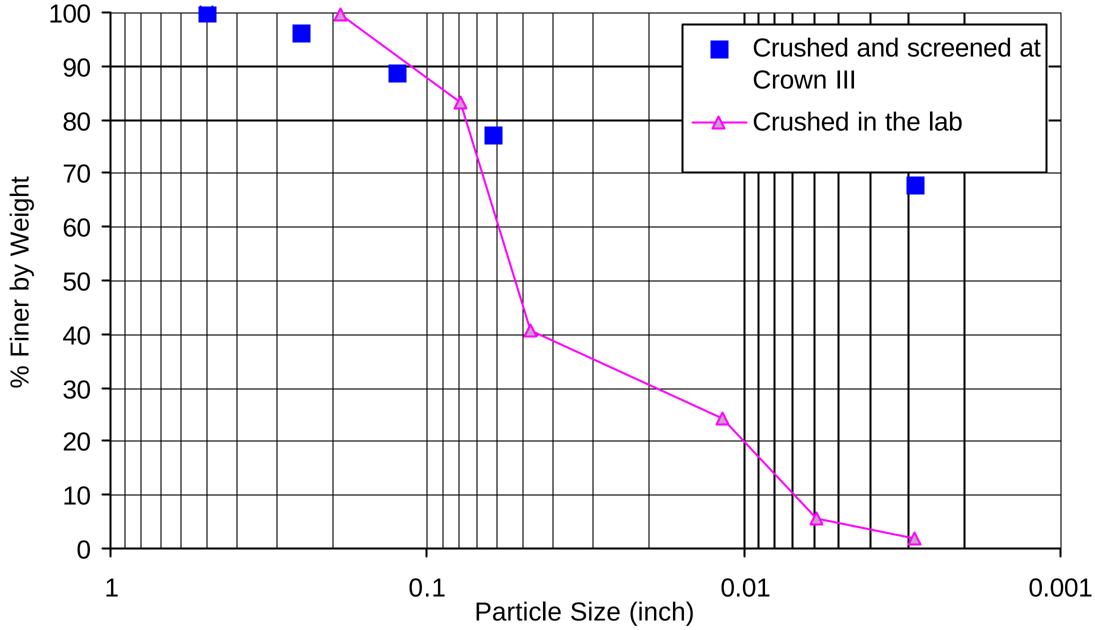


Figure 6. Gob particle size distribution for laboratory and field samples.

In the laboratory, about 20 different mixes were prepared using F-ash, FBC ash, gob and water to achieve a slump height between 9 to 11 inches. For each mix, at least three samples were prepared and tested for 7-day and 28-day strength and elastic modulus as given in Table 2. These data were analyzed using linear regression models with two independent variables such as FBC/Water and F/FBC, and 95% confidence level. Here, σ and E refer to the compressive strength and elastic modulus, respectively and suffix 7 and 28 refer to the number of curing days. In general, a two variable linear statistical model is given in the following equation:

$$y = b_0 + b_1 x_1 + b_2 x_2 \tag{1}$$

where,

y is dependent variable such as strength or elastic modulus
x_i (i = 1, 2) are independent variables
β_i (i = 0, 1, 2) are parameters to be estimated from laboratory data regression analysis

Table 2. Laboratory data for Mix Design.

Sample	FBC/WATER	F-ASH/FBC	s_{7}, psi	E_{7}, psi	s_{28}, psi	E_{28}, psi
1	1.56	0.00	323	18700	484	21667
2	1.56	0.00	214	16500	392	21010
3	1.56	0.00	299	16757	523	29857
4	1.19	0.00	224	21544	384	21468
5	1.19	0.00	187	16250	299	22510
6	1.19	0.00	187	14500	424	29032
7	0.61	0.00	56	11085	150	12500
8	0.61	0.00	77	12339	168	10800
9	0.61	0.00	93	7164	149	10500
10	1.00	0.25	131	11251	423	23855
11	1.00	0.25	130	12875	392	26087
12	1.00	0.25	187	15640	299	17857
13	1.22	0.67	218	20850	356	21088
14	1.22	0.67	265	13424	392	25000
15	1.22	0.67	178	14330	448	26157
16	1.22	0.67	299	19000	523	19230
17	1.22	0.67	224	25000	392	24000
18	1.22	0.67	187	15799	392	25455
19	0.35	1.00	50	8440	84	7037
20	0.35	1.00	56	5125	93	7603
21	0.35	1.00	75	4800	112	8007
22	0.76	1.20	168	13131	298	15257
23	0.76	1.20	155	12727	187	12727
24	0.76	1.20	112	10020	205	22727
25	0.30	4.00	93	7463	131	11429
26	0.30	4.00	149	8333	149	12800
27	1.11	0.50	143	14552	399	22222
28	1.11	0.50	168	18300	348	29167
29	1.35	0.20	221	16580	392	30125
30	1.35	0.20	188	14045	467	24070
31	1.35	0.20	256	23667	485	27631
32	1.35	0.20	199	20048	486	26000
33	1.35	0.20	243	22580	299	25000
34	0.97	0.49	131	11875	429	24880
35	0.97	0.49	195	16000	299	19900
36	0.97	0.49	112	10800	429	21066
37	0.79	1.00	112	8400	354	28571
38	0.79	1.00	142	10000	317	22857

Table 2 (contd.)						
Sample	**FBC/WATER**	**F-ASH/FBC**	**s_7, psi**	**E_7, psi**	**s_{28}, psi**	**E_{28}, psi**
39	0.79	1.00	93	11857	355	21000
40	1.41	0.00	168	20000	429	26617
41	1.41	0.00	252	25080	504	30820
42	1.41	0.00	168	18867	467	28502
43	1.43	0.00	264	17640	504	31250
44	1.43	0.00	193	19235	467	30000
45	1.43	0.00	243	23168	505	35200
46	0.69	1.00	144	10000	205	16755
47	0.69	1.00	93	7550	224	22203
48	0.69	1.00	88	7200	242	22982
49	0.99	1.00	185	14600	392	19540
50	0.99	1.00	205	12230	429	24500
51	0.99	1.00	193	10000	355	26250
52	1.56	0.20	254	26458	504	25000
53	1.56	0.20	299	23875	467	22222
54	1.56	0.20	348	25000	468	20550
55	1.56	0.20	297	15540	504	30481
56	1.56	0.20	212	17560	504	37500
57	1.15	0.20	174	21450	430	23235
58	1.15	0.20	212	20000	467	25309
59	1.15	0.20	147	14222	392	27778
60	1.15	0.20	168	13120	541	28035
61	1.15	0.20	255	27778	542	28866
62	0.98	0.38	119	12255	342	17950
63	0.98	0.38	138	9927	256	21098
64	0.98	0.38	142	10875	268	18760
65	0.79	0.83	168	14830	317	21145
66	0.79	0.83	168	15000	355	21080
67	0.79	0.83	112	11800	280	17833

In this case, β_0 is assumed to be zero because when all independent variables are zero, dependent variables such as strength and elastic modulus must be zero. Since sum of the proportion of three solid components is unity, the following statistical model was developed:

$$Strength = b_1\left(\frac{F}{FBC}\right) + b_2\left(\frac{FBC}{WATER}\right) \tag{2}$$

$$Modulus = b_1\left(\frac{F}{FBC}\right) + b_2\left(\frac{FBC}{WATER}\right) \tag{3}$$

In these models, GOB proportion is indirectly included as a parameter since the sum of the proportions of F + FBC + GOB = 1. However, gob has a significant contribution in acid-base neutralization potential in the mix. In general, gob is acidic with a pH value of 2-4. On the other hand, FBC ash is alkaline and when gob is mixed with FBC in appropriate proportions, the pH of the mix may range between 7-8. Such a mix significantly reduces the possibility of acid mine drainage.

It was found that higher proportions of FBC ash with adequate water significantly increases strength and elastic modulus (Chugh et al., 1998). However, higher amounts of gob (over 50%) adversely affects the strength of the mix. It was determined that the ratio between F ash and FBC ash should be kept below 0.2 to achieve high compressive strength after 28-days of curing (Chugh et al., 1998). These studies proved that the ratio between FBC ash and water content in the mix is the most important parameter for determining strength and elastic modulus. In addition, the ratio between F ash and FBC ash also play an important role in compressive strength for 7-day cured samples. Thus, in this study, two independent variables (F-ash/FBC ash, and FBC ash/Water) are considered to establish relationships with uniaxial compressive strength and elastic modulus for 7-day and 28-day cured samples. Similar relationships are also developed for hydraulic conductivity, durability and swelling strain of the mix.

Regression analysis provided the following relationships for 7-day compressive strength with r^2 and F value of 0.838 and 46.77.

$$s_7 = 13.2\left(\frac{F}{FBC}\right) + 156.95\left(\frac{FBC}{WATER}\right) \tag{4}$$

The coefficient of FBC/WATER passes the t-test with a t value of 27.97, for $\alpha = 0.05$. This variable has significant effect on compressive strength of 7-day cured samples. The coefficient, F/FBC also passes the t-test for $\alpha = 0.05$. F/FBC also contributes to the 7-day compressive strength. Thus, it is a valid statistical model. Figure 7 shows the linear relationship and 95% confidence intervals for 7-day compressive strength with FBC/WATER. This figure also shows that 7-day compressive strength of samples similar to the field mix provides a good match with the predictive model.

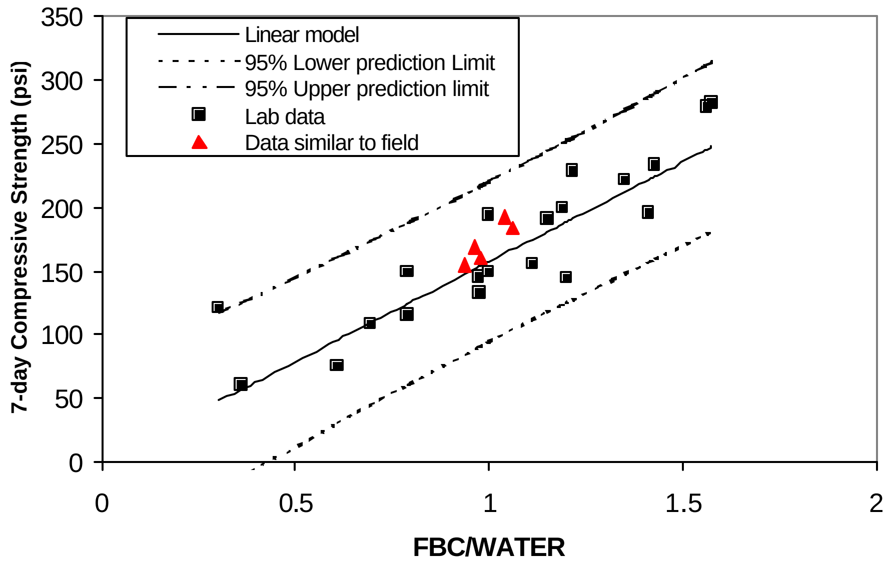


Figure 7. Relationship between 7-day compressive strength and FBC/WATER ratio

The variable, FBC/WATER is significant because FBC ash is cementitious and contains free lime (CaO). In the presence of water, FBC helps to cement F-ash and gob together. However, excessive amounts of water and gob have an adverse affect on the compressive strength as well as on the elastic modulus. Larger particle size gob will also reduce compressive strength due to uneven gob size distribution in the sample.

The relationship between the 7-day elastic modulus with the above mentioned independent variables reveals that F/FBC is statistically insignificant and does not pass the t-test with 95% confidence level. As a result, the relationship with 7-day elastic modulus is given as follows:

$$E_7 = 13723\left(\frac{FBC}{WATER}\right) \quad (5)$$

This relationship has the r^2 and F value of 0.772 and 64.44, respecitvely. Thus, E_7 is significantly dependent on FBC/WATER. Figure 8 shows the linear relationship and 95% confidence intervals for the 7-day elastic modulus and the variable FBC/WATER. This figure also shows a good match for the 7-day elastic modulus with data similar to the field mix.

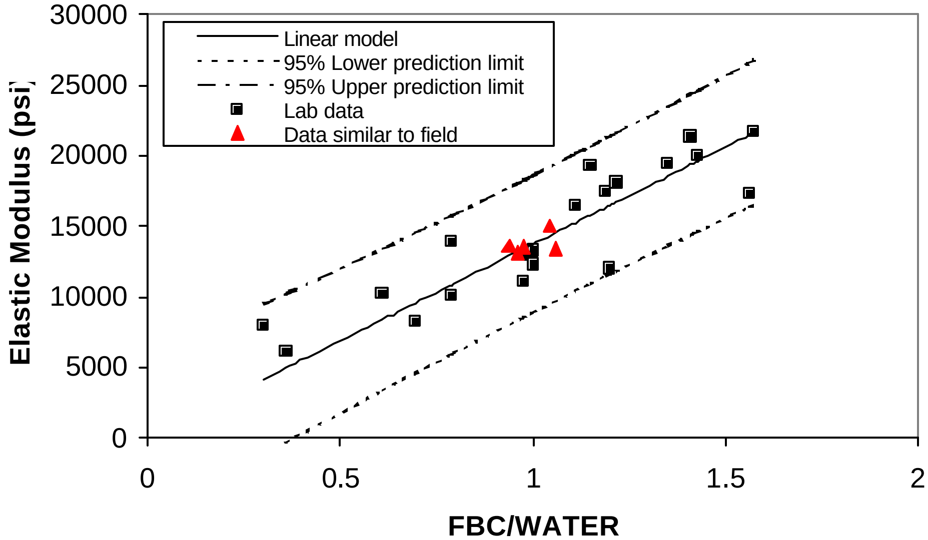


Figure 8. Relationship between 7-day elastic modulus and FBC/WATER

Analysis was also extended to estimate the 28-day strength and elastic modulus using linear models for both variables. F/FBC has insignificant effect and FBC/WATER is the dominant parameter for the 28-day strength and elastic modulus. The following two equations are obtained from regression analysis:

$$s_{28} = 330.10\left(\frac{FBC}{WATER}\right) \tag{6}$$

$$E_{28} = 20442\left(\frac{FBC}{WATER}\right) \tag{7}$$

These two models have r^2 and F values of 0.761 and 60.42 and 0.506 and 19.57, respectively. Both relationships are statistically significant and FBC/WATER does affect 28-day compressive strength and elastic modulus. Figures 9 and 10 show the linear model with 95% prediction intervals. These figures also show that the data for samples similar to the field mix lie close to the linear model.

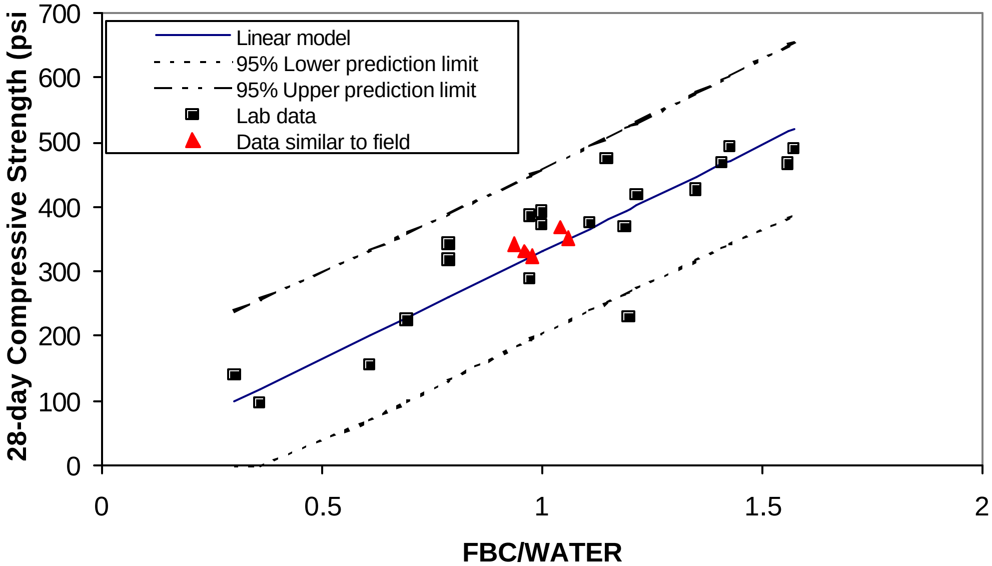


Figure 9. Relationship between 28-day compressive strength and FBC/WATER ratio.

Final Mixes Selection: Freeman United Coal Company and the Industry steering committee selected four (4) final mixes for further consideration. Compositions of the final four mixes are given in Table 3. Engineering properties of final mixes are given in Table 4. In the steering committee meeting on April 15, 1999, it was decided that Mix 25 and Mix 18 only would be considered for underground backfilling. For final underground demonstration, Mix 18 only was selected. The flow characteristics of this mix were excellent even though it contains 40% crushed gob. This mix ("field mix") was slightly modified for field demonstration based on resource availability of different mix constituents. This mix has 33% gob, 53% FBC ash, 14% F-ash, and 50% water.

Table 3. Proportions of raw ingredients of four final mixes.

Components	**Mix 1**	**Mix 25**	**Mix 18**	**Mix 21**	**Mix 26**
Gob, %	33	25	40	45	45
FBC fly ash, %	53%	62.5	50	55	46
F-type fly ash, %	14%	12.5	10	0	9

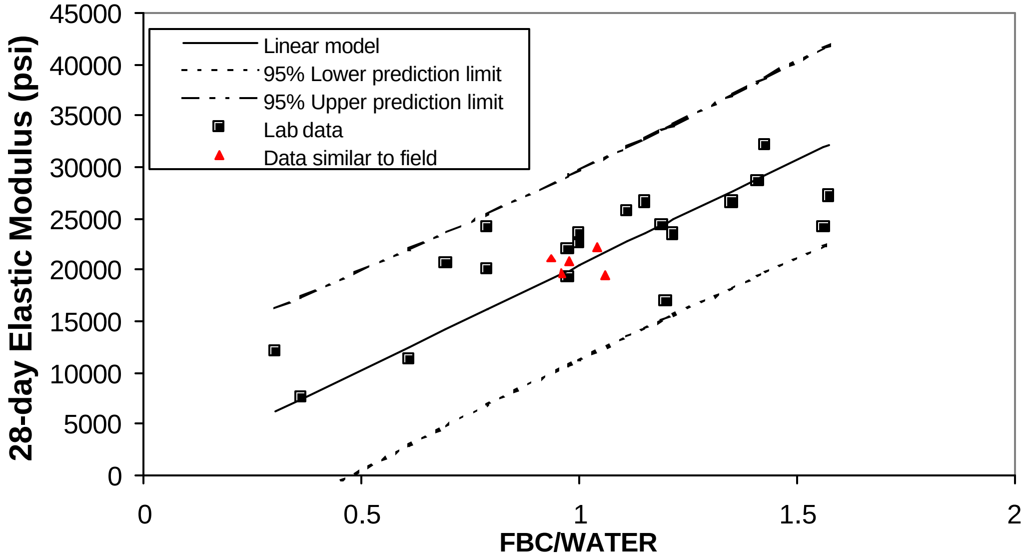


Figure 10. Relationship between 28-day elastic modulus and FBC/WATER

Development of Additional Mixes Similar to the Field Mix: In the laboratory, five mixes were prepared by slightly varying the field mix as shown in Table 5. In this table, mix 1 refers to the field mix compositions. Moisture or water was added to achieve a slump height of 11 inches with a minimal bleed of about 3-4%. For each mix, three samples were prepared and tested for their compressive strength and elastic modulus after 7-day and 28-day curing. Hydraulic conductivity, swelling strain and slake durability tests were also performed for each sample. Analysis based on these samples provides a better estimation of engineering properties of paste backfill injected in Crown III mine. Compressive strength and elastic modulus data of these samples are also used to verify the linear regression models developed earlier.

Table 4. Engineering properties of the final mixes.

Property	**Mix 25**	**Mix 18**	**Mix 21**	**Mix 26**
7-Day strength, psi	479	243	168	430
28-Day strength, psi	523	523	492	579
7-day elastic modulus, psi	22,407	20,000	19,433	26,452
28-day elastic modulus, psi	36,822	27,000	37,156	33,750
Water needed for 9-inch slump, %	40	36	39	40
Bleed for 9-inch slump, %	1.1	1.6	1.6	1.6
CCE, %	45	44	41	36

Table 5. Proportions of additional mixes similar to the field mix.

Mix	Gob	FBC-Ash	F-Ash	Moisture
1*	0.33	0.53	0.14	0.5
2	0.35	0.5	0.15	0.48
3	0.4	0.45	0.15	0.46
4	0.4	0.5	0.1	0.52
5	0.45	0.45	0.1	0.48

*Composition of field mix

- **Construction of Mixing Plant at Crown III Mine**

The concrete mixing plant, used for underground backfilling studies at Pawnee mine, was dismantled and moved to Crown III mine. The mixing plant was set up at Crown III mine in March 1999. Electrical wiring and water pipe connections were finished by June 1999. Figure 11A-B shows the mixing plant and its various components. Several tests were performed to check the conditions of its various components. One of the important factors was to obtain optimum speeds of the three belts, which carry FBC ash, gob and F-ash to the main belt. Table 6 shows the characteristics of these belts. Using the specified speeds, final field mix of the 53% FBC, 33% gob and 14% F-ash was obtained. At the pug mill, water was added to mix solids and then pump it into a pit below. FBC and F-ash belt discharged materials to the main belt through a chute. Gob was discharged directly onto the main belt. FBC ash fugitive dust was controlled using water sprays at four different locations around the chute.

Table 6. Characteristics of conveyor belts.

Belt	Length (ft)	Speed (ft/min)	Capacity (ton/hour)	Weight (lb/ft)	% Weight
Gob	65	66.7	30.02	15.00	32.99
FBC	38	23.1	48.51	70.00	53.32
F-ash	40	17.8	12.46	23.33	13.69
Main	120	66.7	90.99	108.33	100

The peak mixing rate of this plant was designed not to exceed 100 ton/hour of solid and 40 ton/hour of water. This mix provides about 11 inch of slump height. The plant was operated with a water addition rate of 138-152 gallons per minute to obtain a mix of at least 11-inch slump. It was difficult to maintain a constant feed rate of solids into the pug mill. Specifically, the composition of the three components varied slightly depending on the amount of material in the respective hoppers. Moisture content in these components also changed based on weather conditions.

Gob was crushed using a jaw crusher to provide maximum 0.75 inch size material. The size distribution data for gob is given in Figure 6. The crusher was located near the mixing plant. A front-end-loader carried the crushed gob and dumped it into the gob hopper of the mixing plant. F-ash and FBC fly ash were supplied at the plant site and front-end-loader dumped them into the respective hoppers.

Figure 11A. A view of the mixing plant.

Figure 11B. Another view of the mixing plant.

- **Surface Demonstration of Field Mix Flow Characteristics**

A trench was dug with two crosscuts on the surface. Figures 12A and Figure 12B show the schematics and a picture of this trench, respectively. The trench was about 100 ft long, 9 ft wide and 6-10 ft deep. On August 9, 1999, the mix with 40% gob (Mix 18) was pumped into this trench to observe the flow behavior. Representatives from IDCCA, Freeman United, ICCI, SIUC, Ameren CIPS, Old Ben Coal, Peabody Coal, and IDMM were present during this demonstration. The mix flowed in all directions after discharge with little separation of water and solid components. It was also found that the mix flowed under water without much separation.

- **Borehole Camera Survey**

Two boreholes were drilled and cased for backfill injection process. The borehole diameter was 8.5 inch to 9.0 inch with schedule 40, 6-inch diameter steel casing. The annular space between the borehole and casing was filled with cement grout. The steel casing was within 10-ft of the top of the coal seam. Through the steel casing 5.875 inch diameter hole was drilled to intercept the coal seam.

On July 7, 1999, a borehole camera was lowered by OSM staff and found that both the holes were open for backfilling. Representatives from IDDCA, SIUC, Freeman United, and OSM were present during this task. Figure 13 shows the lowering of this camera in the second (alternate) borehole. In the first (primary) hole, the camera was lowered to the mine floor level and the distances of coal pillars from the borehole were measured. There was a roof fall in the second (alternate) borehole. However, from the camera survey, it was found that the entries in three directions were still open.

- **Field Demonstration of Underground Backfilling Using The Mixing Plant-Phase I (First Borehole)**

On August 11, 1999, field demonstration of the underground backfilling operation began though the first (primary) hole. For quality control, several tests were performed every day to check the slump height, strength and modulus of elasticity. Hourly and shift rates of backfill material (solid and water) were recorded from the belt scale and water meter. An hour-meter was installed to estimate net operation time in every shift. This operation was conducted in two eight-hour shifts every day. Day shift began from 6:00 a.m. to 3:00 p.m. and evening shift started at 3:00 p.m. and ended at 10:00 p.m. No work was performed on Saturdays and Sundays. In the evening shifts, regular maintenance of belts, motors, etc. were scheduled.

In every shift, four cylindrical samples were prepared (three samples 3x6 inches, and one 6x12 inches) for strength tests. Slump height was measured every hour to check the water content and flow behavior of the mix. The mix was backfilled underground by dumping it through the borehole under gravity. It was estimated that about 325 to 350 ft of vertical head would be sufficient to make this mix flow underground.

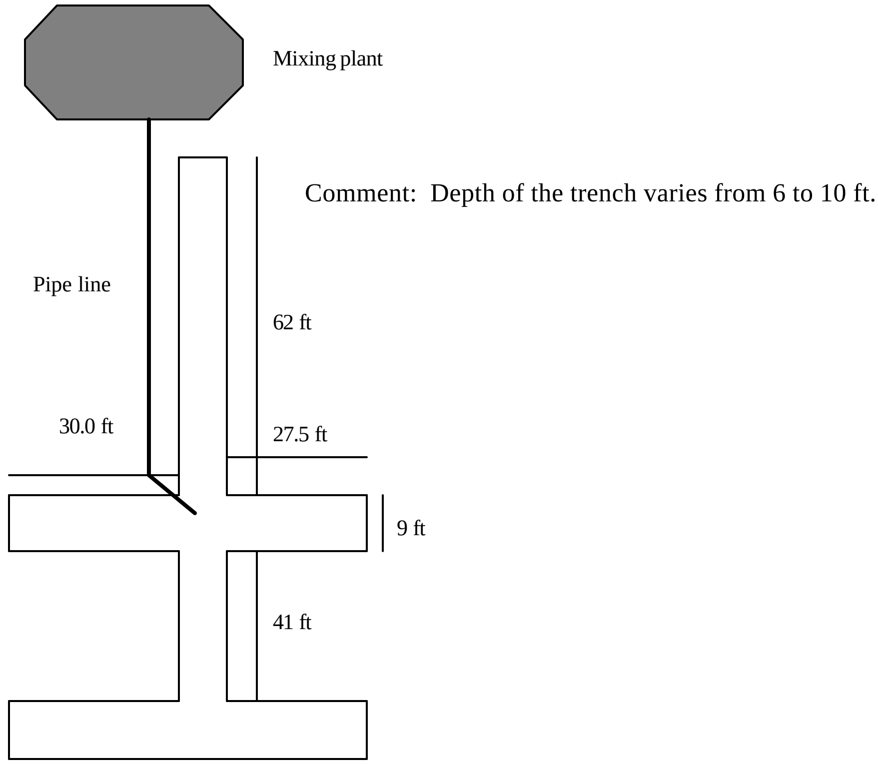


Figure 12A. Schematics of the surface trench.

Figure 12B. Picture of the surface trench.

Figure 13. Borehole camera survey.

After the primary hole choked on September 8, 1999, the backfilling operation was halted for several days. MSHA examiners suggested sealing off the panel underground before resuming any further operation. The panel was sealed by early October and backfilling operation was resumed through the second hole on October 13, 1999. A concrete pump was used to pump mix over 200 ft from the mixing plant to the second hole. Since the capacity of the pump is lower than that of the mixing plant, mixing rate of the plant was reduced to minimize the pump overflow. However, after two days of operation the pump broke down. The pump was repaired within a week. Operation began again on October 22, 1999 and finally it was stopped the following day because this area had to be prepared for CCBs management during winter. During fall 1999, Phase I of field demonstration was completed.

Figures 14 shows the daily and hourly rates of backfilling operation through the primary borehole. The daily average backfilling rate was 627 tons (452 ton of solid and 175 ton of water). The average water to solids ratio was about 40%. With this ratio, 11-inch slump height was achieved. The average hourly rate of mix was 117.1 tons/hour (83.5 ton/hour of solid and 33.6 ton/hour of water). About 8159 tons (5873 ton of solid and 2286 ton of water) of mix was backfilled underground though the primary borehole until the hole choked. Net operational time was 68.4 hours with an average of 3 hours per shift.

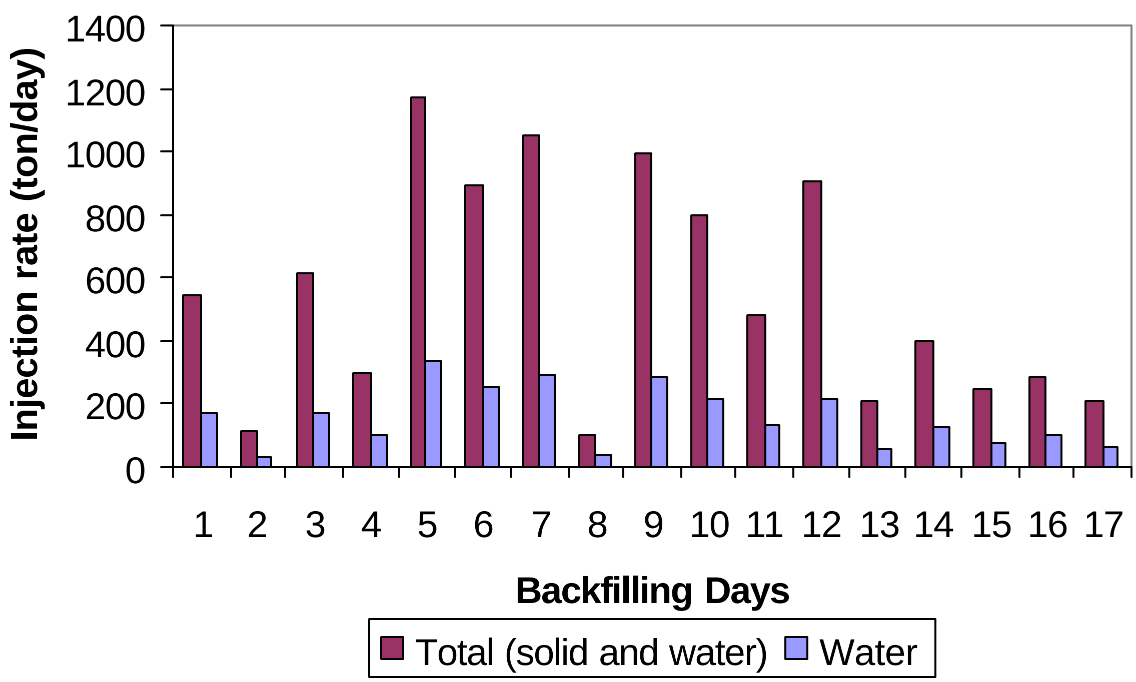


Figure 14. Daily backfilling rate through primary borehole (Phase I).

Only four days of operation was possible through the second borehole. Within these four days, 1134 ton (773 ton solids and 361 ton water) were placed underground in 14.4 net hours of operation (Figure 15). The average hourly rate was 79 ton/hour of solid and water (54 ton/h solid and 25 ton/h water). These rates are lower than those in the primary hole due to the size restriction of the concrete pump. In this case, water to powder ratio is 0.4.

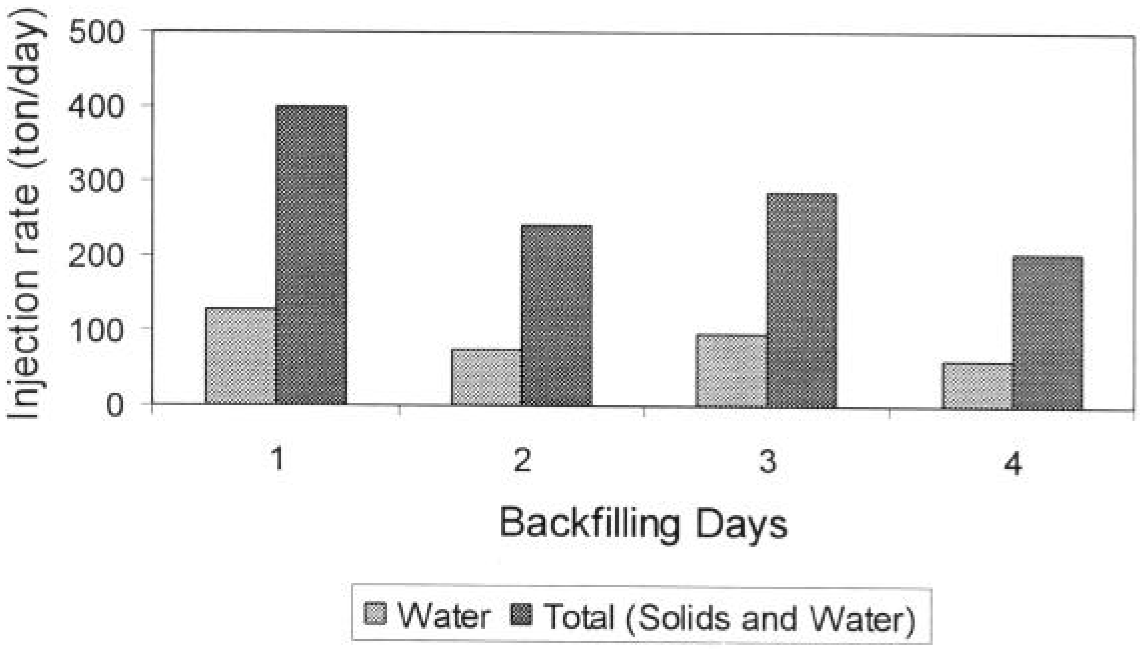


Figure 15. Daily backfilling rate through the secondary borehole (Phase I).

Altogether using both holes, about 9293 ton of solid and water mix were injected underground. The overall water to powder ratio was about 0.40.

FBC fly ash dust control and frequent plant breakdowns were some of the problems encountered in this operation. Due to these problems, the efficiency of the backfilling operation was relatively low. However, better than expected flow characteristics were achieved underground. It was concluded that operating this plant continuously around the clock would have resulted in much higher backfilling rates. However, this arrangement could not be made due to logistical problems such as manpower, equipment and material availability and high possibility of plant breakdowns. As a result, two shift operations were scheduled giving plant maintenance and gob crushing preferences on the evening shift.

- **Underground Backfill Flow Characteristics**

Figure 16 shows the characteristics of underground mix flow (dark shaded region) recorded on August 24, 1999. After this date no underground visits were made. Until this date, about 4770 tons of mix (solid and water) were backfilled underground. It was observed that the mix flowed in all four directions especially to the west and south-west. Mine entries were filled 30-feet from the borehole in all directions to within 1 ft of the roof.

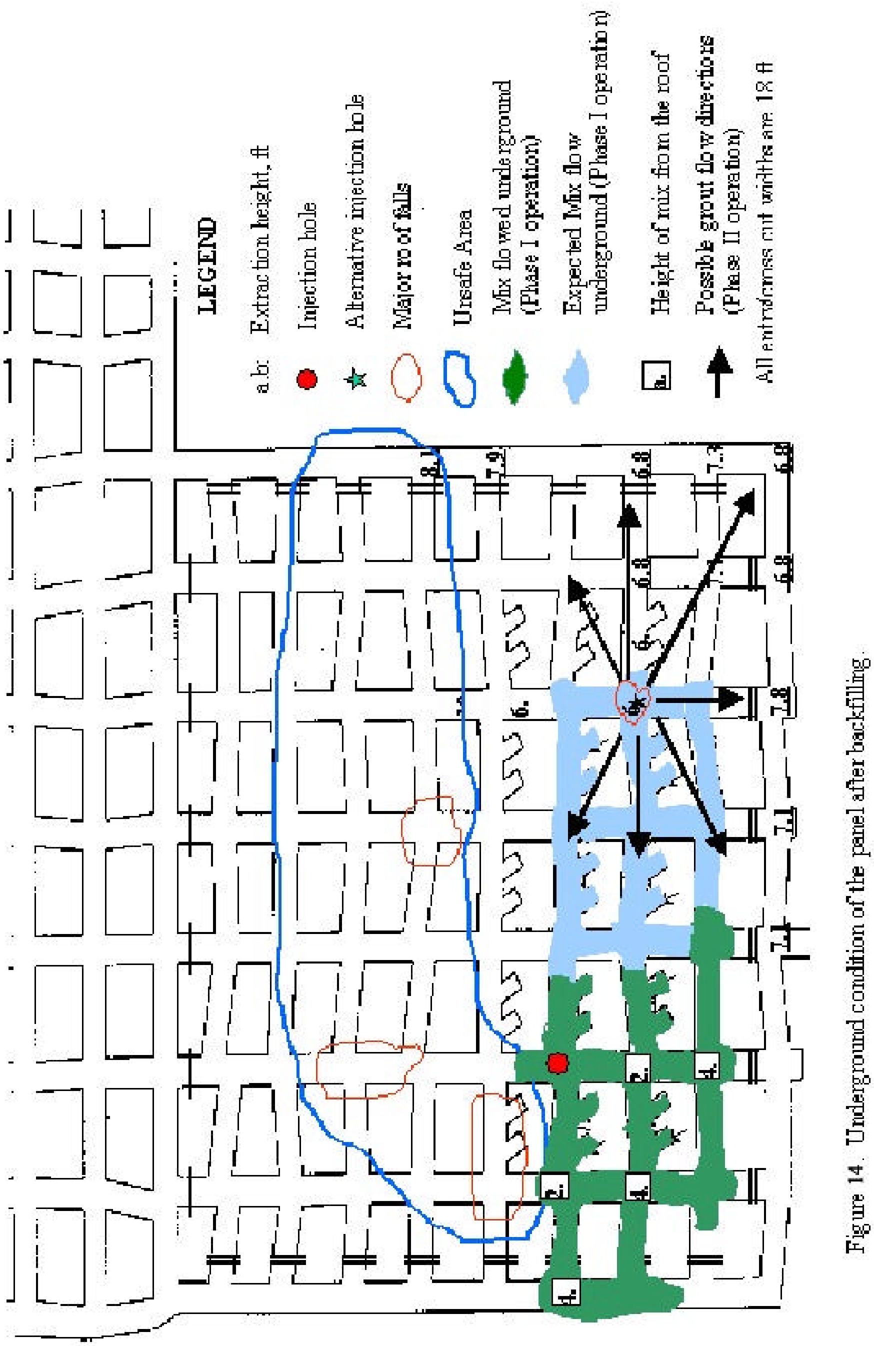


Figure 16. Underground condition of the panel after backfilling.

The backfill gradient was as low as 2 degrees and the mix had flowed a maximum distance of 120 ft. It was also observed that the mix flowed uniformly as a sheet-type-flow and did not form any channels. There was hardly any separation between solids and water. High volume of gob in the mix did not impede underground flow behavior. Moreover, the strength of the mix after twenty-four hours of curing was sufficient to sustain the weight of human beings. Underground observers had no problem walking over the thick sheet of mix.

After August 24, 1999 another 3,389 ton and 1,134 ton of backfill were dumped through the primary borehole and the second borehole, respectively. Mining staff visiting underground indicated that the backfill material flowed at least 300 ft.

- **Field Demonstration of Underground Backfilling Using A High-Speed Auger Mixer-Phase II (Second Borehole)**

After the Phase I operation was completed, the surface area, where the mixing plant was located, was filled with coal combustion by-products. However, the second hole was kept open during this process. The borehole casing was extended vertically upward during the filling process. Phase II of the field demonstration process began on September 27, 2000. In this operation, a high-speed auger with one hopper and conveyor belt system was used to mix solids with water. The following sections describe the mixing plant and backfill injection procedure for Phase II operation.

Borehole Camera Survey: On September 18, 2000 a borehole camera was lowered through the second (alternate) hole to observe the underground conditions at the intersection. This demonstration was performed with the cooperation of OSM as before. It was found that entries in two directions (south and west) were open while the entry towards the east may be closed. The earlier roof fall completely blocked the entry toward north. However, no new roof falls were observed during this survey.

High-Speed Auger Mixing Plant: In this operation, a high-speed 12 inch diameter auger mixer was used to mix solids and water. This mixer is powered by a hydraulic engine operating at 350 rpm. A hopper was set up to deliver premixed solids to the auger mixer using a variable speed conveyor belt. Water was added into the mixer. Figure 17 shows the auger mixer, the borehole and part of the conveyor belt carrying solids into the mixer.

Experiments were conducted to adjust the feed rate of solids and water so that the auger would run without clogging. It was found that the amount of water injected into the auger determines the amount of solids intake. Table 7 shows adjustment of the amount of water and solids to achieve the desired mix.

During this operation, 3x6 inch cylindrical samples were prepared every day for testing compressive strength and elastic modulus. These samples were brought to SIUC laboratory for testing. Samples will be tested after 7 day, 28 day and 90 day curing cycles.

Figure 17. Backfilling operation using auger mixer.

Table 7: Adjusting the speed of conveyor belt based on water feed rate.

Adjustment Date	Conveyor Belt Length (ft)	Conveyor Belt Speed (ft/min)	Capacity (ton/hour)	Water (gal/min)
9/27/00	40	17.78	18.67	35
10/3/00	40	36.92	83.07	75
10/16/00	40	42.86	109.29	85
10/30/00	40	60.00	64.80	100

Underground Backfilling Operation Through Second Borehole: Backfilling operation began on September 27, 2000. As of October 30, 2000, about 6,000 tons of grout (solids 4031 tons, water 1886 tons) were injected underground with a water to powder ratio of 0.47. Figure 18 shows the daily injection rate of this operation. The average daily injection rate was 348 tons of grout (237 tons of solids and 110 tons of water). The net operational time for this operation was estimated to be about 106 hours.

This phase of operation was conducted in one long shift (10 hours) per day. There were a few days that backfilling could not be done due to rain. At one time, the conveyor belt broke down and three working days were lost. Other than these problems, backfill operation went satisfactorily.

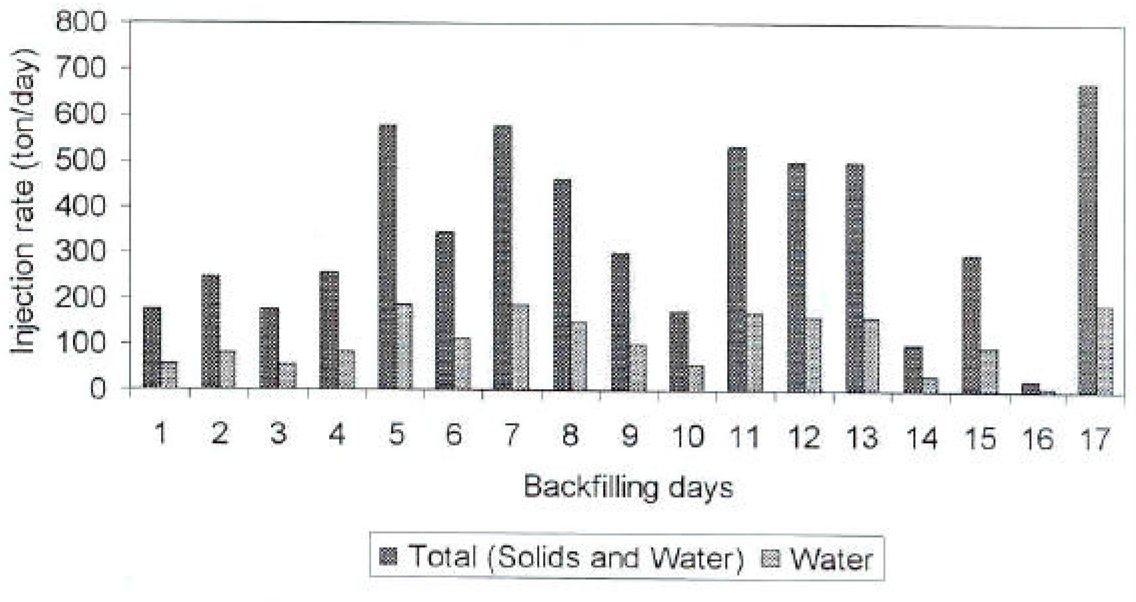


Figure 18. Daily backfilling rate using secondary borehole (Phase II).

The FBC fly ash fugitive dust problem was reduced by using a closed auger mixer instead of conveyor belts and a pug mill as in the case of Phase I backfilling operation. In addition, a water sprinkler was placed where the conveyor belt empties solids to a bin of the auger mixer for controlling dust. The number of personnel required for backfilling operation was reduced to one since only one conveyor belt was needed instead of four. This plant has only two pieces of moving equipment: a conveyor belt and an auger mixer. Thus, chances of equipment breakdown are also reduced significantly from Phase I operation. The mining company strongly favored high-speed auger mixing technology over the concrete mixing plant set up.

Backfill Flow Characteristics: Since the backfilling panel was permanently sealed after Phase I demonstration, no visit could be made underground to look at the extent of grout flow. It was hypothesized that the grout would flow towards the east, west and south directions where the entries were open. Since the second borehole was not choked after injecting about 6,000 tons or 140,000 cft of grout, the grout must be flowing away from the borehole in different directions. It was estimated that the total injected grout would fill 900-1,000 ft of mine voids considering a 20 ft wide entry and 7 ft of opening height. Figure 18 shows the possible directions grout may have flowed underground. It is hypothesized that the grout flowed better in this case as compared to Phase I operation since the grout is composed of F and FBC ash only. There was no gob material in the mix.

- **Engineering Properties of the Field Mixes Recovered During Field Demonstration**

Typical stress-strain curves of 7-day, 28-day and 90-day cured samples are shown in Figure 19. Table 8 shows the strength and elastic properties of these samples. The strength and modulus values of the 28-day cured samples are less than those of laboratory specimens due to the larger size gob particles and more water in the samples. Gob particle size above 0.5 inch was present in the field samples while in the laboratory gob particles below 0.25 inch only were considered. Due to the addition of four water sprinklers for controlling FBC ash fugitive dust the field mix contained more water than the laboratory samples. It was found that the field mix contained about 9.5% water (21% if rain-soaked) before water was added at the pug mill. Therefore, elastic properties of larger cylindrical samples (6x12 inch) may provide close approximation of the properties of the backfill material.

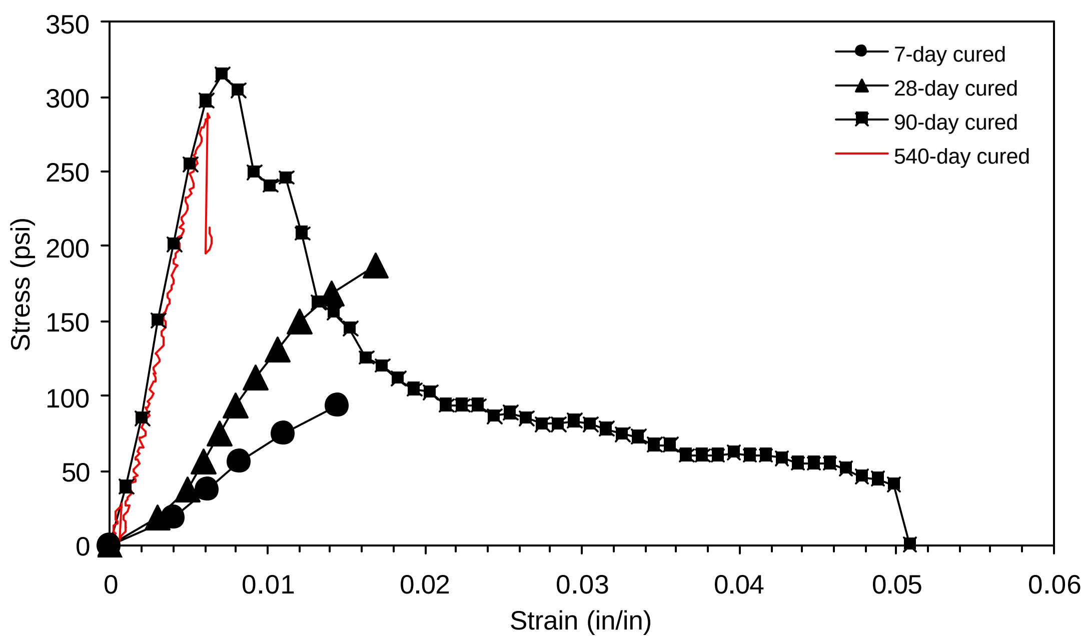


Figure 19. Stress strain relationship for field samples for different curing times.

It is observed that strength and stiffness after 90 days, and 540 days of curing are much higher than those of 7-day and 28-day curing. Table 10 shows the strength and modulus of field samples after 90-days curing and 540-days of curing. The final strength and elastic modulus of injected backfill may be close to 90-day cured samples, because the sample may not be able to dry out due to high humidity.

Field slump values suggest that for the entire backfilling operation a slump value of 11 or above was used for the backfill. The composition of the mix was kept close to the "wet" side because it was backfilled under the action of gravity. Bleed tests of this mix show that it was under 4% while backfilling through the primary hole and over 10% while backfilling through the second hole. More water was added in the latter case to ease the pumping of the mix from the mixing plant to the second borehole. The bleed test was also performed in the laboratory using F-ash, FBC and gob collected from Crown III mine. It was found that for the same slump size the bleed is higher in the samples made from the field components (Figure 20). This is attributed to higher amounts of larger size gob particles (more that 0.25 inch) in field samples than laboratory samples and moisture content in F-ash were also higher compared to laboratory samples.

Table 8. Engineering properties of field samples.

Curing Days	**Strength (psi)**	**Elastic Modulus (psi)**
7	103	6,750
14	112	9,218
28	177	11,929
90	340	40,278
540	559	85,958

Table 9. Compressive strength of 90-day cured field samples.

Sample	Strength, psi	Elastic Modulus, psi
1	667	49,290
2	266	17,890
3	357	80,675
4	315	44,242
5	159	23,940
6	279	26,634

Slake durability, permeability, swelling, bleed, slump, leachate and strength tests were performed in the laboratory for various mixes similar to the field composition.

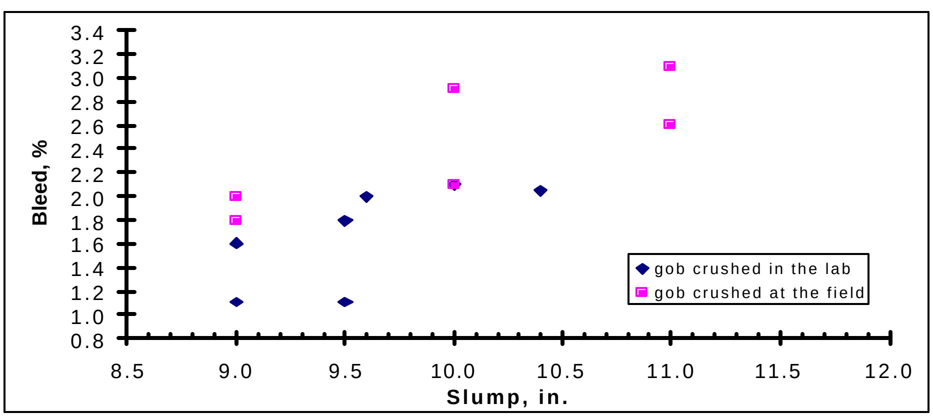


Figure 20. Comparison of slump vs. bleed for laboratory and field.

Slake Durability: The slake durability index for samples similar to the field mix ranges from 81.59 to 88.17%. Field samples show a durability index with an average of 82.0 %. Figure 21 shows the relationship between FBC/WATER and the durability index. As this ratio increases, durability index also increases. However, since the gob particle size is larger and water content is higher in field samples, the slake durability index is lower than that of laboratory samples. If the mine is flooded, the injected grout will not disintegrate readily based on the slake durability data above.

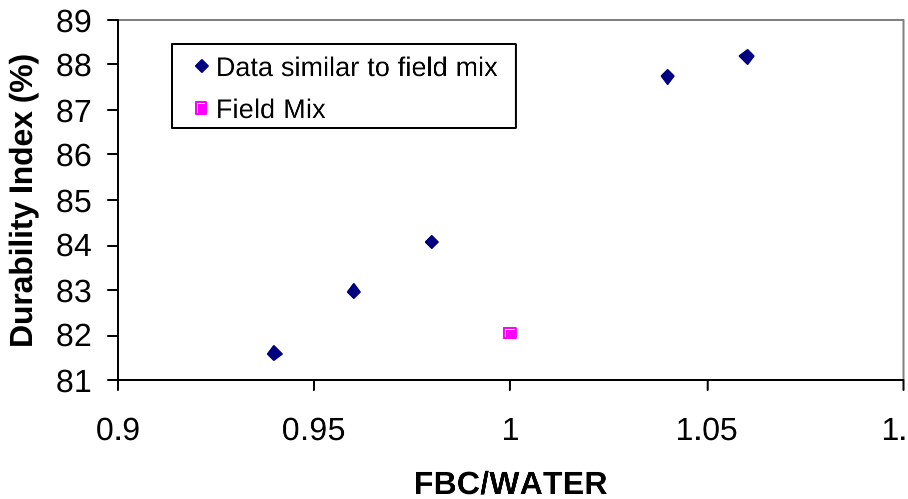


Figure 21. Variation in slake durability index with FBC/WATER ratio.

Swelling Strain: Swelling strain does not increase significantly with FBC/WATER ratio as shown in Figure 22. For the samples similar to the field mix, swelling strain ranges from 8.55 to 14.64% and the average swelling strain for the field samples is 7.4%. Swelling strain is lower in the field samples due to higher water content as explained earlier. FBC ash is completely hydrated in the field samples due to excess water and thus the sample does not swell as much as the laboratory samples. That means that the injected grout underground will have a maximum swelling strain of less than 10% if it is completely saturated with water.

Permeability or Hydraulic Conductivity: Laboratory tests show that hydraulic conductivity of the samples similar to the field mix increase nominally with the FBC/WATER ratio as shown in Figure 23. Since the particle shape of FBC ash is plate-like, the compaction index is high and thus porosity is low. However, higher gob and F-ash content in the mix will increase the hydraulic conductivity and thus may not be suitable for an application where permeability is a major issue. In Crown III mine, the floor is composed of a weak claystone layer and it is weak in the presence of water. Injected grout underground contains about 53% of FBC ash and is not a good conductor of water. Thus, hydraulic conductivity of the mix is low. From this figure it is estimated that if the mine is wet (flooded) then the injected grout will transmit water into the mine floor at a rate of 0.01-0.06 inch/day.

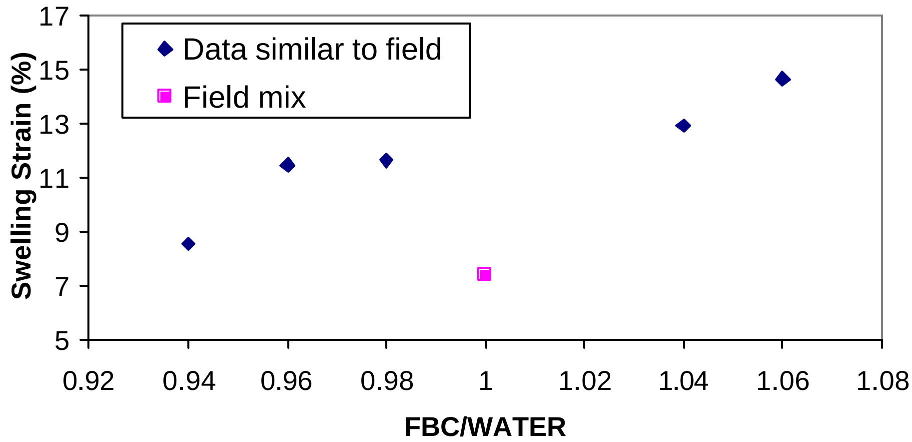


Figure 22. Variation in swelling strain with FBC/WATER ratio.

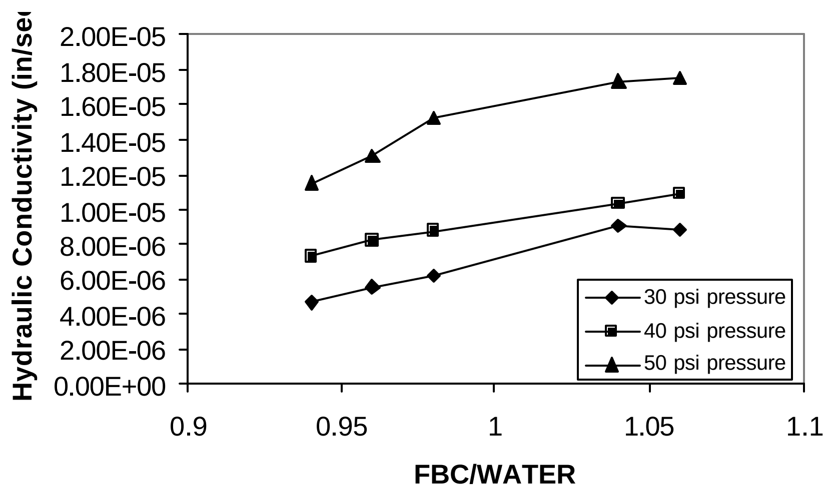


Figure 23. Variations in hydraulic conductivity with FBC/WATER.

Comparison of Compressive Strength and Elastic Modulus for Field Samples and Developed Statistical Model from Laboratory Samples: Tables 11 and 12 show 7-day, and 28-day cured compressive strength and elastic properties of field samples along with prediction limits from the regression models, respectively. In the field operation, FBC/WATER ratio was kept around one (1). It is seen that in most of the cases the actual values lie within the 95% prediction limits. As mentioned earlier, the strength and modulus values of 28-day cured sample are less than those of laboratory specimens of 300 to 400 psi due to the larger gob particles and more water in the samples.

Table 10. Actual compressive strength and elastic modulus of 18-month (560 days) cured field samples.

Sample	**Strength, psi**	Elastic Modulus, psi
1	294	65, 136
2	789	1, 34, 192
3	594	58, 546

Table 11. Actual and predicted compressive strength and elastic modules of 7-day cured field samples.

Sample	Strength psi	Strength Predicted, psi	Elastic Modulus psi	Elastic Modulus Predicted, psi
1	93	94~219	6,000	8,868~18,577
2	112	94~219	10,000	8,868~18,577
3	112	94~219	8,750	8,868~18,577
4	112	94~219	7,500	8,868~18,577
5	93	94~219	7,500	8,868~18,577
6	131	94~219	7,500	8,868~18,577
7	93	94~219	6,000	8,868~18,577
8	112	94~219	7,500	8,868~18,577
9	149	94~219	10,000	8,868~18,577
10	112	94~219	9,375	8,868~18,577
11	93	94~219	8,751	8,868~18,577

Trace Element Concentration in Field Mix Leachate: The ASTM shake test was performed for the field mix to obtain elemental concentration (in ppm) of various trace elements. Table 13 compares data for individual components and the field mix. This table also provides Class I ground water (GW) standards for comparison. Most of the trace elements the field mix contains are lower amount than the standard. Thus, the injected grout underground is an environmentally benign mix. The pH of the mix is also on the alkaline side indicating that the acidic gob is neutralized. The mixes were designed with 25 % more alkalinity than would be required to neutralize all the acidity due to gob in the mix.

Table 12. Actual and predicted compressive strength and elastic modules of 28-day cured field samples.

Sample	Strength psi	Strength Predicted, psi	Elastic Modulus psi	Elastic Modulus Predicted, psi
1	168	203~457	13,333	11,238~29,647
2	187	203~457	10,526	11,238~29,647
3	261	203~457	23,273	11,238~29,647
4	224	203~457	21,667	11,238~29,647
5	168	203~457	18,571	11,238~29,647
6	131	203~457	16,000	11,238~29,647
7	168	203~457	18,600	11,238~29,647
8	242	203~457	20,833	11,238~29,647
9	206	203~457	22,203	11,238~29,647
10	224	203~457	22,982	11,238~29,647
11	280	203~457	10,273	11,238~29,647
12	149	203~457	10,500	11,238~29,647
13	243	203~457	22,301	11,238~29,647
14	206	203~457	15,625	11,238~29,647
15	187	203~457	22,727	11,238~29,647

IV. AN ANALYSIS OF THE EFFECT OF BACKFILLING ON WEAK FLOOR AND COAL PILLAR STABILITY

Subsidence associated with weak floor strata deformation is a well recognized problem in the Illinois Coal Basin. One of the objectives of this project was to stabilize mine openings and coal pillars through backfilling so that potential for future subsidence is minimized. The No 6 coal seam at Crown III mine is associated with 2.0 to 4.5 feet of weak floor strata which is known to result in a small amount of floor heave after mining a panel even with 55% extraction ratio. Increasing the extraction ratio further will increase the potential for floor heave and surface deformations. Backfilling the voids increases the stability of mine openings as well as coal pillars. Coal pillar stability is enhanced through confining pressure on pillar ribs while opening stability is enhanced through reduced water permeability to weak floor strata, and increased resistance to shear movement through stiff backfill. Degradation of weak floor strata through increased moisture content is significantly reduced which promotes the load carrying capacity of floor strata. Finally, backfilling also reduces the chance of water inrush into the panel. The stability of the panel should reduce water drainage from overlying and underlying strata into mine workings.

Table 13. Elemental concentrations (in ppm) in the leachate of ASTM shake test

Element	Class I	Detection Limit	F ash Coffeen Fly Ash	Gob	FBC ADM Fly Ash	Field Mix
Ag	0.05	0.007	0	0.01	0	0.01
Al		0.045	0.24	0.03	0.02	0.13
As	0.05	0.053	0.04	0	0	0.01
B	2	0.012	23.39	0.04	0.53	1.35
Ba	2	0.0013	0.01	0.08	0.10	0.12
Be	0.004	0.00027	0	0	0	0.004
Ca		0.010	241	368	1163	669
Cd	0.005	0.0025	0	0.01	0	0
Co	1	0.007	0	0.10	0	0.01
Cr	0.1	0.0061	0.06	0.05	0.08	0.02
Cu	0.65	0.0054	0	0.06	0.09	0.01
Fe	5	0.012	0.02	0	0	0
Mg		0.00015	6.93	25	0.17	0.45
Mn	0.15	0.0014	0	2.28	0	0.04
Mo		0.012	1.12	0	0.09	0.04
Ni	0.1	0.015	0.04	1.24	0.03	0.01
Pb	0.0075	0.042	0	0	0	0
Sb	0.006	0.032	0.06	0.05	0.04	0
Se	0.05	0.075	0.06	0.20	0	0.02
Ti	0.002	0.120	0.01	0.01	0.02	0.01
V		0.0075	0	0	0	0.03

Zn	5	0.0018	0.07	1.96	0.67	0
Sulfate	400	50	857	688	1140	948
TDS	1200		942	1247	6850	1540
pH			8.8	3.2	12.8	11.23

A ‘0’ in a cell indicates below detection limit of ICP

Analytical Studies

The effect of backfilling on mine stability was analyzed with the SIUPANEL.3D program modified for backfilling analysis and verified by the finite element method. The details of the development of a 3D structural analysis model (SIUPANEL.3D) that can efficiently determine the effect of backfilling on the stability of a room and pillar mining system are discussed here. Further verification of the hypothesis adopted in the 3D structural model with backfilling was required using the finite element method. The results of these studies are presented in the following sections.

SIUPANEL3D Analytical Model: This analytical model (Pytel and Chugh, 1990) is based on the flexural theory of thin plates resting on inelastic foundations. The physical problem consisting of overburden plates, coal seam, and floor strata is transformed into an equivalent mechanistic problem. The system is divided into small blocks through a grid network depending on the size of pillars and openings. Each block may have different coal measure rock properties and loading conditions. The overburden strata associated with the coal seam is transformed into a composite plate with stepwise varying flexural stiffnesses. The uniformly distributed overburden load is transmitted to the weak floor strata through segmented rectangular footings representing panel pillars. The immediate floor stratum is modeled as an equivalent visco-elastic rock. The contact stresses at the rock-plate interface are approximated by rectangular areas of uniform stresses, which are transformed into equivalent concentrated forces acting at the center of the plan area of each element. Coal pillars are represented by a set of non-linear strings sandwiched between the upper overburden strata plate and lower deformable weak floor strata

Weak floor strength parameters in SIUPANEL.3D are determined using the two basic approaches, namely 1) Vesic model (1963) and 2) Pytel-Chugh (1991) model. The data is used to calculate traditional pillar and floor safety factors, floor bearing capacity, and the probability of the failure of pillars for the entire panel. Since geotechnical properties for geologic materials are highly variable, structural analyses in the model also involve probabilistic design. The load bearing capacity of floor strata is primarily governed by immediate floor stratum moisture content, its thickness, and its angle of internal friction, and similar data for the more competent bed below the immediate weak floor strata. The SIUPANEL.3D can provide failure probability for each pillar based on available data for the above variables.

To incorporate the effect of backfilling in room and pillar mining, a simple structural hypothesis was developed and incorporated in the SIUPANEL.3D program (Chugh, 1999). The hypothesis postulates that backfilling redistributes the load to be carried by the backfill, which increases the effective width of the pillar and a decrease in the height of the opening. Both should result in increased safety factors for the pillar as well as for the floor. Figure 24 (A, B and C) shows the stress redistribution at different stages of mining and after backfilling. The load acting on the backfill material is due to its weight and the pressure from the sides of the pillar. No shear stress acts along the surface of the backfill. The pressure from the side of pillar is always higher than the weight of the backfill material. Therefore, the major principal stress plane is oriented along the surface of backfill material in the horizontal direction. The minor principal stress plane is perpendicular to it. Figure 25 depicts the state of stress in the backfill material by Mohr's circle. From the triangle ABC, the angle, α can be determined as,

$$a = \frac{p}{4} - \frac{f}{2} \tag{8}$$

The effective pillar width at the base due to backfilling is determined using the following formula (Figure 1),

$$W = W_p + 2\left[H_f \tan\left(\frac{p}{4} - \frac{f}{2}\right)\right] \tag{9}$$

where, W is the effective width of the pillar after backfill
H_f is the backfill height
W_p is the original width of the pillar
f is the angle of internal friction of the backfill material

The effective height of excavation after backfilling is then given by,

$$H = H_p - \frac{H_f}{3} \tag{10}$$

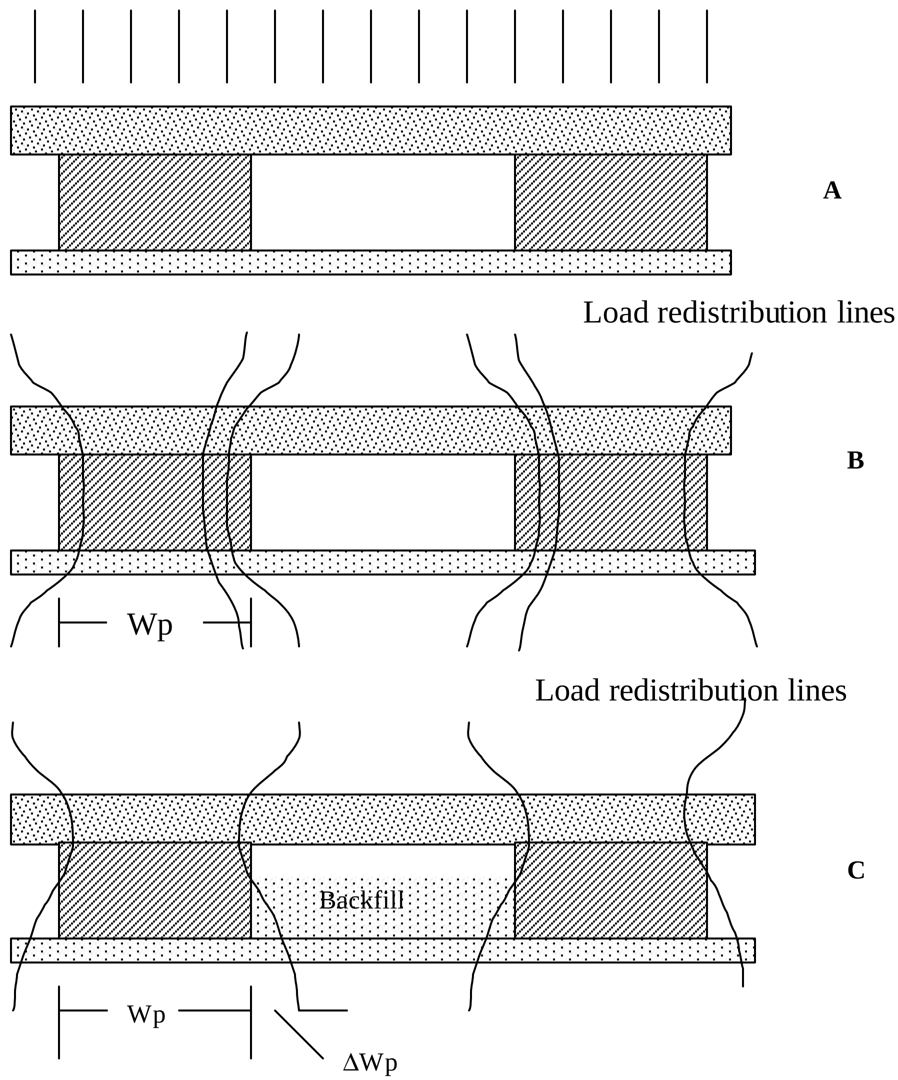
where, H_p is the height of the pillar.

If the backfill material is cohesive, it may bond with the pillars, or have frictional strength. The backfill material will carry some of the vertical load transmitted from the pillars. It is difficult to determine the explicit mathematical equation to represent this kind of partial load transmission mechanism. The above equations are proposed here based on simple geometric relationships and their validation was done using finite element analyses.

After backfilling, the strength of pillar is calculated using Holland's formula as mentioned below (Brady and Brown, 1985),

$$S = s_c \sqrt{\frac{W_p + 2\left[H_f \tan(\frac{p}{4} - \frac{f}{2})\right]}{h}} \qquad (11)$$

where, s_c is the compressive strength of the critical size cube.



ΔWp = increase in pillar width due to load redistribution through backfill

Figure 24. A schematic diagram depicting stress redistribution due to backfilling (A and B = before backfilling; C = after backfilling).

The average stress acting on a pillar is given by (Brady and Brown, 1985),

$$s_p = \frac{P + \partial P}{\left[W_p + 2\left\{H_f \tan(\frac{p}{4} - \frac{f}{2})\right\}\right]\left[W_p + 2\left\{H_f \tan(\frac{p}{4} - \frac{f}{2})\right\}\right]} \quad (12)$$

where, P is the total load acting on the pillar before backfilling, and ∂P is the additional load applied due to subsidence and after opening is backfilled. Using Equation (11) and (12), the improved factor of safety after backfilling is given by,

$$\text{Pillar safety factor} = \frac{s}{s_p} \quad (13)$$

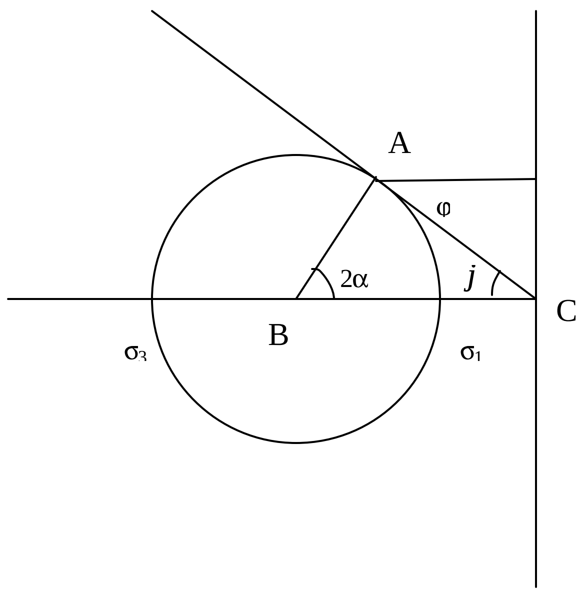


Figure 25. Mohr's circle for a theoretical backfill material.

Brown and Meyerhof (1969) proposed the following equation for the determination of bearing capacity for foundations on a soft stratum lying above a hard stratum, which is the case when the immediate floor consists of underclays overlying a harder limestone or claystone:

$$q_0 = s_1 N_m \quad (14)$$

where, s_1 = unconfined shear strength of the weak stratum
N_m = modified bearing capacity factor

Vesic (1970) proposed the following equation for the determination of N_m:

$$N_m = \frac{KN_c^*(N_c^* + b - 1)[(K+1)N_c^* + (1+Kb)N_c^* + b - 1]}{[K(K+1)N_c^* + K + b - 1][(N_c^* + b)N_c^* + b - 1] - (KN_c^* + b - 1)(N_c^* + 1)} \tag{15}$$

where, K = ratio of the unconfined shear strength of the lower hard layer (S_2) to the upper weak layer (S_1)

$N_c^* = E_c N_c$; $N_c^* = 6.17$ for $\phi = 0$ (16)

E_c = shape factor

N_c = bearing capacity factor for the weak layer

$$b = \frac{BL}{[2(B+L)H]} \tag{17}$$

B = width of foundation

L = length of foundation

H = thickness of the weak layer

Speck (1981) found a good correlation between the natural water content and triaxial strength of underclays. He modified Vesic's equation to include underclay content:

$$q_0 = (N_m)[2070 - (167)(MC)](RF) \tag{18}$$

Using Equation (12) and (18), the floor factor of safety against bearing capacity failure is given by,

$$\text{Floor safety factor } = \frac{q_0}{s_p} \tag{19}$$

Model Validation: The modified SIUPANEL.3D program was validated for data from Crown III mine. The following example gives application of the updated SIUPANEL.3D program in analyzing the influence of backfilling on the pillar and floor stability. A regular mining panel is given in Figure 26 with the pillar size of 80 ft x 80 ft (solid), opening width of 20 ft, and extraction ratio of 55.6%. The coal seam was 6.0 feet high with the overburden depth of 350 feet.

In this example, different backfill heights (backfill height/excavation height = 0, 0.15, 0.3, 0.5, 0.65, 0.8, 0.95, 1.0) were modeled. The backfill material had an angle of internal friction of 25 degrees, unit weight of 110 pcf, cohesion of 175 psi and Poisson's ratio of 0.35. The mean and standard deviation for input parameters are listed in Table 14.

Table 14. Input parameters for probabilistic analysis.

Parameter	Mean	Standard-deviation
Qu (psi)	650	145
Mc1 (%)	8.5	1.0
Mc2 (%)	2.0	0.0
∅1 (degree)	1.0	0.1
∅2 (degree)	30	0.1
H (feet)	4.0	1.0

Where, Qu is the unconfined compressive strength,

Mc1 is the moisture content of the weak floor strata,

Mc2 is the moisture content of the competent floor strata,

$\varnothing 1$ is the angle of internal friction of the weak floor strata,

$\varnothing 2$ is the angle of internal friction of the competent floor strata, and

H is the thickness of the weak floor strata

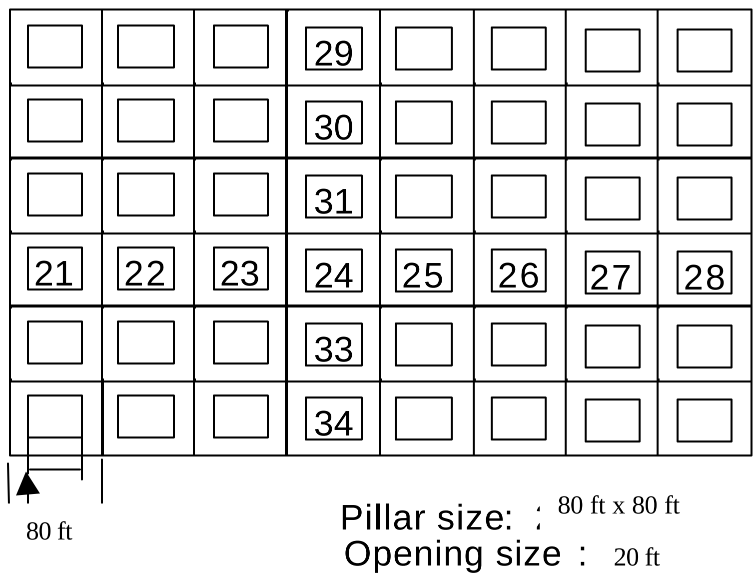


Figure 26. Room-and-Pillar mining geometry for the validation mine

The effect of backfilling was investigated for pillars 21 through 28 and pillars 29 through 34 as shown in Figure 26. The failure probability of pillar 24, located at the center of the panel is shown in Figure 27. As the backfilling height increases, the failure probability of pillar decreases. Figures 28 and 29 show the increase in safety factors and the bearing capacity, respectively of the floor as well as the pillar at pillar no. 24. Figures 28 and 29 suggest that as the backfilling height increases, the floor and pillar safety factors and the bearing capacity of floor increase.

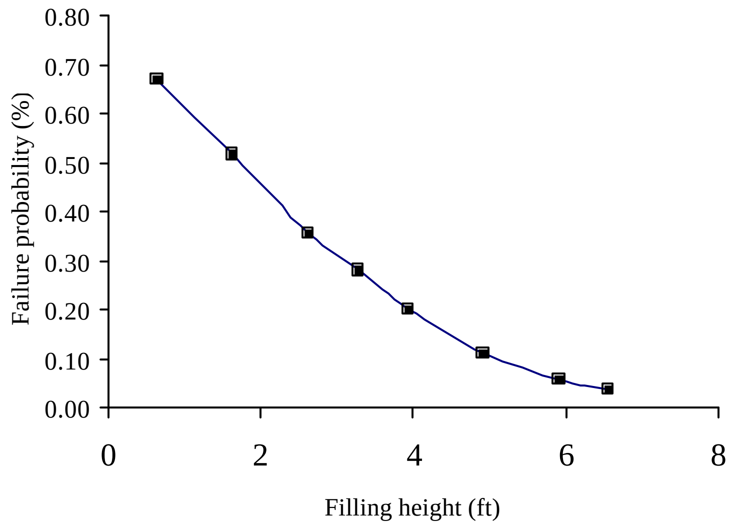


Figure 27. Influence of backfill height on failure probability of pillar 24.

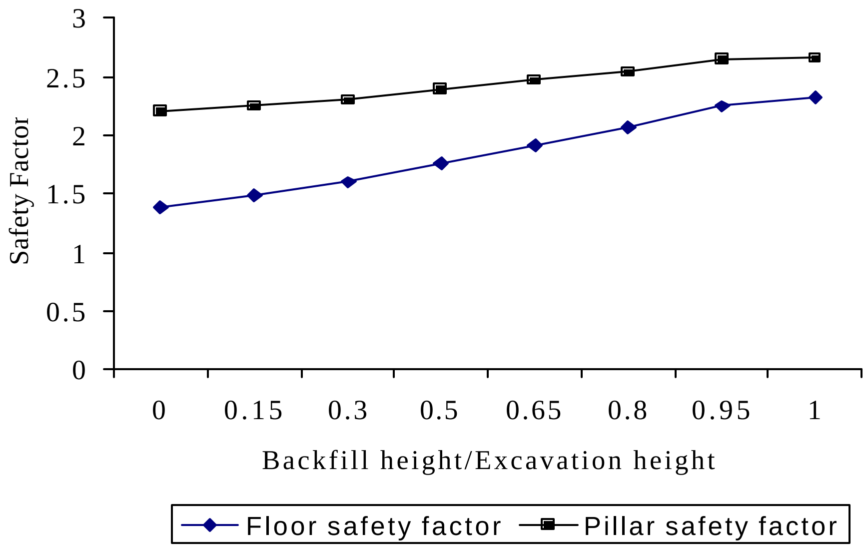


Figure 28. Variation of pillar and floor factor of safety with the backfill height

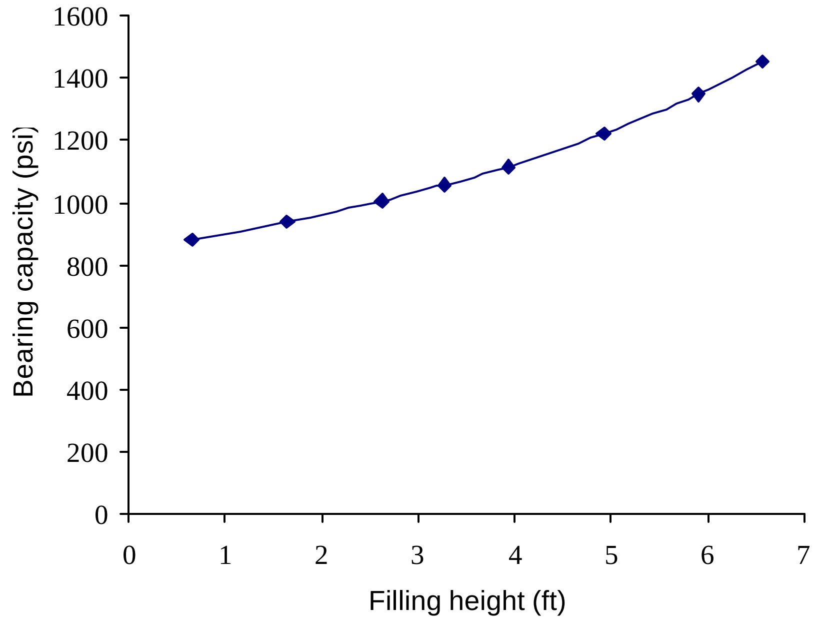


Figure 29. Variation of floor bearing capacity with backfill height.

In a practical mining situation, the center of the panel experiences the maximum overburden load. The SIUPANEL.3D program was used to investigate whether the backfilling had any significant effect at the center of the panel. Figures 30 and 31 depict the pillar and floor safety factors for the complete panel before and after backfilling operation. The results suggest that pillar safety factor improves by about 24% at the center, and by about 26% near the barrier pillar after backfilling. The floor safety factor improves by about 110% at the center, and 130% at the sides of the panel. Therefore, the backfilling has larger effect on the floor safety factor than on the pillar safety factor. The results also suggest that the improvement of safety factors after backfilling are higher near the barrier than at the center of the panel.

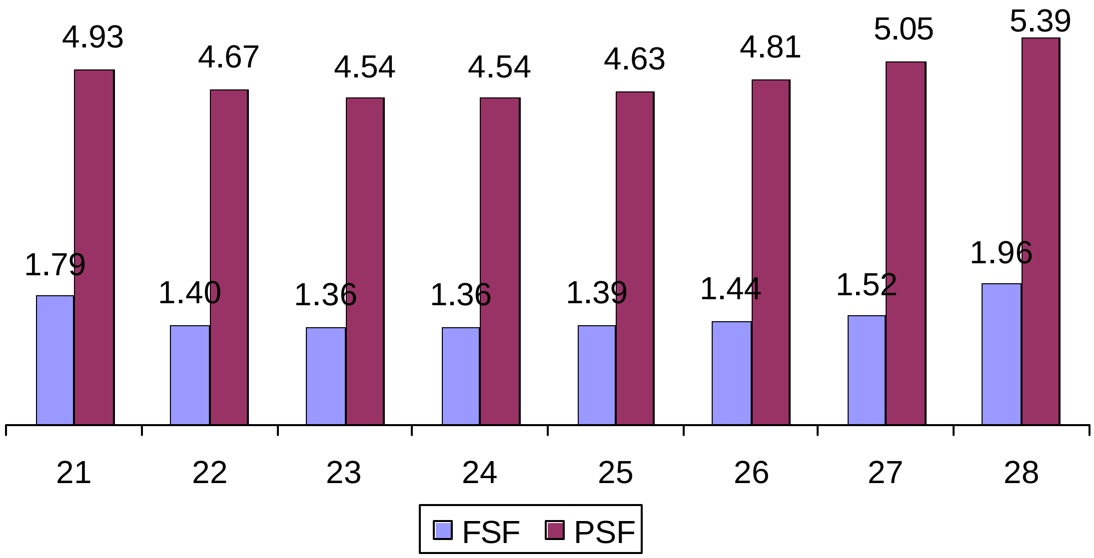


Figure 30. Variation of pillar and floor safety factors in a panel without backfilling.

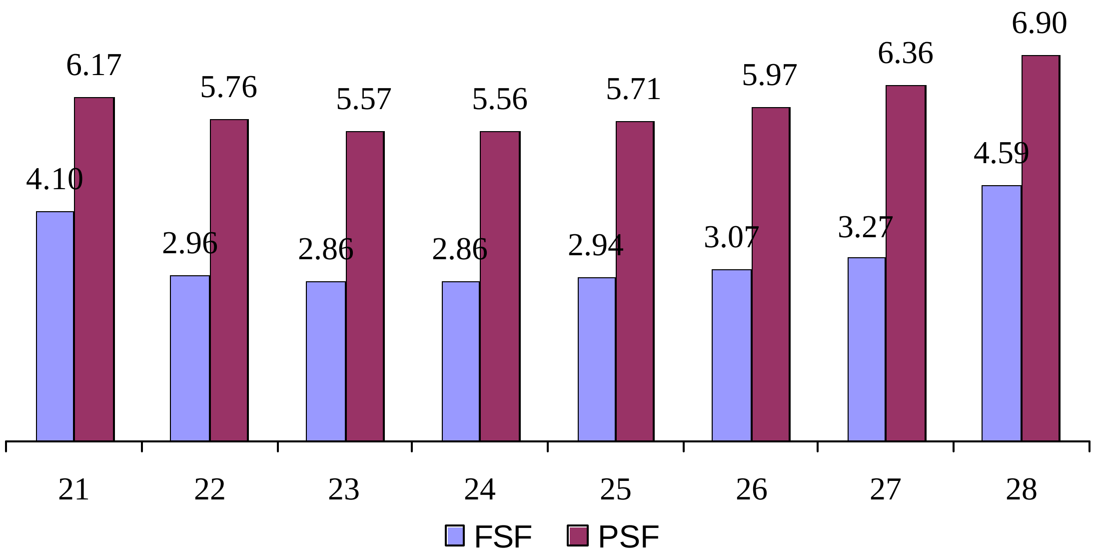


Figure 31. Variation of pillar and floor safety factors in a panel after backfilling.

Validation of Modeling Hypothesis Through Finite Element Analysis

To simulate underground room and pillar backfilling and verify Chugh's hypothesis (1999), several 2D finite element models were developed in *Phase*^{2} finite element software. The analysis was carried out for depth of 350 feet under gravity loading with the ratio of horizontal to vertical stress of 1/3. The thickness of the coal seam considered, was 6.0 feet, with the pillar width and opening width of 80 feet (c-c) and 20 feet, respectively. Only the weak floor was considered as an isotropic and a perfectly plastic material. The immediate roof strata above the coal seam and the strong floor strata underneath the weak floor stratum were considered as isotropic, homogeneous and elastic material. The strengths of the different materials were computed using the Hoek-Brown failure criteria.

Analysis Methodology: The two-dimensional, linear elastic-plastic finite element method was selected to investigate the behavior of consolidated backfill in room and pillar mining. Stress concentrations around an excavation in two-dimensions is larger than that analyzed in three-dimensions cases. The performance of backfill in two-dimensional cases can also be safely correlated to the equivalent three-dimensional cases (Yun-Yan, et al., 1983).

Simulation was done for the weak floor strata condition. Since we are most interested in ultimate deformation rather than time-dependent deformations, the immediate floor beneath the coal seam was considered as a plastic material. The aim of this study was to show that the when the excavation was backfilled with a filling material to different heights the filling material would carry some vertical load at both sides of the backfill near the pillar ribs.

The analyses were carried out for five different stages in the mining process. In the first stage, modeling without mine openings was considered. The other four stages were analyzed with respect to this base case. In the second stage, an excavation 6.0 feet high and 20 feet wide was created. In each of the third, fourth and fifth stages, the excavation was backfilled with the filling heights of 1.5 feet, 3.0 feet and 4.0 feet, respectively. In case of the backfill material, the initial element loading was achieved through body force only, whereas for other rock masses in the roof, floor and coal seam, it was achieved through both field stress and body force.

As might be expected, the backfill material will not assume any load, unless a small surface load (traction) or nodal displacement is applied at the ground surface level. Since it was convenient to apply traction in Phase^2 instead of nodal displacement, traction equivalent to 4 inches of displacement at the ground surface level was applied at stages 3, 4 and 5. To calculate the equivalent traction (i.e., traction equivalent to 4 inches), average elastic modulus was computed first. Then, the strain was calculated as the ratio of displacement (4 inches) to the model height. Finally, the traction was computed as the product of the average elastic modulus and strain. *Phase^2* uses traction per meter depth, and therefore, it was computed as 0.89 MN/m. In each new case, the model adjusted to the new conditions and calculations continued until the unbalanced load approached zero.

Results and Discussions: Differential stresses and displacements with respect to the first stage (unmined case model) were computed. This was done to eliminate the elastic rebound at the near surface level. Figures 32 and 33 denote contours of major principal stress (s_1) and minor principal stress (s_3), respectively. In the figures, different stages are shown by A, B, C and D, that is, A denotes an excavation only, B denotes an excavation with backfill height of 1.5 ft, C denotes an excavation with backfill height of 3.0 ft, and D denotes an excavation with backfill height of 4.0 ft. At stage 1, s_1 varied from 745 psi to 1,005 psi along the sides of pillar (Figure 32 A). There was high stress concentration of 1,263 psi to 1,523 psi in the pillar near the roof level. Along the sides of a pillar, s_1 acted vertically downwards (Figure 32 A). The stress trajectories of both the major and minor principal stresses were superimposed on the major principal stress contour. The long axis represents the orientation and the value of the major principal stress (s_1) and the short axis represents orientation and the value of the minor principal stress (s_3). At stage 2, s_1 varied from 850 psi to 1,655 psi at the sides of the pillar. At this stage, backfill material carried some load at both the sides in the range of 445 psi. At the center of the backfill material, it had no load. The high stress at the sides of the backfill-material was due to the effect of lateral pressure offered by the pillar. The orientations of principal stress trajectories showed that they made an angle with the vertical in the backfill material. This angle was calculated, and was found to be 32.5 degrees (Figure 32 B). However, on closer inspection, it was found that the angle was not constant, rather it varied with depth of the backfill material. This is because the principal stresses in the different layers of the backfill material were not constant. The major principal stress (s_1) at the top layer of the backfill material was approximately 0.30 psi, and at the bottom layer of the backfill material it was approximately 168 psi. The minor principal stress (s_3) at the top layer of the backfill material was approximately 1.2 psi, and at the bottom layer of the backfill material it was approximately 45 psi. Due the varying stress at different layers of the backfill material, the Mohr's envelope would be a curved line, leading to a varying angle. However, for approximation, the average value of 32.5 degrees can be used.

At the center of the backfill material, the principal stress trajectories were horizontal. This was because of the horizontal pressure from the pillar ribs. At this point, the vertical stress was due to gravity. Further increase of backfill height to 3.0 ft showed the zone of high s_1 values at the sides of the backfill material. The directions of stress trajectories remained the same (Figure 32 C). At even higher backfill height (Figure 32 D) of 4.0 ft, there was no increase of s_1 values along the sides of the backfill material. Nevertheless, at the upper layer of the material, horizontal pressure offered by pillar was less than the vertical stress of the backfill material. This was because, the excavation was deformed and the load from upper strata acted on the backfill material. Figure 32 shows that at about 50% of the backfill height, the backfill material carried maximum load along its sides. From the angle of stress trajectories and the backfill height, the effective increase in pillar width was determined.

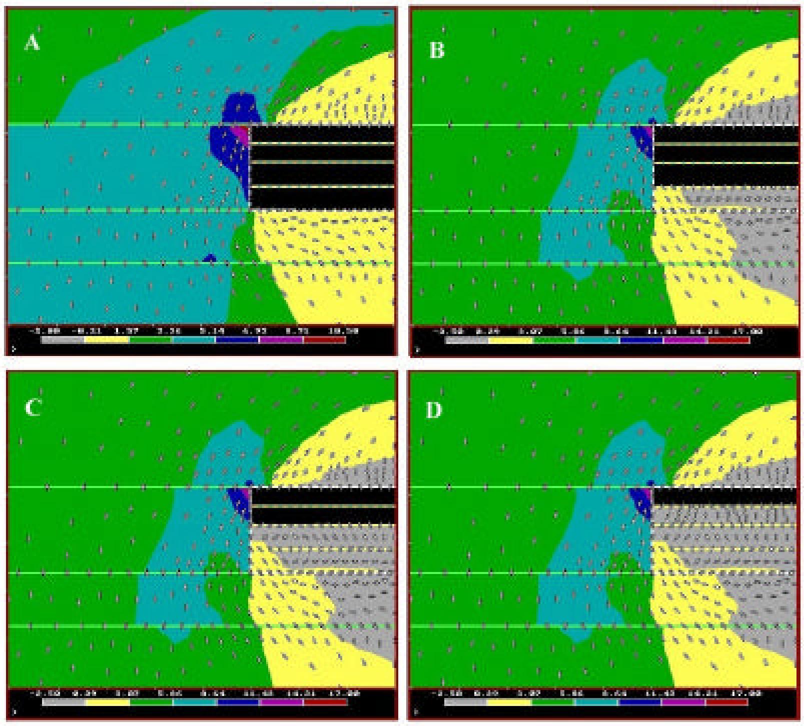


Figure 32. The contour of major principal stress (s_1) at four different stages.

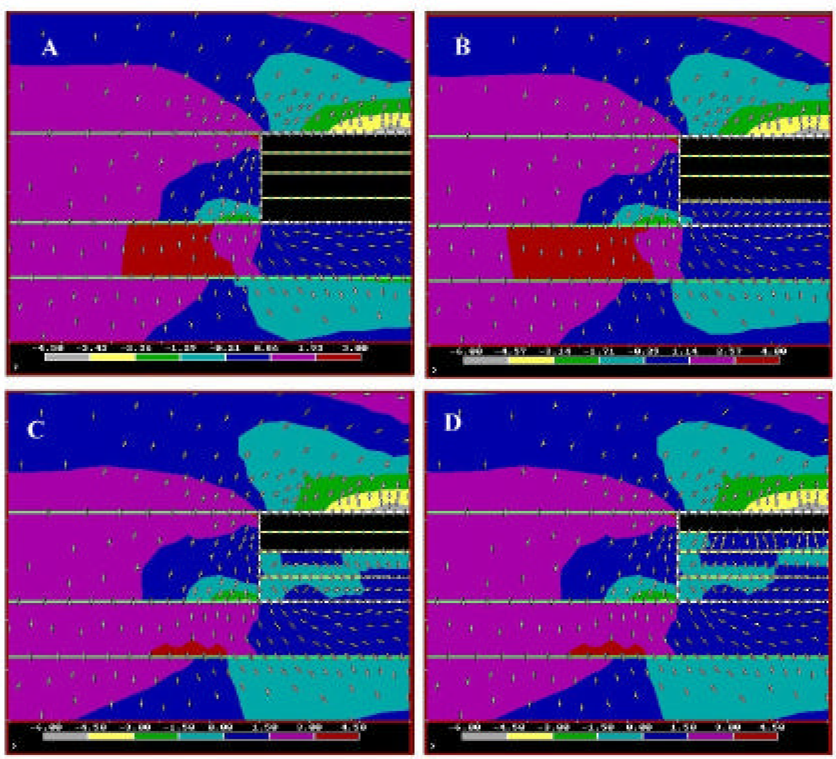


Figure 33. The contour of minor principal stress (s_3) at four different stages.

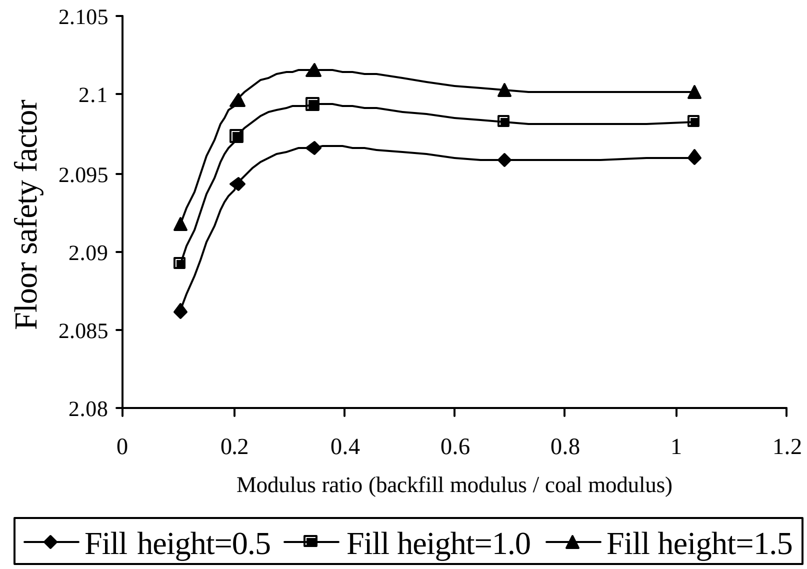


Figure 34. The effect of modulus ratio on floor safety factors.

From the analysis of finite element modeling, the effect of modulus ratio (ratio of the backfill modulus to coal modulus) on the floor safety factor was investigated. Figure 34 shows the effect of modulus ratio on the floor safety factor as well as on the backfill heights. At modulus ratio of 0.3, the safety factor increases rapidly. Beyond this point, it remains constant. Therefore, the optimum value of the backfill modulus can be taken as 1/3 of the coal modulus.

V. ENGINEERING ECONOMIC ANALYSIS

An engineering economic analysis was conducted on the feasibility of managing coal processing waste and coal combustion by-products into an active underground mine with extraction of additional coal through design of short-life pillars. This analysis was based on site specific location at Freeman United Coal Company's (Freeman) Crown III Mine located near Farmersville, IL. The following assumptions were made in the analysis:

1. Revenue from CCBs management - $ 5.00 /ton
2. Selling price of coal - $ 18.00 /ton
3. Incremental production cost of coal - $ 9.00/ton
4. Amount of gob managed annually - 550,000 tons
5. Amount of FBC ash managed/year - 800,000 tons
6. Only panels will be backfilled and efficiency of backfilling will be 80%.
7. For a panel 900 feet wide, two boreholes will be drilled for injection. Each borehole will have an influence area of 450 ft x 450 ft. Thus, backfill will be expected to flow about 300 feet all around an injection borehole.
8. Each injection borehole will be steel cased throughout the length of borehole.
9. Land acquisition cost of $ 3,000/acre.
10. Underground injection royalty cost of $ 0.15/ton
11. The backfill system will be designed for 200 tons per hour peak backfill rate. Three injection boreholes will be operating at any one time.
12. The backfill will be dry mixed near the processing plant, transported in off-highway trucks to the injection borehole where water will be mixed with it and the backfill will be placed underground without pumping.
13. Coal recovery will be increased about 8%.
14. Road development cost for truck transport is included. Water pipelines will be laid along the developed roads for water transport.
15. Analysis was performed for 15 year project life.
16. Approximate cost of cored borehole - $10,000.

Table 15 shows the capital requirements for this project totaled $12.5 million, which includes the following:

Table 15. Capital cost requirements for backfilling project.

Item	**Amount ($)**	**Years spent**
Ash Plant	5,000,000	1
Ash/ slurry pond	1,500,000	1
Equipment	1,625,000	1
Equipment	1,400,000	5
Equipment	1,625,000	10
Equipment	1,400,000	15
Total	12,550,000	

Of this total $8,125,000 was considered initial capital, which was spent in Year 1. The remaining capital ($4,425,000) was replacement capital for the equipment during later years. Table 16 shows the operating cost requirements for the backfilling project. The estimated operating cost totaled $5.99 per ton.

Table 16. Breakdown of operating costs.

Item	**Amount ($)**
Land	810,000
Royalty	200,000
Trucking	1,780,000
Road development	375,000
Site development	610,000
Site setup	65,000
Site operation	1,110,000
Site tear down	65,000
Site reclamation	200,000
Road reclamation	375,000
Subtotal	5,590,000
Ash plant	1,800,000
Subtotal	7,390,000
Contingency (20%)	700,000
Total	$8,090,000

The coal company also sought the professional services of an independent consultant to perform economic feasibility of the project in 2001. The results are summarized below.

Results of Engineering Economic Analysis

1. The costs for surface management are slightly lower than underground management ($5.99/ton). However, this could change as experience is gained with underground management.

2. Land and site development costs at this site are high because of prime agricultural lands involved. If these costs can be reduced, underground management costs may be lower than surface management cost. This could be achieved by initially backfilling areas where land is controlled by the Company.
3. Underground management minimizes slurry and gob areas development, management, and closure costs. However, underground management requires modification of the coal processing plant to handle and process gob as part of the backfill mix.
4. Each site must be evaluated separately for short-term and long-term costs involved. Underground management has significant potential particularly for new mines which have long life.
5. The Company has interest in commercializing the paste backfill technology if capital requirements can be met. The pending review of environmental issues associated with mine fills by the U.S. Environmental Protection Agency is creating some uncertainty.

A recent report by the National Academy of Engineering recommends research into alternate methods of managing coal processing waste. Underground management of coal processing waste in conjunction with coal combustion byproducts is an environmentally sound approach to minimize waste pond development, acid mine drainage, and surface subsidence.

VI. GROUT FLOW SIMULATION IN UNDERGROUND MINE WORKINGS

Introduction: An economic evaluation of mine backfilling in this and earlier studies indicates that the amount of grout injected through each borehole is an important variable. This is particularly true where prime agricultural lands are involved, since the land acquisition and reclamation costs are very high. Therefore, a thorough understanding of grout flow in underground partial extraction mine workings is extremely important. Stiles (1999) of the West Virginia University developed an approximate mathematical model of grout flow in room-and-pillar mine workings. This portion of the study was undertaken to develop a better understanding of grout flow in room-and-pillar coal mine workings typically encountered in Illinois. It was thought that this study would help to optimize spacing between boreholes for mine backfilling projects.

Task Objectives: The overall goal of this task was to develop a better understanding of grout flow phenomenon in flat and slightly pitching coal seams and relative importance of variables, such as grout yield stress, grout hardening, and slope of the coal seam. The specific objectives of the task were:

1. To study grout flow phenomenon in a single entry and three-entry, and develop a sensitivity analysis of different variables such as pumping rate, grout yield, stress, entry slope, and grout hardening.
2. To simulate grout flow in the demonstration panels at Crown III mine of Freeman United Coal Company.

***Groutnet* Mathematical Simulation Model:** The program *Groutnet* was specifically developed to simulate injection of high solids concentration grouts into underground room-and-pillar mines (Stiles, 1999). The model permits solution of highly complex problem in a very reasonable length of time as compared to more precise models such as Atkinson (1995) and Reddy (1997). These authors utilized a commercial computer program *Phoenix*. The mathematical analysis utilizes simplified three-dimensional flow equations to study flow of Bingham and Newtonian fluids in open channels. In contrast, *Groutnet* utilized finite difference approximations to solve the simplified equations. In brief, *Groutnet* offers the following significant capabilities:

- Injection in partial extraction mine workings with variable slope of the floor in both directions,
- Variable shear stress of grout as a function of time,
- Grout hardening at the end of each work period,
- No slip boundary conditions along the excavation walls,
- Variable plastic viscosity,
- Turbulent flow criterion.

A more detailed discussion of the theory and computer program development is given in Stiles (1999).

In practical situations, the program, ***Groutnet*** can be used to analyze the spread of grout and carry out the sensitivity analyses on the following:

- Location of the bore-hole for grout injection,
- Rheological and material properties of the injected grout,
- Rate of grout hardening,
- Effect of mine geometry,
- Effect of mine floor slope,
- Effect of working and rest period,
- Rate of grout injection.

To investigate the sensitivity analyses of the above parameters for the spread of grout in the Crown III room and pillar mine, several simulations were conducted for single entry, three entry and full-scale underground mine situations. The results of these analyses are presented in the following sections.

Description of the Developed Models: The simulations were conducted for single entry, three-entry and full-scale underground mine situations. Figure 35 A-B shows the layout of the single entry system and its *Groutnet* equivalent, respectively.

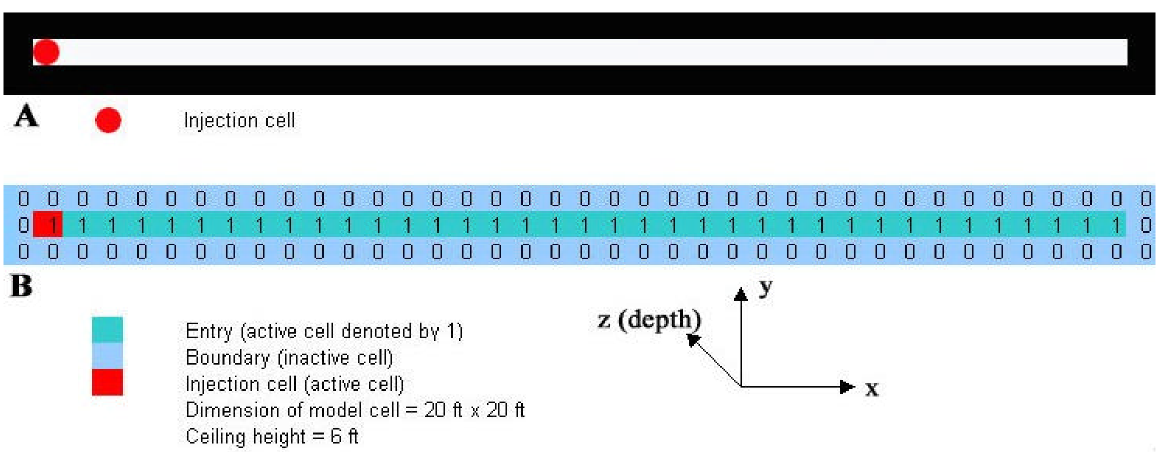


Figure 35. Physical channel (A) and its *Groutnet* equivalent model (B) for the single-entry system.

In single entry model, the total number of cells used in the horizontal and vertical directions, respectively were 100 and 3. Width of each cell considered, was 6.096 m (20 ft). The active cells through which the grout flows are represented by 1, and the boundary and inactive cells are represented by 0. Therefore, a total of 98 active cells were used to model the 597.40 m (1960 ft) of channel (in Figure 35, only 37 active model cells are shown due to page constraints). The height of the channel considered was 1.8288 m (6 ft). The injection cell was located at the left-middle cell designated by column 2 and row 2, and marked by red dot. In the first model, the slope of the channel considered, was taken 0 degree in both the x and y directions, and the rate of grout injection was 0.01716 m^3/s (120 tons/ hour). In the second model, a 1% slope was introduced with the same rate of grout injection. Then in the next models, the injection rate was varied to 0.01716 m^3/s (120 tons/ hour), 0.02152 m^3/s (150 tons/ hour) and 0.0286 m^3/s (200 tons/hour), and the flow behavior was plotted for continuous injection. Table 17 shows the material properties of the injected Bingham grout (fly ash, gob, and water mixture).

Figure 36 A-B shows the three-entry system and its *Groutnet* equivalent, respectively. In three-entry system, the number of rows and columns are 52 and 9 respectively, that is the simulation was conducted for 1000 ft wide panel, almost half the width of the single entry system to save the simulation time. It was found that 1000 ft long three-entry system can represent the grout flow behavior well. The number of active model cells used is 214. Figure 36 shows the layout of the three-entry system (only 40 columns are shown due to page constraints). The injection point was located at the center of the model (i.e., row 5 and column 26, and marked by red dot), and grout was injected continuously as shown in Figure 36. The three-entry system were also simulated as was done in case of the single-entry system using the same material properties as given in Table 17.

Besides the single entry and the three-entry system, the simulation was also conducted for full-scale mine geometry of the Crown III mine, where grout was injected. Figure 37 A and B show the mine geometry and its *Groutnet* equivalent model for the full-scale grout injection model, respectively. The description of the mine was already given in the previous sections. Only the *Groutnet* model for this mine are discussed here.

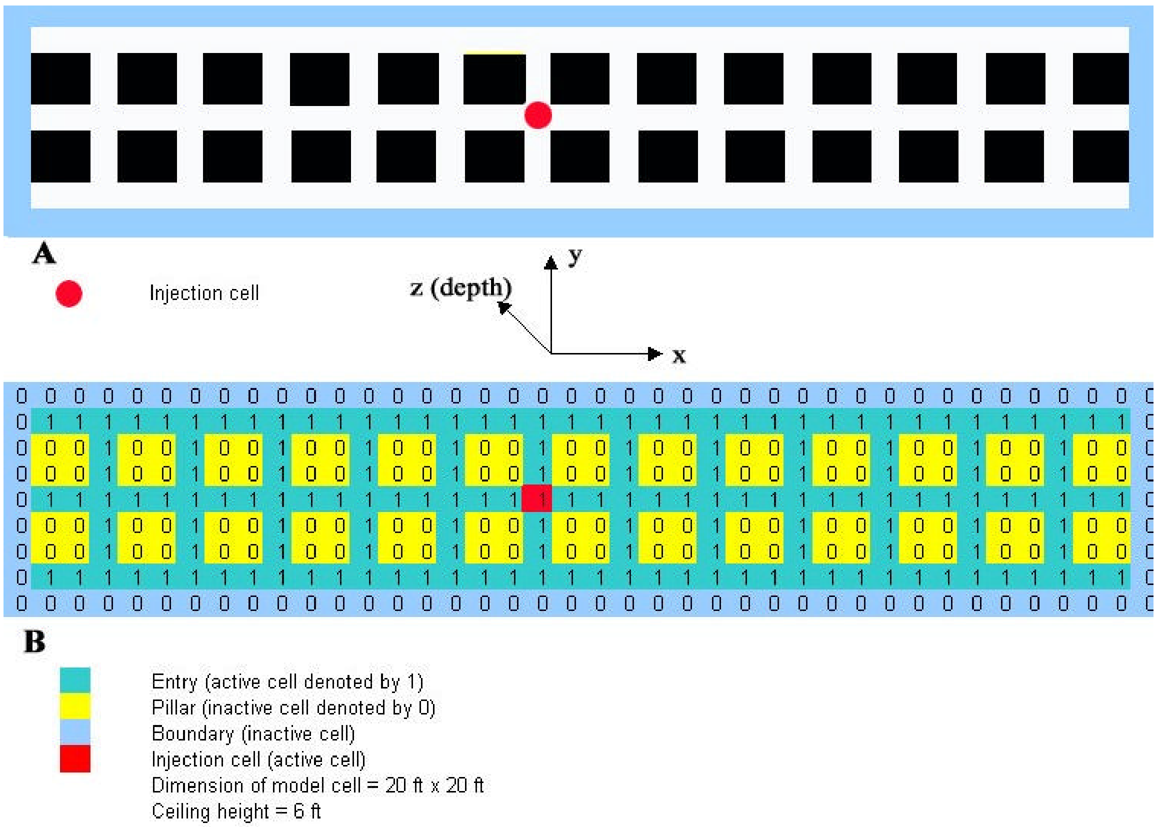


Figure 36. Physical channel (A) and its Groutnet equivalent model (B) for the three-entry system.

Table 17. Material properties of the injected grout.

Modeling unit	**Ft-lbs-s**	**SI**
Model cell size in both direction	20 ft	6.096 m
Total number of rows and column	3, 100	3, 100
Number of active model cells	99	99
Number of injection cells	1	1
Maximum time step	600 s	600 s
Minimum time step	0.6 s	0.6 s
Computer used for calculation	Pentium II	Pentium II
Total grout injection rate	120 t/hr, 150 t/hr, 200 t/hr	0.01716 m^3/s, 0.02152 m^3/s, 0.0286 m^3/s
Specific weight	110 lbs/ft^3	1762.101 kg/m^3
Yield stress	1.04 lbs/ft^2	50 Pa
Grout hardening rate	Non-hardening, hardening	Non-hardening, hardening
Plastic viscosity	0.83 $lbs\text{-}s/ft^2$	40 Pa-s
Slope of the mine	0% and 1%	0% and 1%

Figure 37 A-B shows the full-scale grout injection area of the Crown III mine and its *Groutnet* equivalent, respectively. To model this area in *Groutnet*, a total number of 25 rows and 58 columns were used, that is analysis was carried out for an area of 580 ft long and 250 ft wide. The dimension of a single cell was 10 ft x 10 ft, and a total number of 718 active model cells were used to model the above area realistically. In this model grout was injected for 10 hours a day followed by 14 hours of rest period. During this 14 hours of rest period, grout was allowed to harden. The slope of the area was 1.8% in the south-east (SE) direction, and was considered in the model. The injection points were located in two areas, marked by the red squares in Figure 37B. The properties of the grout material are listed in Table 17. Initially, grout was injected through the first borehole located at the left-hand side of the panel at the rate of 110 tons/hour. The aim of this study was to check how much grout could be injected through the first borehole and its flow profile. If the underground opening is not filled after the injection schedule through the first bore, further injection will be conducted through the second borehole, located at the right-hand side of the panel.

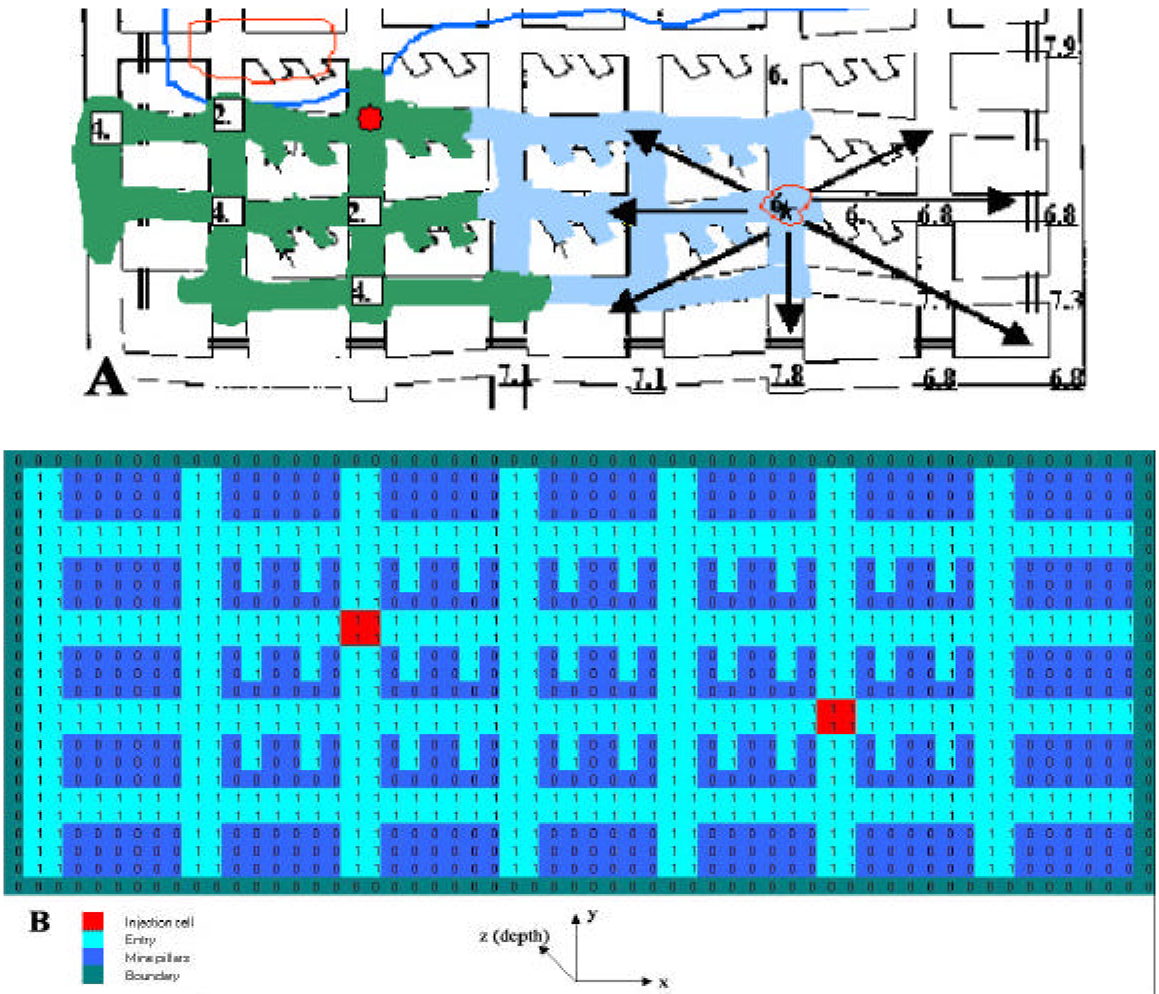


Figure 37. The Crown III room and pillar mine (A) and its *Groutnet* equivalent model (B) for grout flow analysis.

Results and Discussion: The single entry simulations were conducted to investigate the effect of 1) slope, 2) hardening rate, 3) pumping rate and 4) yield stress. Previous investigation by other researchers and the authors indicated that by reducing both yield stress and plastic viscosity, the flowability of the material can be improved (reports already submitted), and hence they have not been incorporated in this report. The simulations of the single entry system were conducted to calibrate the *Groutnet* program and its accuracy. The knowledge gained from the single entry system was then further applied to a three-entry system to further calibrate the program. Finally, the full-scale underground mine was simulated. The results of all the simulations are presented in the following sections.

A. Single Entry Analysis Results

Calibration of the Model: The simulation was allowed to proceed until the entire channel was filled with grout. After the model cell near the injection area was filled, the injected grout flowed under pressure. Figure 38 shows the spread of grout profile at different time periods, the total time taken to fill the complete channel was 5 days 13 hours 52 minutes and 8 seconds and the amount of grout material used was 14,430 tons. According to the injection rate, the channel should have been filled with grout after 4 days 12 hours 52 minutes and 36 seconds. However, *Groutnet* took 5 days 13 hours 52 minutes and 8 seconds to fill the complete channel. The discrepancy is due to the volume error of 23% of the actual volume. The volume error was not consistent in different simulations. It varied from 1% to about 18% in different simulations. Keeping this in mind, further simulations were carried out.

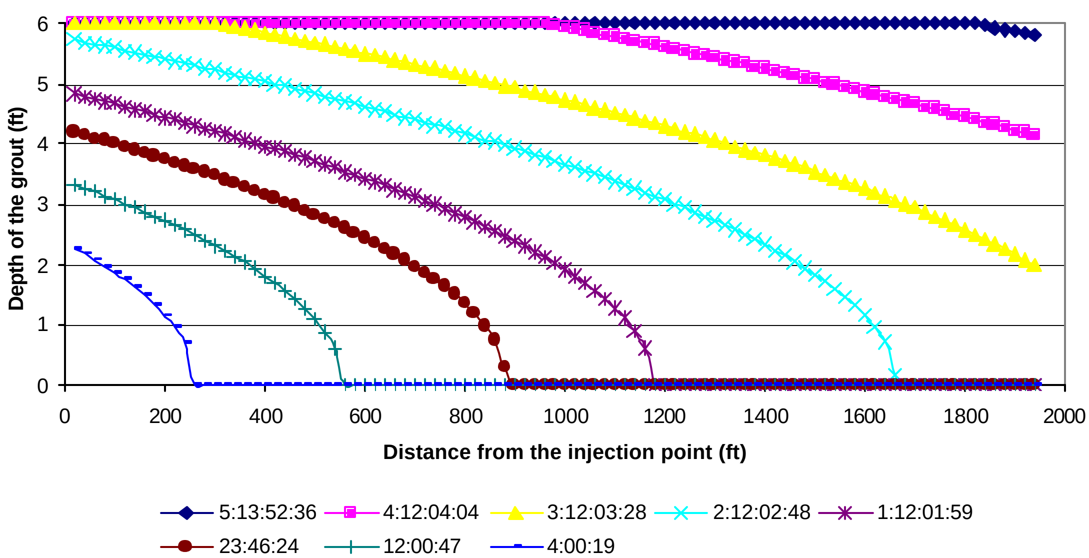


Figure 38. Grout profile of 5 ½ days of grout injection at 0% slope.

Effect of Channel Floor Slope: The floors of underground mines typically follow the slope (dip) of the coal seam, and are usually not level. The dip can be more than 10%. Under uniform, laminar and open channel flow conditions, the slope of the energy grade line is equal to the slope of the channel. In such a situation, the velocity of the flow is directly proportional to the slope. To investigate the effect of the channel slope on the spread of grout, a parametric study of channel slope was carried out.

Figure 38 and 39, respectively compare the effect of 0% and 1% slope of the channel floor after about 5 ½ days of grout injection. Figure 39 shows that with 1% channel floor slope, the flow behavior is normal up to about 12 hours of grout injection. No oscillation was recorded during this period. During this time, the grout flows up to about 1200 ft. This deposition of grout on the channel floor raises the slope such that grout starts to fill up at the extreme right end of the channel slope, and then approaches the left-hand side of the channel (Figure 39).

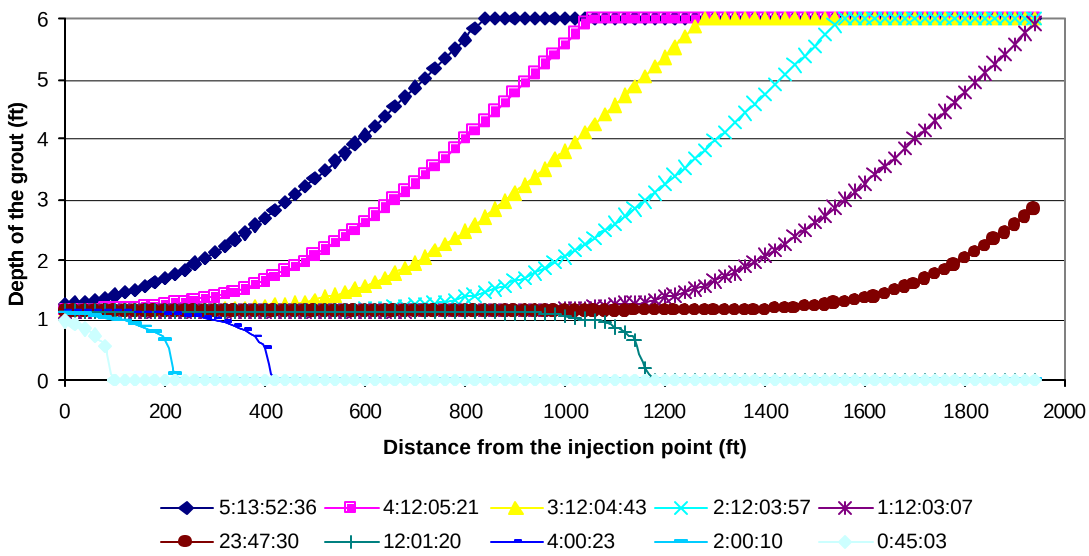


Figure 39. Grout profile after 5 ½ days of grout injection at 1% slope.

The amount of material used for 1% channel slope was 14, 430 ton, which is the same as if it were a 0% slope. The amount of time required to fill the channel for 1% slope is also approximately same as 0% slope. However for 1% slope, theoretically, the channel should have been filled with less material in a less time than 0% slope. This cannot be explained at this time.

Effect of Pumping Rate: Figures 40, 41 and 42 show the effect of the pumping rate at 120 ton/hour, 150 ton/hour and 200 ton/hour, respectively. Figure 40 shows that when the pumping rate is 120 ton/hour, it takes about 5 days 13 hours 52 minutes and 8 seconds to fill the channel. Further increasing the pumping rate to about 150 ton/hour, it takes about 4 days 12 hour 35 minutes and 8 seconds. By further increasing the pumping rate to 200 ton/hour, it takes 3 days 6 hours 11 minutes and 33 seconds.

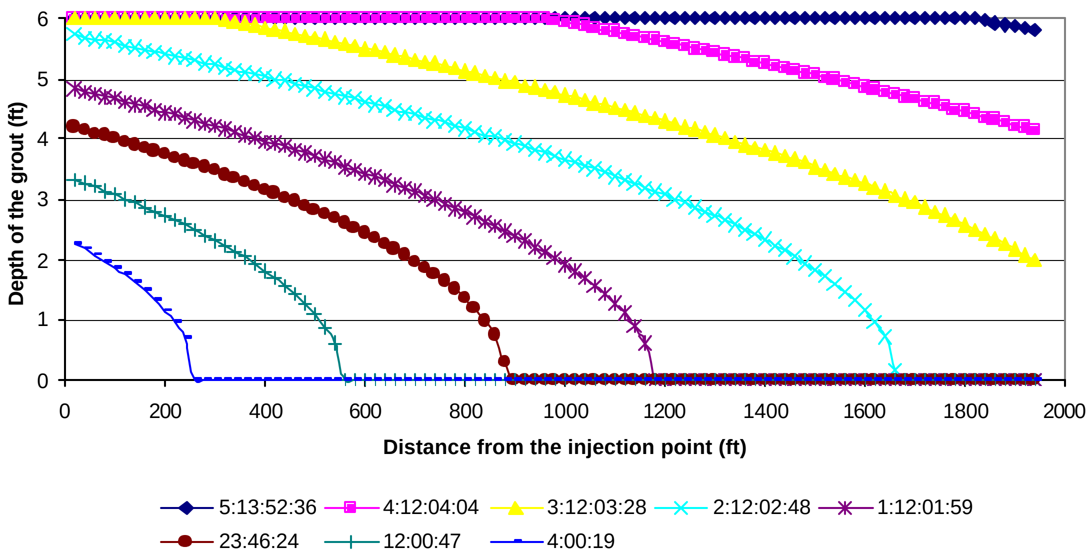


Figure 40. Grout profile of 5 ½ days of grout injection at 120 tons/hour pumping rate ($0.0176\ m^3/s$).

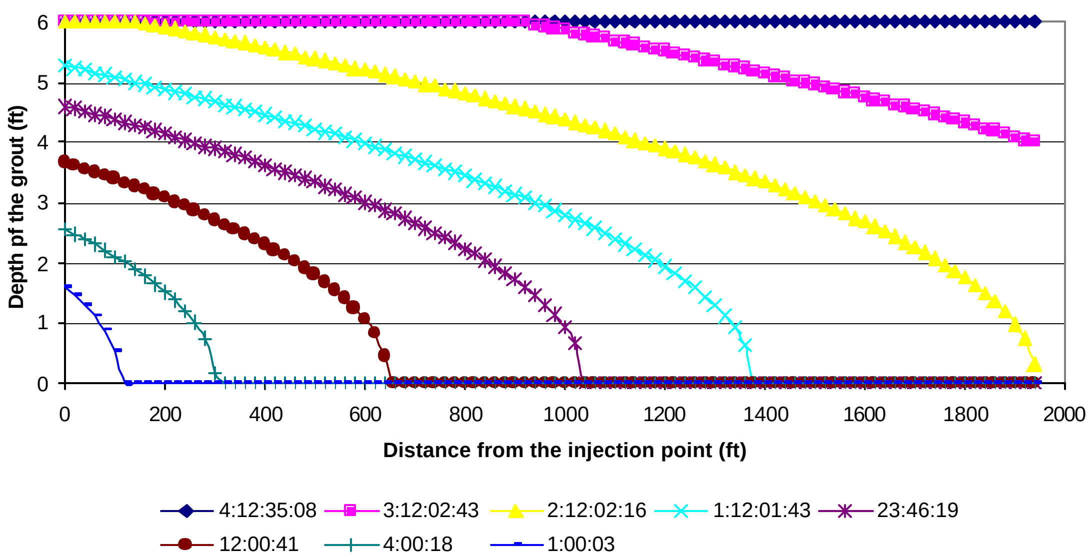


Figure 41. Grout profile of 5 ½ days of grout injection at 150 tons/hour pumping rate ($0.02152\ m^3/s$).

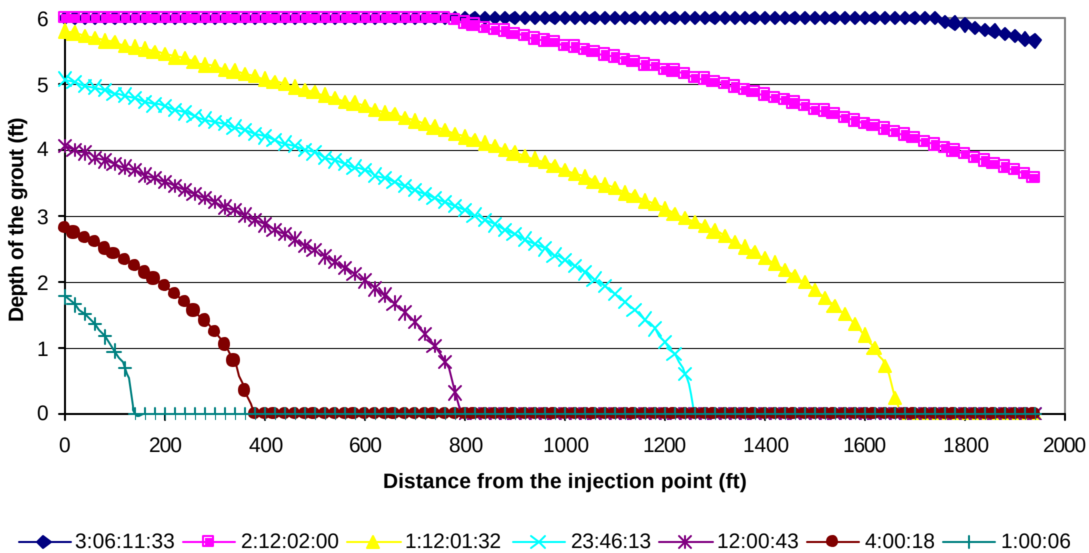


Figure 42. Grout profile of 5 ½ days of grout injection at 200 tons/hour pumping rate ($0.0286\ m^3/s$).

The amount of grout material pumped and the flow distance at different pumping rate were also studied, and are shown in Figures 43 and 44, respectively. Figure 43 shows that if pumping rate increases, the amount of grout required to fill the channel becomes more only up to a certain pumping rate (160 tons/hour). After this, the amount of material required to fill the channel is reduced. This may be due to choking of the grout at the injection point.

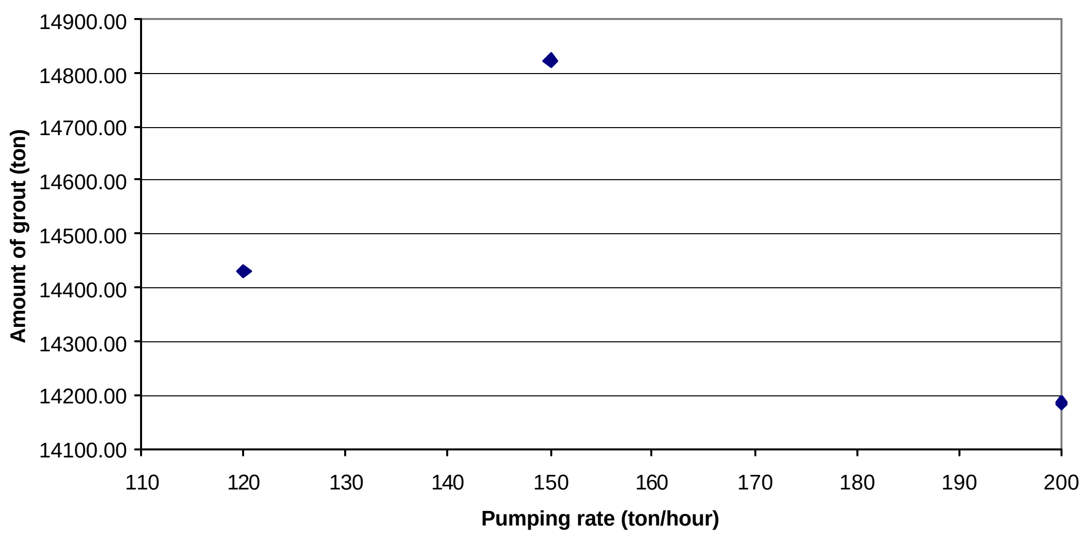


Figure 43. Amount of grout used vs. rate of pumping.

Figure 44 shows the pumping rate vs. the distance traveled by the grout at the end of 1 ½ days. The result shows that if pumping rate increases, the distance traveled by the grout also increases.

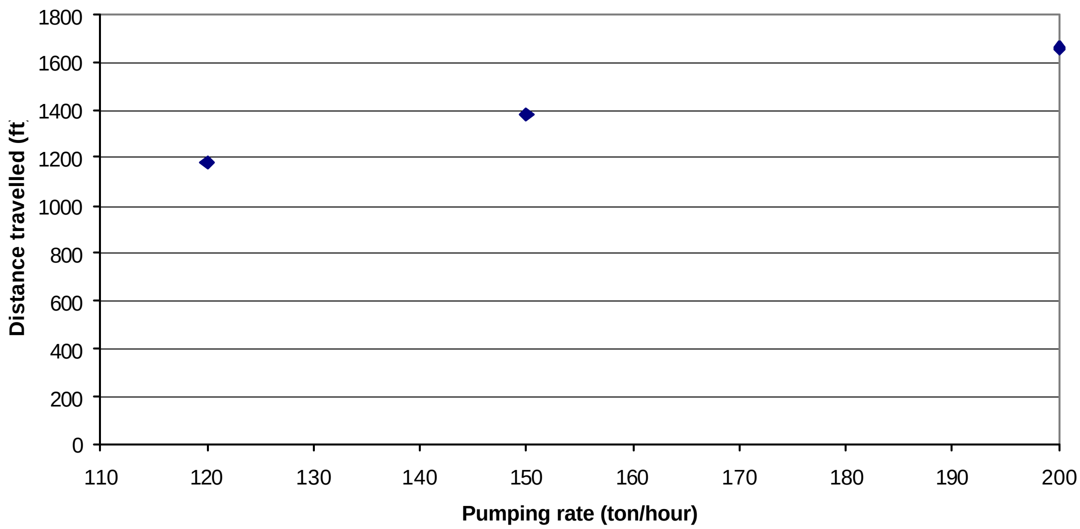


Figure 44. Amount of grout used vs. traveled distance of the grout.

Effect of Grout Hardening: In *Groutnet*, the simulation of grout hardening can be accomplished two different ways, namely 1) by allowing the injected grout to harden completely after each injection period, and 2) using marker particles in the injected grout to calculate the age of the injected grout. The age of the injected grout is used to calculate the yield stress of the grout material at any given point of time. In the present simulation, the second method was used. The model parameters and their values are listed in Table 17.

In the simulation it was assumed that the injected grout would harden after four (4) days of injection. From the literature, it was found that the yield stress at the beginning of the second day's injection is 100 Pa (Stiles, 1999). Therefore, the ultimate yield stress was considered to be 400 Pa in the simulation.

Figure 45 shows the simulation result (grout profile) of the grout hardening process. Figure 45 shows that at the end of 3 days 23 hours 48 minutes and 47 seconds, grout hardens up to a distance of 680 ft from the injection point, and the openings in this area are completely filled. Therefore, no further grout injection could be done at the end of approximately four days. However, further injection, in such a situation, will require pressure to be applied to the hardened grout.

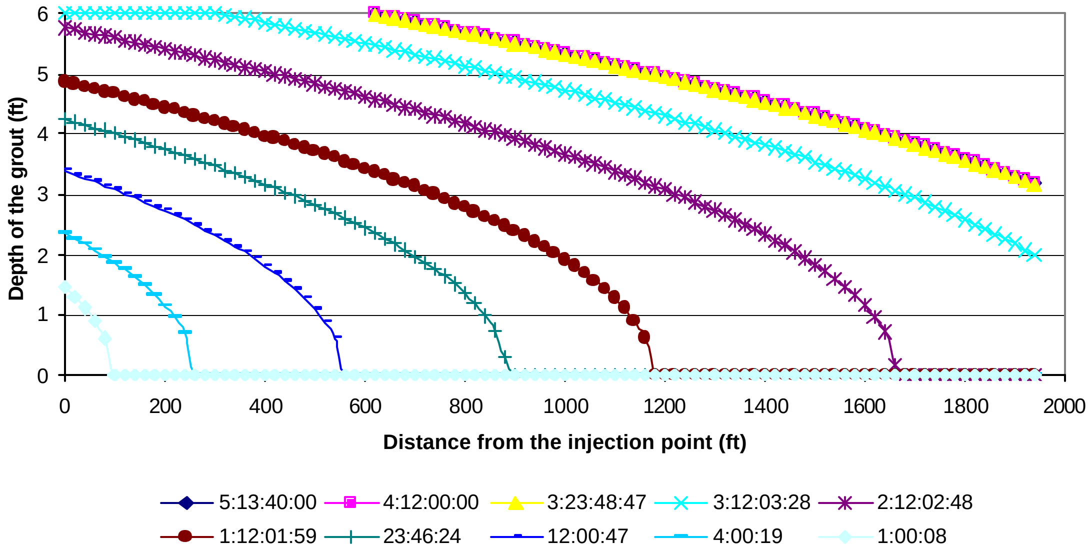


Figure 45. Grout profile for the grout hardening (the grout starts hardening at the end of third day).

B. Three Entry Analysis Results

Figures 46 through 48 show the results of the three-entry analysis at different time intervals. Figure 46 shows that the average depth of the grout after 12 hours of injection is 1.38 ft. Figure 47 shows the flood map of the spread of grout after 1 day 15 hours 2 minute and 50 seconds of injection. The average depth of grout after this time period varied from 0.32 ft to 2.5 ft.

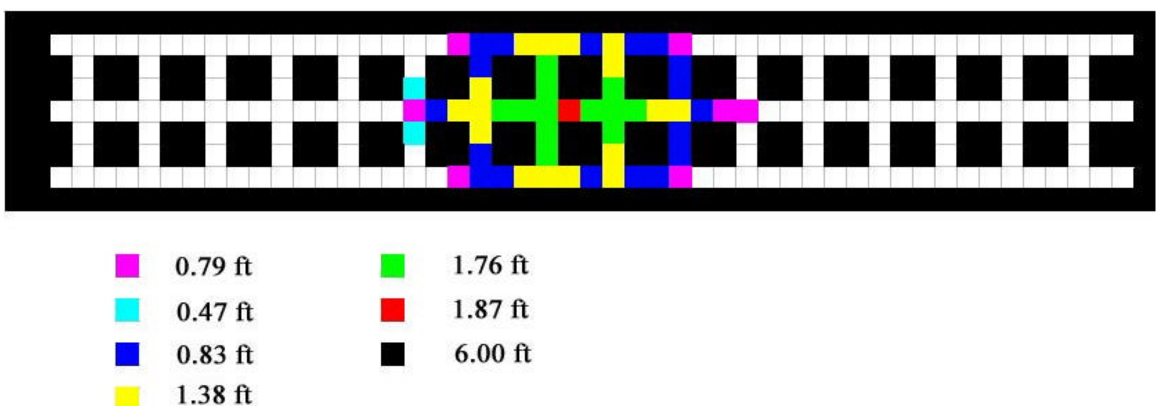


Figure 46. Flood map of grout profile after 12 hours and 1 minute of grout injection.

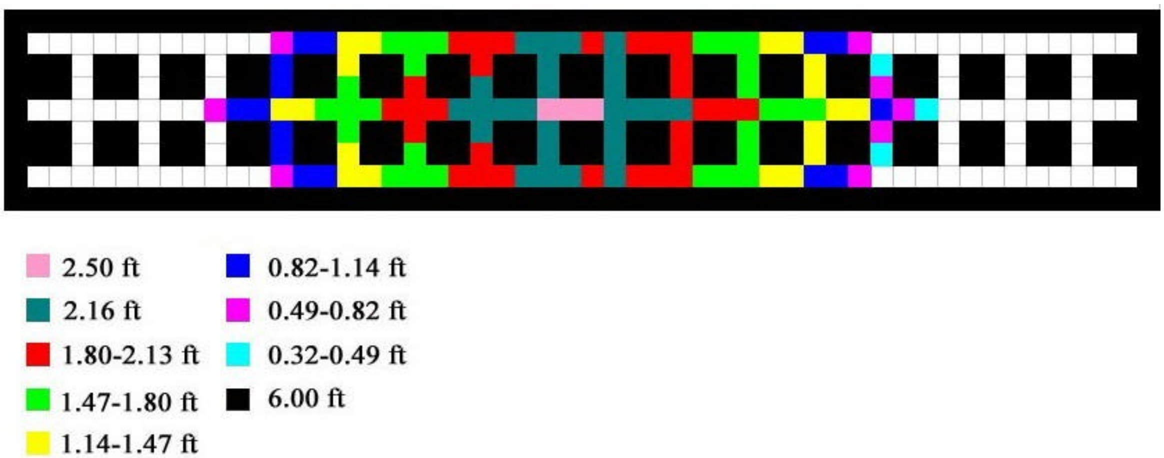


Figure 47. Flood map of grout profile after 1 day 15 hours 2 minutes and 15 seconds of grout injection.

Figure 48 shows the flood map at the end of 10 days 18 hours and 12 minutes (assuming 1 day = 24 hours for continuous injection) for the three-entry system. The simulation stopped at 10:18:12:24 days. According to the rate of injection, the three-entry system should have been filled after 9 days 21 hours 45 minutes and 5 seconds. The amount of grout used was 27832.39 tonne (15795.00 m^3). Theoretically, the amount of grout required to fill the complete three-entry roadway (14543.53 m^3) is 25627.17 ton. Therefore, there is a volume error of approximately 8% associated with the simulation. This volume error is due to the finite difference approximation of the Groutnet procedure.

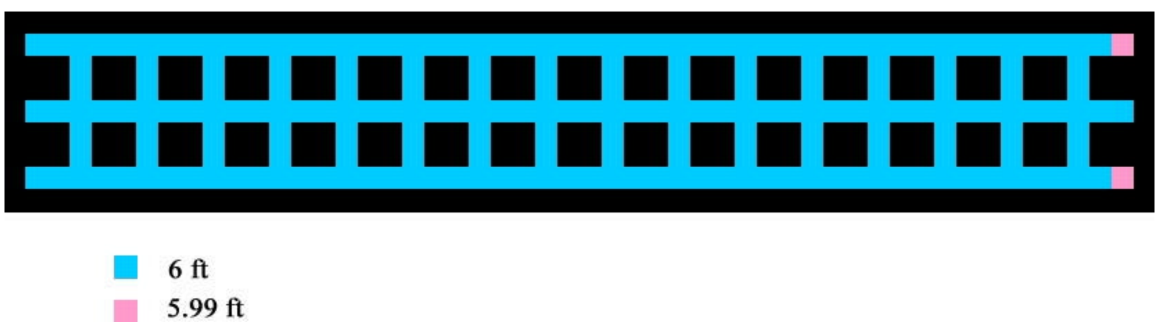


Figure 48. Flood map of grout profile after 10 days 18 hours 12 minutes and 24 seconds of grout injection.

C. Crown III Mine Analysis Results

Figures 49 through 57 show the results of the Crown III mine simulation after 1, 2, 3, 4, 5, 6, 7 and 10 days, respectively. After one day of grout injection, the grout spread up to 220 ft from the injection point in the horizontal dip direction and 80 ft in the horizontal rise direction. Along the rise direction, it spread up to 100 ft. The maximum depth of grout was near the injection point and was about 2.51 ft (Figure 49). The amount of grout pumped after one hour was 1100.06 tons.

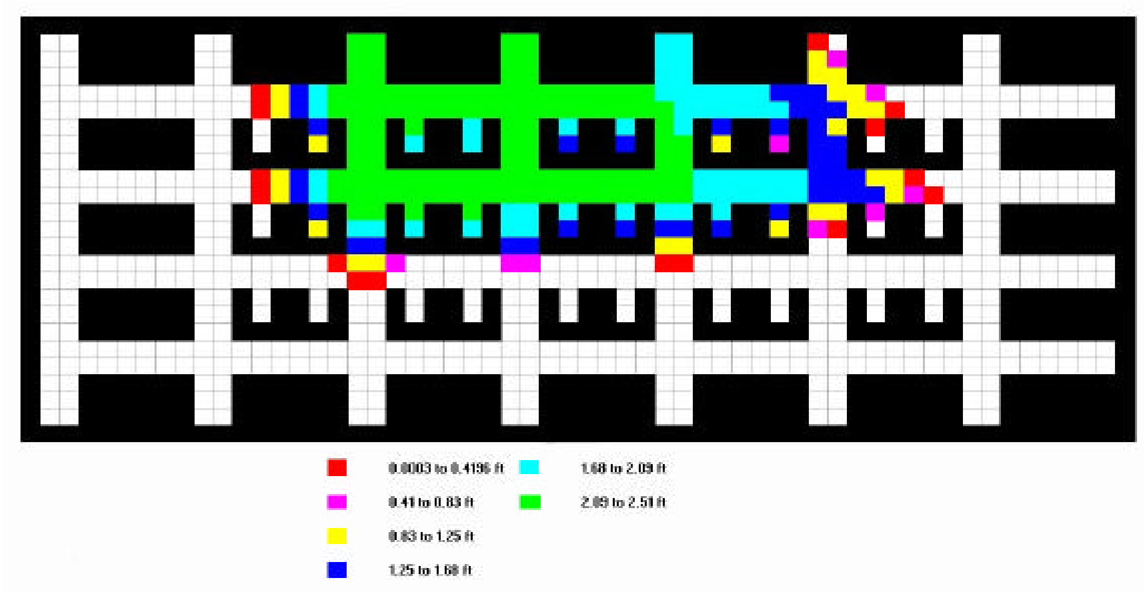


Figure 49. Flood map of grout profile after 1 day of grout injection.

After two days grout flowed further from the injection point in all the directions. In the horizontal direction, it spread maximum of 300 ft in the horizontal direction, and 100 ft in the vertical direction from the injection point. In the rise direction, the maximum depth of grout was about 3.75 ft. The amount of grout used after two days was about 2199.98 tons (Figure 50).

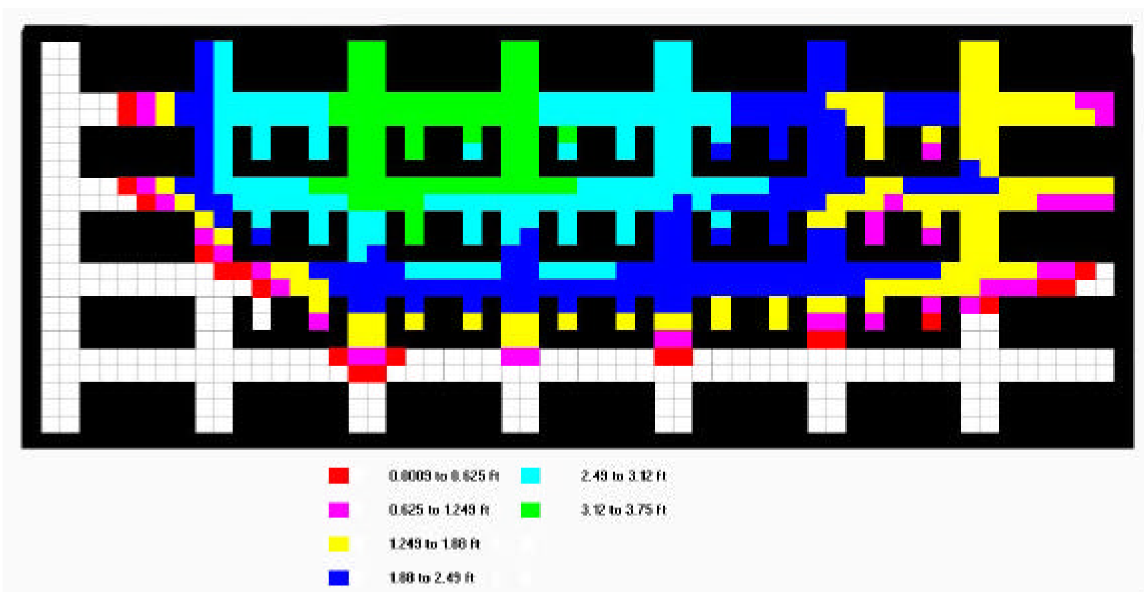


Figure 50. Flood map of grout profile after 2 days of grout injection.

After three days of grout injection, grout flowed in almost all the areas, except some places in the dip side. The maximum grout was accumulated in the North-East direction in the dip side. The maximum depth of grout near the injection point was from 4.13 to 4.96 ft. The amount of material used after three days was about 3300 tons (Figure 51).

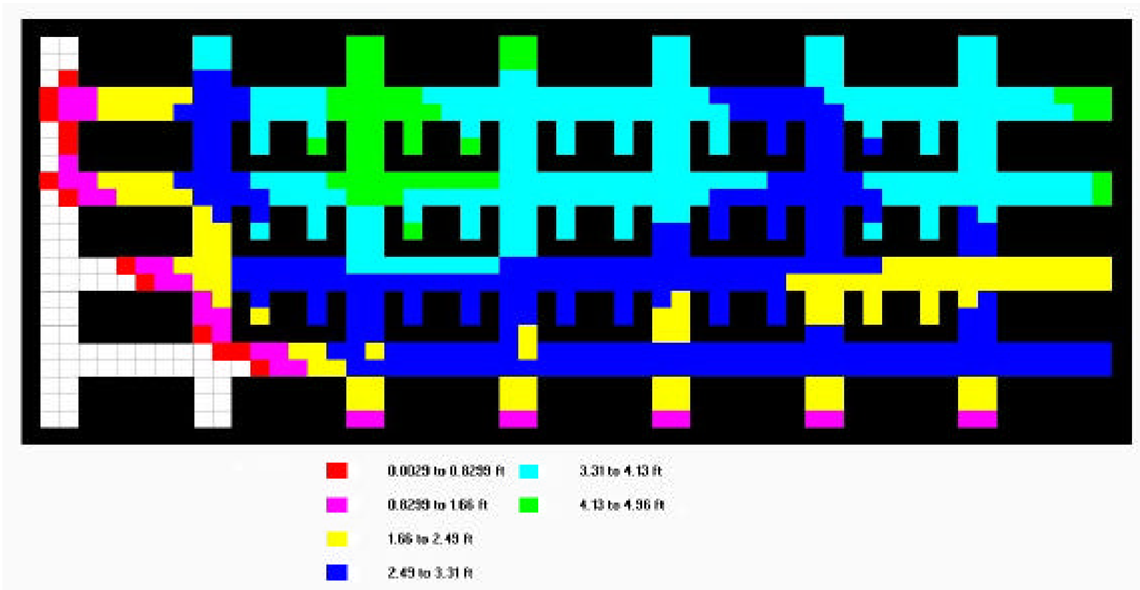


Figure 51. Flood map of grout profile after 3 days of grout injection.

After four days of grout injection, grout flowed in almost all the areas, except some places in the rise side. The maximum grout was accumulated near the injection point. The maximum depth of grout near the injection point was from 5.01 to 6 ft. The amount of material used after four days was about 4400 tons (Figure 52).

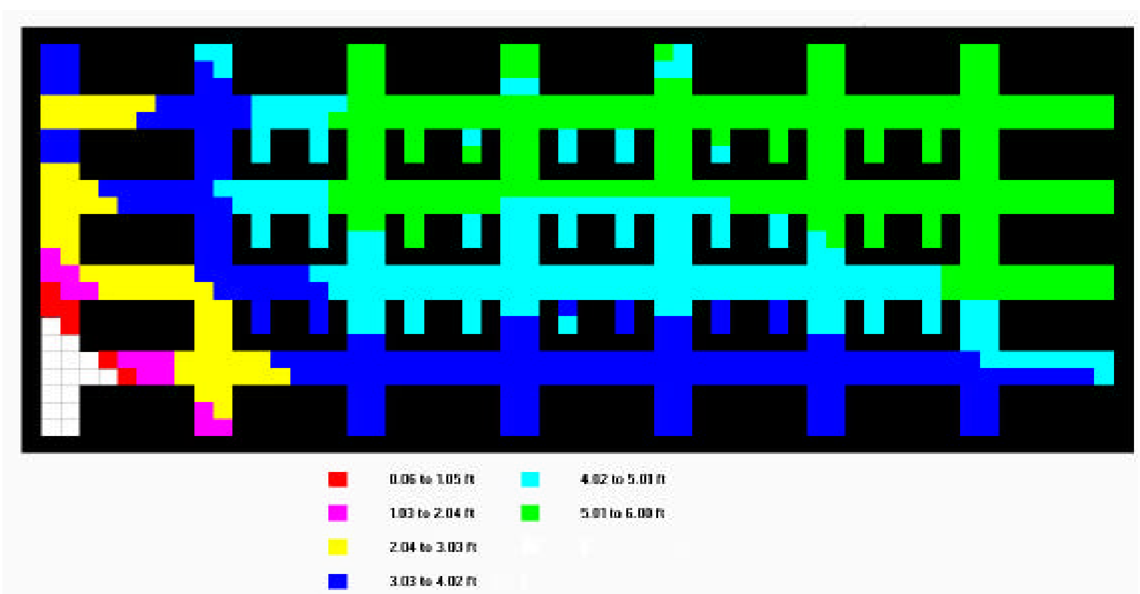


Figure 52. Flood map of grout profile after 4 days of grout injection.

After five days of grout injection, grout flowed in all the areas, and the mine was filled to about 90% of the areas. The maximum grout (5.3ft to 6 ft in depth) was accumulated near the right side of the injection point. The maximum depth of grout near the left side of the injection point was from 3.91 to 5.30 ft. The amount of material used after five days was about 5500 tons (Figure 53).

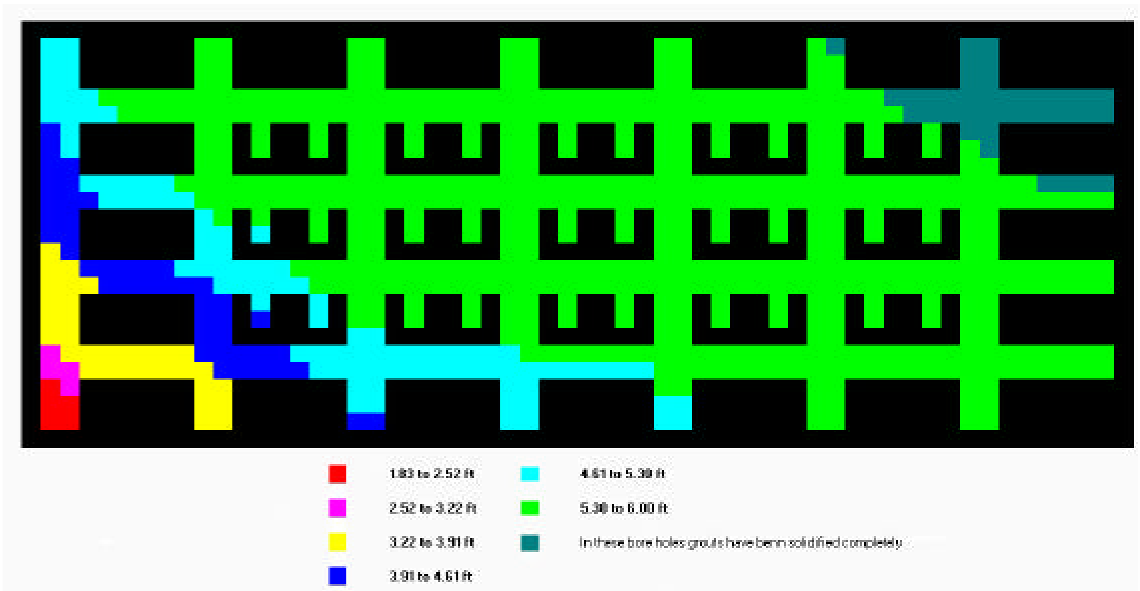


Figure 53. Flood map of grout profile after 5 days of grout injection.

After six days of grout injection, grout flowed in all the areas, and in about 90% of the areas, all the grouts were filled and solidified. No further flow occurred in these areas as well as the injection. However, there was some amount of low in the left side of the injection point in the dip side. The maximum depth of grout in these areas was also very high (5.52 ft to 6 ft in depth). The amount of material used after six days was about 6600 tons (Figure 54).

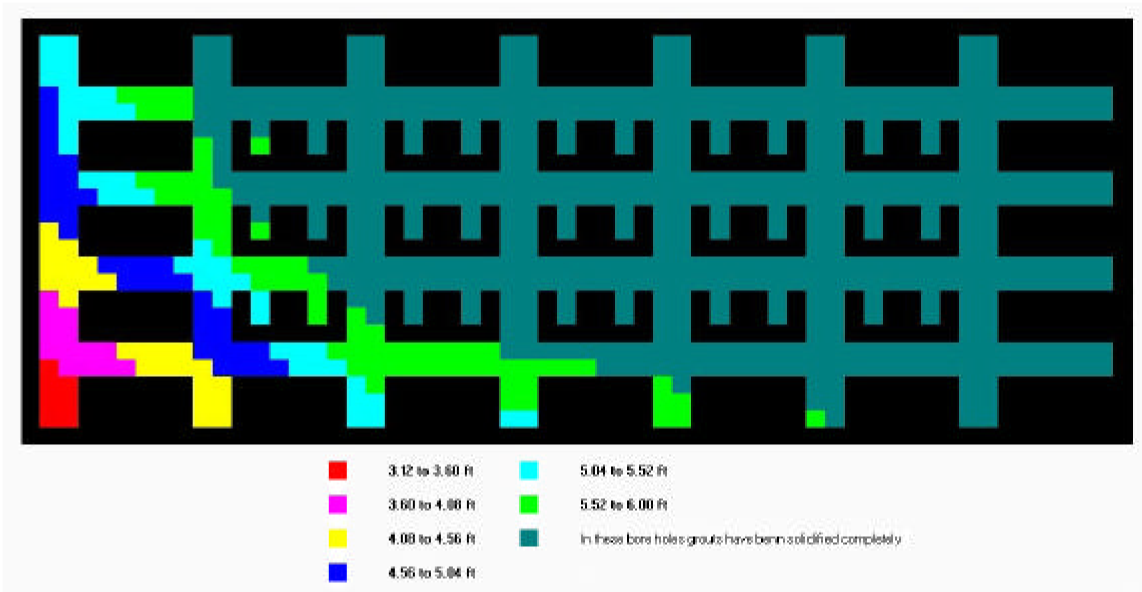


Figure 54. Flood map of grout profile after 6 days of grout injection.

As the flow continued, the profile obtained after 7 days of the grout injection was not significantly different than day 6 (Figure 55). The amount of material used at the beginning of the 7th day was 6601 tons. No further was done after end of the end of 6 days, nd the grout was was not solidified flowed internally. That is why profile obtained after the 6th and 7th days are almost identical.

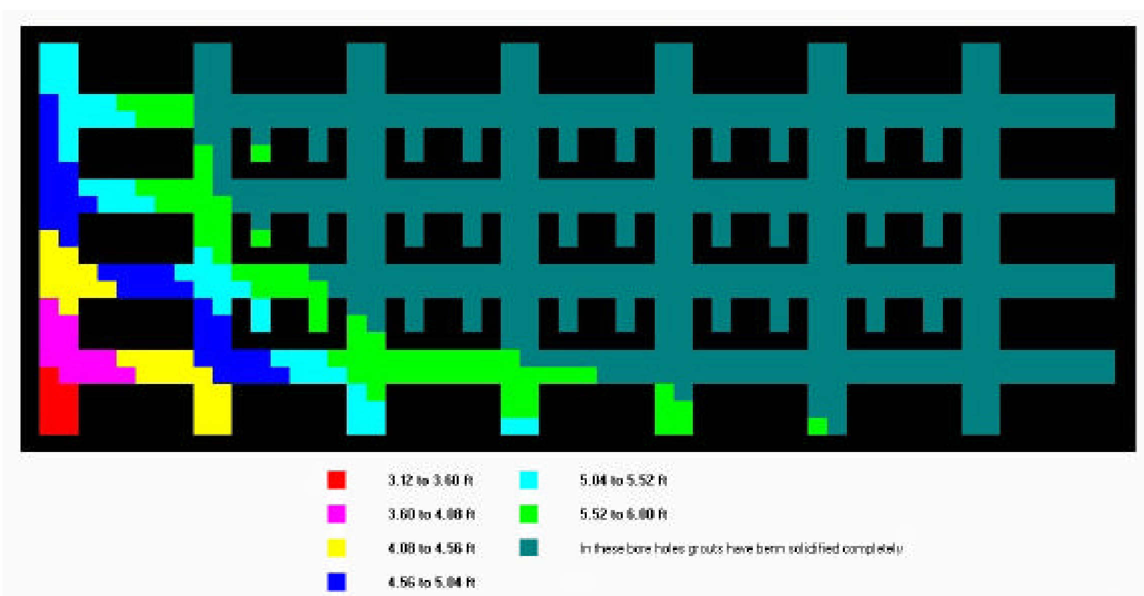


Figure 55. Flood map of grout profile after 7 days of grout injection.

The simulation still continued because of the very little internal flow of grout and ended on day 10. At this point the most of the underground voids were filled with solidified grout. The maximum depth after the end of 10th day was 5.52 to 6 ft (Figure 56).

The most of the grout flowed in the dip direction first. This is obvious for the sloping mine floor, and the same behavior was observed in one entry slope analysis. The total tonnage used after the end of ten days was approximately 6605 tons (Figure 56). This coincides well with the calculated tonnage.

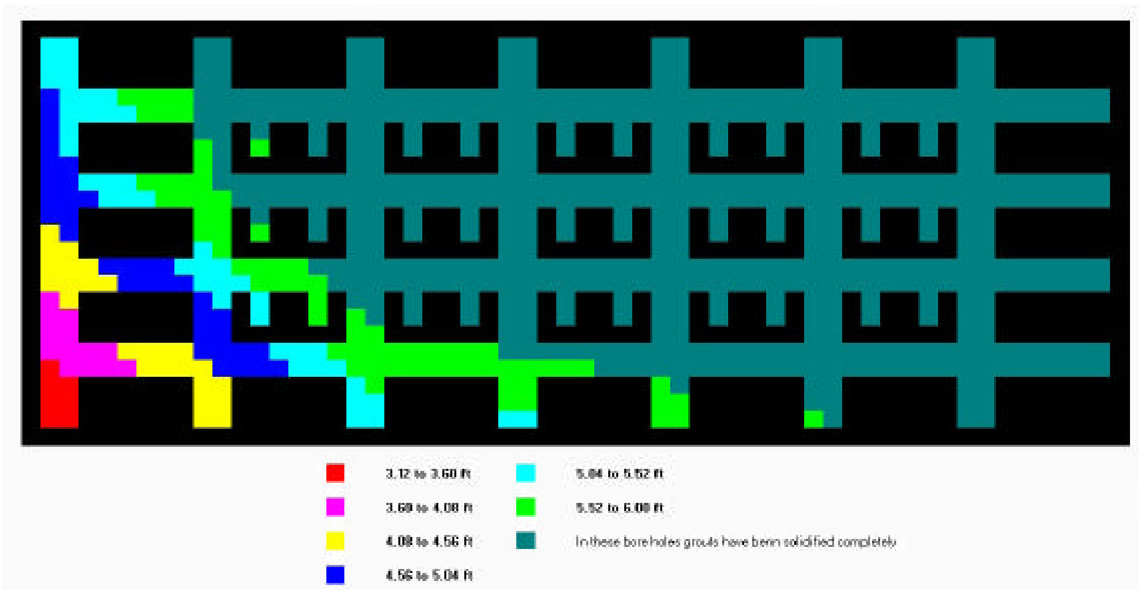


Figure 56. Flood map of grout profile after 10 days of grout injection.

Simulation was also carried out for the same for continuous grout injection at the rate of 110 tons per hour. In this case, the model converged after 5days 1 hour 18 minutes and 28 seconds of grout injection, and the result is shown in Figure 57 in the form of a flood map. Based on the simulation, the model consumed 13,343 tons of grout. According to the calculation, the model should consume 13,343 tons of grout. There is an about 0.84% volume error associated with this simulation. The maximum depth of grout was found in the dip direction and was about 6 ft. This simulation compared favorably, and the volume error is less in this case compared to other simulations, because the grid size is less in this simulation.

In the present study, the yield stress considered was 50 Pa and injection rate of 110 tons/hour. The simulations were also conduced for yield stress of 60 Pa and 70 Pa and the flow rate of 120 tons/hour. From the analyses, it was found that for large scale-simulation like Crown III mine grout injection, there is no significant difference of the slightly increased yield stress as well as injection rate. From the overall analyses of the Crown III room and pillar mine simulation, it was found that the *Groutnet* program simulated the injection schedule well, which compares favorably well with the existing conditions.

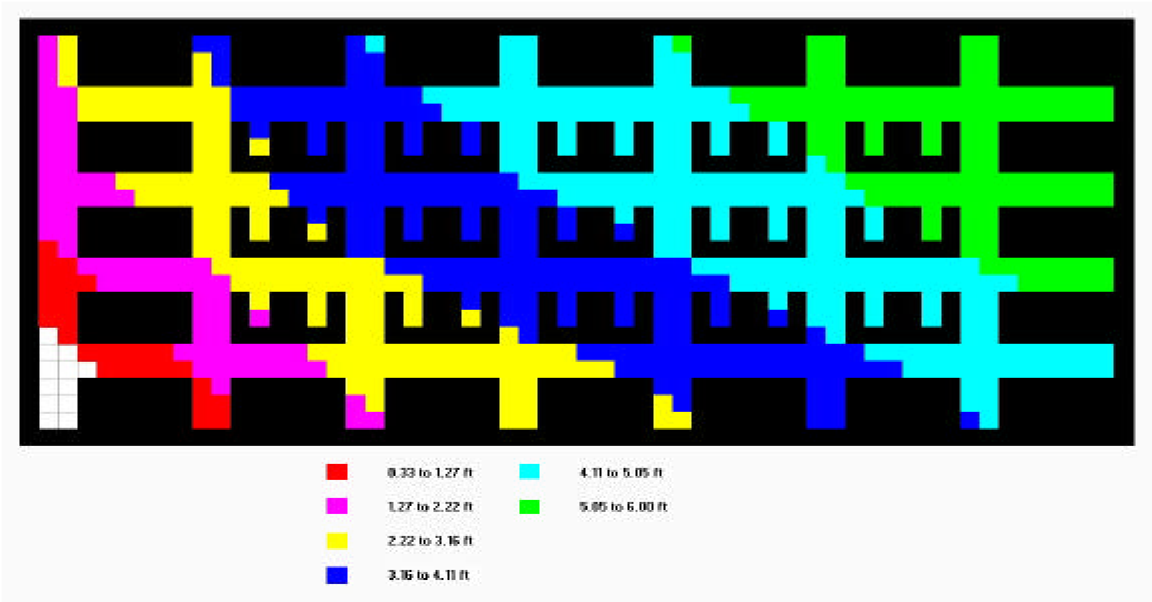


Figure 57. Flood map of continuous grout injection after 5days 1 hour 18 minutes and 28 seconds of grout injection.

VII. CONCLUSIONS AND RECOMMENDATIONS

Conclusions:

- Engineered mixes, consisting of coarse processing waste, and coal combustion byproducts, can be developed that are environmentally benign, and have appropriate strength and flow characteristics for backfilling underground.
- Two technologies, one using a concrete mixing plant, and the other using a high speed auger mixer, for making paste backfill mixes were successfully demonstrated for underground backfilling.
- About 16,000 tons of paste backfill were successfully managed underground without appropriate commercial equipment. Much larger amounts could have been managed if appropriate equipment was available.
- Short-life pillars can be designed with lower safety factors in room-and-pillar mine design. Such pillars permit higher extraction ratio from a mining area, and allow ample time for backfilling to be done to achieve long-term ground stability in the area.
- Backfilling should be planned for 24-hour operation so that the paste backfill does not have a chance to set up.
- Underground management of coal processing waste and CCBs have significant economic and environmental advantages over surface management. However, the management costs could be slightly higher where land acquisition costs are high such as in central Illinois.
- *Groutnet* appears to provide reasonable simulation of flow behavior of grout in room-and-pillar mine workings. Some modifications to the model (modeling large areas with limited computer time) would be very beneficial.
- The model predicted that a typical fly ash grout will flow at least 300-ft from the injection point.
- The model suggests that about 13,000 tons of grout may be pumped from a single borehole in a typical room-and-pillar mine before refusal.
- Pumping rate provides significant advantages in terms of the amount of grout pumped and the time to pump.
- Continuous pumping of grout without stoppages is highly recommended.

Recommendations

- The cooperating coal company has an interest in commercializing the developed technology if appropriate finances can be arranged. ICCI and OCDM/DCCA should assist the company to demonstrate the technologies over an extended period of time.

- Regulatory agencies, such as Illinois EPA, Illinois Office of Mines and Minerals, Office of Surface Mining, and Mine Safety and Health Administration should be contacted to develop appropriate guidelines for permitting and implementation of underground management of coal processing waste and coal combustion byproducts.

Recently completed research and development related to underground management of coal processing waste and CCBs in Illinois should be discussed with USEPA and Illinois EPA. This may influence regulations related to mine fills that are being considered by USEPA.

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