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Disposal of Fluidized-Bed Combustion Ash in an Underground Mine to Control Acid Mine Drainage and Subsidence

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Abstract

This project evaluated the technical, economic and environmental feasibility of filling abandoned underground mine voids with coal combustion byproducts (CCBs) to control acid mine drainage and subsidence. The project contained three phases. Phase I consisted of a laboratory phase to determine the flowability of CCB grouts and to determine their ability to control acid mine drainage. During this phase it was determined that one yard of highly flowable grout could be obtained from 1000 lbs of FBC ash, 80 lbs of bentonite and 125 gallons of water. The bentonite was used to provide for greater flowability and stability of the grout. Phase II consisted of a small field test of the Phase I grout design. A total of 1000 yards was injected into the Fairfax Mine owned by Anker Energy Corporation. The grout flowed 635 feet from a single borehole. During Phase II a field testing procedure for grout flowability was developed (ASTM STP 1311). The test allows for a flowability check of thin grout mixes (slump greater than 10 inches). Also during Phase II a pneumatic device for the injection of CCBs into underground mine voids was developed. The device, the Burnett Ejector, is attached to the bottom vertical pipe and uses a supersonic air stream to stow materials horizontally into the mine void. Phase III of this project consisted of the full scale demonstration of both pneumatic and hydraulic stowing techniques to control acid mine drainage and subsidence. The Sherman Helms No. 2 mine owned by Anker Energy was used to demonstrate the pneumatic stowing of CCBs into a mine void while the 11 acre Longridge Mine owned by Anker Energy was used to demonstrate both the pneumatic and hydraulic techniques. Due to limited availability of the FBC ash the grout mix in Phase III was changed to include 75% class F ash from the Albright Power Station mixed with 25% cement kiln dust from Capital Cement Corporation. The solids were mixed with an equal weight of water to produce a highly flowable grout mix. The grout mix flowed over 2,000 feet from a single borehole and exited the mine with the mine drainage. To control the placement of the grout, the pneumatic technology was employed to stow barriers made of limestone into the mine void. The area behind each barrier was completely filled with the grout to control subsidence and block the acid forming materials from contact with air and water. The flow of acid mine drainage from the Longridge Mine was reduced by 97%.

Executive Summary

This document reports on the findings of a study to determine the technical, economic and environmental feasibility of using coal combustion byproducts to control acid mine drainage and subsidence in an underground mine. Phase I involved a laboratory study of grout mixes while Phases II and III tested and demonstrated techniques of stowing coal combustion byproducts in an underground mine. The report is arranged according to the major tasks undertaken during the project team at West Virginia University. Task 1 Grout Formulation was completed by Dr. William J. Head. The objective of the task was to create a grout mixture from FBC ash. Tasks 2 and 3 were completed by Dr. Donald D. Gray and are concerned with the computer modeling and bench scale testing of Grout Rheology or flowability. The objective of this work was to optimize the grout designed by Dr. Head for flowability. Task 3b and 3c were concerned with the design and bench scale testing of a pneumatic device to stow coal combustion byproducts in underground mine voids. Task 4 documents the water quality at the field demonstration sites before during and after field operations. Tasks 5 and 6 were completed by Dr. Hema J. Siriwardane. These tasks were concerned with Subsidence Control and Contaminant Transport respectively. Tasks 7, 8 and 9 report on the results of the Phase II field operations at Anker Energy's Fairfax Mine. Task 11 reports on the Phase III field operations at Anker Energy's Longridge Mine. Task 12 is the Economic Analysis while Task 14 is a Regulatory Analysis of the use of CCBs in mine environments. Task 13 also completed by Dr. Siriwardane is a water quality model of water around the mine after grouting, it is included with the Task 6 results.

The project successfully demonstrated that grouts can be made of coal combustion byproducts and injected in underground mines to control acid mine drainage and subsidence. Furthermore, this project demonstrated that pneumatic stowing techniques can be employed with grouting to make the grouting more efficient. The efficiency is gained by getting full roof contact of the grout in sections of the mine. This allows for the elimination of water flowing through the mine and a reduction in acid mine drainage. The acid load from the Longridge Mine was reduced by 97% after the installation of 43,000 yards of coal combustion byproduct grout into the mine void.

This project supported seven Masters Level graduate students and one Ph.D. student. This project supported the development of an ASTM Standard Technical Paper No. 1311 and numerous technical papers listed below.

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Introduction

The United States currently has over 480 billion tons of mineable demonstrated reserve base of coal. Two thirds of this reserve is mineable by underground methods. Unfortunately coal contains a number of components such as sulfur and nitrogen that form undesirable oxides during combustion. Coal also contains incombustible mineral matter that is converted into ash and leads to suspended particles in air and degradation of downstream components. Several technologies designed to prevent the release of sulfur dioxide and nitrogen oxides have been developed. One such technology is Circulating Fluidized-Bed Combustion (cFBC). The cFBC technique combines limestone and coal in a circulating combustion chamber. The limestone burnt with the coal will capture the sulfides and a low combustion temperature prevents nitrogen oxides from being formed. However, this technique produces a large quantity of waste that has free lime or slaked lime with a large portion of fly ash. This material can present a disposal problem, and due to its high pH, landfill costs can be quite high.

Coal mining areas have been subjected to a series of problems. The legacy of coal mining has been ground collapse due to subsidence, and acid mine drainage due to dissolved pyrite forming sulfuric acid where abandoned mine workings drain underground water pools. The combined subsidence and acid mine drainage problems due to abandoned coal mines prompted the enactment of the Surface Mining and Control and Reclamation Act of 1977. This act was passed in order to accomplish the reclamation of abandoned coal mine lands. This reclamation includes the abatement of acid mine drainage, the prevention of mine subsidence, and reclamation of surface mines. The act also places strict regulations on coal companies, requiring them to take responsibility for the lands left behind after mining.

The combined problems of advanced coal combustion byproduct disposal, mine subsidence, and acid mine drainage create a ready made market. The problem of FBC ash disposal can be solved by stowing the material in abandoned mines. The material may be made into a grout or steowed pneumatically. When stowed pneumatically, the ash, which is typically is highly pozzolonic, will achieve a reasonable compressive strength in the humid underground environment without additives. In addition, the neutralizing capability of the material makes it an ideal candidate for acid mine drainage prevention. When grouts are utilized, the flowability characteristics of the FBC grout will fill the complex mine voids. This solution carries the multiple benefit of creating a market for the FBC byproducts and providing mine closure engineers and abandoned mine authorities a new tool to combat abandoned mine land problems. The potential market for the FBC byproducts includes every active coal and non-coal mine, and every abandoned mine that state and federal agencies are in the process of reclaiming. The potential market is such that future coal contracts may include clauses to backship the FBC material to the mine. Full implementation of this application of the FBC material will change it from a waste product to a valuable product to be sold. This will help electric companies and other large and small scale coal burners to make the decision to use FBC or other Advanced Coal Combustion technologies.

This report will individually report on the main tasks of this research project. The first section

is Task 1 Grout Formulation. Dr. William J. Head was the principal investigator for Task 1. Experiments have been conducted to determine the optimum mix of ash, water, and admixtures to give the grout optimum flowability and strength. Dr. Donald D. Gray was the Principal Investigator for Task 2 Grout Rheology Model and Task 3 Bench-Scale Rheology Tests. Task 2 involves the computer modeling of grout flow given formulations provided by Dr. Head. The model will also aid in the design and verification of the Bench-Scale testing. Under Task 3 Dr. Gray and his team were concerned with the flowability of the grout and modeling it on a bench scale. Task 4 Water Quality Monitoring was concerned with gathering background water quality data for the Phase II and Phase III field demonstration sites. Dr. William A. Sack was the Principal Investigator for Task 4. Task 5 Subsidence Control and Task 6 Contaminant Transport research efforts were lead by Dr. Hema J. Siriwardane. Dr. Siriwardane's research focused on determining the minimum strength requirement for the grouts to be placed into the mine void and to develop the borehole drilling patterns for both Phase II and Phase III mines. Task 6 modeled the change in transport of contaminants into the groundwater below the mine before and after grouting. Additional work performed includes investigating the use of pneumatic devices to inject FBC into a mine void. Phase II and Phase III field reports are included as well as an Economic and Regulatory Analysis of using coal combustion byproducts to control acid mine drainage and subsidence in underground mines.

TASK 1 GROUT FORMULATION

Purpose

The prime purpose of the work reported here is to develop grouts that are suitable for injection into selected underground mines. The grouts consist of fluidized bed ash, water, and other ingredients that may be necessary to achieve desirable mixture characteristics.

Methodology

The laboratory program associated with grout development focused on attainment of mixtures which were structurally adequate, appropriately durable, placeable in the field, and economical. Assessments of underground cavities with the aid of finite element computer codes aided in determination of desirable mixture properties. Such properties were adequate strength for subsidence control and appropriate permeability which affects grout durability and transport of contaminants through grout media. Considerations of flow and spreadability of fluid grout informed the development program of properties necessary for fresh grouts so that they may be properly and economically placed in underground cavities. Effort was expended in the program to employ water from mine pools as grout mix water as an example of an economic consideration.

The laboratory program entailed mixing varying proportions of Morgantown Energy Associates (MEA) ashes with water and other ingredients and casting the mixtures into cylinders and beams for future testing. To date, the test have dealt with strength, strength development, consistency/flowability, and durability. Strength tests were unconfined compression test of three inch diameter by six inch high cylindrical specimens conducted at selected time intervals after casting. Specimens were covered after casting to prevent egress of water. Triplicate samples were tested and average strengths were reported. A few beam specimens were fabricated to determine representative flexural strengths of selected mixtures. Strength development was assessed by monitoring in compressive strength with time. Based on previous experience, a target consistency of ten to twelve inch slump of the fresh grout was selected for all mixtures. Tests were also performed to determine changes in mixture consistency with time.

In sum, employed in the laboratory program were tests to determine relationships among grout water content, age, strength, consistency, flowability, and durability. Strength tests were conducted in accordance with provisions found in ASTM Standard test methods C 39 (compression test) and C 78 (flexure test). Consistency (slump) test were conducted according to ASTM C 143.

Results and Discussion

Results of the laboratory program appear in Figures 1-18; a discussion of the Figures follows. Figures 1 and 2 represent particle size distribution curves for the coarse fraction (3/8 in to # 200 sieve) and the fine fraction (-200 sieve) of the MEA fluidized bed ash. The coarse fraction resembles a sandy particulate material while the fine fraction resembles a silty substance.

Information from MEA plant personnel indicates that the average production of ash is about evenly split between coarse (bottom ash) and fine (fly ash) particles. However, production of ash is nonuniform both in terms of particle sizes available and with respect to physical characteristics of the ashes. The former point is illustrated in Figure 1, the latter in following Figures.

Figure 3 is a plot of compressive strength as a function of water content and age for specimens whose solids contents were 50 percent bottom ash and 50 percent fly ash. The results are typical: strength increases with age for a given water content and decreases with increasing water content for a given age. Tap water was used in fabricating these specimens. Water content here and in following plots was measured in terms of percent of total weight of specimen.

Figure 4 is a plot of compressive strength versus age for specimens fabricated with a 38 percent tap water and 4.6 percent bentonite, a commercially available clay material. Bentonite was used to enhance the flow characteristics of the grout mixture. Strength increased with age as anticipated.

Figure 5 contains plots of compressive strength versus age for specimens containing various combinations of water content and admixtures. WRDA-19 is a commercially available high-range water reducer employed on occasion in concrete technology to enhance consistency or strength characteristics of concrete mixtures. WRDA-19 was used here in attempts to enhance flow characteristics of grouts. Darafill is a commercially available air entraining admixture used to entrain air bubbles in concrete mixtures. It was used here to enhance mixture flowability. The plots indicate that the water reducing agent had little effect on mixture strength while the air entraining agent produced notably weaker mixtures compared to the mixtures containing tap water only. The fourth plot indicates that the mixtures containing bentonite may gain strength relatively slowly. The higher (39 percent) water content for this latter mix was necessary to achieve the target slump of ten to twelve inches for the fresh grout.

Figure 6 indicates that grouts can be successfully fabricated using mine water as mixing water. Specimens were made using 31 percent acidic water from the Longridge mine. The strengths of such specimens were essentially the same as strengths of specimens made using tap water both in the short term and over about a six-month period.

Figure 7 strength versus age plots are shown for specimens fabricated from ashes collected from the MEA plant at various times. Evident here is that ashes differ in cementitious characteristics.

Figure 8 portrays strength/age plots for specimens whose solids component were fly ashes only. Such mixes generally require more water than mixes containing bottom ash; additionally, strengths of fly ash only mixes varies with production time. Two mixes were fabricated with small (three or six percent) amounts of Portland cement. Such mixes were not as strong as a mix containing less water. The cement containing mixes required a relatively high water content to achieve the specified initial slump.

In Figure 9 are shown results of flexure tests. Flexural strengths vary directly with age and with

composition. Flexural strength of grouts is relatively low which reflects the brittle behavior associated with cement bound materials.

Figure 10 portrays slump as a function of water content for a mix consisting of a 50-50 blend of bottom ash and fly ash. As anticipated, slump varies directly with water content. Water contents of about 33 to 35 percent of total weight of specimen are required to achieve target slumps of ten to twelve inches for mixes containing both bottom ash and fly ash only.

In Figures 11 and 12 are shown slump versus age or time after mixing for various grouts. As anticipated, slump decreases with time after mixing. Noteworthy is that significant slump (about three inches) remained for most mixes after two hours had elapsed.

Figure 13 contains plots of compressive strength versus age for specimens whose solids content were 50/50 bottom ash/fly ash and with 31 percent water contents. Specimens were made with tap water. Some specimens were simply covered to prevent water loss until tested while other specimens were submerged in acidic mine water until tested. No differences were noted between the submerged and unsubmerged specimens.

Compared in Figures 14 and 15 are results of slump and flow tests for grouts containing tap water, high-range water reducer, an air entraining admixture, and the clay material, bentonite. The flow used here employed a cylindrical container, four inches in diameter and six inches high. The container was filled with fresh grout and then moved vertically upward allowing the grout to flow outward. The diameter of the sample after flow ceased was the flow. This test was believed to provide a better measure of grout flowability than the slump test. As indicated in the plots, the water reducing and air entraining admixtures did enhance flow characteristics slightly. Of note was that mixes containing bentonite and water reducer exhibited both enhanced flow and cohesion compared to the other mixes. Cohesion is the property of a mix to "stick together" and not experience particle segregation.

Illustrated in the last three plots, Figures 16, 17, and 18 is the swelling tendency of mixes with various ingredients as a function of age. The plots indicate that the grouts have little propensity to change volume with increasing age.

Conclusions

Based on results discussed above and observations made in the laboratory, the following tentative conclusions appear to be warranted:

1. The MEA ashes, products of modern coal-burning and gas cleaning technologies, are self-hardening materials. The ashes will form a hard, coherent, concrete-like substance when mixed with water. Setting and hardening times range from several tens of minutes (initial stiffening and significant loss of initial slump) to a few days (material capable of supporting small load), respectively. Setting and hardening times are functions of water content and composition of the

ashes which is variable. The coarse ash (bottom ash) has particle size distributions which are similar to coarse sands while the fine ash (fly ash) resembles silty soils.

2. Most of the blends of MEA ashes employed in the laboratory have consisted of 50 percent bottom ash and 50 percent fly ash as percent of total weight of ash. These proportions were chosen because they represent the average production quantities from the MEA plant.

3. Compressive strengths of blends of MEA bottom ash and fly ash are functions of water content, time, ash composition and type and quantity of admixture present, if any. Flexural strengths of such mixtures are probably affected by similar variables in similar ways.

4. Mixture water contents, as percent of total weight of sample, to achieve ten to twelve inch slump range from about 31 percent to about 43 percent for "straight" mixes (no admixtures) and mixes containing bentonite, respectively.

5. Mixes containing three to five percent bentonite seem to exhibit superior flow and cohesion characteristics when compared to straight mixes. Compressive strengths of bentonite mixes are lower than strengths of straight mixes at similar ages. However, attainment of 500 psi unconfined compressive strength over short (30 days or less) time periods appears feasible with properly constituted bentonite mixes. Five hundred psi was chosen as a benchmark compressive strength based on previous experience with strength levels needed to ameliorate subsidence problems in shallow underground cavities.

6. Incorporation of conventional air entraining agents into grouts at moderate to high dose rates seems to offer little advantage in enhanced consistency when compared to straight mixtures. High-range water reducers may enhance consistency to a useful degree; however, such admixtures are expensive and would be required in large dose rates. Utilization of a high-range water reducer appears not to be cost-effective at this time.

7. It seems feasible to employ mine pool water as mixing water. Strength appears to be unaffected when mine pool water is used as mixing water. In addition, mixtures will harden when submerged in mine pool water.

8. Consistency (slump) of mixes decreases with time. However, the time rate at which mixtures stiffen appears to be moderate. Consequently, mixes should remain pumpable in the field for sufficient time to allow proper installation of the grouts in underground cavities.

9. Grouts made with MEA ash experience little change in volume with time.

10. At the present time, the following "recipe" is recommended for disposal of MEA ashes in underground cavities: the ash component should consist of about 50 percent each of bottom ash and fly ash. The water content should be approximately 43 percent of the total weight of mixture with adjustments made on site to yield a grout with ten to twelve inch slump. Three to five percent of the total weight of the mixture should be bentonite. These proportions were employed in a recent

field trial which demonstrated that such grouts could be successfully mixed and that they exhibited satisfactory flow and cohesion characteristics. It must be emphasized that these proportions are tentative; final proportions will be established based on results of current flow characterization test and test of other admixtures.

Figure 1 Grain Size Distribution of MEA Bottom Ash.

Figure 2 Grain Size Distribution of MEA Fly Ash.

—□— 7 Days + 14 Days * 21 Days ■ 28 Days × 90 Days

Figure 3 Compressive Strength vs. Water Content at Various Specimen Ages.

Figure 4 Compressive Strength vs. Age of 38% Tap, 4.6% Bentonite Specimens.

Figure 5 Compressive Strength vs. Age of Various Lot 5 FA, Lot 5 BA Specimens.

Figure 6 Compressive Strength vs. Age of 31% Tap & AMD Specimens.

Figure 7 Compressive Strength vs. Age of Various 31% Tap Specimens.

Figure 8 Compressive Strength vs. Age of Various Fly Ash Only Specimens.

Figure 9 Flexural Strength vs. Age of Specimens.

Figure 10 Slump vs. Water Content of Specimens.

Figure 11 Slump vs. Age of Specimens.

Figure 12 Slump vs. Age of Specimens.

Figure 13 Compressive Strength vs. Age of Normal & AMD Submerged Specimens.

Figure 14 Comparison of Slump and Flow Tests.

Lot 5 FA, Lot 5 BA

Figure 15 Comparison of Slump and Flow Tests.

Figure 16 Swell vs. Age of 31% Tap Specimens.

Figure 17 Swell vs. Age of 31% AMD Specimens.

Figure 18 Swell vs. Age of 38% Tap, 4.6% Bentonite Specimens

TASK 2 GROUT RHEOLOGY MODEL

Purpose

Throughout much of Appalachia, abandoned coal mines have caused subsidence damage by collapse, and environmental damage by acid mine drainage into local watersheds. In addition, the local use of coal combustion for generating electricity has led to a large quantity of ash waste which is currently being disposed of in landfills. It has been proposed that these environmental problems may be eliminated by injecting a grout material, made from the ash waste product, into the abandoned mines. The grout would prevent subsidence by filling the empty mines, would reduce acid mine drainage by preventing groundwater and oxygen from coming into contact with exposed minerals and by neutralizing acid that is produced, and would provide an environmentally safe location for disposal of the coal combustion waste ash. The grout, a mixture of fly ash, bottom ash, water, and possibly other additives, would be injected into the underground mines through bore holes that would be drilled from the surface. Such a project would be economically feasible if the cost of disposing of the ash waste as a grout is less than the cost of the present practice (landfill disposal). Drilling the bore holes is the costliest part of this project, so to minimize project cost, the required number of bore holes must be minimized. To this end, a reliable computer model is needed to predict how the grout will flow in the subsurface environment, and to predict the total number and location of bore holes that will be required for successful placement of the grout in the mines. Once a suitable prediction method is developed, an efficient grout placement strategy can be developed.

The grout that is investigated in this project is produced from the ash waste of the Morgantown Energy Associates-Beechurst Avenue power plant. The MEA-Beechurst power plant uses an atmospheric pressure fluidized bed reactor for the combustion of coal and gob, and the waste that is generated is called FBC-ash. Gob, a combination of rejected rock, waste coal, and other impurities from the mining process, cannot be used in all coal combustion power plants. Limestone is added to the coal-gob mixture during combustion to reduce sulfur emissions. The FBC-ash particles have a higher calcium content and a more irregular shape than ash generated by traditional combustion processes. The grout mixtures made from the FBC-ash are similar in appearance to other flyash slurries, cement mortar, and concrete. All of these fluids exhibit non-Newtonian behavior. The behavior of these fluids has been described by other investigators with the power-law, Bingham, or Herschel-Bulkley fluid models.

The computer model used for this project is an adaptation of the commercially available code, PHOENICS. PHOENICS is a computational fluid dynamics code which has the capability to simulate three dimensional, unsteady, multi-phase flows with a free surface. However, user generated subroutines were needed to describe the non-Newtonian flow behavior exhibited by the grout mixtures. Rheological testing of the grout suggests that a Herschel-Bulkley fluid model may be used to describe the shear stress-shear rate relationship in the grout mixtures. The Herschel-Bulkley fluid model has been encoded in a subroutine added to the PHOENICS code. The parameter values obtained by laboratory testing of the grout can be used in the PHOENICS

computer model to correctly simulate the physics of flowing grout. Numerical simulations have been compared to analytical solutions and bench scale laboratory experiments. The computer model is intended as a design tool to make decisions about the mine filling process.

Background

The FBC-ash grout is a suspension of fly ash, bottom ash, and other additives in water. The flow and deformation of suspensions may be drastically different than that of Newtonian fluids as shown by a great deal of previous research.

RHEOLOGY OF SUSPENSIONS

When dealing with suspensions of solid particles, it is sometimes possible to treat the suspension as a homogeneous mixture by ignoring the particle-fluid interaction and ascribe effective fluid properties to the suspension. *Darby* (1984) stated that suspensions containing greater than 30-40% solids concentration by volume may be considered homogeneous and subject to description by the laws of continuum mechanics. *Jeffery and Acrivos* (1976) have described when suspensions may be treated as homogeneous fluids and have identified some of the factors that are responsible for non-Newtonian behavior. *Jeffery and Acrivos* further noted that even when suspensions may be considered homogeneous, it is not sufficient to simply modify the viscosity of the suspending medium as a function of the solid particle concentration. *Gupta* (1994) states that for all but very small volume fractions, suspensions will exhibit non-Newtonian behavior, and something more complicated than a viscosity value will be required to adequately describe the suspension behavior. Both *Jeffery and Acrivos* and *Gupta* have described the fundamental origin of the non-Newtonian behavior as a competition between hydrodynamic and non-hydrodynamic forces. Depending upon their size, shape, and concentration, particles in suspension may be subjected to a variety of non-hydrodynamic forces such as thermal (Brownian) forces, electrical forces due to surface charges, London-van der Waals forces, and hydrodynamic (i.e. viscous) forces from shearing at the particle-fluid boundary. As a consequence, the behavior of the suspension will often depend not only upon the concentration of particles but also on the particle size and aspect ratio, size distribution, the strength of the non-hydrodynamic forces, and the strength of the viscous forces. Fortunately, when it is valid to treat the suspension as a homogeneous fluid, the underlying reasons for the non-Newtonian behavior can be ignored, and attention can be focused on the macroscopic behavior of the homogeneous fluid. The macroscopic description of flow and deformation is therefore usually not based on theoretical considerations, but rather on empirical observations of behavior. Such observations are the foundation of the field of rheology: the science of deformation and flow.

Patel (1984) provides a useful survey of rheology. In the field of rheology, a fluid is defined as any material that deforms continuously for some range of applied shearing stress. Although the classical definition of a fluid is a material that exhibits continuous deformation under *any* applied shearing stress, there are materials that deform continuously only after a "yield" stress is exceeded. Below the yield stress, the material behaves like a solid. Such materials are still treated as fluids in

the field of rheology. The macroscopic description of a fluid's behavior is summarized by a constitutive law which relates the stress in a material to the rate of strain experienced by the fluid. It is therefore the primary goal of rheology to identify the constitutive law that is appropriate for a given fluid material.

Examples of constitutive laws may be found in any textbook on continuum mechanics, see for example *Mase and Mase* (1992). It is desired to find a constitutive law that can be used to describe the state of stress in a fluid by relating the stress tensor to the rate of deformation tensor. The state of stress is given by the stress tensor which can be decomposed as follows:

$$\sigma_{ij} = -p\delta_{ij} + \tau_{ij}$$

where σ_{ij} is the total stress in the fluid, p is the pressure, δ_{ij} is the Kronecker Delta, and τ_{ij} is the viscous stress tensor. The viscous stress tensor is a function of the gradients of fluid motion and, for a fluid without a yield stress, must be equal to zero when the fluid is at rest or in solid body motion. For a yield stress fluid, there may be a non-zero stress residual when the fluid is at rest because the fluid behaves like a solid until the yield stress is exceeded. For yield stress and non-yield stress fluids, the function that describes the stress tensor is called the constitutive law. It is usually assumed that the viscous stress tensor is related to the fluid motion through some function involving the rate of deformation tensor,

$$\tau_{ij} = f_{ij}(D_{kl})$$

where D_{ij} is the rate of deformation tensor defined by the velocity gradients:

$$D_{ij} = \frac{1}{2} \left(\frac{\partial v_i}{\partial x_j} + \frac{\partial v_j}{\partial x_i} \right)$$

The rate of deformation tensor is symmetric, thus $D_{12}=D_{21}$. If the function is linear, the fluid is called a Newtonian fluid and the relationship may be expressed as:

$$\tau_{ij} = C_{ijkl}D_{kl}$$

With the assumption that the fluid is isotropic, the fourth order tensor C_{ijkl} is an isotropic tensor. Because of the symmetry properties of D_{ij} and the isotropic nature of C_{ijkl} , the 81 components of C_{ijkl} may be reduced to two. The viscous stress tensor appropriate for an isotropic Newtonian fluid is:

$$\tau_{ij} = 2\mu D_{ij} + \lambda D_{kk}\delta_{ij}$$

The two constants, μ and λ , are properties of the fluid. μ is the dynamic viscosity and λ is the dilatational or second coefficient of viscosity. The compressibility of most liquids is negligible, so the trace of the rate of deformation rate tensor is zero. Consequently the last term in the above equation is usually neglected, leaving:

$$\tau_{ij} = 2\mu D_{ij}$$

With this relation, the constitutive equation for an incompressible Newtonian fluid may be expressed as:

$$\sigma_{ij} = -p\delta_{ij} + 2\mu D_{ij}$$

The Newtonian model described by this constitutive law is valid for many fluids, including water. The behavior of such materials may be described by the single parameter, μ , the dynamic viscosity. Each component of the viscous stress tensor, τ_{ij} , may be seen to be proportional to the

corresponding component of the rate of deformation tensor. The dynamic viscosity, μ , is the constant of proportionality, for instance:

$$\mu = \tau_{12}/2D_{12}$$

There are, however, many non-Newtonian fluids whose flow behavior cannot be adequately described by one constant. There exists a large class of materials whose constitutive relations fall under the class of generalized Newtonian fluids. For this class of materials, an *apparent* viscosity, η , is defined by analogy with the constant Newtonian viscosity and is used in place of the constant μ . The apparent viscosity is assumed to be a function of the magnitude of the rate of deformation. *Crochet and Walters* (1983) give the stress tensor for a generalized Newtonian fluid as:

$$\sigma_{ij} = -p\delta_{ij} + 2\eta(I_2)D_{ij}$$

where I_2 is a suitable form of the second invariant of the rate of deformation tensor, D_{ij} . Different forms of the function η describe different fluid behavior. *Patel and Govier and Aziz* (1972) list many of the common forms of the function η .

Relationships between a component of the stress tensor and the corresponding component of the rate of deformation tensor may be displayed graphically in stress vs. strain rate plots (see Figure 19). In such a plot, the slope of the curve is the viscosity. For Newtonian fluids, the line passes through the origin and the slope is constant. Pseudoplastics are fluid materials whose viscosity decreases with increasing rate of strain. Dilatant materials exhibit an increase in viscosity with increasing rate of strain. Dilatant and pseudoplastic behavior are often modeled with the power-law model. The power-law model is a two parameter model that relates the viscous stress to the magnitude of the rate of deformation rate tensor. In two dimensions, with one non-zero component ($D_{12} = D_{21} = \gamma/2$) of the rate of deformation tensor, the power-law model is:

$$\tau = k\gamma^n$$

where k is the consistency index, n is the behavior index, and τ here represents the only non-zero component ($\tau_{12} = \tau_{21} = \tau$) of the viscous stress tensor. For $n > 1$, the behavior is dilatant, for $n < 1$ the behavior is pseudoplastic, and for $n = 1$, the power-law model reduces to the Newtonian model. In general, there will be more than one non-zero component of the viscous stress tensor and the rate of deformation tensor. *O'Donovan and Tanner* (1983) give the appropriate form of the generalized Newtonian constitutive relation in three dimensions. The symbol, γ , then represents the magnitude of the strain rate, and is interpreted as the square root of the second invariant of the rate of deformation tensor. γ is given by:

$$\gamma = [2(D_{ij}D_{ij})]^{1/2}$$

The expansion of the second invariant is:

$$2D_{ij}D_{ij} = 2(D_{11}^2 + D_{22}^2 + D_{33}^2) + 4(D_{12}^2 + D_{23}^2 + D_{13}^2)$$

By analogy with the Newtonian definition of viscosity, the apparent viscosity is defined by using the power-law model as follows:

$$\eta = \frac{\tau}{\gamma} = \frac{k\gamma^n}{\gamma} = k\gamma^{(n-1)}$$

The apparent viscosity can be interpreted as the slope of a chord from the origin to a point on the

Figure 19 Stress vs. Rate of Strain

plot of stress vs. strain rate, as shown in Figure 20. The general three dimensional stress tensor for a fluid whose behavior may be described with the power-law constitutive relation is:

$$\sigma_{ij} = -p\delta_{ij} + 2(k\dot{\gamma}^{n-1})D_{ij}$$

For those fluid models that include a yield stress, such as the Bingham and Herschel-Bulkley models, the stress vs. strain rate curve does not go through the origin (see Figure 19). The intercept on the stress axis denotes a stress value that must be exceeded in order to initiate continuous fluid deformation. The Bingham fluid model describes a fluid behavior that is linearly dependent on the strain rate once the yield stress has been exceeded.

The two dimensional form of the Bingham constitutive relation is given by:

$$\begin{aligned} \tau &= \tau_y + k\dot{\gamma} & \text{for } \tau > \tau_y \\ \dot{\gamma} &= 0 & \text{for } \tau < \tau_y \end{aligned}$$

where k is the plastic viscosity and τ_y is the yield stress. As the yield stress approaches zero, the Bingham model will also reduce to the Newtonian model.

The Herschel-Bulkley (HB) fluid model uses the power-law model to describe fluid behavior after the yield stress is exceeded. In two dimensions, the viscous stress is given by:

$$\begin{aligned} \tau &= \tau_y + k\dot{\gamma}^n & \text{for } \tau > \tau_y \\ \dot{\gamma} &= 0 & \text{for } \tau < \tau_y \end{aligned}$$

An apparent viscosity can also be defined for the yield stress fluids. For the Herschel-Bulkley model, the apparent viscosity is defined as follows:

$$\eta = \frac{\tau}{\dot{\gamma}} = \frac{\tau_y}{\dot{\gamma}} + k\dot{\gamma}^{(n-1)} \quad \text{for } \tau > \tau_y$$

The appropriate extension of the Herschel-Bulkley model to three dimensions is:

$$\sigma_{ij} = -p\delta_{ij} + \left(\frac{\tau_y}{\dot{\gamma}} + k\dot{\gamma}^{n-1}\right)\left(\frac{\partial v_i}{\partial x_j} + \frac{\partial v_j}{\partial x_i}\right) \quad \text{for } \tau > \tau_y$$

where again the symbol, $\dot{\gamma}$ represents the magnitude of the rate of deformation tensor. Notice that the Herschel-Bulkley constitutive law will reduce to the power-law model when $\tau_y \rightarrow 0$, to the Bingham model when $n \rightarrow 1$, and to the Newtonian model when $\tau_y \rightarrow 0$ and $n \rightarrow 1$. It is

Figure 20 Apparent Viscosity

important to emphasize that the above constitutive laws are empirical descriptions of observed fluid behavior. This means that it may be possible that more than one model of fluid behavior could be used to describe how a particular fluid behaves depending on the range of shear stresses. The power-law, Bingham, and Herschel-Bulkley models have all been used to describe the behavior of suspensions. Other fluid models such as the Casson and the Prandtl-Eyring fluid models have also been used in attempts to describe such materials. *Bunn, Chambers, and Goh* (1991) have measured the rheological properties of fly ash slurries with solids concentration greater than 60% by weight. They showed that both the power-law and Bingham models could be used to fit data obtained with a tube viscometer. *Panda, Parida, Murty and Mitra* (1994) used the Bingham model to describe the behavior of a slurry of coarse and fine coal particles. *Håkansson, Hässler, and Stille* (1992) showed that the Bingham model was also able to describe flow of cement grouts. *Sakuta, Yamane, Kasami, and Sakamoto* (1979), and *Mori and Tanigawa* (1992) also used the Bingham model to characterize the flow of fresh concrete. *Hanks and Hanks* (1984) fit the Herschel-Bulkley model to experimental data for slurries of coal particles.

Although the Bingham and Herschel-Bulkley rheological models are widely used, some controversy appears in the literature as to the validity of the yield stress concept. *Barnes and Walters* (1985) have argued that if measurements can be made accurately enough, then continuous, but very slow, deformation will be observed at even the smallest of applied stresses. They contend that previous experimenters have not carefully evaluated the fluid behavior at low shear stresses, but have merely extrapolated a curve through the stress axis on a plot of experimentally obtained stress vs rate of strain data. They claim that a yield stress only defines what cannot be measured under the conditions of experimentation. Further controversy arises when considering the

existence of unyielded regions within a flow domain. O'Donovan and Tanner investigated the location of a yield surface within a parallel-plate plastometer and discuss the disagreement in the literature. They point out that the existence of an unyielded region, although required for a yield stress fluid, is kinematically impossible. Therefore, while yield stress models are a useful empiricism, care should be given when declaring a yield stress value for a fluid.

UNSTABLE SUSPENSIONS

The fluid models described above are based on the assumption that the fluid is homogeneous, and as such, are not adequate to describe the behavior of settling (unstable) suspensions in which the constitutive relation may change with position. Lombardi (1985) investigated the rheology of unstable grouts. As the solid particle concentration increases, friction will develop between the solid particles that are in contact. This friction is an additional resistance to flow and must be included in the rheological model of the grout. Lombardi modified the two dimensional Bingham fluid model to account for internal Coulomb friction in an unstable grout suspension:

$$\tau = \tau_y + \eta\dot{\gamma} + p\mu_f \quad \text{for } \tau > \tau_y$$

with p denoting the fluid pressure, and μ_f is the Coulomb friction coefficient. The friction coefficient is the tangent of the angle of internal friction between the solid particles. The above rheological model is the same as the Bingham model except for the addition of the last term on the right side. Similarly, the Herschel-Bulkley model could be modified to account for internal friction with the inclusion of the same extra term. The extra term shows that an additional stress is present that is proportional to the local fluid pressure and the tangent of the internal friction angle. This implies that the shear stress will differ throughout the fluid because the pressure will vary throughout the flow domain. An additional complication arises because the internal friction is a function of solid particle concentration which varies in space and time. The yield stress and plastic viscosity are also dependent upon solid particle concentration, and as such, will not remain constant. The result is a constitutive law that is a function of time, space, and flow strength. It is more illustrative to write the two dimensional constitutive law of an unstable suspension as an explicit function of space, x , time, t , and flow strength as measured by the magnitude of the rate of deformation, $\dot{\gamma}$:

$$\tau(x,t,\dot{\gamma}) = \tau_y(x,t) + \eta(x,t)\dot{\gamma} + p(x)\mu_f(x,t)$$

Thus, the rheological model needed to describe unstable suspensions is much more complex than the corresponding constitutive relation for stable suspensions.

Any yield stress fluid that is pumped, with a finite initial pressure head, into a closed, infinitely long channel will inevitably come to rest. As the pressure head is converted into velocity head, the loss of head due to friction will cause the stress in the fluid to continually decrease downstream. At some distance downstream, the stress in the fluid will fall below the yield stress and a solid plug will completely fill the channel resulting in cessation of flow. The Coulomb friction present in unstable suspensions will further limit the extent of flow. Lombardi compared the distances that

a stable and unstable grout could be made to flow in idealized, two-dimensional, bounded channels. Equations that describe the pressure drop and maximum flow distance were derived from a balance of forces on a fluid element that has stopped moving. For stable grouts, the maximum flow distance is a function of initial pressure, channel size, and yield stress. For unstable grouts, the maximum flow distance depends also on the coefficient of friction, and is much smaller than the maximum flow distance of stable grouts. Lombardi's results are not accurate because axisymmetric and two dimensional results were incorrectly combined.

THE WALL BOUNDARY CONDITION

Non-Newtonian fluids mechanics may also differ from Newtonian fluids mechanics by the inclusion of wall slip into the statement of the wall boundary condition. *Schowalter* (1988) used the term *slip* to describe behavior which violates the usual wall boundary condition of Newtonian fluid mechanics, which states that there must be no relative motion between an impenetrable boundary and a flowing fluid. Schowalter warns that the behavior of non-Newtonian fluids can be so unpredictable near a solid boundary that the statement of the boundary condition is often part of the solution being sought. Schowalter further claims that when dealing with complex fluids, continuum mechanics must be coupled to microscale physics because macroscopic effects can be caused by the microscopic physical and chemical nature of the boundary.

Some of the possible microscopic mechanisms responsible for slip are reviewed in *Silliman and Scriven* (1980). First, while not slipping locally on the microscopic irregularities of the actual surface, a fluid could appear to slip at a mathematical surface passed through the mean elevation of the solid surface roughness. Second, the normally submicron scale of intermolecular forces between the solid and fluid may locally reduce the apparent viscosity. Third, in fluids whose viscosity is highly dependent upon shear rate, apparent slip could occur in a thin layer of high shear near a solid surface. Fourth, the presence of solid boundaries may result in a migration of suspended solid particles away from the walls. As a result a thin low-viscosity fluid layer is formed near the wall in suspensions whose apparent viscosity is a function of solid particle concentration. The last two mechanisms create a "lubrication" layer that enhances flow without causing true slip. Fifth, "true" slip may occur at the molecular level, implying that there may be an appreciable departure from thermodynamic equilibrium within a few molecular diameters. Layers of molecules may slide past each other in a coordinated manner. Finally, apparent slip may occur in suspensions due to contact between the suspended solid phase and the solid boundary surface. While the velocity of the suspending fluid may still be zero at a solid wall, the solid particles in the suspension may slide on the wall. A combination of any of these mechanisms may generate a macroscopic appearance of slip at the wall.

The most easily observed macroscopic effect of wall slip are flow rates that are higher, for a given pressure, than predicted assuming no slip. *Barkat, Shaughnessy, and Clark* (1988), *Lawal, Kalyon, and Yilmazer* (1993), *Yilmazer and Kalyon* (1992), *Yoshimura and Prud'homme* (1988), and *Cohen and Metzner* (1985) have all reported apparent slip of suspensions in capillary and torsional viscometers. These researchers have concluded that the formation of a thin lubrication

layer is the cause of the apparent slip. It has been shown that above a critical shear stress, a plug of nearly undeformed fluid can be made to flow through a capillary due to slippage at the wall. As a consequence of the variation of slip layer thickness compared to capillary diameter, experimental pressure vs. flowrate data from different diameters of capillary or torsional viscometers will not overlap. An attempt to empirically quantify the magnitude of the wall slip is given by Navier's slip condition:

$$\beta u_s = \eta \frac{\partial u}{\partial n}$$

where β is Navier's formal slip constant, η is the apparent viscosity, u_s is the slip velocity, and n is the normal to the surface into the fluid. Barkat, Shaughnessy, and Clark have shown that particle migration does occur in pipe flows of cement slurries, and have measured the parameter, β , for cement slurries.

Methodology

PHOENICS

The computer simulations were performed using a PC version of the commercially available PHOENICS software. The computer had an Intel Pentium cpu operating at 90 MHZ, with a 500 MB hard disk, 16 MB RAM, and the MS-DOS operating system. PHOENICS is a general purpose Computational Fluid Dynamics (CFD) code developed by *Ludwig, Qin, and Spalding* (1989). Figure 21 illustrates the organization of the program. The flow parameters, domain, grid, and boundary conditions are specified using PHOENICS Input Language (PIL) in the Q1 file. SATELLITE is a pre-processor that translates the user's specification of a flow simulation into a format that is understood by EARTH, the computational program that solves the conservation and transport equations. The user has access to the solution process at specific points via the GROUND subroutine, which is written in FORTRAN. A specific GROUND subroutine provided by the vendor, GREX3.FOR (Ground EXample 3), contains many of the most frequently needed capabilities. By modifying GREX3.FOR, the user can customize the solution method, implement various fluid models, change fluid properties, or solve for user defined variables. AUTO PLOT and PHOTON are two post-processors that provide graphical display of the simulation results. The numerical solution is performed by EARTH. The relevant equations are solved using the well known finite volume method of discretization and the SIMPLE algorithm for coupling the velocity and pressure.

Figure 21 The PHOENICS Structure

THE FINITE VOLUME METHOD

In order to obtain a numerical solution it is necessary to replace the continuous governing differential equations with discrete equations for the dependent variables at a finite number of locations within the calculation domain. The locations where the dependent variables are calculated are called grid points. The equations that relate the grid point values to each other are finite difference equations, and the process by which the finite difference equations are extracted from the differential equations is called discretization. The discretization method employed by PHOENICS is the finite volume method. The finite volume method is well explained by *Patankar* (1980). In the finite volume method, the calculation domain is discretized into a number of finite volumes or cells. The primary grid points are located at the geometric center of the cells, and secondary grid points are located at the centers of the cell faces. These are the locations where the values of the dependent variables are calculated according to the staggered grid convention. Figure 22 shows a typical finite volume discretization in two dimensions.

Figure 22 The Staggered Grid Arrangement

The finite difference equations are created from the differential equations by integrating the differential equations over the finite volume of a cell and, for transient problems, a finite time interval. The integrals are evaluated by making assumptions about how the dependent variables change between grid points. A significant advantage of the finite volume method is that it insures integral conservation of the dependent variables over every cell.

The authors of PHOENICS wished to create a general discretization scheme that could be applied to all of the governing differential equations. Therefore, all of the governing equations are cast into the following peculiar form:

$$\frac{\partial}{\partial t}(\rho\phi) + \nabla \cdot (\rho \bar{V}\phi - \Gamma \nabla \phi) = S_\phi$$

where ρ is the density, $\mathbf{V}$ is a velocity vector, Γ is the diffusion coefficient, S_ϕ is a source term, and ϕ may represent any conserved quantity such as enthalpy or a velocity component. The quantities Γ and S_ϕ have unique meanings for a particular meaning of ϕ . The first term on the left hand side

is called the transient term, the second term is the advection term, the third is the diffusion term, and the term on the right hand side is the source term. The continuity equation is obtained by setting $\phi=1$ thus obtaining:

$$\frac{\partial}{\partial t} \rho + \nabla \cdot (\rho \bar{V}) = S_\phi$$

When ϕ denotes the x-velocity component, the associated meanings of Γ and S are:

$$\Gamma = \eta_{app}$$

$$S_\phi = -\frac{\partial p}{\partial x}$$

where η_{app} is the apparent viscosity. With this notation, the general equation becomes the x-direction momentum equation:

$$\frac{\partial}{\partial t}(\rho u) + \nabla \cdot (\rho u \bar{V}) = -\frac{\partial p}{\partial x} + \frac{\partial}{\partial x}(\eta_{app} \frac{\partial u}{\partial x}) + \frac{\partial}{\partial y}(\eta_{app} \frac{\partial v}{\partial y})$$

The conservation and transport equations cannot be solved directly. They need to be discretized. The discretization of the general equation is accomplished by formally integrating over a control volume. For a steady flow, the transient term is zero and the integration over a control volume yields:

$$\int_{\delta V} \left[\frac{\partial}{\partial x}(\rho u \phi - \Gamma \frac{\partial \phi}{\partial x}) + \frac{\partial}{\partial y}(\rho v \phi - \Gamma \frac{\partial \phi}{\partial y}) \right] dV = \int_{\delta V} S_\phi dV$$

The above result is for two dimensions such where $dV=\delta x \delta y$, but all of the following results can be extended to three dimensional transient simulations by integrating over δx , δy , δz and a finite time step, δt . The restriction to two dimensions is only to simplify the explanation.

The left hand side may be converted to a surface integral by the application of Gauss' theorem:

$$\begin{aligned} \int_s \left[(\rho u \phi - \Gamma \frac{\partial \phi}{\partial x})_e - (\rho u \phi - \Gamma \frac{\partial \phi}{\partial x})_w \right] dy \\ + \int_s \left[(\rho v \phi - \Gamma \frac{\partial \phi}{\partial y})_n - (\rho v \phi - \Gamma \frac{\partial \phi}{\partial y})_s \right] dx = \int_{\delta V} S_\phi dV \end{aligned}$$

The integration limits, e, w, n, s refer to the east, west, north, and south cell faces as defined in Figure 22. To evaluate the integrals in the above equation, some assumptions must now be made about how ϕ varies between grid points. The first assumption is that the fluxes through the control volume faces can be expressed as the product of a mean flux at the cell face center times the cell face area. This enables the previous equation to be written as:

$$[(\rho u \phi - \Gamma \frac{\partial \phi}{\partial x})_e - (\rho u \phi - \Gamma \frac{\partial \phi}{\partial x})_w] \Delta y + [(\rho v \phi - \Gamma \frac{\partial \phi}{\partial y})_n - (\rho v \phi - \Gamma \frac{\partial \phi}{\partial y})_s] \Delta x = S_\phi \Delta V$$

The diffusive fluxes are then approximated as follows:

$$\begin{aligned} \left(\frac{\partial \phi}{\partial x} \right)_e &= \frac{\phi_E - \phi_P}{\Delta x_e} & \left(\frac{\partial \phi}{\partial x} \right)_w &= \frac{\phi_P - \phi_W}{\Delta x_w} \\ \left(\frac{\partial \phi}{\partial y} \right)_n &= \frac{\phi_N - \phi_P}{\Delta y_n} & \left(\frac{\partial \phi}{\partial y} \right)_s &= \frac{\phi_P - \phi_S}{\Delta y_s} \end{aligned}$$

This ultimately leads to central differencing being applied to the diffusion terms.

The advective fluxes; $\rho u \phi$, are discretized using either the second order exponential scheme (Patankar, 1980) or traditional first order upwinding. This combination is called the HYBRID scheme. The Peclet number

$$Pe = uL / \Gamma$$

is a ratio of the strengths of advection and diffusion. The Peclet number is evaluated for each cell.

When $|Pe| < 2$, diffusion dominates the flow and the HYBRID scheme uses exponential differencing for the advection terms. When $|Pe| > 2$, advection is strong and the HYBRID scheme reduces to upwind differencing of the advection terms. As a consequence, the discretization applied to the advective fluxes may vary through the computational domain. The solution order of a method is the rate at which errors decrease as the cell size decreases. A method whose solution error decreases linearly with the cell size is called a first order method, while a method whose solution error decreases like the square of the cell size is called a second order method, and so on. Therefore, the solution order will also vary from first to second order throughout the computational domain.

The source term, S_ϕ is assumed to be constant within a cell. Non-linear source terms must be linearized because the solution method used on the system of algebraic equations is for linear equations. Therefore, the discretized equations must be linear. Source terms are linearized as follows:

$$S_{\phi,p} = S^*_{\phi,p} + \left(\frac{\partial S_{\phi,p}}{\partial \phi_p} \right)^* (\phi_p - \phi_p^*)$$

where $S_{\phi,p}$ is the source value of ϕ at the p^{th} cell and the starred values are those from the previous iteration. The above equation is put into the following form:

$$S_{\phi,p} = S_c + S_p \phi_p$$

where:

$$S_c = S_{\phi,p}^* - \left(\frac{\partial S_{\phi,p}}{\partial \phi_p} \right)^* \phi_p^*$$

$$S_p = \left(\frac{\partial S_{\phi,p}}{\partial \phi_p} \right)^*$$

For unsteady simulations, all values are assumed to be constant within a cell and for each cell, the integral over a finite time step is replaced with:

$$\int \frac{\partial}{\partial t} (\rho \phi) dt = \frac{[(\rho \phi)_{\text{new}} - (\rho \phi)_{\text{old}}]}{t_{\text{new}} - t_{\text{old}}}$$

With the above approximations, a set of algebraic equations may be constructed relating the value of the dependent variable at the P^{th} grid point, ϕ_p , to the values of the dependent variable at the surrounding grid points and the value at the previous time step. The algebraic equations are of the following general form (in two dimensions):

$$a_p \phi_p = a_E \phi_E + a_W \phi_W + a_N \phi_N + a_S \phi_S + b_p$$

The 'a' coefficients are calculated for each cell using:

$$a_E = \left[-F_e, D_e - (F_e/2), 0 \right]$$

$$a_W = \left[F_w, D_w - (F_w/2), 0 \right]$$

$$a_N = \left[-F_n, D_n - (F_n/2), 0 \right]$$

$$a_S = \left[F_s, D_s - (F_s/2), 0 \right]$$

$$a_p = a_E + a_W + a_N + a_S + (F_e - F_w) + (F_n - F_s)$$

$$b_p = \text{contribution of source terms}$$

where:

$$F \equiv \rho \phi \quad \text{and} \quad D \equiv \Gamma / \delta x$$

The function $\llbracket x, y \rrbracket$ means the maximum of x and y , and is analogous to the AMAX1 function in FORTRAN. This set of equations is must be solved iteratively. Each meaning of ϕ has it's own set of equations for which a solution must be obtained.

The SIMPLE Method

A special method is required to solve for the above equations when ϕ represents a velocity

component. This is because of the indirect link between the velocity field and the pressure field. While the pressure gradients appear in the momentum equation source terms, there is no an explicit equation for solving the pressure field. As a consequence, the coupling of pressure and velocity is accomplished by using the Semi-Implicit Method for Pressure Linked Equations (SIMPLE) method described by Patankar (1980). The SIMPLE method is based upon the idea that the velocity field will satisfy the continuity equation only when the correct pressure field is used to solve the momentum equations. The staggered grid approach must be followed to appropriately represent the pressure gradient term in the momentum equations and the velocity gradient terms in the continuity equation. Use of the staggered grid arrangement avoids unrealistic checkerboard solutions and divergence. Pressure and velocity correction equations are used to adjust the pressure and velocity values during the iterative SIMPLE procedure as follows:

1. Guess the pressure field.
2. Solve the momentum equations.
3. Solve the pressure correction equations with the current velocity values.
4. Calculate new velocity values that satisfy continuity.
5. Solve for all other dependent variables.
6. Use the corrected pressure field as the next guess and return to step 2.

This iterative procedure is continued until pressure and velocity values are obtained that satisfy both the momentum equations and the continuity equation within a set tolerance.

CONVERGENCE CRITERIA

PHOENICS solves the flow equations using the algebraic equations and the SIMPLE method described above. The algebraic equations are implicit. Therefore, PHOENICS will "sweep" through the simulation domain many times, updating the dependent variables in each iteration. During three dimensional simulations, PHOENICS sweeps through the domain in a slab-wise fashion in the z-direction, solving for the dependent variables at each grid point in one x-y slab, then for each grid point in the next x-y slab, and so on. For transient problems, many sweeps are performed for each time step so that a converged solution is obtained for each time step. The sweeps through the domain continue until the specified maximum number of sweeps is reached, or the residuals fall below the convergence criteria. For this project, the convergence criteria was chosen to be 10^{-4} . During some simulations, it was not possible to make the residual fall below the convergence criteria, even though further iterations produced no change in the values of the dependent variable. Such solutions appeared to be converged, but no explanation can be given for the high residuals.

GRID REFINEMENT STUDY

A grid refinement study was performed to assess the accuracy and verify the solution order of a PHOENICS solution. In such a study, numerical solutions are computed for a particular flow using two or more grids of different grid spacing. By computing the difference between the

numerical solutions, their dependence upon grid spacing can be quantified. The grid refinement ratio, GR, is the ratio between the coarse grid spacing, Δx_c , and the fine grid spacing, Δx_f , as follows:

$$GR = \frac{\Delta x_c}{\Delta x_f}$$

The relative difference, ϵ , between ϕ_f , the fine grid solution, and ϕ_c , the coarse grid solution, is expressed as:

$$\epsilon = \frac{(\phi_f - \phi_c)}{\phi_f}$$

This quantity by itself is not a good estimate of the error in the solution because it does not take into account the grid refinement ratio, GR, and the solution order, p , of the numerical method. For instance, with a very small GR, the two solutions would be very close and ϵ may falsely indicate that the fine grid solution is very close to the converged solution. Roache (1993) proposes the use of a Grid Convergence Index (GCI) that includes both GR and p . Roache states that within the CFD community, there is confidence that the ϵ obtained by a grid doubling with a second order method (ie: $GR=p=2$) is indicative of the solution error if both solutions are within the asymptotic range. Thus, the idea behind the GCI is to relate the ϵ obtained from any grid refinement study with arbitrary GR and p , to the ϵ that would be obtained with the same base grid but with GR and $p=2$. The GCI defined as:

$$GCI = 3 \frac{\epsilon}{(GR^p - 1)}$$

where ϵ , GR, and p are the values obtained from the actual grid refinement study. If the solution order is unknown, it can be found with solutions from three grids as follows:

$$p = \frac{\ln\left(\frac{\phi_1 - \phi_2}{\phi_2 - \phi_3}\right)}{\ln(GR)}$$

The three solutions, ϕ_1 , ϕ_2 , and ϕ_3 , are obtained from two grid reductions with the same GR such that:

$$GR = \Delta x_1 / \Delta x_2 = \Delta x_2 / \Delta x_3$$

This definition of the GCI is only valid for solutions that are within the asymptotic range. Determination of whether or not the asymptotic range has been achieved can be accomplished as

follows. If an exact solution is known for the simulated flow, then an Error Indicator, E , is defined as:

$$E = \frac{\phi_{exact} - \phi_{numerical}}{\Delta x^p}$$

and is monitored as the grid spacing, Δx , is decreased. If E remains constant the solutions are within the asymptotic range. If an exact solution is not known, then the numerical solution must be calculated with three different grid spacings. From these two GCI values are computed. If the solutions are within the asymptotic range, the following relationship will be true for GCI_{12} (the GCI found using solutions ϕ_1 and ϕ_2) and for GCI_{23} (the GCI found using solutions ϕ_2 and ϕ_3):

$$GCI_{12} = (GR)^p \cdot GCI_{23}$$

The ϕ values used to compute the GCI may represent any of the solution variables or any function of the solution variables. For example, ϕ may represent individual velocity values at a specific location or a flow rate obtained by integrating a velocity profile. The only restriction is that the comparison must be made at the same physical location within the domain. Thus care must be taken when choosing GR to insure that each grid contains nodes for at least a few of the same points in physical space.

The grid refinement study was performed with results from a steady, axi-symmetric pipe flow simulation. The pipe flow was used in the grid refinement study because the exact solution for this flow is well known, even for fluids that exhibit non-Newtonian behavior. The simulated pipe flow domain and boundary conditions were as follows:

Dimensions:

radius = 1.0 m
length = 25 m

Boundary Conditions:

uniform entrance velocity = 2 m/s
exit pressure = 0.0 Pa
no slip condition at the pipe wall

Fluid Properties (Newtonian):

density = 1.0 kg/m³
dynamic viscosity = 0.1 N·s/m² (SAE 30 oil @ 38°C)

These parameters resulted in a Reynolds number of $Re = 20$, which guarantees laminar flow. The entrance length was found to be 3 meters using an empirical relationship in *White* (1991). Simulations of this flow were performed with three different grids: 3x12, 9x36, and 27x108 cells in the radial and downstream directions. In this grid refinement study, both refinement ratios are 3. PHOENICS grid points are cell centered, thus the first grid point is only a half grid spacing from

the boundary and a grid refinement ratio of three was necessary to insure that nodes existed in the coarse and fine grids which represented the same points in physical space. The results from the three PHOENICS simulations and the exact solution are shown in Figure 23. The velocity and flowrate data were used to determine the solution order and the rate at which grid refinement leads to a converged solution. Figure 24 shows root-mean-square (RMS) error vs. grid spacing for the velocity profiles.

Figure 23 Comparison of PHOENICS and Exact Solutions.

Figure 24 RMS error vs. Grid spacing.

The solution order, determined using velocity values at several locations, was found to vary across the pipe radius within the entrance length from $p=1.0$ at $r=0.167$ (near the pipe centerline), to $p=1.5$ at $r=0.500$, and $p=1.85$ at $r=0.833$ (near the pipe wall). The solution order is nearly first order where convection terms dominate the flow (near the centerline), and closer to second order where viscous terms dominate the flow (near the pipe wall). This behavior is attributed to the hybrid discretizing employed by PHOENICS. The solution order variation found within the entrance region is consistent with this explanation and is within the theoretical limits for the solution order of this method. In the fully developed region, the solution was second order for all positions across the pipe. This is expected because the convective terms are negligible when the flow is fully developed. For these flow conditions, the HYBRID scheme is functioning as a second order central differencing scheme. The fully developed velocity profiles were integrated using Simpson's rule and compared to the flowrate specified at the inlet boundary. The error indicator between the computed and exact flowrate decreased as a second order method such that:

$$E = \frac{Q_{exact} - Q_{numerical}}{\Delta x^2} = constant$$

The same result was obtained when flowrate values and discrete velocity values were used in the above equation.

The results of this grid refinement study clearly indicate that the solution is second order within the fully developed region, and is between first and second order throughout the rest of the domain. While the GCI does not provide information about the overall solution, the computed GCI can be used as an estimate of how close the fine grid solution is to the converged solution. Using the method of Roache (1993), the GCI was computed with data from the flowrate values obtained by integrating the fully developed velocity profiles. The GCI computed for the finest grid using $GR=3$ and $p=2$ was 0.002346. This indicates that the fine grid solution is within 0.23% of the converged solution. The GCI was computed at a location within the entrance region using velocity values and was found to be higher than the GCI in the fully developed region. Within the entrance region, the GCI was found to vary from 0.024 near the wall (where $p=2$) to 0.118 near the centerline (where $p=1$). This implies that the finest solution at this sampling location within the entrance length may be expected to lie within 2.4% of the converged solution near the pipe wall, and within 11.8% of the converged solution near the centerline. The solution was more converged in the developed region than in the developing region. This verifies that the HYBRID discretizing scheme is functioning as a second order method in the developed region, but as a lower order method within the entrance length.

TRACKING A FREE SURFACE

PHOENICS has two built in methods for tracking a moving interface between two fluid phases during a transient flow simulation. The two methods are the Scalar Equation Method (SEM), and the Height Of Liquid method (HOL). Both are coded within the GREX3.FOR subroutine, and may be "turned on" by making the appropriate setting within the Q1 file. Both the SEM and HOL

methods treat the flow as one phase, i.e. two sets of velocity and pressure are not calculated. Rather, the methods permit fluid properties such as density and viscosity to vary throughout the domain to represent the spatial distribution of the two phases. A definite interface must exist, therefore dispersed phase simulations are not possible with the SEM and HOL methods. There are advantages and disadvantages to each model, but for flow simulations where both models are applicable, they yield the same result.

SCALAR EQUATION METHOD

The SEM simulates the transient convection of a scalar variable throughout the simulation domain. The scalar variable is given a value of unity in fluid A, and a value of zero in fluid B. Use of this kind of marker variable is analogous to seeding a flow domain with a dye (fluid A is purple, fluid B is clear), except that the numerical dye is transported only by convection. The convection of the scalar variable, ϕ , is governed by:

$$\frac{\partial \phi}{\partial t} + u \frac{\partial \phi}{\partial x} + v \frac{\partial \phi}{\partial y} + w \frac{\partial \phi}{\partial z} = 0$$

The solution of the flow is obtained by solving the governing differential equations in conjunction with the transport equation for the scalar marker variable. The simulation is a one phase simulation of a fluid whose properties vary throughout the domain. The value of the scalar variable at each grid point will establish the fluid properties for that control volume. For control volumes with intermediate values of the marker variable, fluid properties are assumed to vary linearly. The variation of the density, ρ , and viscosity, μ , are:

$$\begin{aligned}\rho &= \rho_a + (\rho_b - \rho_a)\phi \\ \mu &= \mu_a + (\mu_b - \mu_a)\phi\end{aligned}$$

Upwind discretization of the above transport equation creates unphysical smearing of the discontinuity at the interface. The upwind scheme used in PHOENICS evaluates the flux at a cell face by using the information at the cell that lies "upwind" of the cell to be evaluated. For instance, to evaluate the advective flux at the east face of cell P:

$$\text{for } u_e > 0 ; \phi_e = \phi_P \qquad \text{for } u_e < 0 ; \phi_e = \phi_E$$

and for the west face:

$$\text{for } u_w > 0 ; \quad \phi_w = \phi_w \qquad \text{for } u_w < 0 ; \quad \phi_w = \phi_p$$

The smearing associated with the above discretization scheme is combated with the discretization scheme proposed by *Van Leer* (1981). The Van Leer approach is an explicit scheme which modifies the first order upwind formulation using characteristics. The flux at the east face is modified as follows:

$$\begin{aligned} \text{for } u_e > 0 : \quad \phi_e &= \phi_p + \left(\frac{\partial \phi}{\partial x} \right)_p (dx - u dt) \\ \text{for } u_e < 0 : \quad \phi_e &= \phi_E + \left(\frac{\partial \phi}{\partial x} \right)_E (dx + u dt) \end{aligned}$$

The gradient of ϕ is based upon the values at the start of the time step, thus making the scheme explicit. Therefore, the Courant criterion places a restriction upon the size of a time step to avoid instability. Physically the time step restriction prevents the interface from moving through a full cell during a time step. The following limit must be satisfied for every cell in the computational domain:

$$dt < \min \left(\frac{dx}{u}, \frac{dy}{v}, \frac{dz}{w} \right)$$

The Van Leer scheme is incorporated into the SEM subroutine in the GREX3.FOR file. The SEM has been used successfully by *Hirt and Nichols* (1981) and *Chan, Pericleous, and Cross* (1991). Both have interpreted a surface of $\phi=0.5$ in the computational solution as the location of the fluid interface. This interpretation of the surface location was verified by measuring the area under the $\phi=0.5$ curve for a simple two dimensional test case that will be discussed later.

HEIGHT OF LIQUID METHOD

The HOL method determines the location of a fluid interface by evaluating the mass of liquid that exists in a vertical column of cells. An "up" direction must be specified along which only one intersection of the interface is permitted. Flows with overturning surfaces can not be simulated using the HOL method. A volume fraction of liquid, ϕ , is defined for each cell. The inflows and outflows of fluid to each column of cells in the up direction are calculated from the convective mass fluxes multiplied by the upwind values of ϕ . The net flux of fluid mass into a column of cells is used to calculate the total mass within the column. Since the cell volumes are known, the height of liquid within a cell can be calculated by dividing the total mass by the density and cell volume. Cells below the interface have a volume fraction of unity (filled with fluid A), and cells above the interface have a volume fraction of zero (filled with fluid B). The cell where the interface occurs is given an intermediate volume fraction defined by:

$$\begin{aligned} \phi &= (\text{Total Mass} - \text{Lower Mass}) / (\rho_a * \text{cell volume}) \\ \text{Total Mass} &= \text{Old mass in column} + \text{inflow} - \text{outflow} \\ \text{Lower Mass} &= \text{Mass below the interface cell} \end{aligned}$$

Fluid properties will have an intermediate value at the interface cell and are defined by the same linear variation as used in the SEM. A surface of $\phi=0.5$ is interpreted as the interface of the two fluids and was verified with a simple test case that will be discussed below. The HOL method does not generate numerical smearing of the interface, nor does it have a restriction on time step. The coding for the HOL method is included in the GREX3.FOR file.

DAM BREAK SIMULATION

The following "dam break" simulation was used to evaluate the two surface tracking methods in PHOENICS. An imaginary restraining wall is instantaneously removed and the volume of initially static fluid contained behind it is permitted to slump under the influence of gravity. The simulation domain is 1.0 m in the x-direction and 0.3 m in the vertical y-direction. Initially, the fluid occupies a 0.2 m wide by 0.275 m high area at the left end of the domain. The simulation begins just after the wall has been removed. The simulation is transient and two dimensional. The grid has 50 uniform cells in the x-direction and 20 non-uniform cells in the y-direction expanding exponentially at a rate of 1.4 in the positive y-direction. The Herschel-Bulkley fluid model was used with the following parameters: $n = 0.8$, $k = 1.0 \text{ (N}\cdot\text{s}^n/\text{m}^2)$, $\tau_y = 5 \text{ Pa}$. The time step was 0.001 s and 200 iterations were specified per time step. A 0.5 second simulation required 5 hours and 9 minutes of cpu time when the SEM was used. The same simulation has been run using the SEM and the HOL method.

At each time step, the surface of $\phi = 0.5$ is used to mark the location of the fluid interface as shown in Figure 25. To check the validity of this as a true indicator of the free surface, the area under each of the curves was computed by numerical integration using Simpson's rule. The area under the $\phi = 0.5$ curve at each time step was the same as the initial area occupied by the fluid. Thus, the $\phi = 0.5$ curve is a good representation of the free surface location.

The results predicted by the SEM and the HOL method agree. The smoothness of the free surface predicted by the HOL method is more susceptible to grid coarseness. A coarse grid may produce a free surface that has steps coinciding with the grid columns. This is because the HOL method "fills up" each columns of cells. When the grid is finer, the HOL method produces free surface curves that are as smooth as those produced by the SEM. To achieve realistic results with the SEM, small times steps are needed. If the Courant criterion is exceeded for any cell in the domain, large errors occur that spread quickly throughout the domain, ultimately leading to divergence. This can be a problem when the velocity magnitudes cannot be predicted before the simulation is run because the size of

Figure 25 Dam-Break Simulation Results Using SEM

the time steps must be set before the run. PHOENICS does not have the ability to change time step duration once the simulation is started. Thus, both methods have a dependency on the discretization of the continuum. The HOL method requires a fine spatial grid while the SEM requires fine temporal discretization. With the appropriate grids, both methods yield the same results.

A second test case to evaluate the SEM and HOL methods used a flow that is similar to the mine filling flow. Fluid is introduced into the domain by a fixed mass flux boundary condition through the "roof" of the axisymmetric domain shown in Figure 26. Gravity is in the negative y-direction. The SEM is able to satisfactorily simulate the flow, but the HOL method cannot be used in such a simulation. The reason is that no correct "up" direction can be used. For each of the four possible choices, $\pm x$ -direction and $\pm y$ -direction, unrealistic results are produced. The reason for the unrealistic results is that the HOL method always "fills up" columns of cells from the bottom based upon the specified "up" direction. For this simulation (and the mine filling simulation) some of the columns of cells are actually filled from the "top". Mine filling cannot be represented using the HOL method. Therefore, the SEM must be used in the mine filling simulations.

Figure 26 Axisymmetric Simulation Domain

SAMPLE Q1 FILE

This section presents a typical Q1 file which is used to specify the flow simulation parameters for PHOENICS. The Q1 file is an ASCII file that PHOENICS SATELLITE reads and interprets for use by EARTH. There are 24 groups in a Q1 file, some or all of which may contain information. The groups only serve to assist the user in organizing the simulation data. Commands may be placed anywhere within the file, but subsequent commands supersede earlier commands. SATELLITE will only recognize commands that begin in the first or second space; all other commands are interpreted as comment lines. The meaning of all the PIL commands and their associated groups may be found in the CHAM manual TR200a (Ludwig, et.al.). The Q1 file shown below contains all the information that must be submitted to SATELLITE so that EARTH will simulate the "dam break" problem described above using the SEM method to track the interface. The command that activates the SEM subroutine is SURF=T. If the HOL method was used, the appropriate subroutine would be activated by replacing SURF=T with HOL=T. The other settings that are necessary for both methods to function properly are described in the PHOENICS documentation.

```
TALK=T;RUN( 1, 1);VDU=VGAMOUSE
```

```
GROUP 1. Run title and other preliminaries
TEXT(DAM BREAK ; H-B FLUID ; NO Friction
TITLE
```

```
GROUP 2-5. Grid specification
STEADY=F
GRDPWR(T,500,0.5,1.0)
GRDPWR(X,50,1.0,1.0)
GRDPWR(Y,20,0.30,1.4)
```

```
GROUP 7. Variables stored, solved & named
STORE(DEN1,PRPS,ENUL,GENK);SOLVE(VFOL,SURN,btau)
SOLUTN(P1,Y,Y,Y,N,N,N)
SOLUTN(U1,Y,Y,N,N,N,N)
SOLUTN(V1,Y,Y,N,N,N,N)
```

```
GROUP 8. Terms (in differential equations) & devices
** activate the "gas-and-liquid algorithm", ie volumetric
continuity equation, and allow convection fluxes to be
modified in GROUND
GALA=T
TERMS(VFOL,N,N,N,N,P,P)
TERMS(SURN,N,N,N,N,P,P)
GENK=T
SURF=T
```

```
GROUP 9. Properties of the medium (or media)
RHO1=GRND10 ; ENUL=grnd6
enula=0.00083 ; enulb=0.0042 ; enulc=0.8
```

```
GROUP 11. Initialization of variable or porosity fields
FIINIT(P1)=0.0;FIINIT(V1)=0.0;FIINIT(SURN)=0.0
FIINIT(DEN1)=1.2;INIADD=F
** place the initial "pile" of liquid of which the slumping
is to be simulated
PATCH(LIQUID,INIVAL,1,10,1,ny-1,1,nz,1,1)
COVAL(LIQUID,SURN,ZERO,1.0)
COVAL(LIQUID,DEN1,ZERO,1200)
```



```

GROUP 13. Boundary conditions and special sources
** try an actual open top boundary
OUTLET(TOPOUT,NORTH,1,NX,1,NY,NZ,NZ,1,LSTEP)
** provide for the gravity-force source of w1
PATCH(GRAV,PHASEM,1,NX,1,NY,1,NZ,1,LSTEP)
COVAL(GRAV,v1,FIXFLU,-9.81)
** time patch for transient term
PATCH(TCON,CELL,1,NX,1,NY,1,NZ,1,LSTEP)
COVAL(TCON,SURN,GRND,GRND)

GROUP 15. Termination of sweeps
LSWEEP=500
LITER(SURN)=1

GROUP 16. Termination of iterations
RESREF(P1)=1.E-4 ; RESREF(V1)=1.E-4 ; RESREF(u1)=1.E-4

GROUP 17. Under-relaxation devices
relax(p1,linrlx,0.3)
RELAX(u1,FALSDT,0.0001);RELAX(v1,FALSDT,0.00008)

GROUP 19. Data communicated by satellite to GROUND
IPRPSA=68 ; IPRPSB=0 ; IDISPA=10 ; CSG1=y
VARMIN(SURN)=0.0 ; VARMAX(SURN)=1.0
RLOLIM=0.4 ; RUPLIM=0.6

GROUP 22. Spot-value print-out
TSTSWP=50 ; ECHO=F ; IYMON=NY/2 ; IZMON=NZ/2 ; IXMON=NX/2

GROUP 23. Field print-out and plot control
** provide contour-plot output for EXCELL manipulation
PATCH(tbosurn,CONTUR,1,1,1,ny,1,NZ,1,LSTEP)
PLOT(tbosurn,SURN,ZERO,10)
OUTPUT(P1,Y,n,n,n,Y,Y);OUTPUT(u1,Y,n,n,n,Y,Y)
OUTPUT(V1,Y,n,n,n,Y,Y);OUTPUT(SURN,Y,N,N,N,N,N)
OUTPUT(DEN1,Y,N,N,N,N,N);OUTPUT(vf01,y,N,N,N,N,n)
OUTPUT(btau,y,N,N,N,N,N)
STOP

```

MODIFICATIONS TO THE PHOENICS CODE

SUBROUTINE FOR HERSCHEL-BULKLEY FLUID MODEL

The capability to simulate power-law and Bingham fluid behavior is already contained within PHOENICS. However, preliminary rheological testing indicated that the Herschel-Bulkley fluid model would be needed to describe the flow behavior of the FBC-ash grout. It was therefore desired to implement a procedure within PHOENICS to simulate flows of Herschel-Bulkley fluids. The Herschel-Bulkley fluid model in two dimensions is as follows:

$$\tau = \tau_y + k \dot{\gamma}^n \quad \text{for } \tau > \tau_y$$

The apparent viscosity that corresponds to the Herschel-Bulkley constitutive law is:

$$\eta_{app} = \frac{\tau_y + k\gamma^n}{\gamma} = \tau_y \gamma^{-1} + k\gamma^{n-1} \quad \text{for } \tau > \tau_y$$

The behavior expressed by the above constitutive law is included in a flow simulation by evaluating the apparent viscosity at every cell and using that value as the viscosity when solving the momentum equations. The form of the apparent viscosity equation is very similar to that of the power-law and Bingham fluid models which are already included in a PHOENICS subroutine called GXPROPS.FOR. The subroutine GXPROPS.FOR is called by the GREX3.FOR subroutine when the laminar viscosity, ENUL, is set equal to a "ground flag" rather than a viscosity value in the Q1 file. A ground flag is a PHOENICS device for calling a specific subroutine. Ground flags may be used for calling a wide range of subroutines for many of the flow simulation parameters that need to be specified in the Q1 file. The PHOENICS manuals CHAM tr200a and CHAM tr200b (Ludwig, et.al.) provide in-depth discussions of the ways to use the GREX3.FOR subroutine to call other subroutines.

These viscosity settings in the Q1 file invoke the following fluid models and parameters.

ENUL = GRND4 → Power-Law Apparent Viscosity Calculated

ENULA = k

ENULB = n

ENUL = GRND5 → Bingham Apparent Viscosity Calculated

ENULA = k

ENULB = τ_y

The ground flags (e.g. GRND4, GRND5, etc.) specify unique subroutines. The subroutines that calculate an apparent viscosity according to the power-law and Bingham fluid models were used as patterns for developing a subroutine to calculate an apparent viscosity value based upon the Herschel-Bulkley model.

The existing GXPROP.FOR file was copied to GXPROP2.FOR, and the GREX3.FOR file was copied to GREX4.FOR. The original files are in the \D_PHOE20\D_EARTH directory. All modifications were made to the two copied files in the \D_PHOE20\D_JOHN directory, thus leaving the original files unmodified. The new subroutine was given a ground flag of GRND6. It appears in the GXPROP2.FOR file immediately following the GRND5 code. The FORTRAN variables used in the new subroutine are:

VISL = η_{app}

LGEN1 = γ^2

ENULA = k

ENULB = τ_y

$$ENULC = n$$

The symbol, γ , denotes the magnitude of the rate of strain. In PHOENICS, the rate of strain is the square root of the variable LGEN1. The calculation of apparent viscosity with these variables is obtained with the following:

$$VISL = ENULA * LGEN1^{0.5 * (ENULC - 1)} + ENULB * LGEN1^{-0.5}$$

This is broken up into four steps:

1. $EXPN = 0.5 * (ENULC - 1)$
2. $VISL = ENULA$
3. $VISL = VISL * LGEN1^{EXPN}$
4. $VISL = VISL + ENULB * LGEN1^{-0.5}$

Using built in PHOENICS function statements, these steps become:

1. $EXPN = 0.5 * (ENULC - 1)$
2. $CALL FN1(VISL, ENULA)$
3. $CALL FN37(VISL, LGEN1, EXPN)$
4. $CALL FN35(VISL, LGEN1, ENULB, -0.5)$

These function statements are explained in appendix 4 of CHAM tr200b.

Printout of stress is enabled by computing the stress as BTAU according to the actual constitutive law. The following three steps compute the stress magnitude for each cell in the domain.

1. $EXPN = 0.5 * (ENULC - 1)$
2. $CALL FN1(VISL, ENULB)$
3. $CALL FN35(VISL, LGEN1, ENULA, EXPN)$

The GXPROP2.FOR file is partially listed below. Modifications to the file are in bold.

```

C!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!!
C**** SUBROUTINE GXENUL is called from section 6, group 9 of
GREX3,
C    and is entered only when ENUL is set to GRND options.
C
C.... The arguments VISL, and TEMP1 are integer names used in the
C    SATELLITE, indicating which whole-field store will be used
C    for the laminar kinematic viscosity and the temperature of
C    phase 1; ENUL is the laminar kinematic viscosity, and set to
C    GRND (n) .

```

.

.

.

REMOVED FOR BREVITY OF PRESENTATION

```

C.... ENUL=GRND4 selects laminar kinematic viscosity as a linear
C      function of the strain rate to the power 0.5*(ENULB-1), as
C      is
C      appropriate for a power-law non-Newtonian fluid.
C.... ENUL=GRND5 selects Bingham laminar kinematic viscosity.
C.... ENUL=GRND6 selects Herschel-Bulkley viscosity law.
C      NOTE that ENUL or VISL must be stored in Q1. This is checked
C      for in GREX.
      ELSEIF (GRNDNO(4, ENUL) .OR. GRNDNO(5, ENUL) .OR. GRNDNO(6, ENUL))
THEN
      IF (ISWEEP.EQ.FSWEEP) THEN
        IF (QEQ(FIINIT(VISL), READFI)) THEN
C.... On first sweep of restart, copy viscosity from internal 2D
C      store KDUMM to prevent 'kick'
          CALL FN0(VISL, -KDUMM)
        ELSE
C.... On sweep 1, set VISL to ENULA, as no sensible strain rate
C      available
          CALL FN1(VISL, ENULA)
        ENDIF
      ELSE
C.... ENUL=GRND4: power-law laminar kinematic viscosity
        IF (GRNDNO(4, ENUL)) THEN
C      FN1(Y,A) Y = A ; FN37(Y,X,A,B) Y = Y * A*X**B
          CALL FN1(VISL, ENULA)
          IF (NEZ(ENULB) .AND. ONE(ENULB, 1.0)) THEN
            CALL FN22(LGEN1, TINY)
            CALL FN37(VISL, LGEN1, 0.5*(ENULB-1.))
          ENDIF
        ELSEIF (GRNDNO(5, ENUL)) THEN
C.... ENUL=GRND5: Bingham laminar kinematic viscosity
          CALL FN22(LGEN1, TINY)
C      Option to printout square-root of the second invariant of
C      deviatoric stress tensor, i.e. tau = enulb+enula*sqrt(lgen1)
C      where enulb=yield stress & enula=Bingham plastic viscosity
          IF (LBNAME('BTAU').NE.0) THEN
            CALL FN1(VISL, ENULB)
            CALL FN35(VISL, LGEN1, ENULA, 0.5)
            CALL FN0(LBNAME('BTAU'), VISL)
          ENDIF
C      Compute kinematic viscosity, visl =
[enula+enulb/sqrt(lgen1)]/rho
C      FN(Y,A) Y=A ; FN35(Y,X,A,B) Y=Y+A*X**B
          CALL FN1(VISL, ENULA)
          CALL FN35(VISL, LGEN1, ENULB, -0.5)
        **
        ** Modification on 16-May-1995 to simulate H-B fluid behavior
        **
      ELSEIF (GRNDNO(6, ENUL)) THEN

```

```

      CALL FN22(LGEN1,TINY)
      EXPN=0.5*(ENULC - 1.0)
C      Option to printout square-root of the second invariant of
C      deviatoric stress tensor, i.e. tau = enulb+enula*sqrt(lgen1)
C      where enulb=yield stress & enula=Bingham plastic viscosity
      IF(LBNAME('BTAU').NE.0) THEN
        CALL FN1(VISL,ENULB)
        CALL FN35(VISL,LGEN1,ENULA,EXPN)
        CALL FN0(LBNAME('BTAU'),VISL)
      ENDIF
C      Compute kinematic viscosity from HB fluid model
C      FN1(Y,A) Y=A ; FN35(Y,X,A,B) Y=Y+A*X**B ; FN37(Y,X,A)
Y=Y*X**A
      CALL FN1(VISL,ENULA)
      CALL FN37(VISL,LGEN1,EXPN)
      CALL FN35(VISL,LGEN1,ENULB,-0.50)
    ENDIF

**
** End of modification from 16-May-1995
** Original grnd6 routine has been removed
**

      ENDIF

**

```

A modification was also required in the GREX3.FOR file to include storage for LGEN1 and the velocity gradients needed for calculating LGEN1 when GRND6 is called from the Q1 file. A logic statement on line 172 is modified to include GRND6. The following is an excerpt from the GREX4.FOR file that contains the modification, indicated in bold face.

```

C-----LGEN1
C.... Provision of storage for square-of-velocity-gradient
C      expressions for viscous-dissipation or turbulence-
C      energy source. First ensure that strain-rate is calculated
C      for ENUL=GRND4 and GRND5.
C
      GENK=GENK.OR.((GRNDNO(4,ENUL).OR.GRNDNO(5,ENUL).OR.GRNDNO(6,ENUL))

```

The modification activates a call on line 216 which is also modified as follows:

```

C-----EASP5 TO EASP18
C
      IF(GRNDNO(4,ENUT).OR.GENK.OR.DUDX.OR.DVDX.OR.DWDX.OR.
1      DUDY.OR.DVDY.OR.DWDY.OR.DUDZ.OR.DVDZ.OR.DWDZ) THEN

```

Once the two files were modified, a "private" version of the executable, EARTH, needed to be compiled. This was accomplished by updating the BLDEAR.BAT batch file in the \D_PHOE20\D_UTILS directory. By including the path and file name of the new subroutines to be included, a new EARTH file was "built" in the \D_PHOE20\D_JOHN directory by running the BLDEAR.BAT batch file. The FORTRAN compiler used to

compile and assemble the various subroutines was the Salford Software FTN77 compiler version 2.74. When the user tries to run EARTH, a choice of "public" or "private" is now given to permit selection of the EARTH version that is desired. The public version is the original commercial version of PHOENICS, the private version contains the modifications discussed above.

FULLY DEVELOPED PIPE FLOW

The exact solution is known for the fully developed velocity profile and flowrate of a Herschel-Bulkley fluid in a pipe. A pipe flow simulation was performed to evaluate the accuracy of the new subroutine. The velocity profile and flow rate are functions of fluid properties, pipe radius, and pressure gradient, and are given by the following equations.

r = radial coordinate

R = tube radius

r_{plug} = radius of the unsheared plug

$\Delta P/\Delta L$ = pressure gradient

For $R > r > r_{\text{plug}}$, the velocity profile is:

$$u = \left(\frac{1}{k}\right)^{\frac{1}{n}} \left(\frac{-2\Delta L}{\Delta P} \right) \left(\frac{n}{n+1} \right) \cdot \left[\left(-\frac{r\Delta P}{2\Delta L} - \tau_y \right)^{\frac{n+1}{n}} - \left(-\frac{R\Delta P}{2\Delta L} - \tau_y \right)^{\frac{n+1}{n}} \right]$$

When $r < r_{\text{plug}}$ the fluid moves as a plug. The plug velocity is obtained by replacing r with r_{plug} in the above equation. The plug radius is given by:

$$r_{\text{plug}} = \frac{2 \tau_y \Delta L}{\Delta P}$$

The flowrate is:

$$Q = \frac{\pi R^3 n}{k^{(1/n)}} \left(\frac{R\Delta P}{2\Delta L} \right)^{-3} \left(\frac{R\Delta P}{2\Delta L} - \tau_y \right)^{\frac{n+1}{n}} \cdot \left[\frac{\left(\frac{R\Delta P}{2\Delta L} - \tau_y \right)^2}{1+3n} + \frac{2\tau_y \left(\frac{R\Delta P}{2\Delta L} - \tau_y \right)}{1+2n} + \frac{\tau_y^2}{1+n} \right]$$

A FORTRAN program named PIPEFLOW.FOR was written to compute the exact velocity profiles for flow of an HB fluid, Bingham fluid, power-law fluid, and Newtonian fluid for comparison to the numerical model.

The Q1 file that was used previously in the pipe flow grid refinement study was changed to call the HB subroutine. The pipe radius is 0.1m and the pipe length is 5 m. A uniform entrance velocity of 2.5 m/s is specified at the upstream end of the pipe. Thus, the specified velocity corresponds to a volume flowrate of 0.07854 m³/s. From the exact solution, the pressure gradient that would produce a flowrate of 0.07854 m³/s is 39525 Pa/m for fully developed flow. This is the pressure gradient used to compute the exact velocity profile. The pressure at the pipe exit was set to zero. A uniform grid with 9 divisions in the radial direction and 60 divisions in the axial direction was used in the numerical simulation.

Care must be taken to properly correlate the material property values specified in the PHOENICS simulation with those used in the corresponding analytical solution. All three non-Newtonian subroutines, power-law, Bingham, and Herschel-Bulkley, return a value for the kinematic viscosity. This means that the parameters used as input in the Q1 file must be interpreted as kinematic properties. To illustrate this, the four dummy variables DENS, VISC, EXP, and TAU_Y are used to relate the PHOENICS input with the values used in the exact solution. Note that VISC has different meanings for different fluid models.

	<u>Q1 File</u>	<u>Exact Solution</u>
Newtonian:	RHO1=DENS ENUL=VISC	$\rho = \text{DENS}$ $\mu = \text{DENS} * \text{VISC}$
Power-Law:	ENUL=GRND4 ENULA=VISC ENULB=EXP	$k = \text{DENS} * \text{VISC}$ $n = \text{EXP}$
Bingham:	ENUL=GRND5 ENULA=VISC ENULB=TAUY	$k = \text{DENS} * \text{VISC}$ $\tau_y = \text{DENS} * \text{TAUY}$
Herschel-Bulkley:	ENUL=GRND6 ENULA=VISC ENULB=TAUY ENULC=EXP	$k = \text{DENS} * \text{VISC}$ $\tau_y = \text{DENS} * \text{TAUY}$ $n = \text{EXP}$

If the above correlation between the PHOENICS input and the exact solution is not made, the numerical and exact solutions will not agree.

In the axisymmetric pipe flow simulation of a Herschel-Bulkley fluid, the following fluid properties were specified:

$$\begin{aligned}\rho &= 1200 \text{ kg/m}^3 \\ k &= 12 \text{ N}\cdot\text{s}^n/\text{m}^2 \\ n &= 0.8\end{aligned}$$

$$\tau_y = 1200 \text{ Pa}$$

The appropriate variables in the Q1 file are:

```
DENS=1200.
VISC=0.01
EXPN=0.8
TAUY=1.0
```

Following is the Q1 file used to simulate the flow described above of a HB fluid in a pipe:

```
TALK=T;RUN( 1, 1);VDU=VGAMOUSE
TEXT(non-Newtonian Pipe Flow
REAL(LENGTH,RAD,WIN,REYNO,VISC,DENS,EXPN,TAUY,flsfact,falsev,fals
ew)

RAD=0.10 ; LENGTH=5.0
NY=9 ; NZ=60

DENS=1000.0 ; VISC=0.01
TAUY=1.0 ; EXPN=0.80
WIN=2.5

GROUP 2-5. Grid specification
STEADY = T
CARTES = F
GRDPWR(Y,NY,RAD,1.0)
GRDPWR(Z,NZ,LENGTH,1.0)

GROUP 7. Variables stored, solved & named
SOLVE(P1,V1,W1) ; SOLUTN(P1,Y,Y,Y,N,N,N)
STORE(ENUL,BTAU,GENK,lgen1)
genk=t

GROUP 9. Properties of the medium (or media)
RHO1=DENS
ENUL=grnd6
enula=visc ; enulb=tauy ; enulc=expn

Group 11. Initial Values
INIADD=F
fiinit(w1)=win ; fiinit(v1)=0.0

GROUP 13. Boundary conditions and special sources
Inlet
INLET(IN,LOW,1,1,1,NY,1,1,1,1)
VALUE(IN,P1,WIN*RHO1)
VALUE(IN,W1,WIN)
Outlet
PATCH(OUTLET,HIGH,1,1,1,NY,NZ,NZ,1,1)
COVAL(OUTLET,P1,1000,0.0)
```



```

      North wall
      WALL(PSIDE,NORTH,1,1,NY,NY,1,NZ,1,1)

      GROUP 15. Termination of sweeps
      LSWEET=5000
      RESREF(P1)=1.E-4;RESREF(V1)=1.E-4;RESREF(W1)=1.E-4

      GROUP 17. Under-relaxation devices
      varmax(v1)=0.00
      RELAX(P1,LINRLX,0.6)
      flsfact = (length/nz)/(1.5*win) ; flsfact
      falsev = 0.01*flsfact ; falsev
      falsew = 0.1*flsfact ; falsew
      RELAX(V1,FALSDT,falsev) ; RELAX(W1,FALSDT,falsew)

      GROUP 22. Spot-value print-out
      IYMON=ny/2 ; IZMON=3*nz/4
      TSTSWP=-1

      GROUP 23. Field print-out and plot control
      NXPRIN=1 ; NYPRIN=ny/5 ; NZPRIN=nz/10
      OUTPUT(P1,N,N,N,N,Y,N)
      OUTPUT(V1,N,N,N,N,Y,N)
      OUTPUT(W1,N,N,N,N,Y,N)
      OUTPUT(ENUL,N,N,N,N,N,N)
      OUTPUT(GENK,N,N,N,N,N,N)
      OUTPUT(BTAU,N,N,N,N,N,N)
      OUTPUT(lgen1,N,N,N,N,N,N)
      patch(tbop1,contur,1,1,ny/2,(ny/2)+1,1,nz,1,1)

      plot(tbop1,p1,0.0,10)
      patch(tboend,contur,1,1,1,ny,nz-6,nz-2,1,1)
      plot(tboend,w1,0.0,10)
      plot(tboend,genk,0.0,10)
      plot(tboend,enul,0.0,10)
      ECHO=F
      STOP

```

Figure 27a shows that the velocity profile calculated with the Herschel-Bulkley subroutine agreed very well with the exact solution. The RMS error between the exact and numerical velocity profile was 1.601%. The numerical velocity profile was integrated using Simpson's rule to obtain a flowrate of 0.07898 m³/s. The relative error between the exact and numerical flow rate is 0.555%. Although the velocity profile agrees very well with the exact solution, the relative error between the exact and numerical pressure gradient is large. The pressure gradient predicted by PHOENICS was 45857 Pa/m, but the exact pressure gradient is 39525 Pa/m. The relative error between the PHOENICS and exact pressure gradient is 13.8% for the 9x60 grid solution. The large discrepancy has been attributed to insufficient grid resolution. To verify this, solutions were also obtained with a 6x40 grid and a 3x20 grid. Figure 27b shows the relative error decreasing as the as the

number of grid point increases. The 9x60 grid is sufficient to demonstrate that the Herschel-Bulkley subroutine is functioning properly, but an even more accurate solution would be obtained with a more refined grid.

Figure 27a Numerical and Exact Solution of Herschel-Bulkley Velocity Profile

Figure 27b Relative Error in Calculated Pressure Gradient vs. Grid Spacing

Results

UNSTABLE GROUT

It has become apparent from rheological testing, that the FBC-ash grouts under investigation are unstable unless suitable additives are used. The concentration of solid particles varies with time and space because of gravity settling, implying that the grout becomes inhomogeneous. As the concentration of particles increases at the bottom of the grout mixture, contact develops between the solid particles. This permits Coulomb friction to act as an additional resistance to flow of the grout which must be included in the rheological constitutive law that is used to define the grout mixture. The Herschel-Bulkley model for two dimensional parallel shear flow with the effect of Coulomb friction included is:

$$\tau = \tau_y + k(\dot{\gamma})^n + P\mu_f$$

When there is no flow (i.e. $\dot{\gamma}=0$) the residual shear stress may be as large as:

$$\tau = \tau_y + P\mu_f$$

The following analysis uses this as a starting point for determining the maximum distance that an unstable grout may be pumped.

RELEVANCE TO PROJECT

The project goal is to fill an underground mine with FBC-ash grout using as few drilled holes as possible, because drilling the injection holes is the costliest part of the project. As a consequence, the distance underground that the grout may be made to flow is of direct importance to the project outcome. It is anticipated that as grout is injected into the mine through the mine roof, the grout will form a conical pile that will eventually reach the mine roof. The pumping pressure will then expand the pile in a radial direction until the maximum spreading distance is reached as shown in Figure 28. The maximum flow distance of unstable grouts may be so limited that the project is rendered impractical due to the large number of injection wells needed will be cost prohibitive.

Figure 28 How Grout May Fill Mine Passage

PUMPING DISTANCE

A look at how stable and unstable grouts behave in an idealized channel provides a preliminary evaluation of the flow of the FBC-ash grouts in underground mine voids. The underground cavities are idealized as a two dimensional and an axisymmetric cavity of finite

Figure 29 Simplified Geometry Used in Deriving Maximum Flow Distance

thickness as shown in Figure 29. The floor and ceiling are considered but not the walls.

As discussed earlier, any yield stress fluid, stable or unstable, that is injected into a pipe or closed channel with a finite initial pressure head, will eventually stop moving. Inevitably there will be a location downstream where the stress in the grout will fall below the yield stress at all points across the cross section, a solid plug will completely fill the pipe or channel, and flow will cease. For an unstable grout, Coulomb friction will further limit the extent of flow. The hardening of the grout has been neglected because the time required for hardening is much greater than the time scale of the flow.

Lombardi compared the distances that stable and unstable grout can be made to flow in idealized bounded channels, but axisymmetric and two dimensional results were incorrectly combined. Lombardi's results have been corrected here to find the equations that describe the pressure gradient along a channel and the maximum flow distance. The equations have been derived from a balance of forces for a fluid that has stopped moving. When flow has stopped, the wall shear will be balanced by the internal stress in the grout. The internal shear stress in the grout is found from the constitutive relation given previously. All of the derived equations are presented below. It can be seen that the pressure gradient along the channel for both the two dimensional and axisymmetric cases is a function of initial pump pressure, channel thickness, and yield stress. For unstable grouts, the pressure gradient depends also on μ_f , the coefficient of friction.

The equations for pressure as a function of downstream distance and for the maximum flow distance are given below for stable (no Coulomb friction) and unstable (with Coulomb friction) grout in a two dimensional geometry and in an axisymmetric geometry.

The following variables appear in the maximum flow distance equations:

P_o = inlet pressure
 r_o = inlet pipe diameter
 h = $\frac{1}{2}$ channel thickness.
 d = maximum flow distance
 $\mu_f = \tan(\phi)$
 τ_y = yield stress
 $a = \mu_f/h$
 $b = -\tau_y/h$

1. Stable Grout, yield stress only:

2D Case

$$P = P_o - \frac{\tau_y}{h}(r - r_o)$$

$$d = \frac{P_o h}{\tau_y}$$

Axisymmetric Case

$$P = \frac{r_o}{r} P_o - \frac{\tau_y}{2h} \left(r - \frac{r_o^2}{r} \right)$$

$$d = \sqrt{r_o^2 + \frac{2h r_o P_o}{\tau_y}}$$

2. Unstable Grout, yield stress and Coulomb friction:

$$P = -\frac{\tau_y}{\mu_f} + \frac{1}{\mu_f} \frac{\text{2D Case}}{(\tau_y + \mu_f P_o)} \exp\left[\mu_f \frac{(r_o - r)}{h} \right]$$

$$d = \frac{h}{\mu_f} \ln\left[\frac{\tau_y + \mu_f P}{\tau_y + \mu_f P_o} \right]$$

Axisymmetric Case

$$P = \frac{b}{a} \left(1 - \frac{1}{ar}\right) - \frac{b r_o}{a r} \left(1 - \frac{1}{ar_o}\right) \exp[a(r_o - r)] + P_o \frac{r_o}{r} \exp[a(2r_o - r)]$$

A closed form solution for the maximum flow distance was not found for the unstable grout in axisymmetric coordinates. However, the maximum distance may be found by plotting the pressure, P , vs. the distance, r . The maximum flow distance corresponds to the value of r for which the pressure is zero.

PARAMETRIC STUDY

A parametric study was performed with the equations listed above to determine how variations in the channel height, $2h$, the pumping pressure, P_o , the yield stress, τ_y , and the friction coefficient, μ_f , affect the maximum pumping distance. Each of the parameters was varied over a wide range and the corresponding flow distance computed with the following constants: $P_o = 1$ MPa, $h = 0.75$ m, $\tau_y = 50$ Pa, and $\mu_f = 0.20$. The results are shown in Figures 30-39. It was found that the maximum pumping distance was least sensitive in changes to pumping pressure, and most sensitive to changes in the yield stress and friction coefficient. As can be seen from all the figures, the maximum pumping distance of an unstable grout is always less than that of stable grouts, usually substantially so.

Figure 30

Figure 31

Figure 32

Figure 33

Figure 34

Figure 35

Figure 36

Figure 37

Figure 38

Figure 39

ILLUSTRATIVE EXAMPLE

Grout is to be injected at the intersection of two mine corridors. The geometry of the mine corridors and coal pillars is idealized as shown in Figure 40. It is desired that the grout injected at the center of the intersection shall fill the mine corridors at least half the distance to the next intersection so that injection wells can be drilled only at the corridor intersections. The flow of the stable and unstable grout in the vicinity of the intersection is described by the axisymmetric flow equations given previously.

The following example uses the same data for calculating the stable and unstable flow distances:

$$P_o = 0.5 \text{ MPa}$$

$$r_o = 0.076 \text{ m}$$

$$2h = 1.524 \text{ m}$$

$$\tau_y = 100 \text{ Pa}$$

$$\mu_f = 0.5$$

Maximum Flow Distance:

$$\text{Stable Grout:} \quad r_{\max} = 22.3 \text{ m}$$

$$\text{Unstable Grout:} \quad r_{\max} = 2.89 \text{ m}$$

Thus the stable grout could be made to fill the desired volume, but the unstable grout would not even flow out of the intersection area. This example does not take into account the slope of the mine floor, debris and obstructions in the mine, and segregation of the water and solids due to impact of the grout on the mine bottom. In addition, the actual pumping pressure available at the injection site may be significantly greater than assumed here, and the material property values are uncertain.

Figure 40 Idealized Mine Intersection (Plan View)

IMPACT ON SIMULATIONS

It was intended that knowledge of the grout rheology could be incorporated into a computer model in order to obtain a numerical simulation of the injection process. PHOENICS can simulate the behavior of several non-Newtonian fluid models. However, these fluid models treat the grout as a one phase homogeneous fluid. The FBC-ash grouts that are unstable cannot be modeled as a single phase due to the inhomogeneities that arise from settling. In order to accurately simulate flows of such unstable suspensions, the fluid model and the computer simulation must take into account the interaction between the suspended phase and the liquid phase within the grout. To be accurate, this would entail simulating the movement of each solid particle within the grout mixture. Such a complex task is outside the capabilities of the PHOENICS software and is beyond the scope of this project. PHOENICS can only be used to simulate the flow of stable FBC-ash grouts.

STABILIZATION

It is clear that unstable grout mixtures are not desirable for practical application and are beyond the scope of the numerical model. Therefore, a method must be found to stabilize the grout mixtures. The addition of bentonite to the grout mixtures has been shown to stabilize the grout (Putnam, 1995). Preliminary rheological testing of the stabilized grout indicates that the power-law fluid model may represent the stabilized grout's flow behavior as well as the Herschel-Bulkley fluid model. The simulation results that will be presented make use of both the power-law fluid model and the Herschel-Bulkley fluid model to simulate flow of the stabilized grout.

ATTEMPT TO DETERMINE GROUT PROPERTIES WITH PHOENICS

The grout mixtures used in preliminary testing were all prepared with the same recipe. The initial grout samples were mixed with water fractions ranging from 31 to 37 percent. The water fraction is defined as the ratio of the weight of water to the total weight of the grout mixture. The solid material contained equal parts by weight of fly ash and bottom ash. The size of the fly ash particles ranged in size from 5 to 50 μm . The maximum particle size in the bottom ash is approximately 1 cm. No additives were used when preparing the initial grout mixtures.

The desire to simulate the flows of the FBC ash grouts has been frustrated by difficulties in fully characterizing the grout rheology. Several attempts were made to quantify rheological constants such as yield stress, consistency index, and flow behavior index, by performing well known viscometric flow experiments. Before additives were used, the ash solids would settle very quickly and the subsequent inhomogeneous mixture was not amenable to traditional rheological testing. In addition, the equipment used to perform the viscometric experiments, the cone and plate, parallel plate, and squeeze plate rheometers, are too small to permit large particles in the grout. While the grout to be used in the field

will contain large particles from the bottom ash, rheological testing can only be performed with grouts devoid of large particles. Therefore, it was suggested that the material properties could be obtained by reversing the traditional procedure of *first* defining the constitutive relation, quantifying the rheological constants with viscometric flows, and *then* using the constants as input in a numerical model for more complex flows. The proposed method was to match numerical simulations to experimental results. PHOENICS would be used to simulate an actual flow experiment. The fluid properties used as input in PHOENICS would be updated until the PHOENICS output closely matched the actual flow. The use of PHOENICS to determine the flow properties of the grout would generate confidence that PHOENICS simulations can accurately represent the grout behavior. Unfortunately this method was not successful.

A bench scale flow model of the "dam break" flow (Figure 25) was constructed of plexiglass and is described by *Putnam*, (1996). The bench scale model was 0.2 m in width, 0.3 m in height, and 1.0 m in length. Grout flow in the bench scale model was recorded with a video camera such that the flow as a function of time was clearly captured. The PHOENICS software was then used to simulate flow with the same geometry. Two and three dimensional models were used in conjunction with the Herschel-Bulkley subroutine. Attempts to match the experimental data were not successful. However, the inability to match the experimental data provided illuminating insight into some grout characteristics that were previously unrecognized.

First, the flow behavior (and thus also the grout properties) were different each time a grout sample was mixed. The grout samples were all prepared by combining equal parts by weight of fly ash, bottom ash, and water. The same grout recipe was used for all grout samples and the same procedure was used to prepare each grout sample. The physical flow test was performed many times, but flow time, flow distance, and shape of the free surface varied greatly. The variability has been attributed to variations in the solid material (ash) used to prepare the grout. Although all the ash material used in this project originated in the same power plant, the power plant does not discharge a uniform waste product. The power plant continually varies the amount of lime, coal, and gob in the combustion reactor to maximize efficiency and minimize pollution. In addition, larger bottom ash is extracted from the reactor and stored at a different rate than the finer fly ash. The trucks that are used to haul the waste material away are loaded with varying amounts of fly ash and bottom ash. Thus, within a given sample of ash, solid properties and size distribution may vary greatly. The bench scale flow test has demonstrated the sensitivity of flow to the variations in solid particle properties.

Second, it was discovered that several combinations of fluid properties produced the same or nearly the same results. In other words, a specific flow behavior could be independent of some of the fluid model constants. The time required to impact the far tank wall was the flow property that was used to initially evaluate simulation results. Comparisons were also made of velocity at impact and free surface shape. The Herschel-

Bulkley model was used when trying to match the experimental results. This model has three flow parameters, the yield stress, τ_y , the consistency, k , and the flow behavior index, n . It was found that within the range of properties used in the parametric study, $\tau_y = 1 - 50$ Pa, $k = 0.1 - 500$ (N·s/m²) ^{n} , and $n = 0.5 - 1.0$, the flow was most influenced by the k value. Varying k resulted in large changes in time required to impact the opposite wall and velocity at impact, but a small changes in free surface shape. The τ_y value did somewhat influence the shape of the interface, but its effect on the flow time was negligible, perhaps due to the small range of values over which it varied. An estimate of the yield stress was provided by measurements of the residual slope of the grout-air interface after flow had stopped. Thus, the actual grout yield stress is thought to lie within the range of values used in the numerical simulations. Variation of the n value had negligible effect on all the flow parameters evaluated. Very low n values generated only a slight increase in impact velocity. However, slight n variations do have a large impact on flow behavior in other geometries, such as the pipe flow geometry. As a consequence, there must be something about the flow in this geometry that negates the influence of n . The n value is responsible for the deviation from a linear stress vs. rate of strain curve. It is the exponent of the rate of strain magnitude. If the magnitude of the rate of strain did not vary greatly across a large portion of the domain, the effect of n would be small. Since most of the shearing occurred in a small region near the tank floor, it is conceivable that the effect of n could be limited to this small region. This is consistent with the slight increase in impact velocity because shear thinning would occur in this small region. The conclusion is that, whatever the mechanism, the flow in the dam break model is not suitable for reliably evaluating fluid constants.

Finally, observations of the actual grout flow indicated that there may be very large wall slip behavior. As soon as the grout material was placed behind the retaining wall (the dam) bleeding began. Bleeding is the separation of water from the grout. Bleeding is identified by the appearance of a layer of water on the surface of the grout mixture. The bleed water drained down the sides of the plexiglass containment walls, and out from underneath the retaining dam. Stabilized grout was not used in any of the physical model tests.

When flow was initiated by removal of the dam, the bleed water acted as a lubricant, and the grout would slide along this lubrication layer largely undeformed. The slip behavior was enhanced by the smooth plexiglass used to construct the dam break model. The amount of bleeding was sensitive to the properties of the solid ash used in the mix, as indicated by the different amounts of bleeding that occurred between several grout batches, all prepared according to the same recipe. Thus if the amount of bleed water released was not constant, the amount of wall slip was not constant. In the extreme case, a quantity of nearly clear water would quickly flow to the opposite wall, while the remainder of the grout would slump part way across the tank and remain stationary. The apparent wall slip had the effect of preventing the numerical model from matching the experimental data because a lubricating layer was not included in the simulations. Flow time across the tank was decreased because of the slip layer. The majority of the motion and deformation of the grout occurred very

quickly. After the initial slumping of the grout column, the motion continually decelerated until impacting the far wall. The actual impact velocity of the grout was less than the flow distance divided by the flow time. When fluid properties were specified in the numerical model to match the observed flow time, the simulated impact velocity was too large. When properties were specified to match the impact velocity, the flow time was too long. The slip layer also reduced the amount of deformation that occurred in the bulk of the grout. As a consequence, no combination of parameters were found that could match the observed free surface. The slip layer was below the spatial resolution of the numerical model. A non-uniform grid was employed in the vertical direction, with the cell size expanding in the positive z axis. The smallest vertical cell dimension was adjacent to the lower domain boundary with a thickness of 1.2 cm. The slip layer on the lower domain boundary is estimated to be on the order of 1 mm. Thus, in order to resolve the behavior of the lubrication layer, several rows of cells would be needed adjacent to the lower domain boundary with a thickness on the order of 0.01 mm. The physics at this scale could no longer be modeled as a one phase continuum, and the numerical model could not be expected to yield realistic results. Thus, PHOENICS was not able to accurately simulate this flow, partly because one of the physical mechanisms, wall slip due to grout instability, is not included in the numerical model.

MINE INJECTION SIMULATION

Predictions about how FBC-ash grout flows during injection into an underground mine must be based upon knowledge of the grout properties and the dimensions and geometry of the mine. Due to the sensitivity of grout rheology to solid ash properties, and the settling of the grout suspension as described above, a conclusive fluid model has not yet been obtained. It is believed that either the power-law or Herschel-Bulkley fluid model can be used to describe the grout rheology. While the grout properties are anticipated to lie within the range of:

$$\begin{aligned}\tau_y &= 0 - 50 \text{ Pa} \\ k &= 0.1 - 1000 \text{ (N}\cdot\text{s/m}^2\text{)}^n \\ n &= 0.25 - 1.0\end{aligned}$$

the actual fluid model constants are as yet unknown. Conversely, the geometry of an underground room and pillar mine is known from mine maps, but is of such irregularity that simplifications and idealizations are needed to facilitate the modeling process. As can be seen from the map of the Fairfax mine in Figure 41, large pillars of unexcavated coal form a nearly regular array. The pillars range in size from 16 m to 50 m per side. It is the series of interconnecting hallways and passageways that are required to be filled. The coal seam in the Fairfax mine averages 1.5 m in thickness, and the mine passageways are assumed to be the same thickness. The mine floor has a slope of approximately 4% that is oblique to most of the passageways. The floor slope has been neglected. The idealized mine passageway intersection shown in Figure 40 is regarded as typical of the mine geometry.

It is anticipated that the points of injection will coincide with the underground mine

corridor intersections. The passageway intersections are from 15 m to 70 m apart. It is assumed that one injection hole per intersection is the maximum drilling density that would be considered. Of course the goal is to minimize the number of holes that need to be drilled, but for the purposes of developing a numerical model, a minimum estimate of flow distance is needed. If the grout rheology is such that it could be made to flow further than the distance to one intersection, it may be possible to plan drilling injection holes at every other intersection, every third intersection, and so on. Figure 42 demonstrates the minimum distances that grout must be made to flow for various injection strategies. These conceptual injection schemes are based upon a mine with uniform geometry, square pillars, and negligible slope. Many factors not included in the conceptual model or the numerical model may impede or enhance grout placement. Such factors are the variability of the actual mine geometry, floor slope, obstructions from partial collapse of mine pillars or roof, and possible pooling of groundwater within the mine. Even with these simplifications, useful knowledge can be obtained by considering injection into the idealized geometry.

FIGURE 41 Map of the Fairfax Mine

Figure 42 Grout Injection Schemes

AXISYMMETRIC SIMULATION

Axisymmetric simulations have been used to investigate how grout may be expected to pile up on the mine floor and spread outward. The axisymmetric cavity in Figure 43 represents one of the passageway intersections that extends outward from the injection point. Because the simulation is axisymmetric, there is no influence of the vertical mine walls. This should be a good approximation for flow within the intersection area where the grout that impacts on the floor should spread axisymmetrically. The influence of the walls is not important until the grout extends more than 5 m from the injection point.

The power-law model was used because the power-law model appears to fit stress-strain rate data from preliminary rheological tests of grout mixtures that have been stabilized with bentonite. Since the exact values of the power-law parameters are still unknown, several simulations have been performed with various combinations of power-law parameters. A simulation was also performed with the Herschel-Bulkley fluid model to compare to the power-law model results.

The mass flux boundary condition in the upper left corner of the domain represents grout injection at a rate of $0.2 \text{ m}^3/\text{s}$. Three simulations were performed with an open outlet, $P = 0$, along the right boundary. The outlet boundary condition on the right side of the domain was then changed to a no flow, wall boundary condition 5 m from the injection point. This wall boundary condition is used to simulate injection into one of the mine

Figure 43 Axisymmetric Simulation Domain

passages between intersections. None of the simulations include resistance from walls that are parallel to the flow. A no-slip condition was specified on the floor and roof for all of the simulations. As mentioned before, the unstabilized grout exhibits apparent slip at the wall which implies that the no-slip boundary condition is not realistic. However, the wall resistance for a real grout flow must lie between the no-slip and no-friction conditions. Thus, simulation results obtained with a no-slip boundary condition and a frictionless wall boundary condition should bracket the real grout flow. The results of one simulation without friction on the floor and roof are included for comparison.

All the following simulations used a uniform grid with 50 cells was used in the radial y-direction, and a non-uniform grid in the vertical with 20 cells expanding exponentially in the positive z-direction. With a time step of 0.005 seconds and 50 iterations per time step, each simulation required more than 7 hours to simulate 10 seconds of injection. The results at six time steps are presented for the seven cases listed in Table 1.

Figure No.	B.C. @ Right	Fluid Model	n	k (N·s ^m /m ²)	τ_y (Pa)
44	Outlet	Power-Law	0.8	100.0	–
45	Outlet	Power-Law	0.8	1000.0	–
46	Outlet	Power-Law	0.3	25.0	–
47	Outlet	HB	0.3	25.0	25.0
48	Outlet	HB (no fric.)	0.3	25.0	25.0
49	Wall	Power-Law	0.8	1000.0	–
50	Wall	Power-Law	0.3	100.0	–

Table 1

The figures all show the grout-air surface as a solid line. The space beneath the solid line represents the grout. The area above the solid line contains only air. PHOENICS treats the entire simulation as a one phase fluid with spatially varying properties delineated by the solid line. The vector arrows shown in the figures illustrate the velocity vectors in either the grout or air phase.

It was assumed that the grout flows would be laminar because the grout is so viscous. No turbulence models were used in any of the simulations. Typically, the Reynolds number is calculated to verify that a flow is laminar, but for this flow geometry there is no obvious choice for a length scale or a representative velocity with which to calculate a Reynolds number. Moreover, even if a Reynolds number were calculated, no data is available for this geometry to suggest at what Reynolds number the transition to turbulence would begin. Thus, no Reynolds numbers have been calculated, and laminar flow has not been verified.

Figure 44a

Power-Law Fluid Model

 $k = 100.0$; $n = 0.80$

step = 150 ; t = 0.5 sec.

→ 12.72 m/s

Min: 1.83E-02 Max: 5.24E+00

2D FILLING FROM TOP ; SEM ; PWR-LAW

Power-Law Fluid Model

$k = 100.0$; $n = 0.80$

step = 210 , $t = 0.7$ sec.

→ 10.18 m/s Min: 1.31E-02 Max: 3.90E+00

2D FILLING FROM TOP ; SEM ; PWR-LAW

Figure 44b

Figure 44c

Power-Law Model $k=100.0$; $n=0.80$

step = 300 ; t = 1.0 sec.

→ 10.18 m/s

Min: 1.15E-02

Max: 3.90E+00

2D FILLING FROM TOP ; SEM ; PWR-LAW

Figure 44d

Figure 44e

The results demonstrate the how strongly the flow is dependent upon fluid properties. Figures 44 - 46 were all computed with the power-law fluid model, yet changing the material property constants resulted in changes in the flow field. As n and k increase, the grout is more viscous, and the depth of the grout on the mine floor increases. The model parameters used to generate Figure 46 simulated a less viscous grout that spreads out in a thin layer on the floor and quickly flows out of the simulation domain. The free surface depicted in Figure 46 is similar in appearance to supercritical profiles in open channel flow. Therefore, the Froude number was checked to determine if a hydraulic jump could occur at some location downstream of the simulation domain. A Froude number of 1.32 was calculated for the solution at 10 seconds at a location 4 m from the axis of symmetry. Although this indicates supercritical flow, the jumps associated with Froude numbers in the range of 1 to 1.5 are small undular jumps and standing waves which would have no influence on the project. Henderson(1966) presents figures and descriptions of these types of hydraulic jumps. All of the other simulations had Froude numbers less than one, indicating subcritical flow.

When the Herschel-Bulkley model is used, Figure 47 shows that the grout does not flow across the floor like the power-law models predict. Rather it piles up until it fills the entire height of the cavity and then begins moving outward as a nearly solid plug. Thus the power-law and Herschel-Bulkley fluid models generate very different predictions about how the mine may fill with injected grout.

Figure 48 illustrates a case in which there is no wall friction. Comparison with Figure 47 shows that the main effect of wall slip is to eliminate the overshoot at the floor and ceiling. With no friction, the leading edge of the spreading grout is nearly vertical, with a tendency to slump down and outward.

The simulations that produced Figures 49 and 50 used a wall boundary condition at the right of the simulation domain instead of an outlet condition. These simulations represent the flow that may occur if an injection well is drilled (perhaps inadvertently) in the middle of a mine passage, rather than in one of the passage intersections. These simulations represent the flow in a plane that is perpendicular to the closest mine pillar wall. Real flows that impact a wall would cease to be axisymmetric. Thus, Figures 49 and 50 do not give realistic predictions for flow after the grout impacts the wall.

It is evident that the choice of fluid model and model parameters have a direct impact on simulation results. This indicates that the choice of fluid model may greatly impact mine filling predictions and design decisions. Obviously it is important to use a numerical model that realistically simulates the actual grout rheology. Thus, thorough understanding of the grout rheology and accurate representation of the rheology in the computer model are vital to the success of the numerical simulation.

TWO DIMENSIONAL SIMULATION

A horizontal two dimensional simulation of a mine passage in plan view was attempted to demonstrate how the grout would flow once it has filled the intersection area and begun to move past the coal pillars. However, it was found that the inlet boundary condition could not be correctly implemented. The two dimensional simulation was to take place in the horizontal x-y plane, but the grout is actually to be injected vertically in the -z direction. In the actual three dimensional domain, the grout that comes down from the roof would impact the floor and begin to spread out in an, axisymmetric manner. A number of

HB Fluid - Outlet B.C. - No-Slip B.C.

Figure 47

Power-law Fluid - Outlet B.C. - No-Slip B.C.

Figure 45

HB Fluid - Outlet B.C. - No Wall Friction

Figure 48

Power-law Fluid - Outlet B.C. - No-Slip B.C.

Figure 46

Figure 49a

Figure 49b

Figure 49c

Figure 49d

Figure 49e

Figure 49f

Figure 49 Power-Law Fluid - No Outlet B.C. - No-Slip B.C.

Figure 50a

Figure 50b

Figure 50c

Figure 50d

Figure 50e

Figure 50f

Figure 50 Power-Law Fluid - No Outlet B.C. - No-Slip B.C.

attempts were made to represent this axisymmetric spreading by declaring mass inflow boundary conditions in the x and y directions near where the grout would impact the floor.

Unfortunately, such boundary conditions did not produce the axisymmetric spreading that was expected. Rather, the grout interface developed irregular lobes as shown in Figure 51 because the grout was being injected into the domain in a non radial fashion. Thus, a two dimensional simulation of the area near the injection point is not meaningful because the inlet boundary condition cannot be defined realistically.

THREE DIMENSIONAL SIMULATION

A three dimensional domain was created to simulate the complete injection process. The symmetry of the mine passage geometry was exploited to minimize the size of the simulation domain. The x, y, and z dimensions of the domain were 50 m, 1.524 m, and 5.0

Plan View of Mine Floor Near Injection Point

Inflow

Mow--+·

Realistic axisymmetric spreading

Unrealistic spreading predicted due to improper inflow B.C.

Figure 51 Unrealistic Simulation Result

m respectively, as shown in Figure 52. A uniform grid was used with 50, 12, and 15 divisions in the x, y, and z direction, for a total of 9000 grid points. A sample of the simulation results after 5 second of injection is shown in Figure 53. The figure depicts the grout-air interface near the injection point as represented by the three dimensional $z=0.5$ surface. Figure 53 reveals that the grout is spreading axisymmetrically after it reaches the mine floor and before it impacts the walls, as expected. The grout properties used in the three dimensional simulation are the same as those used in the axisymmetric simulation represented by Figure 45. Although the results of both are qualitatively similar, a

quantitative comparison of the two results cannot be made because the specified flowrate boundary conditions are different.

Figure 52 Three Dimensional Simulation Domain

Figure 53 Three Dimensional Grout Interface after 5 s.

Although the results appear promising, the three dimensional simulation was stopped after 48 hours of cpu time. The small time steps required by the Scalar Equation Method of marking the grout-air interface, and the large number of computational cells in the three dimensional domain, result in a very slow simulation. With a 90 MHZ Pentium processor, it takes approximately 8 hours to simulate one second of the injection process. Because the time scale of the injection process is expected to be on the order of minutes or possibly hours, the three dimensional model cannot be considered to be a practical design tool with the present generation of personal computers.

Conclusions

1. It has been shown that the maximum flow distance of an unstable grout will always be less than the flow distance of a stable grout. As settling occurs in the unsteady grout mixtures, Coulomb friction between the suspended particles will generate an additional resistance to flow. Such mixtures may be so limited in their ability to flow that they would be impractical for the mine filling application under investigation.
2. A commercially available CFD code, PHOENICS, has been modified to include a specific non-Newtonian constitutive law. The Herschel-Bulkley apparent viscosity has been included in the GXPROP2.FOR subroutine and a new PHOENICS run version has been compiled. The subroutine has been compared to an exact solution and shown to produce accurate results.
3. Axisymmetric simulations of the mine injection process have been performed. The CFD software, in conjunction with the new subroutine, may be used to simulate the initial period of injection of stable grout mixtures into abandoned, underground mines.
4. The computer model has not been able to simulate flow beyond the mine passage intersections where the influence of the coal pillar walls transforms the axisymmetric flow into a truly three dimensional flow. Three dimensional simulations have been performed, but the execution times are excessive on the available hardware. More powerful computers are necessary to make three dimensional simulations a practical tool.
5. Simulation results using several non-Newtonian fluid models display strikingly different results. It is therefore very important that the correct fluid model be used to simulate the injection process, else the quantitative and qualitative information provided by numerical simulations will be useless. With the appropriate non-Newtonian model, fluid properties, and boundary conditions, numerical simulations can provide useful predictions of grout flow behavior.

RECOMMENDATIONS FOR FURTHER STUDY

Several aspects of this investigation are still unresolved. The correct wall boundary

condition is uncertain. It should be determined experimentally and incorporated into the PHOENICS code for future simulations. The variability of solid content from the Beechurst plant must be addressed. If no change is made to the current method of gathering the ash waste, it will not be possible to predict mine filling behavior because the grout properties will be subject to large and unknown variations. If a definitive grout recipe cannot be obtained, the computer model will not be a successful tool for designing injection well placement nor choosing an injection strategy. It is desirable to devise a simple field test, similar to the slump test used for concrete testing, to monitor the grout rheology during large scale injection operations. If the grout rheology can be determined in the field, field personnel may be able to adjust the grout mix and injection well spacing to compensate for variations in the grout ingredients. Finally, the computer model developed here should be used to perform a thorough parametric study covering the range of expected grout property values.

TASK 3 BENCH-SCALE RHEOLOGY MODEL

by Donald D. Gray, Ph.D. and James M. Stiles, Ph.D., P.E.

Purpose

The purpose of this task is to determine the rheological properties of a grout composed of FBC byproducts, water, and bentonite, and also to simulate the flow of the grout into an abandoned mine using both computer models and bench scale lab tests. The goal of this task is to minimize the number of boreholes needed to completely fill abandoned subsurface mines using conventional pumping techniques. Therefore it is necessary to achieve maximum flowability while maintaining grout stability and meeting minimum strength properties.

Methodology

Grout Rheology

The first task was to establish a suitable range of water fractions for the FBC ash grouts. This was done by preparing grout samples at several water fractions and visually inspecting them for flowability. After doing preliminary tests to determine a suitable range of water fractions, rheological tests were performed. After the grout sample was prepared, a portion was tested using the Brookfield LVDV III rotational viscometer fitted with a T-bar spindle and a heli-path mount. A schematic of this setup is shown in Figure 54.

Figure 54: Schematic of viscometer with heli-path mount.

The principle of the rotational viscometer is to measure the torque required to turn a rotor submerged in the test fluid at a constant angular velocity. The heli-path device moves

the entire viscometer up and down periodically so that the spindle cuts a helical path through the sample. This continually puts the spindle in contact with new material. This is of particular importance when testing non-flowing samples where a spindle turned at the same depth could cut a groove into the sample. Tests were performed at variety of angular velocities with torque readings being taken when the T-bar was at a constant depth in the grout in order to eliminate the variation in the torque reading due to the changing length of the shaft that is submerged in the sample. The torque-rpm data were then reduced to shear stress and strain rate data using the mixer viscometer method introduced by *Metzner and Otto (1957)*, who were working with power law fluids, and by *Tattersall and Bloomer (1979)*, who extended this work to include Bingham plastics. This method was further extended by the present authors to include the Herschel-Bulkley fluid model. The mixer viscometer method uses the slope and intercept of the torque versus rpm plot (either on linear-linear or log-log paper) along with two machine constants, which are found by testing fluids of known properties. This method can be used to obtain parameters for Newtonian, power-law, Bingham and Herschel-Bulkley fluid models.

Mixer Viscometer Method

The basic principle of this method is that the torque T needed to maintain a constant rotational speed N is equal to N multiplied by the apparent viscosity of the substance at the corresponding rate of strain (η_a) and G , which is a device constant.

$$T = G\eta_a N$$

Case 1: Newtonian Fluid

For a Newtonian fluid, the constitutive equation for parallel shear flow is given by the equation

$$\tau = \mu \gamma$$

where τ is the shear stress, μ is the dynamic viscosity, and γ is the rate of strain. From this we can define an apparent viscosity which is equal to the shear stress divided by the strain rate. For a Newtonian fluid, this is a constant equal to the dynamic viscosity. By solving for the apparent viscosity, we get

$$\eta_a = \frac{\tau}{\dot{\gamma}} = \mu = \frac{T}{GN}$$

The constant G can be found by testing Newtonian fluids with known viscosities and plotting the data as torque (T) versus rotational speed (N). This plot should yield a "straight" line through the origin with a slope equal $G\mu$. If a fluid of known μ is tested, G

can be determined. This allows μ to be found for any other Newtonian fluid.

Case 2: Bingham Fluid

For the Bingham model, the constitutive equation which relates the shear stress to the strain rate in a parallel shear flow is

$$\tau = \tau_y + \mu_p \dot{\gamma}$$

where τ_y is the yield stress of the fluid and μ_p is the plastic viscosity. Again we can define an apparent viscosity by dividing the shear stress by the rate of strain. Unlike the Newtonian, the apparent viscosity is not a constant, but depends on the rate of strain.

$$\eta_a = \frac{\tau_y}{\dot{\gamma}} + \mu_p = \frac{T}{GN}$$

By assuming that the rate of strain ($\dot{\gamma}$) is equal to a constant K multiplied by N and solving for the torque we get

$$T = \mu_p GN + \frac{\tau_y N}{K}$$

The assumption $\dot{\gamma}=KN$ has been verified for numerous Non-Newtonian fluids (*Skelland, 1983*). Like the Newtonian case, a plot of torque versus angular speed yields a straight line with the slope equal to $\mu_p G$, only in this case the line does not pass through the origin (See Figure 55). The intercept on the torque axis is equal to $\tau_y G/K$. By testing a Bingham fluid with known properties such as an 8% bentonite slurry (*Grim, 1978*), K , which is a device constant, can then be found.

Case 3: Power-law Fluid

For a power-law fluid, the constitutive relationship for parallel shear flow is

$$\tau = r \dot{\gamma}^s$$

where r is the consistency index and s is the power number. By the definition of the apparent viscosity, we have

$$\eta_a = r \dot{\gamma}^{s-1} = \frac{T}{GN}$$

Figure 55: Torque versus rotational speed plot for a Bingham fluid.

Again substituting KN for γ and solving for T we get

$$T = GrK^{s-1}N^s$$

For this case the plot of T versus N does not yield a straight line on linear graph paper, but taking the natural log of both sides of the above equation gives

$$\ln T = \ln(GrK) + s \ln N$$

Plotting this equation on log-log paper then yields a straight line with the slope equal to the power number s and the intercept equal to GrK .

Case 4: Herschel-Bulkley Fluid

$$\tau = \tau_y + r\gamma^s$$

The constitutive equation for parallel shear flow for a Herschel-Bulkley fluid is
For this relationship, the apparent viscosity is equal to

$$\eta_a = \frac{\tau_y}{\dot{\gamma}} + r\dot{\gamma}^{s-1} = \frac{T}{GN}$$

By making the same substitutions as before, and again solving for T, the result is

$$T = \frac{Gr\tau_y}{K} + GrK^{s-1}N^s$$

In order to fit the data to an HB model it is necessary to first have an independent measure of the yield stress of the liquid. This can be obtained by several procedures, of which the vane method is the easiest to perform and also one of the more reliable (*Nguyen and Boger, 1992*). For this test a four-bladed vane is immersed in the liquid. The torque is then increased until the vane begins to rotate. Below this value of torque the material behaves like an elastic solid, and above the value, like a fluid. The maximum torque reached before rotation of the vane begins can be converted to the yield stress by

$$T = \frac{\pi}{2}d^3\left(\frac{l}{d} + \frac{l}{3}\right)\tau_y$$

where T = maximum static torque
 l = length of the vane
 d = diameter of the vane
 τ_y = yield stress

Substituting this value of the yield stress into the previous torque equation and moving that term to the left side of the equation gives

$$T - \frac{Gr\tau_y}{K} = GrK^{s-1}N^s$$

Taking the natural log of both sides of this equation gives

$$\ln \left(T - \frac{Gr\tau_y}{K} \right) = \ln (GrK^{s-1}) + s \ln (N)$$

Plotting the left hand side of the equation versus N on log-log paper gives a straight line with the slope equal to s and the intercept equal to GrK^{s-1} . This can then be solved for s and r if τ_y is already known. Thus all the parameters of the constitutive equation for the

HB fluid are determined.

Grout Stability

Before any viscometric tests could be performed, it was first necessary to produce a grout that was stable. By definition, an unstable grout is one that bleeds greater than five percent of its water during a two hour period. Bleed tests were done in accordance with ASTM standard C 232 - 87. Samples were prepared and placed in round containers which were covered to prevent evaporation. After two hours, any clear water that had bled to the surface was pipetted off and weighed. This weight, divided by the weight of water used in the mix and multiplied by 100 gives the percentage of bleed. Mixtures of pure FBC ash and water were found to be unstable. To combat the settling of the solid particles, various percentages of WYO-BEN 250 mesh bentonite were added to the grout mix. Figure 56 shows that the percentage of bleed increases linearly with an increase in water fraction, while the amount of bleed is decreased by increasing the amount of bentonite added. It was found that 3% bentonite was sufficient to render the grout stable.

Other tests done on the rotational viscometer using the T-bar spindle gave proof that particles in the pure FBC ash - water grouts were settling. First a series of tests were done in which the spindle was rotated at a constant speed while being moved through a helical path in a grout containing only fly ash. Torque readings were taken at five second intervals for the length of the test. When these readings were plotted versus time, the pattern showed that the torque increased as the spindle neared the bottom of its path, and decreased as it neared the top of the path, as seen in Figure 57. This periodic behavior is due to the variation in the length of the submerged shaft, but the increase in amplitude is due to the settling of the particles causing the lower portion of the sample to become more concentrated than the upper portion.

Figure 56: Results of bleed tests on mixes containing equal portions of fly ash and bottom ash.

Tests were then done using different water fractions and different amounts of bentonite. At 7 % bentonite, there was very little settling. Another test where the spindle was rotated at a constant depth was performed with torque readings taken at a specified time interval. Figure 57 shows that with no bentonite used in a fly ash only grout, the torque increased with time, but with 7 % bentonite, the torque at the constant depth showed much less increase. The increase in torque for the no bentonite grout is due to the settling of the ash particles. As the particles come in contact with one another there is an additional frictional

mechanism which increases the torque. Eventually, the T-bar formed a slip surface in the sample which effectively caused the torque to become constant. The sample containing bentonite, on the other hand, has less torque increase, showing that particle settling is not a problem. All of these tests have shown that bentonite is an effective additive in the reduction of particle settling and in increasing the stability of the grout.

Ash Variability

Different samples of ash taken from the Beechurst Avenue power plant not only contain varying percentages of fly and bottom ash, but also vary considerably in flow characteristics. This was discovered when doing the "dam break" experiment. In this test a rectangular box 100 cm long by 20 cm wide by 40 cm high with a removable wall 20 cm from the end was used. Grout was placed behind the wall to a depth of 30 cm. The wall was then quickly removed, and the grout was allowed to flow toward the other end of the box. Tests were done using grouts prepared using identical recipes but with ash taken from two separate ash lots. In one test the grout was very fluid and reached the end of the box in under 2 seconds, while the other sample failed to reach the end of the box at all. This difference in behavior is likely due to the variability of material that the plant is burning and also to the plant's method of collecting the ash. The power plant burns a mixture of pure coal, limestone, and "gob", which is refuse coal found in the overburden. Limestone is added to decrease the sulfur emissions from the exhaust stack. The percentage of gob burned is changed

Figure 57: Plot showing increase in stability due to the addition of bentonite.

according to the output power requirements. Another cause of variability is due to the ash handling at the power plant. Every truck is filled from one of three ash holding hoppers. At any time these hoppers can contain any combination of fly and bottom ash.

Figure 58: Spread versus water fraction for fly ash only samples at varying amounts of bentonite.

The variability will become a problem when actually filling the mine. The operator will not be able to simply use the same grout recipe for each truckload of ash. For this reason work has been done to come up with an acceptable field test that the operator can perform to ensure that the grout has the proper flow characteristics. The proposed test uses a cylinder 3 inches in diameter by 6 inches in height open at both ends. The cylinder is placed vertically on a horizontal surface and filled with grout. The cylinder is then slowly lifted and the grout is allowed to spread in a radial fashion. The distance of spread is then measured in two perpendicular directions and an average is taken. Spread tests were done using varying amounts of bentonite and water. Figure 58 shows how the addition of bentonite reduces the spread of the grout. This reduction in spread may call for the use of a super-plasticizer or air entraining agent to counteract the effect of the bentonite. Work is currently being done to see how these additives will affect the flow of the grout, and to obtain a correlation between the spread and the appropriate flow parameters.

Results and Discussion

Tests have been done using the T-bar spindle on grouts with fly ash as the only combustion byproduct. The ash was taken directly from the ash collection flow stream before it was deposited in the collection hoppers. Work will eventually include grouts containing both fly and bottom ash. These tests have been done with varying water fractions and amounts of bentonite. When using bentonite it is necessary to dry mix the ash and the bentonite before any water is added to insure that the bentonite will be in contact with as much of the ash as possible. Figure 59 shows typical torque

Other tests done on the rotational viscometer using the T-bar spindle gave proof that particles in the pure FBC ash - water grouts were settling. First a series of tests were done in which the spindle was rotated at a constant speed while being moved through a helical path in a grout containing only fly ash. Torque readings were taken at five second intervals for the length of the test. When these readings were plotted versus time, the pattern showed that the torque increased as the spindle neared the bottom of its path, and decreased as it neared the top of the path, as seen in Figure 57. This periodic behavior is due to the variation in the length of the submerged shaft, but the increase in amplitude is due to the settling of the particles causing the lower portion of the sample to become more concentrated than the upper portion.

Figure 56: Results of bleed tests on mixes containing equal portions of fly ash and bottom ash.

Tests were then done using different water fractions and different amounts of bentonite. At 7 % bentonite, there was very little settling. Another test where the spindle was rotated at a constant depth was performed with torque readings taken at a specified time interval. Figure 57 shows that with no bentonite used in a fly ash only grout, the torque increased with time, but with 7 % bentonite, the torque at the constant depth showed much less increase. The increase in torque for the no bentonite grout is due to the settling of the ash particles. As the particles come in contact with one another there is an additional frictional

mechanism which increases the torque. Eventually, the T-bar formed a slip surface in the sample which effectively caused the torque to become constant. The sample containing bentonite, on the other hand, has less torque increase, showing that particle settling is not a problem. All of these tests have shown that bentonite is an effective additive in the reduction of particle settling and in increasing the stability of the grout.

Ash Variability

Different samples of ash taken from the Beechurst Avenue power plant not only contain varying percentages of fly and bottom ash, but also vary considerably in flow characteristics. This was discovered when doing the "dam break" experiment. In this test a rectangular box 100 cm long by 20 cm wide by 40 cm high with a removable wall 20 cm from the end was used. Grout was placed behind the wall to a depth of 30 cm. The wall was then quickly removed, and the grout was allowed to flow toward the other end of the box. Tests were done using grouts prepared using identical recipes but with ash taken from two separate ash lots. In one test the grout was very fluid and reached the end of the box in under 2 seconds, while the other sample failed to reach the end of the box at all. This difference in behavior is likely due to the variability of material that the plant is burning and also to the plant's method of collecting the ash. The power plant burns a mixture of pure coal, limestone, and "gob", which is refuse coal found in the overburden. Limestone is added to decrease the sulfur emissions from the exhaust stack. The percentage of gob burned is changed

Figure 57: Plot showing increase in stability due to the addition of bentonite.

according to the output power requirements. Another cause of variability is due to the ash handling at the power plant. Every truck is filled from one of three ash holding hoppers. At any time these hoppers can contain any combination of fly and bottom ash.

Figure 58: Spread versus water fraction for fly ash only samples at varying amounts of bentonite.

The variability will become a problem when actually filling the mine. The operator will not be able to simply use the same grout recipe for each truckload of ash. For this reason work has been done to come up with an acceptable field test that the operator can perform to ensure that the grout has the proper flow characteristics. The proposed test uses a cylinder 3 inches in diameter by 6 inches in height open at both ends. The cylinder is placed vertically on a horizontal surface and filled with grout. The cylinder is then slowly lifted and the grout is allowed to spread in a radial fashion. The distance of spread is then measured in two perpendicular directions and an average is taken. Spread tests were done using varying amounts of bentonite and water. Figure 58 shows how the addition of bentonite reduces the spread of the grout. This reduction in spread may call for the use of a super-plasticizer or air entraining agent to counteract the effect of the bentonite. Work is currently being done to see how these additives will affect the flow of the grout, and to obtain a correlation between the spread and the appropriate flow parameters.

Results and Discussion

Tests have been done using the T-bar spindle on grouts with fly ash as the only combustion byproduct. The ash was taken directly from the ash collection flow stream before it was deposited in the collection hoppers. Work will eventually include grouts containing both fly and bottom ash. These tests have been done with varying water fractions and amounts of bentonite. When using bentonite it is necessary to dry mix the ash and the bentonite before any water is added to insure that the bentonite will be in contact with as much of the ash as possible. Figure 59 shows typical torque

versus speed data for a fly ash only sample containing 7% bentonite along with the Bingham fit to the data ($\mu_p = 6.7 \text{ Pa}\cdot\text{sec}$, and $\tau_y = 2.9 \text{ Pa}$)

One problem that has been discovered is the shear rates found using the T-bar spindles are on the order of 0.2 s^{-1} . This is far below the rates which will be found in the pumping and placing of the grout. During the placing, the maximum shear rate for the grout flowing in an injection pipe is approximately 60 s^{-1} . Therefore a series of tests have been done using a cylindrical spindle . With this particular geometry, it is likely that wall slip may occur, so that the results may not be reliable, but the shear rates that are generated in this geometry are in the range that will occur in the field. Future work will include tests to determine if wall slip has occurred and to correct for it if necessary. When the concentric cylinder data is combined with the data generated using the T-bar spindle, the best fit is given by an HHB model as shown in Figure 60.

Figure 60: Shear stress versus strain rate for a sample tested using concentric cylinder geometry.

Bingham approximation to actual data. Water fraction = 50% Tested using helical path

Figure 59: Torque versus speed plot for a sample with a water fraction of 50% containing 7% bentonite.

TASK 3B DESIGN OF PNEUMATIC ASH INJECTION DEVICE

TASK 3C BENCH-SCALE TESTING OF EJECTOR

by Mackenzie Burnett, P.E.

Introduction

This project is undertaken to further develop the Burnett Ejector by designing, building and testing an improved supersonic nozzle which will be shockfree flow and to shroud the nozzles so as to induce additional air flow that transports the ash horizontally in the mine void. The project will develop the nozzles and shroud on a bench scale to optimize the performance with the ultimate goal of reducing the required amount of compressed air per ton of ash flow. These tasks are Tasks 3B and 3C from the Statement of Work for this Contract.

Purpose

The purpose of this work is to reduce the cost of injection of fluidized bed combustion by products into underground mine voids. The goal of this task was to improve the performance of the ejector by redesigning it to use less air for the same stowing capability or to increase the distance the material could be stowed using the same air flow as on the original design.

Background

The original design of the Burnett Ejector developed for the U.S. Bureau of Mines (*Burnett*) was tested on a NMLRC contract (*Burnett and Black*). The results of this project indicated that pneumatic injection of fluidized bed combustion by products into underground mines could be very cost effective and environmentally sound. Although the system appeared to be cost effective, it was known that the system could be further developed to reduce the highest cost element of the system operation, the cost of compressed air.

Methodology

The original ejector design tested on the NMLRC contract was designed for stowing gravel into underground mines through boreholes. It was meant to be a very simple low cost device. The supersonic nozzle design used on this ejector was a simple converging-diverging nozzle. This type of nozzle is simple to manufacture, but it has certain deficiencies. Shockwaves in the expanded flow can cause the flow to exit the nozzle in a non-parallel fashion which is detrimental to maximizing the horizontal distance the stowed material is thrown. To eliminate the shockwaves, the supersonic nozzle was redesigned with a contoured exit section which will insure that the flow is parallel to the nozzle centerline.

Results and Discussion

Aerodynamic design

The aerodynamic properties of the supersonic nozzle were calculated. Figure 74 shows the nozzle contour design parameters for various mach numbers. The plot shows the nozzle shape downstream of the throat as a function of the throat diameter. For a converging-diverging supersonic nozzle the air is sonic at the throat. Therefore, the flow is determined by the throat radius. The extent of expansion and the mach number of the outlet air is determined by the exit radius and the pressure ratio across the nozzle. The nozzle throat diameter determines the airflow rate through the nozzle for a given pressure ratio. The nozzle throat and outlet diameter can be determined using the following equations. The conditions at the throat are determined from:

$$\frac{T_c}{T_0} = \frac{2}{k+1} \quad \frac{P_c}{P_0} = \left[\frac{2}{k+1} \right]^{\frac{k}{k-1}} \quad \frac{\rho_c}{\rho_0} = \left[\frac{2}{k+1} \right]^{\frac{1}{k-1}} \quad c_c = V_c = \sqrt{kRT_c}$$

Where: T is the absolute temperature in $^{\circ}\text{R}$
 k is the specific heat ratio dimensionless
 P is the absolute pressure in psi
 ρ is the density lbm/ft^3
 c is the critical sonic velocity ft/s
 V is velocity ft/s
 R is the gas constant $\text{ft lb/lbm } ^{\circ}\text{R}$

and the Subscripts: $_0$ = conditions in the reservoir
 $_c$ = critical condition at throat

The mass flow rate is an input variable to this equation set, and the throat area can be calculated from it using:

$$A_c = \frac{\dot{m}_{\max}}{\rho_c V_c} \quad (51)$$

Where: $\dot{m}$ = mass flow rate lbm/s

The outlet area A is determined from the desired mach number using the following equation:

$$A = A_c \left(\frac{1}{M} \left(\frac{5+M^2}{6} \right) \right)^3 \quad (52)$$

M = mach number at outlet

A = area at outlet

Mechanical design

A drawing of the three nozzles plus the original nozzle design are shown in Figure 75. The contour of the exit section was determined from the parameters shown in Figure 74. The nozzle exit contour was plotted on an Autocad® system and the parts were made directly from this software. All of the nozzles are sized for 50 ft³/min at 100 psig. The throat diameter for all of the nozzles is 0.1774 in. The exit velocity is designed to be mach 2.1 for the C-V nozzle and nozzle number 1. The design mach number for nozzles 3 and 4 is 1.5 and 2.0 respectively. Performance of nozzles 1,2 and 3 provide an exit flow that is straight and parallel to the nozzle centerline.

Nozzle tests

A test fixture, shown in Figure 76, was designed to allow comparison of the original designed C-V nozzle and the new contoured nozzles. Thrust of each nozzle was measured on the test fixture by recording the load measured by the scale. The expected thrust of nozzle design number one was 3.38 lb-force/nozzle.

The test results of the four nozzles are shown in Table 4. All three of the contoured nozzles performed better than the original C-V nozzle design. The original C-V nozzle performed at 92% of its calculated thrust. This is to be expected since the flow coefficient was assumed to be unity, which is not possible. Nozzle number 1 (M=2.1) had 11% greater thrust than the original nozzle design. The other two nozzles had slightly less thrust. Based on these test results it can be concluded that nozzle number 1 (M=2.1) is the design to be used in the next phase of the development.

Table 4 Nozzle performance test results

Nozzle	Thrust lb	Ratio of Measured Thrust to Calculated Thrust	Ratio of Measured Thrust of Nozzle Tested to Measured Thrust of Original C-V Nozzle)
C-V (original)	3.05	.920	1.00
1-M=2.1	3.38	.995	1.11
2-M=1.5	3.12	.940	1.02
3-M=2.0	3.20	.960	1.05

Figure 74 Isentropic nozzle data, wall contour vs. Exit mach number for sharp-cornered, axi-symmetric, isentropic, parallel exit flow nozzles ($\gamma=1.4$)

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Figure 76 Nozzle Test Fixture.

Ejector design

The ejector uses a high velocity jet of air to intercept the material falling by gravity down the feed pipe. This jet of air in a volume of essentially still air such as found in a mine void will cause circulation of the air in that void. The velocity of this induced circulation, however, is low and needs to be as high as possible for good material transport.

Ejectors have been used for many purposes to move gasses and solids. Air blown through a set of supersonic nozzles mounted in a aerodynamic shroud can increase the total flow by at least a factor of two. That is, the flow through the nozzles induces an equal flow through the shroud. This concept can be applied to the ejector, as shown in Figure 77, to increase the flow of the jet. This additional flow contributes to an increased mass flow of air intercepting the falling material and creates a velocity in the mine void that is sufficient to transport material.

The full scale ejector was conceptually designed and is shown in Figure 77. The ejector will have 30 nozzles for a 1,500 cfm ejector. The large number of nozzles is necessary to keep the ejector to a size that will fit down an 8-diam-borehole. The new nozzle L/D ratio (length over nozzle diameter) is significantly longer than the original design which requires scaling down of the nozzle and increasing the number of nozzles to achieve the desired air flow. The large number of nozzles has an additional benefit of insuring an even flow through the shroud which will aid in inducing up to 1,500 cfm of additional air.

Conclusions

The tests of the new contoured nozzles show that a performance improvement can be expected for the complete ejector. The changes to the ejector design will either allow the same stowing performance as the original ejector while using less compressed air or substantially improved stowing performance while using the same compressed air flow as the original ejector. This performance improvement will have a significant benefit to the overall cost of injecting ash in underground mines. Field tests in the spring of 1996 will demonstrate the actual improvement. The two changes over the original ejector design are:

1. improved supersonic nozzle insuring supersonic flow parallel to the nozzle centerline
2. shrouded nozzles allow induced air up to 100% of jet air which will enhance flow of ash in the mine void and allow ash transport to be increased substantially over the 100 ft achieved with the original ejector design.

Figure 77 Three View Drawing of the Full Scale Ejector.

TASK 4 WATER QUALITY MONITORING

by William A. Sack, Ph.D. and D. Courtney Black

Purpose

This purpose of this task is to monitor baseline water quality and quantity of the acid mine drainage (AMD) from the Longridge and Fairfax Mines prior to grouting. The baseline data will be essential to evaluate and document the benefit of the grouting activity over time.

Background

The Longridge Mine is an ideal site for sampling since all water exits the mine from a single opening. It was relatively easy to modify the area near the opening so as to capture all the flow and then to direct it into a pipe for measurement and sampling. Several flow and water quality measurements were made on AMD from the mine in 1982. At that time the flow varied between 10 to 74 gallons per minute (gpm). This information was helpful in choosing a flow measurement system to monitor the expected large variation in flow over the project period. It was decided to set up a continuous sampling station for baseline monitoring of flow and water quality from the Longridge Mine. All water in the active Fairfax Mine has historically been collected in a sump and pumped out of the mine as required. It was planned that baseline samples would be taken of the pumped flow on a bi-monthly basis. However, the Fairfax Mine closed shortly after Phase II operations so no water samples were collected.

Methodology

In order to measure flow from the Longridge Mine, a rock and concrete dam was built at the entrance to back up water leaving the entrance to a depth of about 2 feet. The minewater was directed to flow through a 6 inch diameter PVC pipe built into the bottom of the dam and a variable gate flowmeter was attached to the end of the pipe to continually monitor the flow. Calculations show that the system should be able to monitor flows of from about 0.25 to 180 gpm which will cover the expected flow range both prior to and after the grouting.

The site is visited monthly to inspect the equipment and to collect the data and samples. Flow data from the meter is continuously recorded and a signal is sent to a sampler in order to allow collection of samples in proportion to flow. The flow data is both recorded on a stripchart and stored on Flowlink software for monthly downloading to a portable computer. Samples of minewater are taken and sent to a weatherproof sampling station (heated and cooled) where they are stored in a polyethylene container at 4° C. The sampled AMD is picked up on a weekly basis and transported to the laboratory for analysis. The sampler is programmed to take as many as 100 aliquots of minewater per month based on a preset flow interval such as 1200 gallons. Thus when the flow is high, samples are taken more frequently than when flow falls off. The flow interval between samples is adjusted from time to time depending on the flow at that period so as to spread sample collection over the entire month's period.

Seventeen constituents were analyzed in the monthly samples. The constituents were chosen based on contaminants known to be in AMD and their potential toxicity to the environment. Another factor in the choice were those pollutants which might be leached from the MEA FBC ash. The constituents chosen for analysis are as follows:

acidity	magnesium	specific conductance	arsenic
calcium	pH	acidity	lead
temperature	alkalinity	sulfate	cadmium
manganese	iron	aluminum	boron
selenium			

It may be seen that a variety of major AMD constituents were chosen such as acidity, sulfate and iron as well as 5 trace elements. All analyses were performed according to accepted standard methods and QA/QC procedures rigorously observed.

Results and Discussion

Table 2 presents data from the Longridge mine from December 1996 to November 2000. The concentrations of elements in the Longridge water is typical of other Freeport seam deep mines in Preston County, West Virginia. The flow however is quite high for an eleven acre mine. Typically water is produced at a rate of approximately 0.4 gallons per minute per acre. Thus a typical flow of 44 gpm or so would be expected from the Longridge mine. During the Summer sampling events this is the case but during the Fall and Spring high flow periods as much as 437 gpm was observed. This observation led to a change in the Phase III grouting operations that will be covered later in this report.

Table 3 displays mean values for Longridge water quality. The samples are divided into three groups: 1) prior to grouting, 2) during grouting and 3) post grouting. Prior to grouting the pH averaged 3.0, with an acidity 621 mg/L. The flow averaged 190 gpm while the acid load (acid concentration x flow in gpm converted to lbs/day) was 1437 lbs/day. During the grouting operations the pH dropped to 2.9, the acidity dropped to 586 mg/L but the flow increased to 288 gpm. This caused the acid load to increase by about 43%. After grouting the pH remained at 3.0, the acidity dropped to 425 mg/L and the flow decreased by 90%. This yields an acid load of 47 lbs per day or a reduction in acid load of 97%.

Conclusions

The grouting of the Longridge Mine decreased the acid load by 97%. The reduction in acid load is primarily due to the reduction in flow. The area around the mine has been investigated for new seeps and springs of water quality similar to the Longridge discharge; none have been found. The flow at an abandoned Freeport coal seam surface mine up-dip of the Longridge Mine has increased but the water quality is not acidic.

TABLE 2 Patriot Mining
Longridge mine discharge

Date	pH	Cond.	Alkalinity mg/l	Acidity mg/l	Al mg/l	As mg/l	B mg/l	Ba mg/l	Ca mg/l	Cd mg/l	Fe mg/l	Mg mg/l	Mn mg/l	Pb mg/l	Se mg/l	SO4 mg/l	Avg. Flow gpm	Acid Load lbs per day
12/06/96	2.7	2020	0.0	597.0	43.0	0.4	0.1	<0.001	76.0	<0.001	84.0	47.0	14.0	<0.040	0.3	970.0	148.8	1065.7
12/13/96	3.1	1970	0.0	644.0	50.0	0.5	<0.030	<0.001	92.0	<0.001	93.0	55.0	16.0	<0.040	0.3	1100.0	137.5	1062.3
12/27/96	2.9	2070	0.0	622.0	49.0	0.5	0.1	0.0	96.0	<0.001	92.0	54.0	15.0	0.2	0.4	1200.0	66.6	496.7
01/04/97	2.9	1960	0.0	537.0	42.0	0.4	0.1	0.0	83.0	<0.001	80.0	48.0	14.0	0.1	0.4	1000.0	101.2	652.0
01/11/97	3.0	2050	0.0	585.0	47.0	0.5	0.1	0.0	93.0	<0.001	90.0	52.0	14.0	0.1	0.4	1100.0	53.6	376.1
01/21/97	2.7	2300	0.0	646.0	46.0	0.6	0.1	0.0	110.0	<0.001	110.0	59.0	17.0	0.2	0.5	1100.0	48.4	375.4
02/12/97	2.7	2000	0.0	644.0	41.0	0.5	0.1	<0.001	120.0	<0.001	110.0	78.0	18.0	0.2	0.5	1100.0	66.2	511.9
03/18/97					58.0	<0.175	0.2	0.0	109.0	0.0	102.0	59.0	19.0	<0.040	<0.139		165.4	
04/29/97	2.5	1940	0.0	528.0	37.0	0.4	0.1	<0.001	80.0	<0.001	67.0	49.0	14.0	0.0	0.2	1100.0	59.7	378.1
05/29/97	2.8	1800	0.0	548.0	36.0	0.5	0.2	0.0	85.0	0.0	74.0	46.0	13.0	0.2	0.4	920.0	60.4	397.0
06/16/97	2.7	1920	0.0	633.0	40.0	0.5	<0.030	0.0	91.0	0.0	90.0	54.0	14.0	0.2	0.4	960.0	38.5	292.3
07/28/97	2.7	1740	0.0	533.0	46.0	0.5	0.2	0.0	100.0	0.0	98.0	59.0	14.0	0.2	0.5	1100.0	22.1	141.0
08/29/97	2.7	2150	0.0	757.0	53.0	0.6	0.2	0.0	120.0	0.0	91.0	66.0	16.0	0.2	0.5	1000.0	16.4	149.1
09/25/97	2.7	2280	0.0	868.0	73.0	0.8	0.2	<0.001	160.0	<0.001	150.0	86.0	23.0	0.3	0.6	1500.0	20.0	208.3
10/17/97	2.8	2110	0.0	800.0	67.0	0.7	0.2	0.0	160.0	0.0	140.0	82.0	23.0	0.2	0.6	1800.0	198.0	1900.8
11/19/97	2.8	1880	0.0	707.0	58.0	0.5	0.1	<0.1	120.0	<0.1	130.0	80.0	24.0	0.2	0.4	1600.0	367.0	3113.6
12/22/97	2.5	2065	0.0	606.0	47.0				61.0		116.0	80.0	18.0			1590.0	405.0	2945.2
01/30/98	2.8	1640	0.0	612.0	49.0	0.6	0.2	<0.1	110.0	<0.1	110.0	64.0	18.0	0.2	0.4	1200.0	412.0	3025.7
02/25/98	2.8	1500	0.0	517.0	42.0	0.4	0.2	<0.1	88.0	<0.1	42.0	52.0	76.0	0.2	0.3	900.0	437.0	2711.1
03/27/98	2.7	1650	0.0	503.0	36.0	0.5	<0.1	<0.1	76.0	<0.1	66.0	49.0	14.0	0.1	0.4	840.0	432.0	2607.6
04/28/98	2.9	1590	0.0	524.0	38.0	0.5	0.1	<0.1	83.0	<0.1	68.0	50.0	14.0	0.1	0.4	880.0	200.0	1257.6
05/28/98	2.9	1600	0.0	538.0		0.5		<0.1		<0.1	65.0		12.0	<0.040	0.4	910.0	317.0	2046.6
6/98	2.9		<.1	540.0	33.0	0.4	<.2	<.2	83.0	<.2	68.0	44.0	12.0	<.2	0.4	900.0	366.0	2371.7
7/98	2.8		0.0	563.9	21.0	0.2	0.0	0.0	40.3	<0.1	35.3	23.2	6.3	0.0	0.2	1078.8	292.0	1975.9
8/98	2.8		0.0	727.6	24.0	0.3	0.0	0.0	52.8	0.0	46.3	28.1	7.3	0.1	0.2	1259.9	283.0	2470.9
9/98	4.1	1835	0.0	825.7	30.0	0.3	0.0	0.0	40.3	<0.1	53.6	31.5	7.9	0.1	0.2	2476.8	270.0	2675.3
10/98	2.9	2190	0.0	902.2	67.8	1.0	0.3	<0.1	132.5	<0.1	138.8	81.3	18.7	0.3	0.7	1277.7	246.0	2663.2

Date	pH	Cond.	Alkalinity mg/l	Acidity mg/l	Al mg/l	As mg/l	B mg/l	Ba mg/l	Ca mg/l	Cl mg/l	Fe mg/l	Mg mg/l	Mn mg/l	Pb mg/l	Se mg/l	SO4 mg/l	Ave. Flow gpm	Ac lbs	Ad per day
11/98	2.9	2180	0.0	833.3	73.8	1.0	0.3	<0.1	191.5	<0.1	112.2	87.3	20.2	0.3	0.8	1484.7	235.0		2349.8
12/98	7.2	683	0.0	45.5	73.4	1.1	0.1	0.2	209.2	<0.1	98.1	90.1	20.0	0.1	0.8	1616.4	53.0		28.9
1/99	2.9	2090	0.0	698.3	81.1	0.0	0.1	0.0	155.2	<0.1	172.5	94.3	16.3	0.2	0.5	1095.9	288.0		2413.5
02/19/99	2.8	2530	0.0	701.4	17.5				179.2		77.0	62.5	43.2			3691.6			
04/05/99	3.0	1816	0.0	414.0	35.3				164.4		44.7	42.6	11.7			770.2			
05/07/99	3.1	1891	0.0	529.1	36.3	<.1		0.1	196.2	0.0	50.0	39.9	6.4	0.3	0.4	784.5			
11/30/99	2.9	3250	0.0	555.0	41.2	0.5	0.2	0.0	40.4	0.0	72.0	43.3	3.0	0.2	0.4	2579.0			
	2.9	3250	0.0	559.0	4.8	0.6	0.2	0.0	40.1	0.0	8.6	5.0	1.8	0.2	0.4	2115.0			
12/16/99	2.9	3200	0.0	614.0	46.2	0.5		<0.005	41.6	0.1	134.0	0.7	5.7	435.0	2157.0	0.2			
	2.9	3060	0.0	618.0	46.7	0.5		<0.005	42.4	0.1	139.0	0.6	5.3	437.0	2018.0	0.2			
01/24/00	2.9	4270	0.0	367.0	30.3	0.0	0.2	<0.01	397.0	0.0	30.8	35.8	5.0	<0.001	<0.002	3200.0	10.0		44.0
02/19/00	2.7	2831	0.0	686.4	3.4				404.1		116.6	31.7					10.0		82.4
06/09/00	3.1	2140	0.0	216.7	20.6				417.2		19.9	28.5	0.5				10.0		26.0
07/18/00	3.0	2170	0.0	340.9	19.8				401.0		33.8	28.3	4.5				6.0		24.5
08/17/00	3.3	1864	0.0	226.0	18.8				364.1		29.4	29.6	4.7				7.9		21.4
09/14/00	3.2	1135	0.0	424.1	30.1				358.0		64.9	50.3	8.8				10.0		50.9
10/12/00	3.1	2380	0.0	423.1	31.6				387.8		67.4	55.6	8.0			1473.6	10.0		50.8
11/14/00	2.9	2160	0.0	398.1	34.0				366.3		61.8	52.2	8.2			1820.6	10.0		47.8

Table 3 **Patriot Mining**
Longridge mine discharge- Summary

	Date	pH	Alkalinity mg/l	Acidity mg/l	Avg. Flow gpm	Acid Load lbs per day	% reduction
Pre-grouting	12/96-12/98	3.0	0.0	621.0	190.3	1417.8	
During grouting	1/99 - 9/99	2.9	0.0	585.6	288.0	2023.8	-42.74%
Post-grouting	11/99-11/00	3.0	0.0	425.1	9.2	47.1	96.68%

TASK 5 SUBSIDENCE CONTROL

by Hema J. Siriwardane, Ph.D.

Background

Adequate subsidence prediction, proper usage of damage mitigation techniques, rational mine backfill and mine design are essential prerequisites to control or minimize subsidence potential at abandoned and active mines. Inadequate pillar support due to deterioration of pillars causes the overlying strata to cave into the mine voids. This process proceeds upward and finally reaches the surface. In longwall mining, as soon as the mining operations are completed, surface subsidence might occur within a matter of a few days. In room-and-pillar mining, it takes much longer time for surface subsidence to occur. Subsidence is actually a function of the type of overlying strata, the depth of excavation and the method of mining. The present day commercial requirements have forced higher extraction rates and weaker pillar support in the mines thus, paving the way for larger subsidence potential in undermined areas.

In this study, Finite Element analysis was used for adequate subsidence prediction, and for obtaining the strength requirements of backfill grout. In this study, two actual mines were considered; one is referred to as the Fairfax mine while the other one is referred to as the Longridge mine. Figure 78 shows the location of the two mines in the state of WestVirginia. Computation of surface subsidence due to roof collapse was done at both the Fairfax and the Longridge mine. The analyses included a simulation of near total backfill to partial backfill to study the influence of backfill in the control of surface subsidence. Figures 79, 80, 81 and 82 show the location and area of grouting operations at both mine sites. The cross-sections selected for the Finite Element analyses are also shown in these Figures.

The Fairfax mine which is an active mine, is located in the Preston County in the state of WestVirginia. Due to mine voids approximately 225 feet below the surface, there is considerable potential for subsidence. The Longridge mine which is also an abandoned mine is also located in the Preston County, WestVirginia. In this mine, approximately 395,120 sq ft area of mine voids as shown in Figure 82 is to be backfilled. The mine voids are located at a depth of about 80 feet below the surface.

Objectives

The purpose of this project is to find the strength requirements of grout and grout backfill configurations to be used for mitigation of subsidence potential at the Fairfax and Longridge mines. The Finite Element Analysis was used to compute surface subsidence due to roof collapse, at selected cross-sections at both mine sites. The study also includes a comparisinal study of the effect of 75%, 90% and 95% backfill of mine voids to control surface subsidence.

Figure 78: Map showing the location of Fairfax and Longridge mines in West Virginia

Figure 79 : Locations of cross-sections at Fairfax mine

Figure 80 : Locations of cross-sections at Longridge mine

Figure 81 : Locations of core holes at Longridge mine

Figure 82: Proposed Location For Grouting

Scope of the Study

The scope of the study covers strength calculations for the grout (a mixture of Flyash, Bentonite and Water), which will be used at the Longridge mine site. An estimate of the volume of grout required for backfilling the mine voids made, based on the information extracted from the mine maps is included in this report. A study on the control of subsidence by using complete and partial backfill of mine voids is also presented in the report.

Site Description

Geologic and Geotechnical Characteristics

Figures 83, 84 and 85 show the soil and rock stratigraphy at core hole locations SQ 41, SQ 42 and SQ 43 at the Fairfax Mine; the locations of the core holes are shown in Figure 79. Information pertinent to these core holes was extracted from the bore hole logs obtained from the mining company. The geologic columns at these sites show that the thickness of soft clayey layers overlying the first layers of rock ranges from 19 to 28 feet. The rocks at the bedrock surface appear to be weathered. As shown in Figure 83, there is about 80 feet of weathered shale which can be fairly massive and uninterrupted. Immediately below the top soil is a layer of sandyshale of about 15 to 20 feet in thickness. The coal seam is located at approximately 225 feet below the surface. The thickness of the coal seam is about 5 feet. The coal seam has a dip of about 6° as can be seen in the contours shown in Figure 79.

The ground profile at the Fairfax mine is shown in Figures 86, 87, 88, 89 and 90. The profiles shown in Figures 86, 87 and 88 are along the longitudinal cross-sections A-A, B-B and C-C, respectively. The profiles shown in Figures 89 and 90 are along the transverse cross-sections D-D and E-E, respectively. The ground elevation starts increasing from a distance of 500 feet side and peaks at the center of the mine. The coal contours extended only upto a distance of about 500 feet on either side of the mine in the transverse direction. Therefore the thickness of the overburden could be determined only upto a certain distance on the basis of contour maps. The immediate roof comprises about 10 feet of sandyshale in the vicinity of core hole locations SQ 41 and SQ 42.

Figures 91, 92, 93 and 94 show the type of soils and rock that make up the overburden at core hole locations LR 29, LR 33, LR 34 and LR 42 of the Longridge mine, respectively. The geologic columns at these locations show the presence of fairly thin layers of clayey soils. The rock stratigraphy is not much different from the stratigraphy at the Fairfax Mine. There is about 50 feet of weathered gray sandstone below a 15 feet thick layer of sandyshale. The average thickness of the coal seam is about 7.5 feet, and is located at approximately 80 feet below the surface. The coal seam has a dip angle of about 6° .

Figures 95, 96, and 97 show the ground profile at cross-sections A-A, B-B, and C-C of the Longridge mine, respectively. Cross-section A-A is along the Longitudinal direction of the mine. Cross-sections B-B, and C-C are transverse sections of the mine. The ground surface at the

Longridge mine is essentially a ridge that peaks at the center of the mine. The ground elevation drops rapidly from the center, and regions of outcrop can be seen at distances of about 500 feet from the center of the mine in transverse sections. The immediate roof is a fairly thick layer of weathered gray sandstone.

Figure 83 : Geologic column at location SQ-41 of the Fairfax mine

Figure 84 : Geologic column at location SQ42 of the Fairfax mine

Figure 85 : Geologic column at location SQ-43 of the Fairfax mine

Figure 86 : Ground Profile At Section A-A Of Fairfax Mine

Figure 87 : Ground Profile At Section B-B Of Fairfax Mine

Figure 88 : Ground Profile At Section C-C Of Fairfax Mine

Figure 89 : Ground Profile At Section D-D of Fairfax Mine

Figure 90 : Ground Profile At Section E-E of Fairfax Mine

Figure 91 : Geologic Column at Location LR29 of the Longridge Mine

Figure 92 : Geologic column at location LR-33 of the Longridge mine

Figure 93 : Geologic column at location LR-34 of the Longridge mine

Figure 94 : Geologic column at location LR-42 of the Longridge mine

Figure 95 : Ground Profile At Section A-A Of Longridge Mine

Figure 96 : Ground Profile At Section B-B Of Longridge Mine

Figure 97 : Ground Profile At Section C-C Of Longridge Mine

Material Properties

Table 5 shows the material properties of the rock materials found in the overburden at the Fairfax Mine. The values are typical of weathered overburden. The material properties of the materials making up the overburden at the Longridge mine is reported in Table 6. Since measured values of rock properties at this mine was not available, the material properties were established on the basis of those reported in literature (Ward, 1992).

Literature Review

Subsidence is an adverse surface effect of underground mining operations. Excavation of solid material from the earth's surface disturbs the inherent state of equilibrium of the strata, and results in the creation of voids. Gravitational and/or tectonic forces, act on the strata surrounding the voids, and if the remaining support is insufficient for withstanding the weight of the overburden, the overlying rock mass fractures and collapses in to the voids. The resulting ground movement at the ground surface is termed as surface subsidence.

Information on subsidence prediction becomes more meaningful if some essential parameters such as angle of draw, width-to-depth ratio and slope of the subsidence profile are considered (Moebs, 1982). Table 7 includes definitions of terms frequently used in the literature.

Pillar Stability and Failure

Factors Affecting Pillar Stability

Pillar stability is a function of various factors such as the nature of the geology, the mining geometry, the type and magnitude of underground stresses, and other local effects such as the state of weathering of the overburden and the hydraulic conductivity of the rocks making up the overburden (Horino & Aggson). The overburden at Fairfax and Longridge mines comprises weathered rocks for considerable depths. These rocks have fairly high hydraulic conductivity values of the order 1×10 to 1×10^3 gallons/day/sq.ft (Todd, 1970). The flow of water through these rocks start the deterioration of roof, floor and pillars thus assisting roof collapse (Gray, et.al., 1977; Hartmann & Greenwald, 1941).

Pillar size and shape also affect pillar strength. The size of the pillar that should be tested for strength in laboratory should be representative of insitu pillars and should include all fractures and inhomogeneities. Hustrulid (1976) suggested the use of a critical width of 36 inches. The pillar strength is a function of the ratio of the least lateral dimension to the height (Crouch & Fairhurst, 1973).

Table 5 . Material properties used for the Analysis of Fairfax Mine

Rock Type	Elastic Modulus x 10 ⁶ psi	Poisson's Ratio	Unit Weight pcf	Cohesion c psi	Internal Friction Φ deg	Tensile Strength psi	Coef. of e-pressure at rest k_v
Soil	0.007	0.30	120.0	2	20	2	0.50
Shale	0.960	0.15	160.0	2904.0	33.0	650.0	0.51
SandyShale	2.81	0.257	160.0	4260.0	31.4	15184.0	0.48
Sandstone	5.6	0.15	160.0	2336.0	28.0	851.0	0.46
Coal	0.180	0.30	83.0	1000.0	30.0	207.0	0.48

Table 6 Material Properties Used for the Analysis of Longridge Mine

Rock Type	Elastic Modulus x 10 ⁶ psi	Poisson's Ratio	Unit Weight pcf	Cohesion c psi	Internal Friction Φ deg	Tensile Strength psi	Coef. of e-pressure at rest k_v
Soil	0.007	0.30	120.0	2.0	20.0	2.0	0.50
Shale	0.960	0.15	160.0	2904.0	33.0	650.0	0.51
Sandy Shale	2.81	0.257	160.0	4260.0	31.4	15184.0	0.48
Sandstone	5.6	0.15	160.0	2336.0	28.0	851.0	0.46
Coal	0.180	0.30	83.0	1000.0	30.0	207.0	0.48

Table 7: Definition of some general subsidence related terminology

Parameters that aid in the prediction of subsidence	Definition
Angle of draw	Angle subtended by the edge of the subsidence area with the edge of the mine, measured vertically as shown in Figure 3-1.
Maximum subsidence	Maximum computed value of subsidence at a section.
Subsidence coefficient	It is the ratio of the maximum subsidence to the seam thickness expressed as a percentage.
Width-to-depth ratio	It is the ratio of the width of a mined section to the overburden thickness.
Critical width	The width of a mine section that causes complete subsidence of a point on the surface.

Lowering of groundwater also leads to pillar instability. It is basically due to an increase in effective stress. Pillars are also largely non-homogeneous with different strength and elastic moduli at different locations (Mathur & Mikkilineni, 1982). The existence of non-homogeneities lead to unequal deformation and subsequent instability.

The loading on a pillar is calculated by using the tributary area method (Peng, 1978). The concept involves calculation of the overburden weight in a "tributary area contributed by one half the width of the room on either side of the pillar" (Peng, 1978). The stress distribution on the pillars ranges from a minimum at the pillar periphery to a maximum at the core (Wagner, 1974). Pillars bear the maximum stress at the core due to lateral confinement of the material. As soon as the stresses exceed the yield strength of the pillar, the pillar fails.

Modes of Pillar Failure

Deterioration of pillars reduces its load carrying capacity and this causes eventual pillar failure. There are three common modes of pillar failure (Mathur & Mikkilineni, 1982):

- 1) Pillars undergo bulging at the mid-height if there is significant amount of flow. This allows separation of coal and results in decreased cross-sections.
- 2) The non-homogeneities present in the pillars, in the form of cleats, allow vertical spalling especially at the middle of the pillar as the cross-sectional area is reduced. This results in the formation of a "Waist".
- 3) The third mode of failure is shear failure along the weaker planes.

Pillar failure is a progressive phenomenon, as the excess loads are transferred to the neighboring pillars, thus triggering the collapse of adjacent pillars.

Roof Collapse

Roof collapse triggers surface subsidence. Mine roof is basically designed as a beam capable of withstanding a distributed load (Mathur & Mikkilineni, 1982). The instability in mine roof is a function of various factors such as the characteristics of materials comprising the overburden, the strength and deformation of the immediate and the main roof and fluctuations in moisture and humidity. Roof instability also depends on the presence of non-homogeneities (Mathur & Mikkilineni, 1982). It has been found that the presence of thin layers of shale in the immediate roof causes instability as compared to the presence of stronger strata such as sandstone.

Thus, it can be stated that physical properties, thickness, and the composition of materials in the immediate roof control roof stability. Figure 83 shows that the immediate roof at core hole location SQ-41 of Fairfax mine comprises a fairly thin layer of shale. Due to this nature of the immediate roof there exists a distinct possibility of roof instability at this mine. The main roof above

the immediate roof is a layer of sandstone. There is a possibility for the development an unbalanced stress field at the sandstone/shale interface due to differences in the moduli of the two materials. As compared to the Fairfax mine, the immediate roof at the Longridge mine is a 25 feet thick layer of sandy gray shale and hence there is a lesser possibility of roof instability.

Floor Failure

The thickness, physical properties and the composition of the materials in the underburden affect floor stability (Mathur & Mikkilineni, 1982). Due to weathering, the floor strata becomes relatively soft thus reducing the bearing capacity. Also, the presence of water reduces the floor stability considerably. The basic mode of failure is a bearing capacity failure, when the loads transferred by the pillars exceeds the bearing capacity of mine floor. The pillars in such a scenario, punch through the soft floor layers. Pillar punching is a function of pillar width, with larger pillar widths posing lesser threat of pillar punching.

Progressive Failure

The presence of a weak underburden immediately below the coal seam often results in a bearing capacity failure (pillar punching). The presence of water and non-homogeneties further contributes to this process. Pillar punching results in a gradual transference of loads to adjacent pillars and triggers a progressive failure.

Progressive failure halts as soon as a state of equilibrium is reached through a redistribution of loads. Further deterioration of pillars due to weathering, might start another cycle of progressive failure and the formation of a new state of equilibrium.

Mine Geometry and Layout

The amount of surface subsidence is related to the general geometry and the layout of the mines. Among other things, surface subsidence is related to seam thickness, the depth of coal seam, pillar width, panel width and design layout of room and pillars. Generally, subsidence is higher in mines with thicker coal seams. Subsidence potential is greater if coal is mined out from seams closer to the surface (Mathur & Mikkilineni, 1982). Larger extraction ratios and smaller pillar widths contribute to larger values of subsidence.

Presence of multiple seams also affects subsidence. In mines with multiple seams, the layout of pillars and the thickness of separation influences subsidence. Usually, lesser subsidence potential is noticed if pillars are designed to be directly above one another. In mines where pillar design is haphazard, the thickness of separation plays an important role in providing stability. A minimum separation of 50 feet is essential for stability (Mathur & Mikkilineni, 1982).

Bulking

The immediate and the main roof are subjected to high compressive and tensile stresses. The material comprising the roof might fail due to this stress field, and falls into the voids. This process is accompanied by volume change and is termed 'Bulking'. The subsidence of the surface may be stopped at an intermediate layer if significant bulking occurs in the mine.

In this study, the effect of bulking was considered in the subsidence modeling, by incorporating a bulking factor of 20% in the model. As the maximum subsidence in any mine is not equal to the seam thickness, it is expressed as a percentage of the seam thickness.

In the subsidence model, the prescribed roof displacements are multiplied by the 'roof displacement ratio 'p' to include the effect of bulking. The roof displacement ratio, p, is given by;

$$p = 1 - \alpha$$

where α = bulking factor.

Design Features in Room-and-pillar Mining

Mine Design

In the United States, room-and-pillar-mining has been extensively used in the past (Moebs, 1982). Solid pillars of coal (or mineral) are left as supports for holding the overburden, while extraction is being done. In other words, extraction of coal (or mineral) results in the creation of 'rooms'. The extraction rate in room-and-pillar mining was as low as 30% (Moebs, 1982). However, the rate has gone up in recent times (Moebs, 1982). Subsidence patterns and prediction methods over longwall mining is better understood, and is well documented. However, subsidence patterns over room-and-pillar-mining is somewhat complex and is a function of local geology, overburden thickness, physical properties of materials, presence of non-homogeneities and presence of stress fields (Ward, 1992). The water table as well as subsidence have to be accounted for in the mine design.

Room Design

Mine roof is designed as a beam supported on pillars. This is a criterion used for designing rooms. In other words, the maximum allowable span for stability of mine roof is one of the criteria for room design. Room design should take into consideration that sufficient space should be available for the operation of machinery. The average room dimensions are 40 ft x 40 ft x 5 ft in the Fairfax mine. The room dimensions in the Longridge mine are 30 ft x 30 ft x 7.5 ft as found from the mine maps.

Pillar Design

Pillars are basically designed by estimating the expected pillar loads from the overburden and equating it to the pillar strength. A factor of safety is included to account for local uncertainties in the geology. Some of the important factors that affect pillar design are the stress distribution in the pillars including horizontal stress component, deformation properties of pillars, roof stability, room design and other geologic factors. Pillars can be designed for either complete failure, i.e no yielding of pillars are allowed, or for yielding. Formulas for designing pillars are found in literature.

Grouting

For effective mitigation of surface subsidence, backfilling of mine voids with grout is one of the most practical and economical methods that is available today. As part of the efforts to minimize the adverse environmental impact of abandoned underground mining operations the backfilling of mine cavities with a grout was considered in this study. The grout consisting of a mixture of fluidized bed combustion ash, bentonite and water was selected for this purpose. The common types of grouts that are used include chemical and permanent or cementitious grouts. Chemical grouts are used as support systems to stabilize soil in underground construction projects and are expected to perform for a period of one year. Chemical grouts are further classified as suspension type (ex; slag cement) and solution type (acid reactant). Suspension type grout has higher strength than solution type grout (Yonekura & Kaga, 1992). Strength of these grouts is a function of cement content. Cementitious grouts are used for permanent stabilization of underground projects especially in mining operations.

Effectiveness of Grout as a Function of Rheology

Slump is a measure of grout mobility. Slump of a grout is a function of the shape of the particles and gradation. Grout mobility should be low for effective grouting especially in compaction grouting (Warner, 1992). Another important factor that evaluates the performance of grout is the setting time. Different setting times are necessary for different geologic conditions. Some of the other factors affecting the effectiveness of the grout are rate of injection, low confining pressure in shallow injections, proximity to down slope and presence of water. Lower injection rates, reduction in water content and addition of coarse aggregates increase the effectiveness of grout.

The Use of Bentonite in Grout

Bentonite is a promising sealing material that can be injected in fractures as it has fine particle size, very low permeability and a self-healing ability (Ran & Daemen, 1992). Bentonite increases the adhesive properties of the grout and keeps the grout suspension stable while pumping. Fractures ranging from 9 to 90 microns can be stabilized.

Methods Used in Backfilling

Hydraulic backfilling methods are of three kinds. 1) Controlled Flushing, 2) Blind Flushing, and 3) Pumped-Slurry Injection. Controlled flushing is used in accessible mines. The method

consists of building bulkheads around the periphery for confinement of the fill. One borehole for 4 acres is drilled from the surface to the top of the mine opening. Horizontal dispersal of slurry is done using large 90° pipe elbows. Horizontal dispersion is usually between 300 to 1000 feet. Blind flushing is used in inaccessible or flooded mines. The method consists of injecting slurry material until a conical pile builds up in the mine opening. Blind flushing requires closely spaced injection holes.

Pumped-slurry technique uses energy for injecting material. It is a closed system containing a dynamic suspension of slurry. Water from flooded mines may be used. The success of this technique depends on uninterrupted supply of energy sufficient to create enough velocity to keep the particles in suspension. This method requires less number of boreholes and is easier for use in inaccessible locations. Depending on strength requirements, selection of total or partial backfill can be chosen.

Methodology

Strength calculations are based on the fact that the grout after being injected into the mine void, should develop sufficient compressive strength to withstand the stresses caused by the overburden, along with the pillars. In effect, it should play the role of the excavated coal with its full strength. Traditionally, the vertical stress at the level of coal seam is calculated by multiplying the unit weight of the materials comprising the overburden with the respective thicknesses.

Source Code for Finite Element Analysis

The FORTRAN source code FEAGMEC (Finite Element Analysis in Geomechanics) written by Siriwardane, 1983 (Siriwardane, 1983, 1988, 1991, 1993) was used in this analysis. The output gives information on the stresses due to different loading conditions at any location in the ground. The output also provides the vertical and horizontal displacements due to overburden stresses, at any specified location in the ground.

Steps Involved in the Analysis

The steps involved in the analysis is given below.

- The ground profile, the geologic column and the material properties are established for a selected cross-section at the mine. A cross-section is selected for the analysis.
- The cross-section is discretized into a domain containing a finite number elements. The elements are generally four noded iso-parametric elements.
- The input for the program comprises the material properties, the nodal coordinate information, the element connectivity and the type of loading on the pillars.
- The input also includes information pertaining to the specified location in the ground where the stresses and the displacements are desired.

Finite Element Formulation : a Brief Review

A brief review of the steps involved in the Finite Element formulation is given below. Primarily, there are eight steps in a Finite Element Formulation. These steps are: 1) discretization, 2) selection of approximate model functions, 3) definition of the gradient-unknown relationship, 4) derivation of element equations, 5) derivation of global equations and the application of boundary conditions, 6) solution of primary unknowns, 7) solution of secondary quantities, and 8) interpretation of results. More details of the Finite Element formulation can be found elsewhere (Zienkiewicz, 1977).

Derivation of Element Equations

The governing differential equation in stress deformation problem for a one-dimensional element can be written as:

$$AE \frac{\delta^2 U}{\delta x^2} = f(x)$$

where A is the area of cross-section, E is the elastic modulus and $f(x)$ is the force acting on the body.

Element equations corresponding to this equation can be obtained by using one of the two methods.

- a) Variational methods
- b) Residual methods

Considering the variational methods, the element equations can be derived as:

$$[k](q) = (Q)$$

where $[k]$ is the stiffness matrix for a single element, and (Q) is the load vector.

$$[k] = \iiint_v [B]^T [C] [B] dV (Q) = \iiint_v [N]^T (\gamma') dV + \iint_s [N]^T [T'] dS + (P)$$

The stiffness matrix obtained for each element above is assembled to form the global stiffness matrix. If the chosen element has n degrees of freedom, then the size of the element stiffness matrix would be of the order $n \times n$.

The next step after the assembly of the local stiffness matrices, is the imposition of boundary conditions. The solution for primary unknowns can be obtained by using Gaussian elimination, after imposing the boundary conditions. The values of the primary unknowns obtained can be used for finding the secondary unknowns such as strain and stress at the element level. The results obtained are interpreted with the aid of computer graphics.

Results and Discussion

A quantitative assessment of subsidence potential beneath abandoned mines is essential for studying the adverse surface effects of underground abandoned mining operations. Reasonably accurate prediction of subsidence potential is possible with growing usage of numerical methods like the Finite Element method (Kutuk, et.al., 1994; Siriwardane, 1988, 1991, 1993). In this project, the subsidence computations are done for a two-dimensional idealization of the mine configuration. In other words, a plane strain analysis is performed for subsidence calculations at selected cross-sections.

Subsidence computations at both mine sites were done at selected cross-sections shown in Figures 79 and 80. The computations were done for 75%, 90% and 95% backfill of mine voids. Figure 98 shows grout backfill configurations. A bulking factor of 20% was considered to incorporate the effect of bulking of fallen overburden rock into mine voids.

Subsidence Computations at Section D-D of Fairfax Mine

The cross-section D-D of Fairfax mine is shown in Figure 79. The pillar configuration along this cross-section is shown in Figure 99. The procedure involved in the computations includes the discretization of the domain following the steps outlined in section 4.2. Roof collapse is modeled by prescribing displacements at the roof nodes. The Finite Element mesh used in this analysis is shown in Figure 100.

Due to symmetry, only one half of the mine is considered in the analysis. Figure 101 shows the computed subsidence profile at section D-D of the Fairfax mine. The profile shown in this Figure are for 75%, 90% and 95% backfill of mine voids.

Subsidence Computations at Section E-E of Fairfax Mine

Section E-E is a transverse cross-section as shown in Figure 79. The Finite Element mesh for this cross-section is shown in Figure 102. The computed subsidence profile is shown in Figure 103. At both cross-sections the values of maximum subsidence was almost equal for 75%, 90% and 95% backfill of mine voids. For 50% recovery of coal, the value of maximum subsidence was computed to be equal to 2.0 feet i.e. 40% of seam thickness of 5 feet.

Subsidence Computations at Section B-B of Longridge Mine

Pillar configuration at section B-B of Longridge mine is shown in Figure 104. The Finite Element mesh for this section is shown in Figure 105. The analyses was conducted in two phases. Initially, the ground profile was assumed to be horizontal. The computed subsidence profile for this analysis is shown in Figure 106. In phase two, the ground profile is the actual one as shown in Figure 96. The Finite Element mesh used in this analysis is shown in Figure 107. The computed subsidence profile is shown in Figure 108. The maximum value of subsidence was computed to be

equal to about 1.75 feet, i.e., is 23% of the seam thickness of 7.5 feet in both cases. The ground profile apparently had no significant influence on the maximum value of subsidence.

Subsidence Computation at Section C-C of Longridge Mine

The location of section C-C is shown in Figure 80. The pillar configuration is shown in Figure 109. The Finite Element mesh used in this analysis is shown in Figure 110. The computed subsidence profile is included in Figure 111.

In the analyses conducted at both mines, pillar dimensions were assumed as 20 ft x 20 ft to see the effect of uniform pillar dimensions. The assumed pillar configuration for Fairfax mine is shown in Figure 104. The computed subsidence profile for this analysis is shown in Figure 106. The assumed pillar configuration at Longridge mine is shown in Figure 112 and the computed subsidence profile is shown in Figure 113. The value of the maximum subsidence due to uniform pillar dimensions was found to be slightly higher than the maximum value of subsidence computed due to actual pillar dimensions.

Figure 98 (a): 50% Backfill

Figure 98 (b): 75% Backfill

Figure 98 (c): 90% Backfill

Figure 98 : Actual Pillar Configurations at the Fairfax Mine Site

Figure 99 : Pillar Configurations at Section D-D of Fairfax Mine Site

Figure 100 : Finite Element Mesh for Section D-D at Fairfax Mine

Figure 101 : Computed Subsidence Profile at Section D-D of Fairfax Mine Site

Figure 102 : Finite Element Mesh for Section E-E at Fairfax Mine

Figure 103 : Computed Subsidence Profile at Section E-E of the Fairfax Mine Site

Figure 104 : Assumed Pillar Configurations at Fairfax Mine Site

Figure 105: Finite Element Mesh for Section B-B with Horizontal Ground at Longridge Mine

Figure 106 : Computed Subsidence Profile at Section B-B of Longridge Mine Site

Figure 107: Finite Element Mesh for Section B-B with Horizontal Ground at Longridge Mine

Figure 109: Pillar Configurations at Section C-C of Longridge Mine Site

Figure 110: Finite Element Mesh for Section C-C with Horizontal Ground at Longridge Mine

Figure 111: Finite element mesh used in the analysis of section c-c of the Longridge mine

Figure 112 : Assumed Pillar Configurations at Longridge Mine Site

Figure 113 : Computed Subsidence Profile at the Longridge Mine Site

Grout Strength Requirements

Strength requirements of grout may vary with the method used for backfilling mine voids. The requirements are different for total backfill, partial backfill and for grout columns. Usually grout strength is estimated by calculating the stress caused by overburden at the level of coal seam by multiplying the thickness of the overburden with the respective unit weights. Insitu strength requirement varies from place to place and is a function of the following factors:

- 1) Local geology
- 2) Depth of coal seam
- 3) Stress orientations and Stress concentrations and,
- 4) The method used for placing grout.

Strength requirements found in the laboratory may have to be multiplied by a factor of safety to account for these uncertainties. Without accounting for any of these uncertainties, some preliminary calculations were done for calculating the grout strength requirements. Table 8 shows some of these preliminary study at Fairfax mine. Results of some preliminary calculations for Longridge mine are included in Table 9.

Summary and Conclusions

The results obtained from the Finite Element Analysis at the Fairfax and Longridge mines can be summarized as follows:

- The maximum value of the subsidence computed at the Fairfax mine was almost equal to the prescribed roof displacements in this analysis.
- The subsidence profiles were obtained for varying backfill configurations, i.e. for 75%, 90% and 95% backfill of mine voids.
- The larger the grout volume, the lesser the subsidence, as can be seen from the computed profiles.
- The effect of ground topography on the subsidence profile was not significant as can be seen from the computed values of maximum subsidence at section B-B of Longridge mine for both assumed and actual ground topographies.
- Preliminary calculations for grout strength requirements were performed by calculating the stresses at the level of the coal seam due to overburden height. In the present phase, detailed calculations for required grout strength are underway.

The required strength of grout should be multiplied by a factor of safety to account for the various uncertainties such as potential stress concentrations and segregation of the grout mix during placement as listed previously in this report.

Table 8 Expected Stress Levels On The Grout At Fairfax Mine Site

Core Holes	Stress In psf	Stress In psi
SQ-41	34495	239.54
SQ-42	35035	243.29
SQ-43	32815	227.88

Table 9 Expected Stress Levels On The Grout At Longridge Mine Site

Core Holes	Stress In psf	Stress In psi
LR-29	8982.5	62.37
LR-33	10488.5	72.83
LR-34	5456.25	38.51
LR-42	14905.5	103.51

TASK 6 CONTAMINANT TRANSPORT
TASK 13 WATER QUALITY MODELING
by Hema J.Siriwardane, Ph.D.

The Final Report for these tasks is included in Appendix A at the end of this report.

TASKS 7, 8 AND 9 PHASE II OPERATIONS

by Paul F. Ziemkiewicz, Ph.D. and D. Courtney Black

Introduction

The test injection of the grout developed in Phase I was conducted at the Fairfax Mine in the Preston County, West Virginia on May 20, 1996. One thousand cubic yards of grout was injected over a period of 4 days. A recipe of 34% water content (this is the amount added at the site and does not include the moisture present in the ash that reached the site) and 5% bentonite was adopted for the injection. A 40 yd³/hr capacity concrete pump was used to pump the grout through a 4" diameter injection hole.

The results of the injection were very encouraging and the grout exhibited the required standards of flowability and stability. A spread of 600 ft of running length of the mine corridor 18 feet wide was achieved from a single injection hole. The height to which the grout build up was about 2.5 feet at the injection point to about 6" midway through the length. The onsite spread tests showed a spread of 14". The test also shows that the dip on the floor had a significant influence on the direction of flow.

Field Methods

The cFBC ash was delivered to the Fairfax site in bottom dump (clam shell) trailers. The ash was placed in wind rows and then piled by a front end loader. The ash was placed into a hopper with the front end loader. The hopper emptied onto a short beltline that dumped into a mixer truck (See Photo 1). Bentonite was added to the beltline by a small screw conveyor (See Photo 2). Water was added to the mixer trucks as the dry materials were delivered. Once the materials were thoroughly mixed, the mixer truck poured the grout into a piston style concrete pump which delivered the grout to the borehole (See Photo 3). The grout dropped 230 feet vertically into the mine and began free flowing throughout the mine void (See Photos 4, 5 and 6).

Before entering the field, a mix design was decided upon in the laboratory. This design was modified, given field conditions (namely % of water in fresh FBC ash). The final mix design used for the 1000 yd³ field test is as follows:

14,700 lbs. cFBC ash	
800	lbs. Bentonite
1,150	gallons (9593 lbs.) Water

This mix was chosen due to its high flowability. The only variance from the laboratory design is 4% in water content. The water content of the field mix is approximately 38%. Note that water content for this project is defined as weight of water divided by total weight of the mix.

Photo 1 - Grout Injection at Fairfax Mine.

Photo 2 - Ash is loaded into a hopper and onto a belt conveyor. Bentonite is added to the belt via a screw conveyor.

Photo 3 - Grout is loaded into a piston style concrete pump and placed underground at a rate of 40 cubic yards per hour.

Photo 4 - Grout is pumped into the mine and free flows to fill the mine void.

Photo 5 - Grout is self leveling and free flows to fill the void.

Photo 6 - The grout flowed around pillars and achieved a distance of over 600 feet from the injection borehole.

Observations of Grout Flow from Underground while Injecting

The following observations were made by Dr. Paul F. Ziemkiewicz of the National Mine Land Reclamation Center at West Virginia University. Dr. Ziemkiewicz was underground while the first four truckloads of grout were being injected.

Injection began at 9:35 am, 21 May 1996. Johnnie Nichols, mine superintendent and Paul Ziemkiewicz were in the mine, 75 ft. from the injection borehole when injection began. FBC ash slurry was injected in batch mode from two alternating cement trucks on the surface. Each pour comprised about 9-10 cu yds of slurry. Each pour lasted about 20 minutes and the interval between pours was 5-10 minutes.

Pour #1 flowed 75 feet from the borehole. At the downstream end of the ash lobe its depth was about 1 inch. At the borehole the depth was about 2 inches. The ash front continued to advance until injection stopped. The slurry was very fluid, finding and progressively filling low spots. It formed a leveled channel between 1 to 2 foot wide with the narrower widths associated with higher flow rates over constrictions and overfalls. In incompletely filled headings, slurry flowed down the center line of a channel which was semicircular in cross section, ~4 foot wide and 3 inches high at the center line. These channels eventually became occluded, at which side channels would break out and initiate a new lobe.

Pour #2 advanced a further 75 feet. Upon the subsequent pours, the slurry front remobilized with a roughly 1 minute lag time, initiated by a roughly ½ inch wave which propagated in line with the axis of advance. The wave was parabolic with its apex at the downstream end. The second pour did not ride over the first pour, rather it displaced it from the upstream end pushing the entire mass forward. Shale rocks (4-6 inches long, 2 inches square) were observed carried along with the slurry. Slurry velocity was 12 inches/sec at overfalls and more typically 6 inches/sec on the level floor.

Pour #3 advanced only another foot or so, but it spread out to fill two headings from pillar to pillar (18 ft heading width). At the end of pour #3 the thickness at the downstream end was 2 inches. A zone of bleed water about ½ inches was observed at the downstream end of the slurry. Between pours, when the slurry advance stopped a thin (1-2 mm) layer of bleed water could be seen moving slowly along the top of the slurry. At the end of Pour #3 slurry depths were 2 inches at the downstream end, 6 inches in a low spot at the first spad (center of intersection) and 4 inches at the borehole.

There appeared to be some particle size segregation with higher sand contents at the borehole and in the main channels with more fines at the downstream end and in the side channels and bays.

Slurry flowed down the 1-2% slope of the mine floor. It would dam behind roof falls and floor irregularities then flow over or around. Pouring continued until 5:30 pm on the first day.

About 125 cubic yards of slurry were injected on the first day.

The slurry was warm to the touch but not uncomfortable (like bathwater). It generated a good deal of vapor. Also, since AMD treatment water was used to make up the slurry, and the AMD was treated with NH_3 , contact with the lime caused deprotonation of the ammonium ion and release of more than perceptible amounts of NH_3 .

Day Two

Results of the previous day's injection: About 5,400 sq. Ft. of mine floor were covered by the end of the first day to an average depth of about 8 inches. While floor elevation varied in the order of 1-2 ft from roof fall, the slurry had the effect of leveling to an almost planar surface. From an almost watery consistency the first day the slurry had set up slightly to a Jell-O consistency. At the end of the day one the injection crew ran about 800 gallons of water down the line to clean out the pump. This flushed out the slurry channels leaving them free for the next day's slurry.

Witnessing the injection underground on day two were: Johnny Nichols, Fairfax Fuels; Jeff Kelley, and two engineers from Anker Energy, Inc; Dwayne Maust and Dave Bayles from Patriot Mining, Inc.; Mike Main and Paul Ziemkiewicz from WVU.

Injection began at 7:30 am and while deforming the day one slurry in the immediate vicinity of the injection borehole, the new slurry flowed on top of the old slurry. It did not remobilize the old slurry to any significant extent.

Since the FBC ash had been allowed to sit out overnight in a thunderstorm, it had developed a crust. This was broken up into 2-3 in diameter chunks which were transmitted with the slurry. These could be seen floating by in the slurry channel.

With well developed channels, flow proceeded in surges particularly at constrictions and overfalls. Velocity would decelerate over a period of minutes, stop for a second or so, then release in a surge. The flow would then spread out evenly behind a small (1 in) wave below the overfall.

Injection proceeded to 24 May 1996 until 1,000 cu yds were placed in the mine. The next in-mine visit occurred one week after injection on 31 May 1996.

One Week After Injection

On 31 May 1996 a party consisting of: Johnny Nichols, Fairfax Fuels; Jer-Yu Chang, Scott Renninger, U.S. Department of Energy; Milton Wu, Bruce Leavitt, CONSOL, Inc; Jeff Young, WV Public Radio; Ed Lyons, Maryland DER; Jeff Skousen, Courtney Black, Thirupathi Reddy and Paul Ziemkiewicz, WVU-NMLRC inspected the mine.

By this time the slurry had solidified to the extent that samples had to be chopped out with an entrenching tool. In spots of > 6 inches of slurry it was still moist 3 inches from the top. The slurry had developed cracks which penetrated about 4 inches. These tended to run normal to the direction of ash flow. The ash had flowed about 550 ft from the injection borehole and surrounded four pillars. Headings were filled pillar to pillar and some rooms were filled with 2 ft of slurry (the roof was 4 ft high). At the injection borehole the slurry had coned but was still about 10 inches from the roof.

Task 11 Phase III Grouting Operations **by D. Courtney Black and Paul F. Ziemkiewicz, Ph.D.**

Phase III of the project began in January 1997. The volume of grout required to fill the Longridge Mine was estimated to be 50,000 cubic yards. The use of bentonite in the grout mix at 3 to 5% was a concern for due to cost but the gains in flowability made the logistics of setting up on a few injections boreholes worth the investment. As plans were being made for the start of the injection program it became apparent that the volume of FBC ash needed to create the grout volume would be problematic. Since the inception of the project, FBC ash has proven an excellent alkaline amendment to toxic materials encountered during the surface mining of coal in Northern West Virginia. Anker Energy, the Industry partner on this project, was using 100% of the FBC produced at MEA to prevent acid mine drainage in the active mining operations. At this point alternative grout mixes were being considered.

Lessons learned in the earlier phases of the project led the team to a rheological approach to new grout mixes. Coal combustion products could be used in grouts and flowable fills but in order to fill an abandoned underground mine the grout must be first flowable and second harden to support the load of the overburden. A hardened grout must also be chemically stable so as not to leach any metals that may be contained in the ash. A program was undertaken to develop a new grout using coal combustion products locally available and hardening agents, if any, that were also locally available.

The first candidate ash was an FBC fly ash that Anker Energy had backhauled from a power plant in Florida. Initial tests were favorable but there was not enough of the ash on hand to complete a full scale injection at Longridge. The second candidate ash was a high loss on ignition ash that Anker had backhauled from an industrial boiler in Syracuse, New York. Again, initial flowability, strength and leaching test were favorable but not enough of the ash was available for a full scale test. The third ash considered was the Class F ash from the Albright Power Station owned and operated by Allegheny Power. The ash was readily available but was not self hardening. Several lime products and cementing agents were tried with the ash. Finally a mix of Class F ash and cement kiln dust (CKD) from Capitol Cement Corporation was selected. The CKD provided the hardening while the Class F ash aided the flowability and pumpability of the grout. The content of CKD could be varied to produce varying flowability and hardness.

The final mix design selected consisted of 1:1 solids to water ratio with 75% of the solids by weight being Class F ash and 25% by weight CKD. It was estimated that the mix would perform similar to the Phase II grout (flow greater than 600 feet from a single borehole).

Figure 114 is a map of the Longridge Mine and the injection boreholes that were installed for Phase III operations. The holes are located on approximately 200 foot centers. The injection scheme was to start injecting at the top of the mine (holes 10,11 and 12), monitor grout flow through the remaining holes and inject in the lower holes as needed to fill the mine void.

Injection began in late January 1999. The first hole to receive grout was hole number 10. A total of 3,000 yards was installed until it was noticed that the grout was communicating with the auger hole. The auger hole was where the water quality monitoring had occurred since 1996. Note that the distance from hole 10 to the auger hole is approximately 2000 feet.

The grout that was communicating with the auger hole was very fluid and it was determined was actually the bleed water from the grout being transported to the auger hole by the acid mine drainage that was flowing in the bottom of the mine void. At this point the grouting operation was stopped until the water flow could be managed.

It was decided to employ the Burnett Ejector that was developed during this project to install a pneumatic barrier at hole number 9. Grouting would then ensue in Hole 10 to fill the void up-dip of hole 10. The barrier was to be made of crusher run limestone, a readily available product in the area. A new borehole was drilled in the heading adjacent to hole 9 so that both headings could be blocked and the upper cell filled with grout. Each barrier consisted of 200 tons of crusher run limestone. The Office of Surface Mining lent the use of the down hole camera to ensure that the barriers were intact before grouting began above in the upper cell. A total of 12,000 cubic yards of grout was installed in the upper cell of the Longridge Mine. Once the barriers were intact, the flow at the auger hole decreased to below 10 gpm. Grouting continued in hole 8 after the filling of the upper cell. At this point, over 20,000 cubic yards of grout was installed in hole 8 before the grout began to again communicate with the auger hole. Additional holes were drilled near borehole 15 but the roof had collapsed in that area and pneumatic barrier were not installed. A grout mix consisting of a solids to water ratio of 1:1 with the solids being 50% Class F and 50% CKD was installed in the new holes. Grouting continued in hole 8 until September 1999 when the budget was exhausted.

Conclusions

The Phase III grouting operations were successful in abating acid mine drainage from the Longridge Mine. The acid load of the onridge discharge was reduced by 97%. A total of 43,000 cubic yards of Class F and CKD grout was installed in the Longridge Mine. The pneumatic barriers are critical to ensuring full contact grouting. No adverse environmental effects have been observed from the grouting of the Longridge Mine.

LEGEND

Standing Pool

Puddles, Drips Flowing

Puddles

No Water

Total Obstruction

40% - 60% Obstruction

<15% Obstruction

Holes that were used to generate structure:

Injection Holes: 2, 3, 4, 5, 6, 8, 10, 11, 12, 13, 14, & 15.

Existing Holes: LR-15, LR-16, LR-18, LR-19, LR-20, LR-21, LR-30, LR-42, & LR-43.

Surveyed points: 7 points that were surveyed from Triad, 3 points at 1860' Elev.

Figure 114. Map of Longridge Mine.

TASK 12 ECONOMIC ANALYSIS
by Mackenzie Burnett, P.E.

The cost of disposal of coal ash pneumatically through boreholes in underground mines was presented under Task 3B and 3C. The cost per ton ranged from about \$8 per ton to \$11.50 per ton when the cost of the material was zero. The lower cost was for a material that could be transported in pneumatic bulk trucks.

In attempting to compare the cost of pneumatically stowing CCBs in underground mines with simply disposing of the coal ash in a landfill, it became apparent that the cost of a landfill was very site and situation specific. Landfill design is based on the type of waste to be put in the landfill. The liner and cap system can be as little as - clear the ground of vegetation, put the waste in and cover it up with dirt - to a double composite liner system. The cost of this type of landfill has a wide range of cost which is dependent on the waste that will be placed in it. Historically this cost can vary from as little

as \$1.00 per ton to as much as \$50.00 per ton. More typically, however, the cost of disposing of ash in a landfill will range from \$5.00 to \$15.00 per ton. The cost includes landfill construction, operation, maintenance and regulatory compliance. Typically a landfill designed for ash need only have the topsoil removed, ash deposited and the top soil replaced. There may be other requirements depending on the analysis of the particular ash. Soil may need to be mixed in with the ash as it is placed. As an agricultural property, the long term value of the property may not be changed appreciably. As a site for future building, the land will have decreased value due to the need to address the ability of the ash laden land to support a structure. The actual value of the land is a function of the market. Each case is dependant on the specific site and conditions. However, land value may be enhanced by development of the landfill site. For example

Summary Cost of Stowing 4.8E6 yd ³ in an Underground Mine				
operating	material	total	total cost	total cost
cost	cost	cost	per ton	per ton
			with mat	w/o mat
\$	\$	\$	\$/ton	\$/ton
operating through 2 boreholes at once				
2,677,782	4,624,440	9,900,222	21.41	11.41
2,911,248	5,027,610	10,763,358	21.41	11.41
3,177,062	5,486,850	11,746,412	21.41	11.41
3,511,012	6,063,570	12,981,082	21.41	11.41
4,016,689	6,936,660	14,850,349	21.41	11.41
operating through 12 boreholes at once				
2,089,040	4,624,440	9,311,480	20.14	10.14
2,271,385	5,027,610	10,123,495	20.14	10.14
2,478,377	5,486,850	11,047,727	20.13	10.13
2,738,837	6,063,570	12,208,907	20.13	10.13
3,133,143	6,936,660	13,966,803	20.13	10.13
operating through 12 boreholes at once				
1,774,273	0	3,629,773	7.85	7.85
1,929,169	0	3,946,669	7.85	7.85
2,105,037	0	4,307,037	7.85	7.85
2,326,168	0	4,759,168	7.85	7.85
2,661,892	0	5,445,892	7.85	7.85

Pennsylvania Power and Light developed a golf course on the site of a landfill. Proper management of the site and the ash can result in soil improvement. Generally though, the land is less valuable after a landfill is complete. Allegheny Power has bought land next to a power plant, so the variable cost of disposal can be as low as \$1 per ton. In Philadelphia and Boston, disposal can be as high as \$50 per ton.

Since trucking cost alone to a landfill can run \$1.42 per mile for a truck loaded with 30 tons of ash, a 20 mile round trip will cost \$0.95 per ton for transportation alone. Therefore the farther the landfill is from the power plant the higher the cost and the more incentive there is to consider other means of disposal. Finding a market for the ash is, of course, the best use of the ash. The various state regulations all encourage using the CCBs in a variety of ways such as:

- 1) as a raw material in manufacturing a final product;
- 2) as a stabilization/solidification agent for other wastes that will be disposed;
- 3) as a part of a composting process;
- 4) in uses subject to USEPA procurement guidelines;
- 5) for extraction or recovery of materials and compounds in CCBs;
- 6) as an anti-skid material or road preparation material;
- 7) for use in mine subsidence stabilization, mine fire control, and mine sealing;
- 8) As an additive in commercial soil blending operations, where the product will be used for growth of ornamentals (no food crops or grazed land);
- 9) as daily cover at a landfill;
- 10) as structural fill, defined as an engineered use of waste material as a building or equipment supportive base or foundation and does not include valley fills or filling of open pits from coal or industrial mineral mining;
- 11) as pipe bedding, for uses other than transport of potable water;
- 12) as a construction material for roads or parking lots (subbase or final cover).

Figure 1 shows how the distance a landfill or underground mine site is from a power plant effects the cost. Since trucking cost is about the same for landfill disposal or underground mine disposal, the cost of operating the landfill is the primary factor. The study on underground disposal resulted in a cost of about \$8, thus landfill costs in excess of \$8 would make underground disposal feasible, assuming an underground site was available. Since landfills are site specific, each case must be evaluated separately. Pneumatically stowing of ash in underground mines in some cases may very well be the least costly means of ash disposal, especially when the benefits of reduced or eliminated acid run-off and mine subsidence are gained..

Cost of disposal for landfill and through boreholes into underground mines.

TASK 14 GENERAL SUMMARY OF STATE REGULATIONS

by Mackenzie Burnett, P.E.

For consistency, this report utilizes the term CCBs. The term is intended to generically refer to fly ash, bottom ash, boiler slag, FGD sludge or FBC material. The reader must recognize that each state has different approaches to classification of CCBs and that these respective classifications may limit or expand allowable uses of CCBs. For example, in Pennsylvania CCBs are referred to as "coal ash" which is defined to include only fly ash, bottom ash and boiler slag. Conversely, some states include within the definition of CCBs, wastes which have been combusted with other materials, such as petroleum coke, tire derived fuel and/or wood. In some cases these distinctions are noted herein. However, the reader should not assume that use of the term CCB infers that all types of CCBs are included within the scope of a particular state's regulations.

CCBs are generally exempt from hazardous waste regulations and the states have elected to regulate these materials as solid, special or industrial wastes. States that do not exempt CCBs from hazardous waste regulations require testing to determine hazardousness, and if shown to be non-hazardous, the CCBs are regulated as solid waste.

Most states currently do not have specific regulations addressing the use of CCBs and requests for CCB uses are handled on a case-by-case basis or under generic state recycling laws or regulations. Many states have "generic" laws and regulations which authorize limited reuse and recycling of hazardous and/or solid wastes. These generic laws do not apply specifically to CCBs or any other materials. In general, under these regulations, materials are not considered solid wastes when they can be recycled by being:

Used or reused as ingredients in an industrial process to make a product, provided the materials are not being reclaimed;

Used or reused as effective substitutes for commercial products; or

Returned to the original process from which they are generated, without first being reclaimed. The materials must be returned as a substitute for raw materials feedstock, and the process must use raw materials as principal feedstocks.

The following materials remain regulated solid wastes, even if the recycling involves use, reuse or return to the original process:

- Materials used in a manner constituting disposal, or used to produce products that are applied to the land;
- Materials burned for energy recovery, used to produce a fuel or contained in fuels ;
- Materials accumulated speculatively; or
- Inherently waste-like materials.

In addition, there is little consistency amongst the states regarding use of CCBs in mine applications. Some states have detailed regulations for reuse of CCB's in mine applications (which are discussed herein). Other states address this common use of CCBs by reference to fly ash and FGD material as materials which may be permitted as "discharges" to the mine upon approval by the state mining agency.

In general, the legal and technical requirements for mine applications are complex. For this reason, these regulations are not discussed in detail but are noted so further research can be done in the event the reader is interested in the potential application of these regulations to a proposed project.

States without formal CCB use regulations or guidelines often encourage the use of coal fly ash in cement and concrete applications and products. Additionally, state highway departments are required by the Federal Highway Administration to have specifications conforming to federal procurement guidelines for cement and concrete containing coal fly ash for federally funded projects.

A number of states have adopted laws and regulations or issued policies and/or guidance specifically pertaining to CCB use. The CCB uses authorized within these states vary widely. Some states authorize liberal use of CCBs, while others authorize CCB use only in limited applications. In addition, the level of regulatory control and oversight varies significantly. CCB uses presenting the greatest concern to state regulators are those which involve land application such as use of CCBs in agricultural applications, structural fills, mine applications and embankments. Some states consider these applications to be waste disposal and not reuse or recycling.

Finally, a few states have elected to adopt "industrial solid waste beneficial use" rules intended to authorize use of a variety of material such as coal ash, paper mill sludge and foundry sand.

In summary, laws, regulations, policies and/or guidance authorizing at least limited CCB reuse have been adopted in the following states: Alabama, Alaska, Arkansas, California, Colorado, Delaware, Georgia, Illinois, Indiana, Iowa, Kentucky, Maine, Maryland, Massachusetts, Michigan, Missouri, Nebraska, New Hampshire, New Jersey, New Mexico, New York, North Carolina, North Dakota, Ohio, Oklahoma, Pennsylvania, South Carolina, Tennessee, Texas, Utah, Vermont, Virginia, West Virginia and Wisconsin

Vermont

Under Vermont regulations, fly ash, bottom ash, slag, and flue gas emission control waste are exempt from regulation as hazardous waste. Reuse of CCBs is not specifically authorized under Vermont law. Vermont has adopted generic recycling regulations; however, these regulations may not necessarily exempt a material from regulation.

Detailed Review of CCB Regulations

Under Vermont regulations, fly ash, bottom ash, slag and flue gas emission control waste are exempt from regulation as hazardous waste. VT. REGS. HWM §§7-203.

Currently, reuse of CCBs is not specifically authorized under Vermont law or regulations. Vermont has adopted generic recycling regulations, however, these regulations may not necessarily exempt a material from regulation.

Ohio

Under Ohio regulations, fly ash, bottom ash, slag, and flue gas emission control waste

generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste. Reuse of CCBs is not specifically authorized under Ohio law; however, reuse of "non-toxic" CCBs is authorized under policy documents issued by the Ohio Environmental Protection Agency (OEPA), Division of Water Quality. Non-toxic CCBs may be reused

- 1) as a raw material in manufacturing a final product;
- 2) as a stabilization/solidification agent for other wastes that will be disposed;
- 3) as a part of a composting process;
- 4) in uses subject to USEPA procurement guidelines;
- 5) for extraction or recovery of materials and compounds in CCBs;
- 6) as an anti-skid material or road preparation material;
- 7) for use in mine subsidence stabilization, mine fire control, and mine sealing;
- 8) As an additive in commercial soil blending operations, where the product will be used for growth of ornamentals (no food crops or grazed land);
- 9) as daily cover at a landfill;
- 10) as structural fill, defined as an engineered use of waste material as a building or equipment supportive base or foundation and does not include valley fills or filling of open pits from coal or industrial mineral mining;
- 11) as pipe bedding, for uses other than transport of potable water;
- 12) as a construction material for roads or parking lots (subbase or final cover); and
- 13) other single beneficial uses of less than 200 tons. Certain guidelines may apply to the above uses.

Detailed Review of CCB Regulations

Under Ohio regulations, fly ash, bottom ash, slag and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste. OHIO ADMIN.CODE §§3745-51-04.

Currently, reuse of CCBs is not specifically authorized under Ohio law or regulations, however, reuse of "non-toxic" CCBs is authorized under policy documents issued by the Ohio Environmental Protection Agency ("OEPA"), Division of Surface Water dated November 7, 1994. Under OEPA policy, non-toxic CCBs may be reused:

As a raw material in manufacturing a final product (eg. grout, flowable fill, lightweight aggregate, concrete block, bricks, asphalt, roofing materials, plastics, paint, glass, fiberglass, ceramics, blasting grit and other non-land application products;

As a stabilization/solidification agent, for other wastes which will be disposed;

As part of a composting process when the process is performed in accordance with applicable regulations;

In uses subject to USEPA Procurement Guidelines;

For the extraction or recovery of materials and compounds in CCBs;

As an anti-skid material or road preparation material, if such use is consistent with Ohio Department of Transportation specifications;

For use in mine subsidence stabilization, mine fire control, and mine sealing when authorized

by the Ohio Department of Natural Resources (Note: mine reclamation is not pre-authorized);

As an additive in commercial soil blending operations, where the product will be used for growth of orientals, when the waste constitutes no more than 50% of the mixture, and if the waste does not meet 5 times Ohio's primary drinking water standards, the mixture may not be applied to grazed pastures, home/vegetable/fruit gardens, or used for growing crops or fruit trees;

As daily cover at a landfill if approved by OEPA in the landfill permit;

As structural fill, defined as an engineered use of waste material as a building or equipment supportive base or foundation and does not include valley fills or filling of open pits from coal or industrial mineral mining;

As pipe bedding, for uses other than transport of potable water. Materials used in sanitary sewer projects must comply with OEPA Policy DSW 400.001;

As a construction material for roads or parking lots (subbase or final cover), if approved by a professional engineer, the property and the Department of Transportation where necessary; and

Other single beneficial uses of less than 200 tons.

Certain guidelines apply. For example, written notice must be submitted to OEPA before commencement of a beneficial use project involving structural fill applications. An annual report must be submitted by April 1 of each year summarizing each beneficial use project involving structural fill, road base, and pipe bedding applications completed during the prior calendar year, including a description of the nature, purpose, and location of the project, the type and volume of wastes used, and leachate test results.

OEPA Policy No. DSW004.007

Kentucky

Under Kentucky regulations, CCBs (including fly ash, bottom ash, and scrubber sludge produced by coal-fired electrical generating units) are exempt from regulation as hazardous wastes but are classified as special waste. Excluded from this regulation is boiler slag and residues of refuse-derived fuels such as municipal waste, tires, and solvents. Under Kentucky law, CCBs (as defined above) may be reused under permit by rule regulation

- 1) as an ingredient in manufacturing a product;
- 2) as an ingredient in cement, concrete, paint, and plastics;
- 3) as an anti-skid material;
- 4) as highway base course;
- 5) as structural fill; 6) as blasting grit;
- 7) as roofing granules; and
- 8) for disposal in an active mining operation if the mine owner/operator has a mining permit authorizing disposal of special waste.

Specific conditions for reuse of CCBs include:

- 1) the CCB reuse may not create a nuisance;
- 2) erosion and sediment controls must be undertaken;
- 3) the CCB reuse must be at least 100 ft from a stream and 300 ft from potable wells, wetlands, or flood plains;

- 4) the ash must be "non-hazardous;" and
- 5) the generator must submit an annual report. Mine applications must be specifically authorized under the terms of a permit issued by the Department for Surface Mining, Reclamation and Enforcement.

Mine applications must be specifically authorized under the terms of a permit issued by the Department for Surface Mining, Reclamation and Enforcement. Regulatory requirements to obtain such permit authorization along with operational requirements can be found at KY. REV. STAT. ANN. §§350.270. In summary:

CCBs mixed with low volume waste or material with hazardous waste characteristics may not be used in mine applications;

CCBs generated prior to a certain date may not be used in mine applications unless a satisfactory demonstration is made that the CCBs have not been mixed with low volume waste or material with hazardous waste characteristics;

CCBs may be placed only in the pit or extraction area from which coal has been removed by surface mining. Placement of CCBs in other areas within the permit area may be allowed only upon a satisfactory demonstration, based on site specific conditions and the characteristics of the CCBs, that no adverse environmental impacts will occur. Underground injection of CCBs is not authorized;

The permittee must keep records of the source and amount of CCBs received;

Any material that is not CCB approved for disposal must be removed. The permittee must keep records describing the removed material and its disposition;

The permittee must maintain maps showing each CCB disposal location, and the volume of CCBs disposed of at that location;

An annual CCB lab analysis report must be submitted;

An application to modify an existing permit to initially include CCB disposal will be considered a major permit revision application;

An application for CCB mine disposal must demonstrate the permittee's legal right to conduct such activities. Public notice of the application is required; and

The application must contain specific information such as the annual volume of CCBs that will be received, CCB analytical results, proposed operational procedures, hydrogeologic information, and a groundwater monitoring plan.

Detailed Review of CCB Regulations

Under Kentucky regulations, CCBs are exempt from regulation as hazardous waste but are classified as special waste. Specifically included within the definition of coal combustion by-products classified as special waste is fly ash, bottom ash, and scrubber sludge produced by coal fired electrical generating units. Excluded is boiler slag, and residues of refuse derived fuels such as municipal waste, tires and solvents. KY. REV. STAT. ANN. §§224.50-760(1)(a); 401 KY. ADMIN. REGS. 45:010 §§(4).

Tennessee

Under Tennessee law, fly ash, bottom ash, and flue gas emission control waste generated from the combustion of coal or other fossil fuels must be tested for a hazardous waste determination. If determined to be hazardous, certain hazardous waste generator requirements apply. Upon testing confirmation that the material is not hazardous, fly ash, bottom ash, and boiler slag may be reused under permit by rule regulation 1) in engineered structures for a highway overpass, levee, runway, or foundation backfill; and 2) In other proposed beneficial uses approved on a case-by-case basis. Certain restrictions and requirements apply to "permit by rule" uses and proper written notification of the beneficial use must be submitted to the Tennessee Department of Environmental Conservation and approved.

Detailed Review of CCB Regulations

Under Tennessee law, fly ash, bottom ash and flue gas emission control waste generated from the combustion of coal or other fossil fuels must be tested for a hazardous waste determination. If determined to be hazardous, certain hazardous waste generator requirements apply.

TENN.COMP. R. & REGS. 1200-1-11-.02. Upon testing confirmation that the material is not hazardous, fly ash, bottom ash and boiler slag may be reused under permit by rule regulation as follows:

In engineered structures for a highway overpass, levee, runway or foundation backfill; and

In other proposed beneficial uses approved on a case by case basis. (Note: The Tennessee Department of Environment and Conservation ("TDEC") issued a March 4, 1996 Guidance document setting forth the procedures and requirements to obtain approval of a proposed beneficial use.)

Certain restrictions and requirements apply to "permit by rule" uses. For example, prior written notification of the beneficial use must be submitted to TDEC and approved. The notification must be on forms provided by TDEC. The Permit by rule authorization issued by TDEC must be maintained at the facility.

The project may not be located in wetlands, sink holes or caves, or in a 100 year flood plain unless certain conditions are met. The potential for releases must be minimized and site access controlled. The project may not cause or contribute to the taking of any threatened or endangered species of plants, fish or wildlife or result in the destruction or modification of their habitat. Until development is complete, the area must have a barrier to control unauthorized entry. A geologic buffer of at least three feet with a maximum saturated hydraulic conductivity of 1×10^{-6} cm/sec must be in place between the fill and the seasonal high groundwater table. A permanent benchmark of known elevation must be installed. All boreholes, piezometers and abandoned wells within 100 feet of the site must be filled with bentonite in accord with specific requirements.

Within 90 days of completion of the project at least two feet of compacted soil cover must be in place. Final surface grading requirements apply. Dust must be minimized and there must be equipment present at the times coal is received capable of spreading and compacting the coal ash.

TENN.COMP. R. & REGS. 1200-1-7-.02(c).

Pennsylvania

Under Pennsylvania regulations, fly ash, bottom ash, slag, and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste. Coal ash (defined as fly ash, bottom ash, and boiler slag resulting from the combustion of coal) is regulated under the Solid Waste Management Act and the residual waste management regulations, which authorize the beneficial use of coal ash

- 1) as a structural fill;
- 2) as a soil substitute or additive;
- 3) for reclamation at an active surface coal mine site, a coal refuse reprocessing site, or a coal refuse disposal site;
- 4) for reclamation at an abandoned coal or an abandoned non-coal (industrial mineral) mine site;
- 5) in the manufacture of concrete;
- 6) for the extraction or recovery of one or more materials and compounds contained within the coal ash;
- 7) as an anti-skid material or road surface preparation material (bottom ash or boiler slag only);
- 8) as a raw material for a product with commercial value;
- 9) for mine subsidence control, mine fire control, and mine sealing;
- 10) as a drainage material or pipe bedding; and
- 11) As a stabilized product where the physical or chemical characteristics are altered prior to use or during placement so that the potential of the coal ash to leach constituents into the environment is reduced.

All of these uses must comply with specified State regulations.

Detailed Review of CCB Regulations

Under Pennsylvania regulations, fly ash, bottom ash, slag and flue ash gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste (25 Pa. Code §§261.4). Coal ash is regulated under the Solid Waste Management Act and the residual waste management regulations. In December 1986, this act was amended to authorize the beneficial use of coal ash. Beneficial use of coal ash was implemented through Department of Environmental Protection ("DEP") guidelines under the residual waste management regulations, 25 Pa. Code Chapter 287, which were amended in July 1992 to include the beneficial use of coal ash, 25 Pa. Code §§§287.661-287.666. On January 25, 1997, the beneficial use of coal ash regulations, 25 Pa. Code §§§287.663 and 287.664 were amended to change the requirements concerning groundwater monitoring, reporting to the Department, coal ash beneficial uses, and the amounts of coal ash that could be used at active coal mine and abandoned mine sites.

Coal ash is defined in Pennsylvania's Solid Waste Management Act as fly ash, bottom ash or boiler slag resulting from the combustion of coal and may be beneficially used. Other coal ash that is not beneficially used is a "residual waste."

Pennsylvania residual waste management regulations provided that coal ash may be beneficially used. Other coal ash that is not beneficially used is a "residual waste."

Pennsylvania residual waste management regulations provided that coal ash may be beneficially used:

As a structural fill upon approval from the Department if the person proposing the use complies with specified requirements. (Any other use as a structural fill requires a disposal permit.);

As a soil substitute or soil additive if the person proposing the use complies with specified requirements;

For reclamation at an active surface coal mine site, a coal refuse reprocessing site, or a coal refuse disposal site if the use complies with all specified requirements under 25 Pa. Code §§287.663, the Clean Streams Law and regulations promulgated thereunder, the Surface Mining Conservation and Reclamation act (52 P.S. §§§§1396.1-1396.19a), the Coal Refuse Disposal Control Act (52 P.S. §§§§30.51-30.66), and the applicable provisions of Chapters 86-90;

For reclamation at an abandoned coal or an abandoned noncoal (industrial mineral) mine site if the reclamation work is approved by the Department or is performed under a contract with the Department and the use complies with 25 Pa. Code §§287.664, and the applicable environmental statutes and regulations promulgated thereunder;

In the manufacture of concrete;

For the extraction or recovery of one or more materials and compounds contained within the coal ash;

As an anti-skid material or road surface preparation material, if the use is consistent with Department of Transportation specifications or other applicable specifications. (This use applies to bottom ash or boiler slag only. The use of fly ash as an anti-skid material or road surface preparation material is not deemed to be a beneficial use.);

As a raw material for a product with commercial value, including the use of bottom ash in construction aggregate. (Storage of coal ash prior to processing is subject to specific requirements.);

For mine subsidence control, mine fire control and mine sealing, if the person or municipality proposing the use gives advance written notice to the Department, the pH of the coal ash is in a range that will not cause or allow the ash to contribute to water pollution, and use of the coal ash in projects funded by or through the Department is consistent with applicable Department requirements;

As a drainage material or pipe bedding, if the person or municipality proposing the use has first given advance written notice to the Department, and has provided to the Department an evaluation of the pH of the coal ash and a chemical analysis of the coal ash that meets the specific chemical waste analysis requirements; and

As a stabilized product where the physical or chemical characteristics are altered prior to use or during placement if the person or municipality proposing the use has first given advance written notice to the Department, the coal ash is not mixed with solid waste, unless otherwise approved in writing by the Department prior to use, and the use of coal ash results in demonstrated reduction of the potential of the coal ash to leach constituents into the environment.

New Hampshire

Under New Hampshire regulations, fly ash, bottom ash, slag, and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste. New Hampshire regulations establish criteria and procedures under which "waste derived products" will be certified for distribution and use by rule. Coal ash (boiler slag) is certified as a raw material for industrial and commercial purposes. Waste derived products that meet a published standard (such as ASTM standards) are also deemed certified for distribution and use by rule. Other proposed uses may be certified by application to and approval by the State.

Detailed Review of CCB Regulations

Under New Hampshire regulations fly ash, bottom ash, slag and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste. N.H.CODE ADMIN. R Env-Wm 401.03(b)(4).

New Hampshire adopted revised regulations pertaining to reuse of solid waste on October 29, 1997. The regulations establish criteria and procedures under which "waste derived products" will be certified for distribution and use by rule.

Under the rule, coal ash (boiler slag) is specifically deemed certified for distribution and use as a raw material for industrial and commercial purposes. Env-Wm 3203.14.

Waste derived products that meet a published standard are also deemed certified for distribution and use by rule. Fly ash used as a lightweight aggregate in concrete masonry units meeting the standard published in ASTM C-331-89 are specifically deemed certified for distribution and use by rule. Env-Wm 3203.07.

Other proposed reuses may be certified by application to and approval by the state agency. In general, to obtain Agency approval for other proposed reuses, the applicant must demonstrate:

The benefit of using the waste derived product exceeds the benefit of using a "non-waste derived product." This determination is made based on the following criterion:

The waste derived product:

is comparable in form and function to an existing non-waste derived product and performs as effectively or more effectively, or

satisfies an identifiable and unfulfilled need without violating the Env-Wm 2702 standards;

A buyer or user has been identified; and

There is an identifiable benefit to sites upon which the product is land applied.

The waste derived product will not cause a violation of environmental regulations or pose a risk to health or the environment which is greater than the risk posed by production of an existing comparable non-waste derived product or direct disposal;

The physical and analytical characteristics of the waste used to produce the product must be defined in a specification;

The physical and analytical characteristics of the waste derived product must be defined in a specification; and

The production process must include quality assurance/quality control procedures to assure

the specifications are met.

Maine

Under Maine regulations, fly ash, bottom ash, slag, and flue gas emission control waste generated solely from the combustion of coal, other fossil fuels, or from the combustion of coal and other fossil fuels and wood are exempt from regulation as hazardous waste so long as the CCBs do not exhibit any characteristic for ignitability, corrosivity, reactivity, or toxicity. Under Maine regulations, coal ash is regulated as a special waste, and as such requires special handling, transportation, and disposal procedures. Reuse of CCBs is not specifically authorized under Maine law or regulations. Ash or ash-derived products may be used as a raw material substitute in concrete block manufacturing under specified circumstances.

Detailed Review of CCB Regulations

Under Maine regulations fly ash, bottom ash, slag and flue gas emission control waste generated solely from the combustion of coal, other fossil fuels, or from the combustion of coal and other fossil fuels and wood are exempt from regulation as hazardous waste so long as the CCBs do not exhibit any characteristic for ignitability, corrosivity, reactivity, or toxicity. CODE ME. R. §§6-096-850(3)(x). Under Maine regulations, coal ash is regulated as a special waste. Special wastes require special handling, transportation and disposal procedures. CODE ME. R. §§6-096-400-1(KKK).

Currently, reuse of CCBs is not specifically authorized under Maine law or regulations. Ash or ash derived products may be used as a raw material substitution in concrete block manufacturing under specified circumstances. Code Me. R. §§6-096-409-1(B)(2).

Delaware

Under Delaware regulations, fly ash, bottom ash, slag, and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste. CCBs are classified as nonhazardous industrial waste. The Department of Natural Resources and Environmental Control (DNREC) requires TCLP testing to confirm that CCBs are nonhazardous. Reuse of CCBs is not specifically authorized under Delaware law. DNREC has developed an informal CCB beneficial use program with Delmarra Power, the state's sole electric utility. This program is not included in regulations or written policy, and CCB beneficial use projects are approved on a case-by-case basis. Request for approval, accompanied by TCLP test results, must be submitted in advance.

Detailed Review of CCB Regulations

Under Delaware regulations, fly ash, bottom ash, slag and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste. DEL. REGS. §§261.4. CCBs are classified as nonhazardous industrial waste. The Department of Natural Resources and Environmental Control ("DNREC") requires TCLP testing to confirm CCBs are nonhazardous.

Currently, reuse of CCBs is not specifically authorized under Delaware law or regulations. The DNREC advises it has developed an informal CCB beneficial use program with Delmarra Power, the state's sole electric utility. Currently, the program is not set forth in a regulation or written policy. According to a written summary of the program, the DNREC approves CCB beneficial uses on a case by case basis. To date, approved projects have included highway construction, landscaping, sidewalk and foundation construction, berm materials and sewage sludge stabilization projects. Although a permit is not required, proposed beneficial reuse projects must be approved in advance in writing by the DNREC. TCLP test results demonstrating the CCB is not hazardous must accompany any request to DNREC for CCB beneficial use approval

New York

Under New York regulations, fly ash, bottom ash, slag, and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste. Bottom ash may be used as a component in the manufacture of roofing shingles or asphalt products, or as a traction agent on roadways, parking lots, or other driving surfaces. Fly ash or "gas scrubbing products" may be used as and ingredient in producing lightweight block, lightweight aggregate, low strength backfill material, manufactured gypsum, or manufactured calcium chloride. Fly ash or bottom ash may be used as a cement or aggregate substitute in concrete or concrete products, as raw feed in the manufacturing of cement or as structural fill within building foundations when placed above the seasonal high groundwater table. Other proposed beneficial reuses may be approved by the State on a case-by-case basis.

Detailed Review of CCB Regulations

Under New York regulations, fly ash, bottom ash, slag and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste. N.Y. COMP. CODES R. & REGS. 6 §§371.1.

Under New York regulations, CCBs may be reused in the following manner:

Bottom ash may be used as a component in the manufacture of roofing shingles or asphalt products, or as a traction agent on roadways, parking lots and other driving surfaces;

Fly ash or "gas scrubbing products" may be used as an ingredient in producing lightweight block, lightweight aggregate, low strength backfill material, manufactured gypsum or manufactured calcium chloride (fly ash or FGD material); and

Fly ash or bottom ash may be used as a cement or aggregate substitute in concrete or concrete products, as raw feed in the manufacturing of cement or as structural fill within building foundations when placed above the seasonal high groundwater table (fly ash or bottom ash).

Other proposed beneficial reuses may be approved on a case by case basis. To request approval for a proposed beneficial reuse project not included in the foregoing, a written petition must be submitted New York Department of Environmental Conservation containing specified information regarding the proposed reuse. The Department will determine whether the proposal constitutes a beneficial reuse based on a showing that all the following criteria have been met:

The proposal constitutes a reuse rather than a disposal;

The proposal is consistent with solid waste management philosophy;

The material under review is intended to function or serve as an effective substitute or fuel;

Decontamination of the material must not be required;

A market must exist or be reasonably certain to develop for the proposed use of the material or the product into which the material is proposed to be incorporated; and

Any other criteria established by the Department.

Annual reporting requirements apply. No later than 60 days after each January 1, CCB generators must submit a report to the New York Department of Environmental Conservation identifying the quantities of fly ash, bottom ash and gas scrubbing products it generated during the year. The generator must also specify how much bottom ash and fly ash was sent for reuse and in what manner (eg. cement, roofing shingles). 6 N.Y. COMP. CODES R. & REGS. 6, Ch. IV, subchapter B, Part 360-1.15.

Rhode Island

There are no coal-fired power plants in Rhode Island. State hazardous and solid waste regulations do not specifically address regulatory status or reuse potential of CCBs. CCBs are not automatically exempt from Rhode Island hazardous waste regulations. Reuse of CCBs is not specifically authorized under Rhode Island law. Additionally, under Rhode Island regulations, recycled or reused waste may remain subject to regulations. The Rhode Island Department of Environmental Management reports that it would likely adopt a conservative approach towards CCB use if approval for such a project was requested.

Detailed Review of CCB Regulations

There are no coal-fired power plants in Rhode Island. State hazardous and solid waste regulations do not specifically address the regulatory status or reuse potential of CCBs. CCBs are not automatically exempt from Rhode Island hazardous waste regulations. RHODE ISLAND REGS. HWM §§3.25. Currently, reuse of CCBs is not specially authorized under Rhode Island law or regulations. Additionally, under Rhode Island regulations, recycled or reused waste may remain subject to regulation. RHODE ISLAND REGS. HWM §§3.25. The Rhode Island Department of Environmental Management reports it would likely adopt a conservative approach towards CCP use if approval for such a project was requested.

Maryland

Under Maryland law, fly ash, bottom ash, slag, and flue gas emission control waste generated primarily from the combustion of coal or other fuels are not regulated as a hazardous waste. Maryland law authorizes certain beneficial reuses of "pozzolans." Pozzolans are defined as "the finely divided residue which results from the combustion of ground or powdered coal and is released by combustion gases, as defined by the test methods published by the American Society for Testing Materials." Pozzolans may be used

- 1) for landfill in a manner which complies with sound engineering practices and applicable

permit requirements; and

2) as structural building, soil improvement, agriculture soil conditioning, or land reclamation in compliance with all silt control regulations and permit requirements of the Department of the Environment. Dust and erosion minimization is required. If there is a threat to water quality, an NPDES or state permit may be necessary.

Detailed Review of CCB Regulations

Under Maryland law, fly ash, bottom ash, slag and flue gas emission control waste generated primarily from the combustion of coal or other fuels are not regulated as a hazardous waste.

Maryland law authorizes certain beneficial reuses of "Pozzolans." Pozzolans are defined as "the finely divided residue which results from combustion of ground or powdered coal and is released by combustion gases, as defined by the test methods published by the American Society for Testing Materials." MD. ANN. CODE §§15-407 (formerly MD. ANN. CODE §§7-464) (ASTM class fly ash). Under Maryland law, Pozzolans may be used as follows:

For landfill in a manner which complies with sound engineering practices and applicable permit requirements; and

As structural building, soil improvement, agriculture soil conditioning, or land reclamation in compliance with all silt control regulations and permit requirements of the Department of the Environment. Dust and erosion minimization is required.

Note: In locations where there may be a threat to water quality, a NPDES or state permit to protect groundwater may be necessary. MD. ANN. CODE §§15-407 (formerly MD. ANN. CODE §§7-464).

Indiana

Under Indiana law, fly ash, bottom ash, or such ash when mixed with flue gas desulfurization products may not be regulated if the material is not hazardous and is disposed in a properly permitted and approved facility. Additionally, fly ash, bottom ash, or such ash when mixed with flue gas desulfurization products or boiler slag may not be regulated when used

- 1) for the extraction and recovery of materials and compounds within the ash;
- 2) as an anti-skid material (bottom ash);
- 3) as a raw material in manufacturing another product;
- 4) for mine subsidence, mine fire control, and mine sealing (restrictions may apply under the laws and regulations applicable to mining);
- 5) as structural fill when combined with cement, sand, or water to produce a controlled strength fill material; and
- 6) as a base in road construction. The Indiana Department of Environmental Management has prepared guidelines specifying testing requirements for use in ways other than those specified above.

Detailed Review of CCB Regulations

Under Indiana law, fly ash, bottom ash, or such ash when mixed with flue gas desulfurization

products may not be regulated if the material is not hazardous and is disposed in a properly permitted and approved facility. Additionally, fly ash, bottom ash, or such ash when mixed with flue gas desulfurization products or boiler slag may not be regulated when used in the following manner IND. CODE 13-19-3-3:

The Indiana Department of Environmental Management ("IDEM") has prepared Coal Ash Classification Guidelines specifying testing and other requirements for CCBs proposed for reuse in ways other than those specified above. Although the guidelines are currently being updated, under the current guidelines and other applicable regulations (329 IAC 10-9-4), CCBs may be disposed at Type I restricted waste sites (generally a site designed as a sanitary landfill) without specific testing. CCBs may be disposed of at other waste sites (Types II, III, IV) only if (329 IAC 10-9-4):

EP toxicity results for arsenic, barium, cadmium, chromium, lead, mercury, selenium, and silver are within specified limits;

TCLP results for barium, boron, chlorides, total cyanide, fluoride, Ph, sodium, sulfate, total sulfide and total dissolved solids are within specified limits; and

Resampling is conducted every five years, whenever the characteristics of the coal change, whenever the process generating the waste changes or as otherwise required by IDEM.

Rulemaking pertaining to mine haul back of ash is currently in progress.

Other CCB applications may be authorized upon IEPA's written determination that the proposed use has no greater adverse environmental impact than the beneficial uses specified in the law.

Illinois

Under Illinois regulations, fly ash, bottom ash, slag, and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste. Illinois law specifically authorizes the reuse of CCBs, classified into two different groups: coal combustion waste (CCW) and coal combustion byproduct (CCB). CCW reuse is regulated more stringently than CCB. CCW can be classified as CCB under certain conditions and reused, based on the classification,

- 1) for the extraction and recovery of materials and compounds within the ash;
- 2) as a raw material in the manufacture of cement and concrete products;
- 3) for roofing shingles,
- 4) in plastic products, paints, and metal alloys,
- 5) in conformance with the specifications and with approval from the Illinois Department of Transportation (IDOT);
- 6) as antiskid material, athletic tracks or foot paths (bottom ash);
- 7) as a lime substitute for soils so long as the CCBs meet the IDOT specifications for agricultural lime as a soil conditioner;
- 8) in non-IDOT pavement base, pipe bedding, or foundation backfill (bottom ash);
- 9) as structural fill when used in an engineered application or combined with cement, sand, or water to produce a controlled-strength material; and

- 10) for mine subsidence, mine fire control, mine sealing, and mine reclamation (must meet requirements of both the Illinois Environmental Protection Agency and Department of Mines and Minerals). Other CCB applications may be authorized by IEPA.

Detailed Review of CCB Regulations

Under Illinois regulations, fly ash, bottom ash, slag and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste. 35 ILL. ADMIN. CODE §§721.104.

In 1995, Illinois enacted legislation specifically authorizing reuse of coal combustion waste. 415 ILCS 5/3.94 (P.A. 89-93). P.A. 89-93 creates two classifications of coal ash: coal combustion waste ("CCW") and coal combustion byproduct ("CCB"). CCW is subject to limited management and disposal options. CCB, on the other hand, may be used in multiple applications as discussed below. The term CCW includes fly ash, bottom ash, slag or flue gas or fluid bed desulfurization by-products generated through combustion of coal. The term also includes waste from coal combusted with the following:

Fuel grade petroleum coke, other fossil fuel, both fuel grade petroleum coke and other fossil fuel;
or

Fuel grade petroleum coke, other fossil fuel, or both fuel grade petroleum coke and other fossil fuel in combination with no more than 20% tire derived fuel or wood or other materials by weight of the material combusted. Note: An Agency determination is required that storage and disposal of the resultant wastes will not result in an environmental impact greater than waste from the combustion of coal alone and that the storage and disposal of the resultants wastes will not violate federal law.

CCW can be classified as CCB under certain conditions and reused based on this classification. CCB may be reused as follows:

For the extraction and recovering of materials and compounds within the ash;

As a raw material in the manufacture of cement, concrete, concrete products and concrete mortars;

For asphalt or cement based roofing shingles;

In plastic products, paints and metal alloys;

In conformance with the specifications and under the approval of the Illinois Department of Transportation ("IDOT");

As anti-skid material, athletic tracks or foot paths (bottom ash);

As a lime substitute in the lime modification of soils so long as the CCBs meet the IDOT specifications for by-product lines, and the functional equivalent for agricultural lime as a soil conditioner;

In non-IDOT pavement base, pipe bedding, or foundation backfill (bottom ash);

As structural fill when used in an engineered application or combined with cement, sand or water to produce a controlled strength fill material and covered with 12 inches of soil unless infiltration is prevented by the material itself or other cover material; and

For mine subsidence, mine fire control, mine sealing and mine reclamation.

Certain restrictions apply to reuse of CCBs. The user of CCBs in certain applications must notify the Illinois Environmental Protection Agency ("IEPA") of each project utilizing CCBs, document the quantity of CCBs that will be utilized and certify that the CCBs have not been mixed with hazardous waste prior to use and that the CCBs do not exceed Class I groundwater quality standards for metals when tested utilizing ASTM method D3987-85. Dust generation in fly ash applications must be minimized. CCBs may not be accumulated speculatively. Note: CCBs are not accumulated speculatively if 75% of the CCBs accumulated at the beginning of a calendar year are used during the calendar year.

Mine applications of CCW and/or CCBs must meet the requirements specified in 415 ILCS 5/21(r) and certain guidance memorandum issued by the Illinois Department of Mines and Minerals ("IDMM") and IEPA. IDMM and IEPA have dual jurisdiction over mine disposal of CCBs. IDMM and IEPA have issued joint memorandums detailing the procedures and requirements for mine disposal of CCBs. (Land Reclamation Memorandum 92-11 and Land Reclamation Memorandum 95-89). Groundwater monitoring and liners may be required by IDMM and IEPA in certain applications. In addition, CCW requirements are more stringent than those for CCB. Specifically:

CCW waste disposal must be associated with coal sales (on a company wide basis) of the coal company. A coal company may not dispose of CCW from a company that has not purchased coal from the mine.

CCW disposal may not exceed 35% of coal sales unless information is submitted justifying a higher percentage (According to Scott Schmitz of IDMM, additional volumes will be allowed if it is established that the particular combusted coal generates a greater volume of ash). Initially, this restriction was stated as a limitation applicable to each coal sales source. This requirement was clarified in a November 20, 1995 IDMM memo which states that IDMM is not limiting each CCW source to a 35% disposal limit, but that the 35% limit applies to total coal company sales. The quantity of CCW from each source as well as the total quantity of CCW received at the disposal site must be reported quarterly.

Other CCB applications may be authorized upon IEPA's written determination that the proposed use has no greater adverse environmental impact than the beneficial uses specified in the law.

Connecticut

Connecticut regulations adopt by reference the federal regulation that exempts CCBs (including fly ash, bottom ash, slag, and flue gas emission control waste generated primarily from the combustion of coal) from classification as hazardous waste. CCBs are classified as either special or regulated waste. Reuse of CCBs is not specifically authorized under Connecticut law, although a Solid Waste Beneficial Reuse subcommittee has been formed to assist the Connecticut Department of Environmental Protection in developing policy and regulations to control reuse of solid waste. A General Permit for CCB reuse may be issued in the future.

Detailed Review of CCB Regulations

Connecticut regulations adopt by reference the federal regulation which exempts CCBs from classification as hazardous waste. Exempt from hazardous waste regulation are fly ash, bottom ash, slag, and flue gas emission control waste generated primarily from the combustion of coal. CONN. AGENCIES REGS. §§22a-449(c)-101; 40 CFR 261.4. CCBs are classified as either special waste or regulated waste.

Currently, reuse of CCBs is not specifically authorized under Connecticut law or regulations. The Connecticut Department of Environmental Protection ("DEP") reportedly recognizes the need for such regulation and has formed a Solid Waste Beneficial Reuse subcommittee. The subcommittee is comprised of business and industry representatives. The purpose of the subcommittee is to provide assistance to the DEP regarding development of policy and regulations to control beneficial reuse of solid waste. The subcommittee has received a request for approval of a CCB beneficial reuse project which is currently being reviewed. The DEP reports it may issue a General Permit for CCB reuse in the future.

Massachusetts

Under Massachusetts law, fly ash, bottom ash, slag, and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as a hazardous waste. Ash produced from the combustion of coal, including fly ash, bottom ash, boiler slag, and economizer ash are exempt from regulation as solid waste, if beneficial reused

- 1) as a raw material for concrete block manufacture;
- 2) as aggregate;
- 3) as fill;
- 4) as a base for road construction; or
- 5) in other approved commercial or industrial purposes.

General conditions for reuse apply and storage may be constructed, maintained and operated without being construed as a refuse disposal facility. However, the Department of Environmental Protection (DEP) has jurisdiction to determine that the facility has created a nuisance condition due to odor, dust, fires, smoke or other conditions. Except for reuse as base for road construction or fill, land application of CCBs is prohibited, unless the disposal location has been assigned for such use by the DEP. CCBs may be used as intermediate landfill cover material with approval from DEP.

Detailed Review of CCB Regulations

Under Massachusetts law, fly ash, bottom ash, slag and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as a hazardous waste. MASS.REGS.CODE 310§§30.104.

In Massachusetts, ash produced from the combustion of coal including fly ash, bottom ash, and boiler slag, and economizer ash are exempt from regulation as solid waste, if beneficially reused:

As a raw material for concrete block manufacture;

As aggregate;

As fill;

As a base for road construction; or

In other approved commercial or industrial purposes
(flash fill has been approved).

General conditions for reuse apply. The ash storage location for these reuses may be constructed, established, maintained and operated without being construed as a refuse disposal facility and no approval from the Board of Health or the Department of Environmental Protection ("DEP") is required. However, the DEP has jurisdiction to determine that the facility has created a nuisance condition due to odor, dust, fires, smoke, or other conditions and order the abatement of these conditions.

Except for CCB reuse as base for road construction or fill, land application of CCBs is prohibited, unless the place where the disposal is proposed to occur has been assigned for such disposal by the Board of Health and plans for such disposal have been approved by the DEP. CCBs may be used as intermediate cover material over rubbish at a landfill with the approval of the DEP.

Virginia

Under Virginia regulations, fly ash, bottom ash, slag, and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste. These CCBs are exempt from regulation as a solid waste if beneficially used, and managed in accordance with all applicable state agency requirements,

- 1) for mine reclamation or mine refuse disposal on a permitted mine site
- 2) for soil nutrient additive, stabilization agent, structural improvement, or other agricultural purposes;
- 3) as a traction control material or road surface material (bottom ash or boiler slag);
- 4) as a base, subbase, or fill material under a paved road, the footprint of a structure, a paved parking lot, a sidewalk, walkway, or similar structure;
- 5) when processed with a cementitious binder to produce a stabilized structural fill product that is spread and compacted with proper equipment for the construction of a project with a specified end use; and
- 6) for the extraction or recovery of materials and compounds contained within the CCBs.

Detailed Review of CCB Regulations

Under Virginia regulations, fly ash, bottom ash, slag and flue gas emission control waste generated primarily from the combustion of coal or other fossil fuels are exempt from regulation as hazardous waste.

Under Virginia regulations, CCBs are exempt from regulation as a solid waste if beneficially reused in the following manner and managed in accordance with all applicable state agency requirements:

For mine reclamation or mine refuse disposal on a mine site permitted by the Virginia Department of Mines, Minerals and Energy when used in accordance with the standards developed by the Department of Waste Management;

For soil nutrient additive, stabilization agent, structural improvement or other agricultural purposes under the authority of the Virginia Department of Agriculture and Consumer Services;

As a traction control material or road surface material if the use is consistent with Virginia Department of Transportation specifications (bottom ash or boiler slag);

As a base, subbase or fill material under a paved road, the footprint of a structure, a paved parking lot, sidewalk, walkway or similar structure;

When processed with a cementitious binder to produce a stabilized structural fill product which is spread and compacted with proper equipment for the construction of a project with a specified end use; and

For the extraction or recovery of materials and compounds contained within the CCBs.
9 VAC 20-80-160.

In 1995, Virginia promulgated a regulation specifying the terms and conditions under which CCBs may be reused through land application such as structural fills, mine reclamation or mine refuse disposal (in conjunction with Virginia Surface Mining regulations). The regulation allows for the use of CCBs in structural fills and mine reclamation projects. The regulation also provides for the siting of such projects, the design and construction of runoff and cover systems, the closure of projects, and establishes minimum operational requirements.

CCBs managed under this regulations are not subject to solid waste facility permitting, however, at least 30 days prior to initial placement of CCBs the facility owner must:

Submit certification that the owner has legal control over the proposed site for the project life and closure period, that the location and operation of the site will be in compliance with all local ordinances, and that the owner will allow Department inspections to ensure compliance with applicable regulations;

Provide a description of the intended use, reuse, or reclamation of the CCBs including a description of the site, the estimated beginning and ending dates of the operation, an estimate of the volume of CCBs to be used, and the physical and chemical characteristics of the CCBs including TCLP analyses for specified characteristics;

Certification by a professional engineer that locational restrictions have been satisfied and that the project has been designed in accordance with specified standards; and

An operational and closure plan.

Various location restrictions apply. For example, CCBs may not be placed:

In areas subject to base floods unless it can be shown that the CCBs can be protected from inundation or washout and that the flow of water is not restricted;

With the vertical separation between the CCB and the maximum seasonable water table or bedrock less than two feet;

Closer than 100 feet from any perennial stream, water well, sinkhole or 25 feet from a bedrock outcrop (unless the outcrop is treated to minimize infiltration into fractured zones) or property boundaries;

In wetlands, unless applicable federal, state and local permits are obtained; or

At the site of an active or inactive dump, unpermitted landfill, lagoon or similar facility, even if closed.

In addition, storage and stockpiling of materials must meet specified regulatory requirements.

West Virginia

West Virginia regulations adopt by reference the federal regulation that exempts CCBs (including fly ash, bottom ash, slag, and flue gas emission control waste generated primarily from the combustion of coal) from classification as hazardous waste. Under West Virginia regulations, CCBs may be reused

- 1) as a material in manufacturing another product or as a substitute for a product or natural resource;
- 2) for the extraction or recovery of materials and compounds contained within the CCBs;
- 3) as a stabilization/solidification agent for other wastes if used singly or in combination with other additives or agents to stabilize or solidify another waste product;
- 4) under the authority of the West Virginia Department of Energy;
- 5) as pipe bedding or as a composite liner drainage layer;
- 6) as an anti-skid material (bottom ash, boiler slag);
- 7) as a daily or intermediate cover for certain solid waste facilities;
- 8) as a construction base for roads or parking lots that have asphalt or concrete wearing surfaces.

Detailed Review of CCB Regulations

West Virginia regulations adopt by reference the federal regulation which exempts CCBs from classification as hazardous waste. Exempt from hazardous waste regulation are fly ash, bottom ash, slag and flue gas emission control waste generated primarily from the combustion of coal.

Under West Virginia regulations, CCBs may be reused in the following manner:

As a material in manufacturing another product (e.g. concrete, flowable fill, lightweight aggregate, concrete block, roofing materials, plastics, paint) or as a substitute for a product or natural resource (e.g. blasting grit, filter cloth precoat for sludge dewatering);

For the extraction or recovery of materials and compounds contained within the CCBs;

As a stabilization/solidification agent for other wastes if used singly or in combination with other additives or agents to stabilize or solidify another waste product. Advance written notice must be submitted to the state and the use must result in altered physical or chemical characteristics of the other waste and a reduction of the potential for the resulting established mixture to leach constituents into the environment;

Under the authority of the West Virginia Department of Energy;

As pipe bedding or as a composite liner drainage layer;

As an anti-skid material (bottom ash, boiler slag) if such use is consistent with Department of Highways specifications. The use of fly ash as an anti-skid material is not deemed to be a beneficial use;

As a daily or intermediate cover for certain solid waste facilities if the permit allows for such use; and

As a construction base for roads or parking lots that have asphalt or concrete wearing surfaces, if approved by the West Virginia Department of Highways and the project owner.

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APPENDIX A

**CONTAMINANT TRANSPORT MODELING OF ACID MINE DRAINAGE AT AN
ABANDONED UNDERGROUND MINE**

Final Report

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by

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ABSTRACT

Mine subsidence and acid mine drainage (AMD) are two potential problems caused by abandoned underground coal mines. Backfilling of underground mine cavities with a grout based on fluidized-bed combustion (FBC) ash is considered as one of the methods to reduce AMD problems. The major objective of the study presented in this report was to analyze the contaminant transport patterns around an underground mine cavity using computer models. A contaminant transport model requires several input parameters including the solution from a groundwater flow model and some hydro-geologic properties of the aquifer. In this study, MODFLOW was chosen as the three-dimensional groundwater flow model and MT3D was chosen as the three-dimensional contaminant transport model. The various input parameters for the groundwater flow and contaminant transport modeling tasks of the study were not available for the specific mine site. Therefore, several idealizations and assumptions were made, and the validity of modeling results depends on the accuracy of the assumed material properties and other data.

The simulations of grout backfill configurations show that the contaminant transport occurs largely in the hallways and that the lateral spread of the contaminant plume outside of the mine area is limited. It can also be observed from the results that the concentrations are higher in the mine cavity and that the lateral spread of the contaminant plume is relatively small. The computed contaminant concentration is significantly reduced as the backfill height is increased. The grout serves to reduce the production of AMD and restricts its transport in the mine cavity.

The results suggest that the transport of the contaminant and its concentration may be effectively lowered by covering the entire floor area of the mine with grout backfill. The contaminant transport can be changed by increasing the height of grout fill at certain locations of the hallways. Influence of cracks and fissures around the mine cavity on the contaminant transport was studied by varying the size of fracture zone and its hydraulic conductivity. The results show that the size of the lateral fracture zone would increase the rate of contaminant propagation. The rate at which the contaminants get transported in the mine appears to increase with an increase in the extent of three-dimensional fracture zones around the mine. The fracture zones in the mine layer have a greater influence on the contaminant plume propagation than the fracture zones above the mine layer.

The study shows that computer models can be used to study the contaminant transport patterns in abandoned underground mines and the influence of grouting on contaminant transport. However, it should be noted that the computer models require significant site specific data and material properties, which were not available for the specific mine site. These input data need to be established to have reliable, site-specific predictions. In the absence of site-specific data, the general trends and patterns of contaminant transport around a mine cavity and the influence of different grouting scenarios on contaminant transport can be obtained by using these models.

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Chapter 1

INTRODUCTION

1.1 STATEMENT OF THE PROBLEM

Mining has been practiced by man since times immemorial. It was one of the major industries in the nineteenth century. Several mining methods have been devised over the years for effective extraction of minerals. The underground mining (or sub-surface mining) method is adopted to extract minerals deep below the surface of the earth. The underground mining method can be broadly divided into (a) room-and-pillar method and (b) longwall method [Peng, 1986]. Most of the older mines are based on the room-and-pillar method of mining.

There are many abandoned mines in the Appalachian region. Problems posed by abandoned mines can be divided into two major categories:

1. Mine Subsidence
2. Acid Mine Drainage Contamination

The present study deals with the modeling Acid Mine Drainage (AMD) contamination problem in and around an abandoned coal mine located in northern West Virginia. Water flowing out of this abandoned mine known as the Longridge mine is assumed to be contaminated with Acid Mine Drainage (AMD). The Longridge mine was mined using the room-and-pillar method mentioned above. Chapter 3 of this report contains a detailed description of the Longridge mine area. AMD is produced when sulphide minerals in waste rocks and coal measures are washed by groundwater in the presence of oxygen [Corbett, 1975]. This process is depicted in the schematic diagram shown in Figure 1.1. The outflow of AMD contaminated water from an underground mine

Figure 1-1: Schematic of Acid Mine Drainage in an Underground Coal Mine

constitutes an environmental problem. Acid Mine Drainage from abandoned mines mixes with and pollutes streams and groundwater. It can render the water unusable by lowering its pH value. It can also have a corrosive effect on metals and can kill aquatic life [Maryland DNR, 1969]. Those are a few examples of environmental damage caused by acid mine drainage.

There are two major techniques employed to solve AMD problems. The first technique is by preventing the formation of AMD at the source. Some of the measures used to achieve this are: by neutralizing the acid at the source when it is formed, by preventing the pyrite deposits from being washed by water or by filling the mine cavities with grout [Maryland DNR, 1969]. The second technique is to treat the AMD contaminated water. This is generally the more expensive of the two methods.

The present study is an effort to investigate the influence of grouting on the transport of Acid Mine Drainage by using a computer model. Varied grouting patterns may affect the transport of AMD in different ways. For example, filling mine cavities with grout in the mine cavities all the way to the mine roof may not significantly lower the AMD production and transport in comparison to filling mine cavities with grout half way up to the roof. Grouting certain key locations of the mine cavity, may be almost as effective as grouting the entire mine cavity which is an expensive and a difficult ordeal.

It is unknown whether it may be possible to optimize the amount of grout to be used in an underground mine cavity and to identify some locations where grouting would be more effective. Different scenarios of grout in the Longridge mine cavities were considered for the purpose of this study as shown in Chapter 5.

1.2 OBJECTIVES OF THE STUDY

The objective of the present study is to analyze the patterns of transport of Acid Mine Drainage contaminants in the Longridge mine area by using a computer model. The approach in this study can be broadly described as a two phase approach; the groundwater flow modeling phase and the contaminant transport modeling phase. Background studies were carried out and the literature was reviewed to identify the modeling methods to be adopted, the computer models to be used, the input parameters necessary for the computer models and the physical dimensions of the Longridge mine area.

Different potential grouting scenarios in the Longridge mine were drawn up to be used by the computer models to study their impact on the Acid Mine Drainage contaminant transport. These cases are described in more detail in Chapter 5 of this report. The aim of the computer modeling work was to investigate its potential use in AMD transport around underground mine cavities.

1.3 SCOPE

There are certain factors that limit the scope of the present study dealing with the modeling of contaminant transport at the Longridge mine. These are:

- The irregular geometry of the Longridge mine area and the identified catchment basin around make it a difficult domain to model. Hence, some idealizations were made with respect to the mine and catchment area geometries.
- The computer modeling was carried out with several assumptions made because of the unavailability of important input parameters such as the groundwater table elevations, the hydraulic conductivities of the various materials at the site and several other input parameters

needed for the models. Therefore, the validity of modeling results depends on the accuracy of the assumed site-specific material properties, which were not available.

Chapter 2 of this report contains a description of the literature search conducted for this study. It includes descriptions of the groundwater flow and contaminant transport mechanisms. Specific details on computer modeling of contaminant transport and a general description of the acid mine drainage phenomenon are included in Chapter 2.

The methodology adopted for the study is described in Chapter 3 of this report in a step-by-step manner. Included in Chapter 3 are a description of the site characteristics of the Longridge mine area and the required input parameters for the computer modeling task. A description of the finite-difference grids used in the computer modeling task and the assumed input parameters used can be found in Chapter 4. The results obtained from the computer modeling studies and a discussion of these results are included in Chapter 5. The summary of the present study, the conclusions and some recommendations are presented in Chapter 6 of this report.

1.4 CONTAMINANT TRANSPORT PHENOMENA

The study of propagation of contaminants in saturated porous media is called contaminant transport [Wang and Anderson, 1982]. In the current context contaminants may be defined as "all solutes introduced into the hydrologic environment as a result of man's activities" [Freeze and Cherry, 1976]. In the present study Acid Mine Drainage (AMD) is the major contaminant. Some of the basic mechanisms of transport of contaminants in saturated porous media are: advection, diffusion and dispersion [Wang and Anderson, 1982]. The advection mechanism is dictated by the

of groundwater in the media. The contaminant spreads with the same velocity as the groundwater under the influence of this mechanism.

The diffusion of a contaminant in a medium depends on the concentration gradient distribution. The migration of contaminants under the influence of dispersion takes place from zones of higher concentration to zones of lower concentration. The differences in groundwater flow velocities in a porous medium causes some mixing of the contaminants. This mechanism is called 'dispersion' [Wang and Anderson, 1982].

The phenomenon of contaminant transport is complex and interesting. It has become a major research area in the last decade or so, with a lot of focus on numerical modeling of contaminant transport. [Sarangdhar, 1996; Brainard and Gelhar, 1991; Anderson and Woesner, 1992; Bedient, et al., 1994]. However, no information was found on the contaminant transport modeling of acid mine drainage around underground mine cavities in the literature.

1.5 CONTAMINANT TRANSPORT MODELING

Several different models have been developed over the years to study and achieve a better understanding of the above mentioned mechanisms in the contaminant transport phenomena. Sand tank models, analog models and mathematical models are the three broad based categories into which models can be divided [Wang and Anderson, 1982]. Mathematical models include analytical and numerical models. Numerical models are probably the most widely used kind. The finite difference method and the finite element method are the two most commonly used methods for numerical modeling of groundwater flow and contaminant transport through porous media [Wang and Anderson, 1982]. These models use computer programs to solve the various governing equations

equations and to obtain solutions to the problem at hand.

A contaminant transport model requires several input parameters including the solution from a groundwater flow model and some hydro-geologic properties of the aquifer. It also requires advection, dispersion and chemical reaction parameters associated with the media and the contaminant. Some of these parameters may not be required if a particular mechanism does not play a role in a given model.

The groundwater flow model requires many geo-hydrologic properties as input parameters. It also requires input in terms of the dimensions of the area to be modeled, along with the boundary conditions (wells, drains, rivers, lakes, etc.). A contaminant transport model generally uses the same physical discretization used for the groundwater flow model. There could be errors associated with any numerical model. The most common errors are 'computational errors', 'calibration errors' and 'manual errors'. A proper understanding of these errors is essential to aid in the proper interpretation of modeling results. More details of such errors can be found in the literature [Todd, 1976; Bedient et al, 1994].

The limitations in the capabilities of models must also be taken into account in any modeling task. These limitations may originate from the computer program used or from the assumptions made by the user. Chapter 2 of this report deals with the background study and literature review for this study. The theory behind groundwater flow and contaminant transport modeling are discussed here. Different computer models available for flow and transport modeling studies along with the models used in the present study are also described in Chapter 2.

Chapter 2

BACKGROUND

2.1 OVERVIEW OF LITERATURE REVIEW

This chapter contains an overview of the literature reviewed for the present study and a description of the mechanisms of groundwater flow and contaminant transport. It also includes a summary of the technique used in modeling the transport of contaminants and the Acid Mine Drainage phenomena.

Information pertinent to groundwater flow and contaminant transport was identified and collected from available literature. This provided an invaluable insight into the principles of groundwater flow, contaminant transport modeling and the parameters and the properties required for this task.

It was found from the reviewed literature that Acid Mine Drainage is one of the largest environmental hazards in the Appalachian region [Corbett, 1975; Maryland DNR, 1969; Fetter, 1988]. The Appalachian region was a hotbed for mining activity for several decades. The contamination caused by AMD is a complex problem and large computer models are needed to study the patterns of contamination caused by AMD. The following are a few popular computer models used in groundwater and contaminant transport studies [Reddi et al, 1997]:

- MOC
- PLASM
- BIOPLUME
- FLOWPATH

- GWFLOW
- MODPATH
- PATH3D
- VisualModflow
- SUTRA
- ModelCad
- FemWater

However, in the present study MODFLOW was chosen as the groundwater flow model because MODFLOW is the most widely used groundwater flow modeling code [Fetter, 1988; Reddi et al, 1997; McDonald and Harbaugh, 1985] of the several codes available. The contaminant transport code used in the present study was MT3D [Zheng, 1990].

2.2 CONTAMINANT TRANSPORT

Contaminant transport also known as mass transport is the study of migration of contaminants in saturated porous media. There are several different kinds of contaminants that cause contamination of groundwater. These are broadly listed below as [Bedient et al, 1994]:

- a. Synthetic organic chemicals
- b. Hydrocarbons
- c. Inorganic cations
- d. Inorganic anions
- e. Pathogens
- f. Radionuclide

These contaminants have varying degree's of solubility in water. Some of the organic compounds are only slightly soluble and will exist in both a dissolved phase and an insoluble phase, which can get transported in the porous media. The inorganic substances occur from natural as well as man made sources. Pathogens are disease causing micro-organisms. The radionuclides can occur naturally or from artificial sources such as mining, processing ore, industrial uses and radioactive waste.

Contaminant transport in the saturated porous media occurs through certain processes or mechanisms. Diffusion, Dispersion and Advection are some of the basic processes causing transport. These mechanisms, along with some other basic terms used in the contaminant transport area are defined in the following section of this report. The advection-dispersion equation may be used to describe contaminant transport. It can be expressed as [Fetter, 1988]:

$$D_l \frac{\partial^2 C}{\partial l^2} - v_l \frac{\partial C}{\partial l} = \frac{\partial C}{\partial t} \quad (2.1)$$

where,

l = a curvilinear coordinate direction taken along the flowline,

v_l = average linear groundwater velocity,

D_l = coefficient of hydrodynamic dispersion in the longitudinal direction (along the flowpath),

C = solute concentration, and

t = time,

2.2.1 Mechanisms of Contaminant Transport

Contaminant transport in groundwater involves several mechanisms. More than one of these mechanisms generally contribute to any kind of contaminant transport phenomena. Given below are the definitions of some of the important mechanisms that cause the migration of contaminants.

- *Advection* is defined as "the movement of contaminants along with flowing groundwater at seepage velocity in porous media." [Wang and Anderson, 1982]
- *Diffusion* is defined as "a molecular mass-transport process in which solutes move from areas of higher concentration to areas of lower concentration". [Bedient et al., 1994]
- *Dispersion* is defined as "a mixing process caused by velocity variations in the porous media". [Wang and Anderson, 1982]
- *Adsorption* can be defined as "the partitioning of organic contaminants from the soluble phase onto the soil matrix". [Wang and Anderson, 1982]

2.2.2 Advective Transport

"The transport at the same velocity as groundwater of dissolved solids or of heat is called advective transport" [Wang and Anderson, 1982]. The contaminant moves with the same velocity as the groundwater in the porous media, known as seepage velocity. The seepage velocity may be defined by the equation [Wang and Anderson, 1982]:

$$v_l = \frac{K}{n} \frac{dh}{dL} \quad (2.2)$$

where,

v_l = the seepage velocity of groundwater in the porous media,

K = the hydraulic conductivity of the porous media,

n = the porosity of the porous media, and

dh/dL = the fluid head gradient for groundwater flow.

Advection is the most common contaminant migration mechanism and is the most dominant method of transport in many cases. More information on the Advective transport area can be found in the literature [Lee et al., 1992 and Aldous and Smart, 1988].

2.2.3 Diffusion

"Diffusion is a molecular-scale process that causes spreading due to concentration gradients and random motion" [Wang and Anderson, 1982]. Diffusion causes contaminants to migrate from zones of higher concentration to zones of lower concentration. One dimensional mass transport in porous media due to diffusion is described by Fick's first law of diffusion. This is given as [Bedient et al., 1994]:

$$f_x = -D_d \frac{dC}{dx} \quad (2.3)$$

where,

f_x = mass flux ($M/L^2/T$),

D_d = diffusion coefficient (L^2/T),

dC/dx = concentration gradient ($M/L^3/T$).

2.2.4 Dispersive Transport

Dispersion of contaminants in porous media increases with increasing heterogeneities in the media [Fetter, 1988]. Dispersion may be broadly classified under two categories; (a) longitudinal dispersion and (b) lateral dispersion.

Longitudinal dispersion is the term given to the kind of dispersion that occurs along the direction of fluid flow. Lateral dispersion is the mixing that occurs normal to the direction of fluid flow. Mechanical dispersion is described as "the product of the average linear velocity and a factor called the dynamic dispersivity" [Fetter, 1988].

$$\text{mechanical dispersion} = a_L v_x \quad (2.4)$$

where,

a_L = dynamic dispersivity factor, and

v_x = average linear velocity.

Modeling contaminant transport involving all these techniques is a complex problem. These complexities dictate that the solution technique be numerical in nature.

2.3 COMPUTER MODELING OF CONTAMINANT TRANSPORT

"A model is a tool designed to represent a simplified version of reality" [Bedient et al., 1994].

Models may be used as predictive tools, interpretive tools, as generic or screening tools. The first step in the process of computer modeling of contaminant transport is developing a conceptual model which includes a description of the physical, chemical and biological processes that dominate the

system to be analyzed. This conceptual model can then be translated to a 'numerical model'. The most common numerical methods used in computer modeling are [Bedient et al., 1994; Wang and Anderson, 1982]:

- Finite difference methods
- Finite element methods
- Collocation methods
- Method of Characteristics
- Boundary element methods

Several computer models have been used in the study of contaminant transport [Reddi et al., 1997]. Listed below are some of the typical goals of a modeling study [Domenico and Schwartz, 1990; Bedient et al., 1994]:

- a. Improving knowledge of a given aquifer system or testing a hypothesis.
- b. To achieve a better understanding of physical, chemical and/or biological processes.
- c. To design remediation systems.
- d. To make predictions for the future or the impact of a proposed activity on a groundwater system.
- e. For better resource management.

Having established the objective of the modeling task the conceptual model needs to be developed. The conceptual model, as mentioned above, gives an overview of the transport system of the problem in hand. The next step in the modeling process is the discretization of the physical layout of the area to be modeled into a grid with numerous cells, blocks, or elements depending on whether finite difference or finite element methods are being used by the computer model.

A finite difference computer code was used in the present study. The discretization used in the contaminant transport modeling is generally the same as that used in the groundwater modeling phase. The contaminant transport model requires the solution of the flow model (the fluid heads and fluid velocities) as partial input. The boundary conditions used in the groundwater flow model, such as constant heads, wells and drains may be used in the contaminant transport model as sources or sinks for the contaminants.

Some errors may be introduced into the model by the modeling method adopted. An understanding of these errors allows for a better interpretation of the modeling results. The errors may be broadly classified as:

- (a.) *Computational Errors*: The numerical approximations made by the solvers in the computer code to solve the governing equations generally give rise to these kind of errors. These errors are influenced by the selection and location of the boundary conditions in the system.
- (b.) *Calibration Errors*: These errors arise primarily due to assumptions made in the model, which may be parametric or dimension related assumptions. These errors can be estimated by comparing the values of a parameter obtained in the results of a simulation to the already known values of the same established from field studies [Bedient et al., 1994].

Every model has certain limitations. These limitations are to be taken into account so as to understand the scope of the model. The most common limitations in numerical models are given below.

- The approximation of the governing differential equations which give rise to computational and residual errors.
- Numerical models require a level of knowledge on the part of it's user because of it's

complexity.

- The preparation and input of data into a numerical model could be time consuming.
- Better, faster computers are needed to solve a numerical model efficiently.

2.3.1 MODFLOW Model

MODFLOW is the groundwater flow modeling software used in the present study. It stands for “A Modular Three-Dimensional Finite Difference Ground-Water Flow Model”. MODFLOW was developed by Michael G. McDonald and Arlen W. Harbaugh in 1985 for the United States Geological Survey. The model simulates groundwater flow in three-dimensions. The program consists of one main program (the Basic Package) and several smaller sub-routines called “modules” [McDonald and Harbaugh, 1988]. This structure has several options in terms of selecting a specific module relevant to the problem, and include it in the model. In this model a block-centered finite-difference approach is used to simulate groundwater flow in the aquifer. In MODFLOW either the “Strongly Implicit Procedure” or the “Slice-Successive Overrelaxation” methods can be used to solve the finite-difference equations involved in a flow model. More details of these methods can be found in the literature.

Given below is the generalized governing differential equation solved by MODFLOW [McDonald and Harbaugh, 1988]:

$$\frac{\partial}{\partial x}(K_{xx}\frac{\partial h}{\partial x}) + \frac{\partial}{\partial y}(K_{yy}\frac{\partial h}{\partial y}) + \frac{\partial}{\partial z}(K_{zz}\frac{\partial h}{\partial z}) - W = S_s\frac{\partial h}{\partial t} \quad (2.5)$$

where,

K_{xx} , K_{yy} and K_{zz} = values of hydraulic conductivity along the x, y and z coordinate axes, which are assumed to be parallel to the major axes of hydraulic conductivity (LT^{-1}),
 h = potentiometric head (L),
 W = a volumetric flux per unit volume and represents sources and sinks of water (T^{-1}),
 S_s = the specific storage of the porous material (L^{-1}),
 t = time (T).

This equation describes the three-dimensional groundwater movement through a heterogeneous and anisotropic porous media. Figure 2.1 shows a schematic of the discretization used by MODFLOW. The different layers in the grid can be simulated as confined, unconfined or as a combination of both confined and unconfined [McDonald and Harbaugh, 1988]. As mentioned earlier, different modules of MODFLOW can be used to simulate different characteristics in the modeler's problem. The following are the "modules" or "packages" available in MODFLOW.

- Basic Package
- Block-Centered Flow
- Well Package
- Recharge Package
- Drain Package
- Evapotranspiration Package
- River Package
- General-Head Boundaries Package
- Strongly Implicit Procedure Package

Explanation

----- Aquifer Boundary

● Active Cell

○ Inactive Cell

Δr_j Dimension of Cell Along the Row Direction. Subscript (J) Indicates the Number of the Column

Δc_l Dimension of Cell Along the Column Direction. Subscript (I) Indicates the Number of the Row

Δv_K Dimension of the Cell Along the Vertical Direction. Subscript (K) Indicates the Number of the Layer

Figure 2.1: Schematic of Discretization Used by MODFLOW (McDonald and Harbaugh, 1988)

- Slice-Successive Overrelaxation Package

The *Basic Package* regulates the functions of the whole model. It contains the switches to select the optional packages. Initial conditions, boundaries and time-step lengths are entered in this package. The *Block-Centered Flow* (BCF) package deals with the calculation of the finite-difference equations such as flow from cell to cell and flow into storage [McDonald and Harbaugh, 1988]. Material properties such as hydraulic conductivities and transmissivities are entered in the BCF package.

Boundary Conditions such as wells, recharge, drains, evapotranspiration and rivers can be included in the model by using the *Well Package*, the *Recharge Package*, the *Drain Package*, the *Evapotranspiration Package* and the *River Package*, respectively. The *General-Head Boundaries Package* includes terms representing general head boundaries to the finite-difference equation. The Strongly Implicit technique and the Slice-Successive Overrelaxation technique are two popular finite-difference solution techniques. The *Strongly Implicit Procedure Package* and the *Slice-Successive Overrelaxation Package* named after the two solution techniques are available in MODFLOW to solve the differential equations [McDonald and Harbaugh, 1988].

The MODFLOW output files give the modeler the resulting fluid heads at each cell location for the set of initial and boundary conditions provided. These heads are needed as partial input for the contaminant transport modeling program to study the contaminant plume.

2.3.2 MT3D Model

MT3D is the contaminant transport model used in the present study. MT3D stands for “A Modular Three-Dimensional Transport Model”. It was developed by C. Zheng [Zheng, 1990]. This model uses a mixed Eulerian-Lagrangian approach to solve the three-dimensional advection-dispersion-reactive equation. The partial differential equation describing the transport of contaminants in groundwater in three-dimensions is given below [Zheng, 1990].

$$\frac{\partial C}{\partial t} = \frac{\partial}{\partial x_i} \left[D_{ij} \frac{\partial C}{\partial x_j} \right] - \frac{\partial}{\partial x_i} (v_i C) + \frac{q_s}{\theta} C_s + \sum_{k=1}^N R_k \quad (2.6)$$

where,

C = the concentration of contaminants dissolved in groundwater (ML^{-3}),

t = time (T),

x_i = the distance along the respective Cartesian coordinate axis (L),

D_{ij} = the hydrodynamic dispersion coefficient (L^2T^{-1}),

v_i = the seepage or linear pore water velocity (LT^{-1}),

q_s = the volumetric flux of water per unit volume of aquifer representing sources (positive) and sinks (negative) (T^{-1}),

C_s = the concentration of the sources or sinks (ML^{-3}),

θ = the porosity of the medium, and

$$\sum_{k=1}^N R_k = \text{a chemical reaction term } (\text{ML}^{-3}\text{T}^{-1}).$$

The velocity field ' v ' in the equation (2.6) needs to be obtained by using a groundwater flow model (eg. MODFLOW). MT3D has several 'modules' or 'packages' like MODFLOW. They are:

- Basic Transport Package
- Advection Package
- Dispersion Package
- Sink/Source Package
- Chemical Reactions Package

The Method of Characteristics (MOC), the Modified Method of Characteristics (MMOC) or a hybrid of the two methods (HMOC) are the methods available to solve the governing differential equation [Zheng, 1990]. MT3D has a modular structure very similar to that of MODFLOW.

It has a *Basic Transport Package* that monitors the functioning of the entire model in addition to several supplementary packages. The *Advection Package* is used to simulate the advection component of the plume transport. The dispersion component of contaminant transport can be included using the *Dispersion Package*. The *Sink/Source Package* can be used to simulate contaminated water leaving or entering the domain through a sink or a source like drains and wells for example. Equilibrium-controlled linear or non-linear sorption and first order irreversible rate reactions can be simulated using the *Chemical Reactions Package* [Zheng, 1990].

MT3D uses the flow field from a groundwater flow model such as MODFLOW as partial input. The output from MT3D provides the modeler with the final concentrations at each cell location in the grid.

2.4 ACID MINE DRAINAGE

Permeability and location of underclays and coals greatly influence the groundwater flow in coal measures. The downward flow of water is severely restricted by the characteristic low permeability of clay. This causes water to flow through the coals above, the coals being relatively more permeable and rich in mineral pyrites. Acid Mine Drainage (AMD) is the result of this phenomena. The contaminant to be modeled in the present study is the AMD.

The formation of AMD can be summarized by the following equation:

There are numerous ill effects on the environment, caused by AMD. Man's water usage is adversely affected by AMD. AMD kills aquatic life due to its low pH value. AMD can dissolve concrete, corrode metals and increase water treatment costs. Drainage channels can be clogged by 'Yellowboy', a by product of AMD [Corbett, 1975]. AMD can dissolve highly toxic metals. The sulfates present in AMD are a source of health hazards too [Maryland DNR, 1969].

There are different methods of abatement of AMD, some of them are already published [Corbett, 1975; Driscoll et al., 1988; Cosby et al., 1986; Ebraheem et al., 1990; Warfinge and Sverdrup, 1988; Batarseh, 1992]. The abatement measures for contamination due to AMD are site specific and are dictated by the migration and concentration trends of AMD at a given site.

2.4.1 Grouting of Mine Cavities to Control AMD

As mentioned in Chapter-1 of this report, there are two major ways in which a solution to the

Acid Mine Drainage problem may be approached. The first way is to stop the AMD from being produced at the source, the second way is to treat the water polluted by the AMD [Maryland DNR, 1969].

Grouting mine cavities to prevent the AMD from being produced would fall under the first of the two abatement approaches mentioned above. The advantages of pumping grout into mine cavities maybe listed as follows:

- (a.) The exposed surface area in the mine cavities is reduced by the grout. This reduces the amount of reactive pyrite (present on the exposed surface) available for the AMD production.
- (b.) The amount of oxygen available for the AMD generation process drops. This a result of more space in the mine cavities being occupied by grout.
- (c.) The volume of grout in the mine cavities allows a lesser amount volume of water to flow through the mine rooms. The lesser the amount of water, the lesser the amount of AMD produced.

Grouting can be carried out in such a manner as to divert the flow path of water and minimize it's contact with the reactive pyrites. The chemical composition of the grout could also be directed against the production and spreading of AMD. The soil liming studies conducted by Warfinge and Sverdrup (1988) is a good example.

The methodology adopted in the present study is described in Chapter 3. It also includes a description of the site characteristics at the Longridge mine.

Chapter 3

METHODOLOGY

3.1 OVERVIEW OF THE METHODOLOGY

The modeling of contaminant transport at the Longridge Mine was undertaken in several steps. The following are the broad steps in which this was done;

- A base map of the Longridge mine area was obtained and idealized. The idealized map was used in the computer modeling phase.
- Several input parameters required for computer modeling were not available. These parameters were estimated on the basis of information available in the literature. Some hydrologic computations were made to provide valuable input for the modeling phase.
- The collected and assumed information was used in the computer modeling phase. Several example problems were solved in two parts: (a) the groundwater flow phase, and (b) the contaminant transport phase.
- Contaminant transport at the Longridge mine was investigated for different grout configurations. These steps are described in detail in the following sections.

3.2 SITE CHARACTERISTICS

3.2.1 General Features

The Longridge mine is located in northern West Virginia as shown in Figure 3.1. Figure 3.2 shows a map of the Longridge mine area. Figure 3.3 shows a map of the Longridge mine without the ground contours and other markings. The mined area is about 2,600 feet (792.5

Figure 3.1: Map Showing the Location of the Longridge Mine in West Virginia

Figure 3.2: Map Showing the Longridge Mine Area

Figure 3.3: Map of the Longridge Mine Layout

meters) in length and about 300 feet (91.44 meters) in width. Several boreholes at the Longridge mine were used to obtain information on the geologic stratification of the area. Some of these borehole logs are shown in Figure 3.4, Figure 3.5, Figure 3.6 and Figure 3.7. Figure 3.8 shows the idealized geologic profile used in the present study. The locations of these boreholes are shown in Figure 3.9.

Different stratifications were present in the borehole logs at different locations in the mine. The materials present in the mine area include: brown clay, sandy brown shale, brown sandstone, sandy gray shale, coal, gray sandstone, fire clay and limy gray shale. The thicknesses of layers of these materials vary from place to place in the Longridge mine area. The heterogeneous nature of this terrain necessitates several assumptions and idealizations to be made for the computer modeling task. Listed below are the assumptions made for the computer modeling task based on the available information:

- (a) The input parameters such as hydraulic conductivities used for the computer modeling tasks were based on typical values collected from reviewed literature. Actual values were not available.
- (b) The vertical discretization used in the computer modeling was based on the idealized borehole log shown in Figure 3.8.
- (c) Each layer shown in the borehole log does not have an equivalent layer in the grid used for computer modeling. It was assumed that the layers used in the computer models are representative of the different materials in the borehole log.
- (d) The idealized map of the Longridge mine as shown in Figure 3.10 was the map used in the computer modeling studies.

Figure 3.4: Geologic column showing the Soil and Rock stratigraphy at core hole location LR-29 of Longridge mine

Figure 3.5: Geologic column showing the Soil and Rock Stratigraphy at core hole location LR-33 of Longridge Mine

Figure 3.6: Geologic column showing the Soil and Rock stratigraphy at core hole location LR-34 of Longridge mine

Figure 3.8 : Idealized Geologic Column to be Used in the Computer Modeling Studies

Figure 3.9: Borehole Locations at the Longridge Mine

Figure 3.10 : Idealized Longridge Mine With the Catchment Area

- (e) It was assumed that the same regular geologic stratification extends throughout the area of the Longridge mine. However, the actual topography indicates a hilly terrain.

3.2.2 Contaminant Measurements at the Longridge Mine

Precipitation at the Longridge mine area infiltrates into the ground, enters the mine and flows out at one end of the mine. The Longridge mine has an outflow of water from one end due to a hydraulic gradient almost throughout the year. Precipitation at the Longridge mine has been measured or estimated and samples of water have been collected at this outflow point [Sack, 1997].

Infiltration and runoff values were calculated for the measured precipitation at the mine as described in section 3.3.2 of this report. The infiltrated precipitation reacts with the mineral pyrites overlying the mine in the presence of oxygen to produce Acid Mine Drainage. Calculations pertaining to the precipitation, runoff, infiltration and comparisons of AMD contaminant levels at the mine as compared to those from literature are described in section 3.3.3.

3.3 INPUT PARAMETERS FOR COMPUTER MODELING

The computer modeling task requires several input parameters, but these parameters were not available at the Longridge mine site. A part of these parameters were obtained from hydrologic computations and partly from the literature reviewed. Certain specific properties of materials not found in reviewed literature were assumed based on some typical material property values. The following sections describe these input properties.

3.3.1 Idealization of the Mine Map

An idealized map of the Longridge mine was made for greater compatibility with the computer models as described below.

1. A base map of the Longridge mine was obtained, the pillars were numbered and the pillar and hallway dimensions were measured.
2. The pillar and hallway dimensions were approximated in the map to obtain a more uniform configuration of dimensions for the modeling phase as shown in Figure 3.10.
3. The actual catchment area was approximated by a rectangular area as shown in Figure 3.10.

3.3.2 Hydrologic Computations

The Hydrologic calculations were made to provide valuable input information for the modeling task. A primary requirement for this purpose was the estimation of the amount of water infiltrating into the ground under the idealized Longridge mine map area. The catchment area (or watershed) for the precipitation around the Longridge mine area was determined to facilitate the infiltration calculations. "Runoff occurs when precipitation or snowmelt moves across the land surface, some of which eventually reaches natural or artificial streams and lakes. The land area that contributes surface runoff to any point of interest is called a watershed." [Viessman et al., 1989]

A map with ground contours of the Longridge mine area was obtained for the purpose of identifying the catchment area. This was done by marking an enclosed area on the contour map around the area of interest with a line. The elevations of area enclosed by the line should be such that, theoretically, any precipitation that occurs in the enclosed area should not run-off outside of the area. Such an enclosed area on a surface contour map is termed as a catchment area or a watershed.

The hydrologic computations were based on the standard Soil Classification Society (S.C.S.) method [Haan and Bakersfield, 1978] using the rainfall data at the mine area made available. According to the S.C.S method:

$$Q = \frac{(P-0.2S)^2}{(P+0.8S)} \quad P > 0.2S \quad (3.1)$$

where,

Q = accumulated runoff volume or the rainfall excess in inches,

P = accumulated precipitation in inches, and

S = a parameter given by,

$$S = \frac{1000}{CN} - 10 \quad (3.2)$$

where,

CN = curve number (The curve number of an area indicates the runoff potential of the area).

Table 3.1 shows the monthly rainfall at the Longridge mine area calculated based on the weekly rainfall rates made available for this study. Table 3.2 shows the computed values of runoff and infiltration per month; these values were calculated based on the precipitation rates in Table 3.1 and the Equations 3.1 and 3.2. Table 3.3 shows the monthly precipitation distribution in gallons. Table 3.4 shows the computed monthly precipitation distribution in percentages.

Figure 3.11 shows a schematic representation of the computed average monthly precipitation distribution at the Longridge mine. Figure 3.12 shows the computed distribution of the precipitation in terms of percentage infiltration and percentage surface runoff. Figure 3.13 shows the distribution of the infiltration and surface runoff as a percentage of the precipitation.

3.3.3 Acid Mine Drainage (AMD) Properties

Typical AMD properties were collected from reviewed literature [Corbett, 1975; Maryland DNR, 1969]. Table 3.5 shows the comparison AMD contaminant levels at the Longridge Mine to the highest AMD levels noted from reviewed literature, and the safe drinking water standards. Table 3.6 shows a comparison of the measured AMD levels [Sack, 1997] at the Longridge mine to the safe drinking water standards. It may be observed based on the comparison in Table 3.6 that the AMD contaminant levels at the Longridge mine exceed the permissible levels of contamination.

3.3.4 Geo-Hydrologic Properties

Geo-Hydrologic properties pertaining to the Longridge mine area were not available. Therefore, these properties were estimated to provide input parameters to the computer modeling task. Some of these properties obtained from reviewed literature are shown in Table 3.7 and Table 3.8. It may be noted that these values vary from place to place and the numbers shown in the table are general and not specific to the Longridge mine area.

Table 3.1: Rainfall Data At Longridge Mine**				
Week No.	Rainfall Data (in.)	Weekly Avg. (in.)	Month	Rainfall in the Month
1	1.97	2.61	August '94	10.44
2	1.33			
3	4.52			
4	1.00	0.806	September '94	4.03
5	0			
6	2.26			
7	0.32			
8	0.45	0.307	October '94	1.23
9	0.64			
10	0.09			
11	0.47			
12	0.03	0.362	November '94	1.45
13	0.05			
14	0.58			
15	0.57			
16	0.25	0.812	December '94	4.06
17	0.78			
18	1.28			
19	1.72			
20	0.24			
21	0.04			

Table 3.1: Rainfall Data At Longridge Mine**

Week No.	Rainfall Data (in.)	Weekly Avg. (in.)	Month	Rainfall in the Month
22	1.50	0.885	January '95	3.54
23	0.62			
24	0.91			
25	0.51			
26	0.45	0.447	February '95	1.79
27	0.17			
28	0.63			
29	0.54			
30	0.54	0.505	March '95	2.02
31	0.84			
32	0.00			
33	0.64			
34	0.15	0.404	April '95	2.02
35	0.20			
36	0.67			
37	0.23			
38	0.77	1.06	May '95	4.24
39	1.03			
40	1.59			
41	1.46			
42	0.16			

Table 3.1: Rainfall Data At Longridge Mine**

Week No.	Rainfall Data (in.)	Weekly Avg. (in.)	Month	Rainfall in the Month
43	0.5*	0.93	June '95	3.72
44	1.36*			
45	-			
46	0.82*	0.425	July '95	1.7
47	0.03			
48	0.41	1.095	August '95	4.38
49	3.34*			
50	0.43*			
51	0.2			
52	0.01	0.358	September '95	1.79
53	0			
54	0.47*			
55	1.11*			
56	0.2*			
57	0.79	0.7	October '95	2.8
58	0			
59	1.56			
60	0.45			
61	0.74	0.76	November '95	3.04
62	1.22			
63	0.32			

Table 3.1: Rainfall Data At Longridge Mine**				
Week No.	Rainfall Data (in.)	Weekly Avg. (in.)	Month	Rainfall in the Month
64	0.35	0.712	December '95	3.56
65	0.08*			
66	0.86*			
67	2.03*			
68	0.24*			
69	1.3*	1.32	January '96	3.96
70	1.52*			
71	1.14*			
72	0.7*	0.345	February '96	1.38
73	0.67*			
74	0.01*			
75	0*			
Average		0.824		3.402

** This table was prepared based in part on the rainfall data provided by the National Research Center for Coal and Energy [Sack, 1997, Progress Report Submitted to NRCCE].

Average Daily Rainfall

Average monthly rainfall = 3.402 inches/month

Average daily rainfall = 3.402/30

= 0.1134 inches/day

Average daily rainfall in feet/day = 0.00945 feet/day

Figure 3.11 Schematic of Average Monthly Precipitation At Longridge Mine

Table 3.2: LONGRIDGE RAINFALL AND INFILTRATION DATA

Catchment Area in sq. ft. 2915200 MO= mine outflow 43200
 4.28 =s 0.856 3.424
 12 7.48

Month	P (inches)	P vol (gal.)	Q (inches)	Q (gal)	I (inches)	I (gal.)	MO(gpm)	MO(gal)	Daily Avg.* (in.)
Sep-94	4.03	7323079.57	1.35152616	2455914	2.67847384	4867165.52	27.08	1169856	0.089282461
Oct-94	1.23	2235083.84	0.03005501	54614.194	1.19994499	2180469.65	11.875	513000	0.038707903
Nov-94	1.45	2634854.93	0.07239146	131545.52	1.37760854	2503309.41	9.29	401328	0.045920285
Dec-94	4.06	7377593.81	1.37167504	2492527.4	2.68832496	4885066.4	38.706	1672099.2	0.08672016
Jan-95	3.54	6432680.32	1.03444227	1879727.8	2.50555773	4552952.51	73.445	3172824	0.080824443
Feb-95	1.79	3252682.99	0.16731032	304026.5	1.62268968	2948656.49	58.665	2534328	0.057953203
Mar-95	2.02	3670625.49	0.24887877	452247.89	1.77112123	3218377.6	86.9475	3756132	0.057132943
Apr-95	2.02	3670625.49	0.24887877	452247.89	1.77112123	3218377.6	33.754	1458172.8	0.059037374
May-95	4.24	7704679.25	1.49418789	2715150.6	2.74581211	4989528.68	70.94	3064608	0.088574584
Jun-95	3.72	6759765.76	1.14816573	2086379.4	2.57183427	4673386.35	25.2	1088640	0.085727809
Jul-95	1.7	3089140.27	0.13901952	252618.11	1.56098048	2836522.16	12.175	525960	0.050354209
Aug-95	4.38	7959079.04	1.59130907	2891633.5	2.78869093	5067445.55	11.8475	511812	0.089957772
Sep-95	1.79	3252682.99	0.16731032	304026.5	1.62268968	2948656.49	9.89	427248	0.054089656
Oct-95	2.8	5087995.73	0.60718766	1103345.8	2.19281234	3984649.94	8.745	377784	0.070735882
Nov-95	3.04	5524109.65	0.73791089	1340888.4	2.30208911	4183221.27	20.4	881280	0.076736304
Dec-95	3.56	6469023.15	1.04690951	1902382.5	2.51309049	4566640.61	84.284	3641068.8	0.081067435
Jan-96	3.96	7195879.68	1.3048234	2371048.5	2.6551766	4824831.14	181.506	7841059.2	0.085650858
Feb-96	1.38	2507655.04	0.0571557	103859.99	1.3228443	2403795.05	118.655	5125896	0.047244439
Average	2.81722222	5119290.95	0.7121743	1294121.4	2.10504792	3825169.58	49.07806	2120172	0.06921

* daily average based on infiltration 'I' rates

P = Precipitation, Q = Runoff, I = Infiltration, MO = Outflow from the Mine

Table 3.3: Flow Data at the Longridge Mine

April 2, 1997

Month	P. vol. gal.	Runoff* gal.	Infiltr.* gal.	M.Outfl. gal.	Losses* gal.
Sep-94	7323080	2455914	4867166	1169856	3697309.5
Oct-94	2235084	54614.19	2180470	513000	1667469.6
Nov-94	2634855	131545.5	2503309	401328	2101981.4
Dec-94	7377594	2492527	4885066	1672099	3212967.2
Jan-95	6432680	1879728	4552953	3172824	1380128.5
Feb-95	3252683	304026.5	2948656	2534328	414328.49
Mar-95	3670625	452247.9	3218378	3756132	-537754.4
Apr-95	3670625	452247.9	3218378	1458173	1760204.8
May-95	7704679	2715151	4989529	3064608	1924920.7
Jun-95	6759766	2086379	4673386	1088640	3584746.3
Jul-95	3089140	252618.1	2836522	525960	2310562.2
Aug-95	7959079	2891633	5067446	511812	4555633.6
Sep-95	3252683	304026.5	2948656	427248	2521408.5
Oct-95	5087996	1103346	3984650	377784	3606865.9
Nov-95	5524110	1340888	4183221	881280	3301941.3
Dec-95	6469023	1902383	4566641	3641069	925571.81
Jan-96	7195880	2371049	4824831	7841059	-3016228
Feb-96	2507655	103860	2403795	5125896	-2722101
Average	5119291	1294121	3825170	2120172	1704998

P = Precipitation, *Calculated Values

Table 3.4: Flow Data at the Longridge Mine

April 2, 1997

Month	P (%)	Q (%)*	I (%)*	MO (%)	Losses (%)*
Sep-94	100	33.53663	66.46337	15.97492	50.48845
Oct-94	100	2.443496	97.5565	22.95216	74.60434
Nov-94	100	4.992515	95.00749	15.2315	79.77598
Dec-94	100	33.7851	66.2149	22.66456	43.55034
Jan-95	100	29.22153	70.77847	49.32351	21.45495
Feb-95	100	9.346945	90.65305	77.915	12.73805
Mar-95	100	12.32073	87.67927	102.3295	-14.6502
Apr-95	100	12.32073	87.67927	39.72546	47.95381
May-95	100	35.24028	64.75972	39.77593	24.98379
Jun-95	100	30.86467	69.13533	16.1047	53.03063
Jul-95	100	8.177619	91.82238	17.0261	74.79629
Aug-95	100	36.33126	63.66874	6.430543	57.2382
Sep-95	100	9.346945	90.65305	13.13525	77.51781
Oct-95	100	21.68527	78.31473	7.425006	70.88972
Nov-95	100	24.27338	75.72662	15.95334	59.77328
Dec-95	100	29.40757	70.59243	56.28468	14.30775
Jan-96	100	32.95009	67.04991	108.966	-41.916
Feb-96	100	4.141718	95.85828	204.4099	-108.552
Average	100	20.57703	79.42297	46.20156	33.22142

P = Precipitation, Q = Runoff, I = Infiltration, MO = Outflow from Mine, *Calculated Values

Total Precipitation = 100%

Figure 3.12 : The Calculated Distribution of Precipitation in the Longridge Mine Area

Figure 3.13: The Distribution of Infiltration at the Longridge Mine Area

Table 3.5: Comparison Of Parameters Of AMD At The Longridge Mine To Drinking Water Standards And Other Mine Water

Contaminant/Parameter	Drinking Water Standards	Highest AMD levels (from literature reviewed)	AMD levels in water from Longridge Mine*
pH	6.5-8.5	2.4	2.4
Acidity (mg/L)	-	35000	1264
Acid Load (lbs/day)	-	18283	967
Alkalinity (mg/L)	-	0	-
Total Iron (mg/L)	0.3	7600	238
Sulfate (mg/L)	<250	31000	
Aluminum (mg/L)	-	51	98.3
Manganese (mg/L)	<0.05	675	29.3
Turbidity (TU)	<5	36	-
Dissolved Oxygen (mg/L)	-	4	-
Total Solids (mg/L)	-	103.5	-
Total Hardness (mg/L)	-	13600	2190
Magnesium (mg/L)	-	2990	124
Calcium (mg/L)	-	507	238
Arsenic (mg/L)	0.05	-	0.92
Selenium (mg/L)	0.01	-	0.84
Lead (mg/L)	0.05	-	0.34
Barium (mg/L)	1.0	-	<0.05
Cadmium (mg/L)	0.01	-	<0.05

*The data shown in this table were not all collected on the same day. The table is a representation of the worst case scenario of each parameter from the data available.

Table 3.6: Comparison Of Parameters Of AMD At The Longridge Mine To Drinking Water Standards

Contaminant/Parameter	Drinking Water Standards	AMD levels in water from Longridge Mine*
pH	6.5-8.5	2.4
Acidity (mg/L)	-	1264
Acid Load (lbs/day)	-	967
Alkalinity (mg/L)	-	-
Total Iron (mg/L)	0.3	238
Sulfate (mg/L)	<250	
Aluminum (mg/L)	-	98.3
Manganese (mg/L)	<0.05	29.3
Turbidity (TU)	<5	-
Dissolved Oxygen (mg/L)	-	-
Total Solids (mg/L)	-	-
Total Hardness (mg/L)	-	2190
Magnesium (mg/L)	-	124
Calcium (mg/L)	-	238
Arsenic (mg/L)	0.05	0.92
Selenium (mg/L)	0.01	0.84
Lead (mg/L)	0.05	0.34
Barium (mg/L)	1.0	<0.05
Cadmium (mg/L)	0.01	<0.05

*The data shown in this table were not all collected on the same day. The table is a representation of the worst case scenario of each parameter from the data available.

Table 3.7 : Geo-Hydrologic Material Properties of Various Layers At The Longridge Mine Site
 Properties Based on [Todd, 1976]

Material	Layer Thickness b (ft)	Hydraulic Conductivity K (ft/day)	Transmissivity T (ft ² /day)
Top Soil	28		
Sandy Shale-1	32	0.24	7.68
Sandstone-1	11	8.5	93.5
Shale-1	81	0.03	2.43
Sandy Shale-2	50	0.24	12
Sandstone-2	13	8.5	110.5
Shale-2	5	0.03	0.15
Coal			
Shale-3	15	0.03	0.45

Table 3.8 : Properties Based on [Fetter, 1988]

Material	Layer Thickness b (ft)	Hydraulic Conductivity K (ft/day)	Transmissivity T (ft ² /day)
Top Soil	28	0.0025 (clay)	0.07
		15 (sand)	420
Sandy Shale-1	32	0.05	1.6
Sandstone-1	11	3.0	33.0
Shale-1	81	0.003	0.243
Sandy Shale-2	50	0.05	2.5
Sandstone-2	13	3.0	39
Shale-2	5	0.003	0.015
Shale-3	15	0.003	0.045

Note: All values indicated in these tables are generally observed values and are approximate

3.3.5 Limitations of Input Parameters At The Mine Site

The various input parameters at the Longridge mine site for the groundwater flow and contaminant transport modeling tasks of this study were not available. These values were estimated from reviewed literature. Thus the accuracy of the results of this study depend upon these estimated parameters.

The exact values of the various contaminants that constitute Acid Mine Drainage were not available at the Longridge mine site. Thus the input concentration levels used in all the contaminant transport analyses of this study was 100%. The resulting contaminant concentrations can thus be easily read in terms of the percentage of the initial concentrations used.

The procedure adopted in the modeling of groundwater flow and contaminant transport is described in Chapter 4. This chapter also discusses the selection of the finite difference grid used in the models for this study.

Chapter 4

COMPUTER MODELING

4.1 GRID SELECTION

The Three-Dimensional finite-difference grid used in this study was selected based on criteria such as the impact of the grid size on the accuracy of the results and computer time considerations. More accurate computer modeling results can be obtained by a finer refinement of the grid within reasonable limits. Grid refinement beyond a certain point has very little impact on the solution of the problem, but has a significant bearing on the required computer time and disk space. Therefore, it becomes necessary to strike a balance between grid refinement, accuracy of the results and the computer time taken to solve each problem.

4.1.1 Horizontal Discretization

Two-Dimensional discretization (rows and columns) of the grid was based on the idealized Longridge mine map shown in Figure 3.10. The idealized Longridge mine map includes the rainfall catchment basin around the mine. The catchment basin shown in Figure 3.10 is the area of interest for the computer modeling task.

A non-uniform grid discretization pattern was adopted for this area after careful review as described below. The smallest dimensions of the grid are found in the cells representing the mine hallway areas. This dimension was the same as the hallway width, which is 5 feet (1.524 meters) in the idealized mine map.

The area of interest being about 2600 feet (792.48 meters) long, a uniform cell length of 5 feet

(1.524 meters) in the x-direction would have given rise to a large number of cells, thus greatly increasing the computer time taken to run each simulation. On the other hand, a uniform cell length of about 100 feet (30.48 meters) would mean only 26 cells in the x-direction, but this would mean that up to 2 pillar lengths and a hallway length would fall under a single cell.

Selection of a non-uniform grid discretization [Figure 4.1] would provide a finer grid in the mined area and a coarser grid in the catchment area around the mine. The lengths of the cells outside the mined area vary between 100 feet (30.48 meters) to 200 feet (60.96 meters). The grid used in the groundwater and contaminant transport modeling studies has 54 cells in the x-direction and 13 cells in the y-direction, which brings the total number of cells in each layer to 702.

4.1.2 Vertical Discretization

After the horizontal discretization was completed, the number of vertical layers were to be established to complete the three-dimensional grid to be used in the computer analysis. The height used for the grid was 85 feet (25.908 meters) and this was based on the idealized geologic profile shown in Figure 3.8. Numerical results obtained by using a 4-layered grid, a 5-layered grid and a 6-layered grid were studied to come up with a suitable vertical discretization.

Three wells and a drain were used in each of these cases in the mine layer for the groundwater modeling phase as shown in Figure 4.2; the basis for using the wells is given in section 4.2 of this report. The well locations were used as contaminant injection points and the drains were used as contaminant sinks, in the contaminant transport phase. A simulation time of 500 days was used for the models. This was considered to be a sufficiently long period of time which would not cause a wastage of valuable computer time.

Figure 4.1 Non-Uniform Grid Discretization in the x-y plane

Figure 4.2 Mine Layer Discretization and location of Drain

Four-Layered Case

Figure 4.2 and Figure 4.3 shows two different views of the grid used in the 4-layered case. Tables 4.1a and 4.1b show the properties and boundary conditions used in the groundwater modeling phase of this problem. Table 4.2 shows the input parameters used in the contaminant transport modeling.

Five-Layered Case

The grid used in the 5-layered case is shown in Figure 4.4. The properties and boundary conditions used in this problem are shown in Tables 4.3a and 4.3b. The input parameters used in the contaminant transport modeling are shown in Table 4.4.

6-layered case

Figure 4.5 shows the grid used in the 6-layered case. Tables 4.5a and 4.5b show the properties and boundary conditions used in this problem. Table 4.6 shows the input parameters used in the contaminant transport modeling.

Observation Points

Several pre-selected observation points were chosen [Figure 4.6] on the grid for the contaminant transport simulations described above. This was done to help understand the contaminant migration trends at these different points at different stages of the transport modeling. The concentrations at different observations points were plotted against the time steps to study their trends, Figure 4.7 and Figure 4.8 show the plots of concentration (as a percentage of the initial concentration) versus time at two different observation points.

Figure 4.3 Vertical Discretization for the 4-layered Case

Table 4.1a: Properties Used in the 4-layered Groundwater Modeling Case

Property	Layer 1	Layer 2	Layer 3		Layer 4	Layer 5
			other	hallways		
Hydraulic Conductivity (ft/day)	0.2	0.4	8	64,000	6	-
Bottom Elevation (feet)	60	35	30		10	0
Leakance (/day)	0.004	0.16	0.1		-	-
Top Elevation (feet)	-	60	35		30	-
Transmissivity (ft ² /day)	-	-	-		-	160

*Hydraulic Conductivity used in the hallways

Table 4.1b: Boundary Conditions Used in the 4-layered Groundwater Modeling Case

(Ref: Case 3xlay)

Boundary Condition	Cell Position	Value
Well	(11,7,3)	800 (cu. ft/day)
Well	(15,5,3)	1200 (cu.ft/day)
Well	(15,11,3)	1200 (cu.ft/day)
Drain	(46,8,3)	1000 (conductance)
Constant Head	(54,64)	34 (feet)

Table 4.2: Properties and Boundary Conditions Used in the MT3D Analysis of 4-layered Case

Property	Layer 1	Layer 2	Layer 3	Layer 4
HTOP (feet)	85	-	-	-
Thickness (feet)	25	25	5	30
Porosity	0.3	0.3	0.3	0.3
Longitudinal Dispersivity	0.02	0.02	0.02	0.02
DH/DL	1	1	1	1
DV/DL	1	1	1	1

- 3 well locations from MODFLOW are constant concentration sources
- the drain location from MODFLOW is a constant concentration sink source

Figure 4.4: Vertical Discretization for the 5-layered Case

Table 4.3a: Properties Used in the 5-layered Groundwater Modeling Case

Property	Layer 1	Layer 2	Layer 3 other	3 hallways	Layer 4	Layer 5
Hydraulic Conductivity (ft/day)	0.2	0.4	8	64,000	6	-
Bottom Elevation (feet)	60	35	30		10	0
Leakance (/day)	0.004	0.16	0.1		0.04	-
Top Elevation (feet)	-	60	35		30	-
Transmissivity (ft ² /day)	-	-	-		-	160

Table 4.3b: Boundary Conditions Used in the 5-layered Groundwater Modeling Case

Boundary Condition	Cell Position	Value
Well	(11,7,3)	800 (cu. ft/day)
Well	(15,5,3)	1200 (cu.ft/day)
Well	(15,11,3)	1200 (cu.ft/day)
Drain	(46,8,3)	1000 (conductance)
Constant Head	(54,64)	34 (feet)

Table 4.4: Properties and Boundary Conditions Used in the MT3D Analysis of 5-layered Case

Note: 500-day simulation

Property	Layer 1	Layer 2	Layer 3	Layer 4	Layer 5
HTOP (feet)	85	-	-	-	
Thickness (feet)	25	25	5	20	10
Porosity	0.3	0.3	0.3	0.3	0.3
Longitudinal Dispersivity	0.02	0.02	0.02	0.02	0.02
DH/DL	1	1	1	1	1
DV/DL	1	1	1	1	1

- 3 well locations from MODFLOW are constant concentration sources
- the drain location from MODFLOW is constant concentration sink source

Figure 4.5 Vertical Discretization for the 6-Layered Case

Table 4.5a: Properties Used in the 6-layered Groundwater Modeling Case

Property	Layer 1	Layer 2	Layer 3 other hallways	Layer 4	Layer 5	Layer 6	
Hydraulic Conductivity (ft/day)	0.2	0.4	8	64,000	6	0.2	-
Bottom Elevation (feet)	60	35	30		20	10	0
Leakance (/day)	0.004	0.16	0.1		0.04	0.004	-
Top Elevation (feet)	-	60	35		30	20	-
Transmissivity (ft ² /day)	-	-	-		-	-	160

Table 4.5b: Boundary Conditions Used in the 6-layered Groundwater Modeling Case

Boundary Condition	Cell Position	Value
Well	(11,7,3)	800 (cu. ft/day)
Well	(15,5,3)	1200 (cu.ft/day)
Well	(15,11,3)	1200 (cu.ft/day)
Drain	(46,8,3)	1000 (conductance)
Constant Head	(54,64)	34 (feet)

Table 4.6: Properties and Boundary Conditions Used in the MT3D Analysis of 6-layered Case

Note: 500-day simulation

Property	Layer 1	Layer 2	Layer 3	Layer 4	Layer 5	Layer 6
HTOP (feet)	85	-	-	-	-	-
Thickness (feet)	25	25	5	10	10	10
Porosity	0.3	0.3	0.3	0.3	0.3	0.3
Longitudinal Dispersivity	0.02	0.02	0.02	0.02	0.02	0.02
DH/DL	1	1	1	1	1	1
DV/DL	1	1	1	1	1	1

- 3 well locations from MODFLOW are constant concentration sources
- the drain location from MODFLOW is constant concentration sink source

Figure 4.6 Observation Points

Figure 4.7: Time Vs. Concentration at Observation Point-2

Figure 4.8: Time Vs. Concentration at Observation Point-5

Based on the results of the groundwater and contaminant transport simulations and the computed variation of contaminant concentrations at observation points, it was concluded that the 5-layered discretization is desirable for the modeling study at the mine site. There is also more uniformity in the 5-layered case with two layers above and two layers below the mine layer. Thus, the 5-layer discretization was selected for the purpose of contaminant transport modeling at the mine site considered in this study.

4.2 Simulating Infiltration and Mine Outflow

Wells were used as the source of water into the model. The total flow rate from the wells was the same as the calculated average infiltration rates [Tables 3.2 and 3.3]. These wells were located in the layer with the mine to simulate the same amount of water entering the model. The recharge package available in MODFLOW could have been used to simulate the precipitation. Using the recharge package though would have meant the unnecessary use of valuable computer time. Several changes in properties and boundary conditions would have to be made in the model, and several trial flow and transport analysis would have to be performed before arriving at the appropriate set of input properties. These appropriate set of input properties would ensure that the calculated amount of infiltration enters the mine layer of the model.

Since the mine layer was the zone of interest for this study, it was decided to go with the well package and represent the infiltration reasonably accurately by using wells instead of using the recharge package and spending time on "tuning" (or making several trial runs before arriving at the desired set of conditions and parameters) the model.

Similarly, the drain package was used to simulate flow of water out of the model. The

properties and other boundary conditions in the model were varied slightly until the computed outflow from the drains in the mine layer was significantly close to the measured mine outflow rates made available for this site [Table 3.2]. Hence, the wells in the model simulate the infiltration while the drains simulate flow out of the mine.

The descriptions of the several models used in the groundwater flow and contaminant transport studies are described in chapter 5 of this report. It also contains the results from these studies and a discussion on their implications.

CHAPTER 5

RESULTS AND DISCUSSION

5.1 CONTAMINANT TRANSPORT SIMULATIONS WITH OPEN MINE CAVITIES

Different grids were used to study the contaminant transport phenomena at the Longridge mine. An effort was made to first study the transport trends with no grout in the mine hallways (or cavities) for each of these grids. Described in this section are the simulations with no grout for a three-layered case and a five-layered case.

5.1.1 Three-Layered Case

Figures 5.1, 5.2, 5.3 and 5.4 show different views of the finite difference grid used for the analysis of this problem. The middle layer of the grid is the coal layer where the mine is located. The mine layer has three wells located in the hallways and a drain outside the mined area as shown in Figure 5.3. These wells and drains simulate infiltration into the mine and outflow from the mine as described in section 4.2 of this report.

The hydraulic conductivity in the hallways was assumed to be significantly high to simulate flow through cavities. A hydraulic conductivity was assumed as 64,000 feet/day (19,507.2 meters/day) in mine cavities. This was the highest value of hydraulic conductivity the model could take without causing numerical convergence problems. The highest possible value of hydraulic conductivity was used since the flow of water in the hallways is free, unobstructed flow. Figure 5.5 shows the computed fluid heads of the groundwater flow simulation for this problem. A regular flow gradient can be observed originating from the wells and continuing to the drain. The hydraulic heads

Figure 5.1: Grid Used in the Three-Layered Case

Figure 5.2: Cross-Sectional View of the Grid Used in the Three-Layered Case

Figure 5.3: Plan View of the Mine Layer in Three-Layered Case

Figure 5.4: Side View of the Grid Used in the Three-Layered Case (Y-Z Plane)

Figure 5.5: Fluid Head Contours in the Mine Layer for the Three-Layered Case

are smaller outside of the mined area in the mine layer. Most of the flow occurs in the mine area from the wells to the drain.

The well locations assumed for modeling fluid flow in the mine layer are used as contaminant injection points and the drain is used as a sink in the contaminant transport modeling phase. Figure 5.6 shows the resulting contaminant plume in the mine layer. As can be seen from this figure, the contaminant plume propagated slightly outside the mine area, but the lateral spread was relatively small. The contaminant contours show a higher rate of transport in the hallways, which is consistent with the flow gradient computed in the groundwater modeling analysis. The spread of contaminants in directions other than those leading to the sink is slow and limited in concentration.

5.1.2 Five-Layered Case

The grid used in the five-layered case is shown in Figure 5.7 and Figure 5.8. This case has three wells and three drains in the mine layer. Figure 5.9 shows the fluid head contours in the mine layer from the groundwater flow analysis of this case. These results show a well established regular pattern of flow from the three wells to the three drains in the mine layer. It can be observed from Figure 5.9 that the fluid head contours are similar in shape to those shown in Figure 5.5 for the three-layered case.

The contaminant concentration contours in the mine layer of this case obtained from the contaminant transport analysis for a duration of 500 days are shown in Figure 5.10. The contours from the contaminant transport modeling compare well with the results for the previous case. The contaminant transport is highest in areas in the mine hallways where the groundwater flow

Mttop1 : 500.000

Figure 5.6: Contaminant Concentration Contours in the Mine Layer for the Three-Layered Case

Figure 5.7: Grid Used in the Five-Layered Case

Figure 5.8: Cross-Sectional View of the Grid Used in the Five-Layered Case

Figure 5.9: Fluid Head Contours in the Mine Layer for the Five-Layered Case

Figure 5.10: Contaminant Contours in the Mine Layer for the Five-Layered Case

gradients are the highest. It can also be observed from the figure that the concentrations are higher in the mine area and decrease rapidly outside the mine area. The spreading of contaminants in the lateral direction is slower.

5.2 CONTAMINANT TRANSPORT SIMULATIONS WITH GROUT IN THE MINE CAVITIES

Grout made up of fluidized-bed combustion ash can be used to backfill mine cavities to control the AMD contamination problems. Several grout backfill configurations were simulated in the mine cavities to study their impact on the contaminant transport patterns in the mine. Described in this section are the results from these studies.

5.2.1 Influence of Grout in Different Hallway Locations

Three different cases with different grout locations were simulated to study their influence on the contaminant transport trends. It has been assumed in these studies that the grout will have a low hydraulic conductivity. Details of different configurations are shown in Table 5.1. The grids and the mine layer discretization used in these cases are shown in Figures 5.1, 5.2 and 5.3. It can be seen from Figure 5.3 that the mine layer has three wells located in the mine hallways and a drain located outside of the mine. The simulated wells are the only source of water infiltration into the mine cavity.

In the Case-1 (btop2) (see Table 5.1 for a description of the different cases), all the hallways in the mine layer were completely grouted and there were no cavities. This was simulated using low hydraulic conductivities in the hallways. Figure 5.11 shows the resulting fluid heads from this

Table 5.1: Prefixes Used for Different Modeling Cases and Their Description

S. No.	Case Prefix	Description
1	lxtop2	3-layered case, all hallways grouted in mine layer
2	lxtop3	3-layered case, has the two inner hallways of the four hallways as open cavities, the two outer hallways are grouted [Figure 5.15]
3	lxtop4	3-layered case, has a small part of the two outer hallways grouted [Figure 5.19]
4	hyd1	5-layered case simulating grout fill 1 foot high, in the hallways
5	hyd2	5-layered case simulating grout fill 2 feet high, in the hallways
6	hyd3	5-layered case simulating grout fill 3 feet high, in the hallways
7	hyd5	5-layered case simulating grout fill 5 feet high, in the hallways
8	floor1	5-layered case simulating 25% of the mine floor covered with grout [Table 5.3]
9	floor2	5-layered case simulating 50% of the mine floor covered with grout [Table 5.3]
10	floor3	5-layered case simulating 75% of the mine floor covered with grout [Table 5.3]
11	floor4	5-layered case simulating 100% of the mine floor covered with grout [Table 5.3]
12	fplat1	5-layered case simulating a lateral fracture zone ($K=10$ times more than base case)
13	fplat1b	5-layered case simulating a lateral fracture zone ($K=100$ times more than base case)
14	frvert1	5-layered case simulating a vertical fracture zone directly above the mine layer ($K=10$ times more than the base case)
15	frvert1b	5-layered case simulating a vertical fracture zone directly above the mine layer ($K=100$ times more than the base case)
16	fr3d1	5-layered case simulating a three-dimensional fracture zone ($K=10$ times more than the base case)

Figure 5.11 Fluid Head Contours in the Mine Layer from Case - 1

simulation. It can be observed that there is a small hydraulic gradient around the three wells. This gradient falls gradually as the distance from the well increases and the distance to the drains decreases. The gradient seems to be almost equal in all the directions around the wells.

Figure 5.12 shows the resulting contaminant plume from the contaminant transport phase of this problem. The contaminant transport analysis was carried out for a 500 day period. The well and drain locations from the flow modeling phase are used as contaminant sources and sinks, respectively. It can be observed from Figure 5.12 that the transport seems to be influenced by the flow gradients. The grout fill in the mine cavities apparently limits the spreading of the contaminant in the mine area. There is a higher concentration around the wells where the flow gradient was observed to be high and a lower concentration at other locations in the mine layer.

The Case-2 ("lxtop3") has the two inner hallways of the four hallways in the mine as open cavities. The two outer hallways are simulated to be filled completely with grout. This grout fill scenario is shown in Figure 5.13. Figures 5.14 shows the fluid head distribution from the flow modeling results of this case. It can be observed from this figure that there is a higher flow gradient in the two inner hallways (no grout fill) and outside of the mine area than in the two outer hallways (with grout fill).

The contaminant transport study for this case was carried out for a duration of 500 days. Figure 5.15 shows the contaminant plume from the contaminant transport modeling results of this case. It can be observed that the transport occurs largely in the two inner hallways where there is no grout. The spreading of the plume outside of the mine area is limited. The simulated grout in the outer hallways seems to largely direct the transport of the contaminant to the two inner hallways.

The Case-3 (lxtop4) simulates a case where small parts of the two outer hallways are filled

Figure 5.12 Contaminant Contours in the Mine Layer of Case 1

Figure 5.13: Grouting Scenario in Case-2 (Two Outer Hallways with Grout Fill)

Figure 5.14 Fluid Head Contours in the Mine Layer from Case 2

Figure 5.15 Contaminant Plume in the Mine Layer of Case 2

with grout as shown in Figure 5.16. The fluid head contours of the flow modeling simulations are shown in Figure 5.17. It can be observed from this figure that the flow path (which is perpendicular to the head equipotential contours) occurs straight through the mine from the wells towards the drains until the end of the mine. The simulated grout in the two outer hallways apparently redirects the flow-path to the inner hallways as well as towards the outside of the mine.

The contaminant concentration contours corresponding to 500 days from the contaminant transport analysis of this case are shown in Figure 5.18. It can be observed that the grout causes a bottleneck at end of the mine where the mine has some grout fill in the outer hallways. The grout fill seems to prevent the transport of the contaminant through it and force most of it to the inner hallways. Figure 5.18 also shows that the plume opens out once it passes the grout filled area of the mine. This shows that grout fill has a potential to be used in the mine cavities to stop or to redirect the transport of AMD.

5.2.2 Influence of Exposed Mine Surface Area on Contaminant Transport

Exposed areas in coal mines contain mineral pyrites that aid in the production of AMD. Pumping grout into the mine cavities is an option to reduce the formation of AMD by reducing the exposed mine area. Computed values of the exposed surface areas in the Longridge mine based on the idealized Longridge mine map [Figure 3.10] are shown in Table 5.2. Based on this information, groundwater and contaminant transport studies were carried out to study the impact of grouting on the transport of AMD at the Longridge mine.

Figure 5.16: Grouting Scenario in Case-3 (Part of Two Outer Hallways with Grout Fill)

Figure 5.17 Fluid Head Contours in the Mine Layer of Case 3

Figure 5.18 Contaminant Concentration Contours in the Mine Layer of Case 3

Table-5.2: Exposed Area in the Mine as a Function of the Grout Height

Grout Height (feet)	Exposed Pillar Area (sq. ft.)	Exposed Roof Area (sq. ft.)	Exposed Floor Area (sq. ft.)	Total Exposed Area (sq. ft.)	Hydraulic Conductivity (ft./day)	% of Exposed Area (as a % of maximum exposed area)
0	70,800	200,400	200,400	471,600	64,000	100.00
1	56,640	200,400	0	257,040	51,200	54.50
2	42,480	200,400	0	242,880	38,400	51.50
3	28,320	200,400	0	228,720	25,600	48.50
4	14,160	200,400	0	214,560	12,800	45.50
5	0	0	0	0	8	0

5.3 Influence of Grout Fill Height on Contaminant Transport

In these studies, it was assumed that grouting would be carried out over the entire exposed floor area of the mine. Different grout heights were used in different cases as shown in Table 5.2. Figure 5.19 shows the variation of the "percentage of exposed area" with the "grout height" as established in Table 5.2. Figure 5.19 shows the trends of contaminant concentration that could be expected for different heights of grout fills in the mine. A steep drop can be seen in the percentage of exposed area between grout heights of 0 and 0.5 feet (0.1524 meters). This is because, there is a significant change in the total exposed area in the mine from no grout to 0.5 feet (0.1524 meters) of grout since the mine floor was assumed to be fully covered.

After this point, a gradual fall in the percentage of exposed area can be seen for every foot increase in the height of grout. Then again a sharp drop between the grout heights of 4.5 feet (1.3716 meters) and 5 feet (1.524 meters) can be observed. This is because the mine opening is 5 feet (1.524 meters) high and when the grout fill height reaches 5 feet (1.524 meters), there is effectively no exposed area in the mine. However, complete closure of the mine voids (i.e. 100% fill) may be a practically impossible task.

Different groundwater modeling and contaminant transport studies were carried out based on the data shown in Table 5.2. The hydraulic conductivities used in each groundwater modeling analysis are as shown in this table for different grout heights. The corresponding calculated percentage of exposed areas were used as the initial contaminant levels in the different contaminant transport modeling cases. This was done in concurrence with the assumption that the exposed area in the mine is directly proportional to the concentration of the contaminant generated (AMD) in the mine. Since the initial concentrations of 100% were used when there was no grout pumped into the

Figure 5.19: Height of Grout versus Percentage of Exposed Area

mine (exposed area=100%), the percentage of exposed area as shown in Table 5.2 was used as the initial concentration in the contaminant transport modeling analyses. All the contaminant transport modeling cases described below were simulated for a period of 500 days.

Case-1 (hyd1) corresponds to the scenario where there is a grout fill 1 foot (0.3048 meters) high in the mine cavities throughout the mine. Case-2 (hyd2) corresponds to the scenario where there is a grout fill of 2 feet (0.6096 meters) throughout the mine hallways. The resulting contaminant plumes from the contaminant transport modeling analyses of Case-1 and Case-2 are shown in Figure 5.20 and Figure 5.21. These two figures show similar contaminant transport trends with a marginal decline in the contaminant concentrations from Case-1 to Case-2. This may be because the starting concentrations in Case-2 are slightly lower than those in Case-1.

Figure 5.22 shows the contaminant contours resulting from the contaminant transport analysis of Case-3 (hyd3) corresponding to 3 feet (0.9144 meters) of grout height in the mine hallways throughout the mine. The contaminant concentrations seen in Figure 5.22 are lesser than those in the previous cases as expected. This is because the initial contaminant concentrations are lower as shown in Table 5.2. The contaminant plume seen in Figure 5.22 seems to be restricted to the mine area unlike the plumes from Case-1 [Figure 5.20] and Case-2 [Figure 5.21]. This shows that the lateral extent of the contaminant plume becomes smaller as the grout backfill height is increased.

Figure 5.23 shows the resulting contaminant concentration plume from the contaminant transport analysis of Case-4 (hyd4) which corresponds to the grout fill height of 4 feet (1.2192 meters). This figure suggests that the increase in the simulated grout fill height in the mine cavities causes the contaminant to spread to the outside of the mine area. This may be due to the fact that longitudinal flow may be restricted as the grout backfill height is increased beyond a certain limit. This

Figure 5.20 Contaminant Plume in the Mine Layer of Case-1 (hyd1)

Figure 5.21 Contaminant Plume in the Mine Layer of Case-2 (hyd2)

Figure 5.22 Contaminant Plume in the Mine Layer of Case-3 (hyd3)

Figure 5.23 Contaminant Plume in the Mine Layer of Case-4 (hyd4)

could also be a result of the numerical procedure used to simulate fluid influx into the mine cavity.

Various observation points were chosen in the mine layer in the contaminant transport modeling cases [Figure 4.15] in order to obtain contaminant concentration data at those points during the course of the simulation at various time-steps. Figures 5.24 and 5.25 show the Time versus Concentration plots at 2 different observation points for all the 6 cases in this study. "H0" in the legends of the graphs corresponds to a grout fill height of 0, "H1" corresponds to a grout fill height of 1 foot (0.3048 meters) and so on.

Figure 5.24 shows that the concentrations are identical for the first 350 days for all the different cases in the study. The levels of concentration vary significantly between the 400 day mark and the 500 day mark. This suggests that, at the location of the observation point-3, it takes about 350 days for the concentrations to start increasing in magnitude and spread. The curve that corresponds to the highest concentration is the one that represents the case with no grout fill in the hallways. The curve that corresponds to the lowest concentration at the end of 500 days is the one that represents the case where the hallways are completely filled by grout.

Figure 5.25 shows the concentration variation at observation point-5 for the six cases. It can be observed that the concentrations rise steeply for all the cases for the first 50 days. The concentrations continue to increase uniformly in all the cases though not as steeply for the rest of the 450 days. The bottom-most curve (H5) in the graph represents the case where the mine cavities are completely filled with grout. The second curve from below (H4) represents the case with a grout fill of 4 feet (1.2192 meters) in the mine hallways and so on. The top-most curve (H0) in the graph represents the case where there is no grout fill in the mine hallways.

This graph shows that the contaminant (i.e., AMD) concentration in the mine is strongly

Observation Point - 3

Figure 5.24: Time versus Concentration for Cases With Different Hydraulic Conductivities at Observation Point-5

Figure 5.25: Time versus Concentration for Cases With Different Hydraulic Conductivities at Observation Point-5

dependent on the grout backfill height. The grout serves to reduce the production of AMD and restricts its transport in the underground mine cavities.

5.4 Influence of Floor Area Covered by Grout on Contaminant Transport

Different scenarios described in this section were developed to study the influence of different floor areas covered by the grout backfill on the transport of contaminants. Table 5.3 shows the relationship between different percentages of the floor area covered and the percentage of total exposed area in the mine. Table 5.3 is based on a grout height of 1 foot (0.3048 meters).

Several simulations were made based on the data in Table 5.3 to study the impact of different magnitudes of exposed floor areas covered by grout in the mine. The base flow case for this study is the same as the 5-layered case described in section 5.1.2 of this chapter.

It was assumed in these calculations that the amount of contaminant (AMD) produced is directly dependant on the magnitude of exposed mine area. Thus the constant concentration sources in the contaminant transport simulations were the same as those in the last column in Table 5.3. Figures 5.26, 5.27, 5.28 and 5.29 show the resulting contaminant contours in the mine layer for the contaminant transport analyses of Case-1 (floor1), Case-2 (floor2), Case-3 (floor3) and Case-4 (floor4), respectively.

Case-1, Case-2, Case-3 and Case-4 simulate 25%, 50%, 75% and 100% of the mine floor filled with grout, respectively. The contaminant contours from the four cases all show a similar pattern but have different concentrations at different points in the mine layer. The contaminant plume representing Case-1 [Figure 5.26] is larger than the plumes representing the other three cases. Only

Table-5.3 : Exposed Area in the Mine as a Function of Grout Covered Floor Area*

% of Floor Area Covered by Grout	Exposed Pillar Area (sq. ft.)	Exposed Roof Area (sq. ft.)	Exposed Floor Area (sq. ft.)	Total Exposed Area (sq. ft.)	% of Exposed Area (as a % of maximum exposed area)
0	70,800	200,400	200,400	471,600	100.00
25	67,260	200,400	150,300	417,960	88.62
50	63,720	200,400	100,200	364,320	77.25
75	60,180	200,400	50,100	310,680	65.88
100	56,640	200,400	0	257,040	54.50

*For a 12 inch Grout Fill Height in the Mine Cavities

Figure 5.26 Containment Contours in the Mine Layer Corresponding to Case-1 (floor1)

Figure 5.27 Containment Contours in the Mine Layer Corresponding to Case-2 (floor2)

Figure 5.28 Containment Contours in the Mine Layer Corresponding to Case-3 (floor3)

Figure 5.29 Containment Contours in the Mine Layer Corresponding to Case-4 (floor4)

25% of the mine floor is covered by grout in Case-1 allowing for the transport of a greater amount of contaminants. The resulting concentrations are on the decline from Case-1 to Case-4 as expected. The initial concentration of contaminants for Case 1 has the highest value of all four cases.

These results suggest that the transport of the AMD contaminant and its concentrations may be effectively lowered by covering the entire floor area of the mine with grout fill. This method of covering the mine floor with grout fill probably has a greater impact on the concentrations of the AMD rather than its transport. While the grout (over the entire floor) covers a significant area that would otherwise participate in the production of AMD, it may not be as effective in arresting its transport. It could probably be made more effective in controlling the transport by increasing the height of the grout fill at certain locations in the hallways as discussed in section 5.3 of this report. These locations would have to be determined by carrying out parametric studies.

5.5 INFLUENCE OF CRACKS AND FISSURES AROUND THE MINE CAVITY ON CONTAMINANT TRANSPORT

Underground mines develop cracks and fissures both, when mining activity is ongoing as well as when it is abandoned. "Destressing of the immediate roof above a mine opening produces a dilated rock zone of fracturing and strata separation, with resultant rapid localized draining of permeable materials" [Booth, 1986]. These fracture zones significantly increase the hydraulic conductivity of the material and thus influences groundwater flow. The nature and behavior of these fracture zones are irregular and difficult to predict. The cracks and fissures developing in these fracture zones are influenced by the method of mining adopted.

This was a factor considered in the computer modeling studies. An attempt was made to

study the impact of cracks and fissures on the contaminant transport patterns in the Longridge Mine. The base case was assumed to be the same case as the case described in section 5.1.2. The influence of cracks and fissures on contaminant transport in the mine were classified under three different scenarios as described below.

5.5.1 Lateral Fracture Zone

These studies were carried out by assuming a lateral zone (two-dimensional zone) of fracture in the mine layer as shown in Figure 5.30. Since hydraulic conductivities increase in fracture zones, the objective of the study described in this section was to observe the contaminant plumes in zones of high hydraulic conductivities.

Case-1 (flat1):

The hydraulic conductivity in the fracture zone in Case-1 was increased by 10 times that of the value used in the base case, to simulate flow through fractures. Figure 5.31 shows the computed fluid heads in the flow model for this case in the mine layer. A large flow gradient can be observed in this figure from the wells to the drain. A small amount of flow seems to occur towards the outside of the mine. Most of the flow apparently occurs directly from the wells to the drains. This may be because of the high hydraulic conductivity field in the mine area forcing the flow in the direction of the drains.

Figure 5.32 and Figure 5.33 show the contaminant concentrations at the end of 250 days and at the end of 500 days (the end of the simulation) of transport, for this case. The plume shown in Figure 5.32 which represents the extent of contaminant transport after 250 days is smaller than the plume in Figure 5.33 which represent the extent of transport after 500 days. The concentrations in

Figure 5.30: Fracture Zone for Case-1 (Frlat 1)

Figure 5.31 Fluid Head Contours in the Mine Layer from Case-1 (frlat1)

Figure 5.32 Contaminant Concentrations Contours in the Mine Layer After a Period of 250 Days, from Case -1 (frlat1)

Figure 5.33 Contaminant Concentrations Contours in the Mine Layer After a Period of 500 Days, from Case -1 (frlat1)

the 500-day plume are also higher than those in the 250-day plume. It may be interesting to note that the plume for the 250-day case seems to have traveled quickly and reached the simulated drains.

Case-2 (fflat1b):

The fracture zone described in Case-1 was the one that was used in this case too. The hydraulic conductivities used here in the fracture zone were 100 times that of those used in the base case (10 times more than that in Case-1). The fluid heads from the output of the flow simulation for Case-2 are shown in Figure 5.34. This figure illustrates a high flow gradient between the wells and the drains through the mine, very similar to the flow pattern from Case-1 [Figure 5.31].

The contaminant concentrations in the mine layer at the end of 250 days and at the end of 500 days (end of the simulation) of transport are shown in Figure 5.35 and Figure 5.36, respectively. These results seem to indicate that the contaminant propagates faster in Case-2 than in Case-1 because of the increased hydraulic conductivities in the fracture zone. It can be observed that the 250-day plume in this case is bigger and carries a higher concentration than the 250-day plume in Case-1. The 500-day plumes for Case-1 and Case-2 look very similar but the concentrations in the 500 day plume from Case-2 are higher than those in Case-1. This appears to be because of the increased hydraulic conductivities in the fracture zone from Case-1 to Case-2. The analysis of the lateral fracture zone suggests that cracks and fissures would increase the rate of propagation of contaminants in the Longridge mine.

5.5.2 Vertical Fracture Zone

This scenario was used to simulate a fracture zone extending in the vertical direction. Most

Figure 5.34 Fluid Head Contours in the Mine Layer from Case-2 (frlat1b)

Figure 5.35 Contaminant Concentrations Contours in the Mine Layer After a Period of 250 Days, from Case -2 (frlat1b)

Figure 5.36 Contaminant Concentrations Contours in the Mine Layer After a Period of 500 Days, from Case -2 (frrlat1b)

cases of fracture zones around underground mines occur above the roof of the mine and in the mine layer. The cracks and fissures may also develop below the floor of an underground mine.

Case-3 (frvert1)

Case-3 was used to simulate a fracture zone in the layer, directly above the mine layer. The hydraulic conductivity in the overlying layer was assumed as ten times that of those in the base case to simulate flow through the fracture zone. Figure 5.37 shows the resulting fluid heads from the flow modeling analysis for this case. The figure shows a strong flow field in the mine from the wells to the drain very similar to those in Case-1 and Case-2.

Figure 5.38 and Figure 5.39 show the contaminant concentrations from the contaminant transport modeling study for this case at the end of 250 days and 500 days of transport, respectively. The contaminant plumes from Figures 5.38 and 5.39 are not significantly different from the plumes in Case-1. This may be due to the fact that the source of the flow of water is within the mine layer and the fractures in the layer above may not be significantly influencing the flow field in the mine layer. Thus, the fracture zone above the mine layer may not significantly affect the transport of the contaminant in the mine layer.

Case-4 (frvert1b)

The fracture zone in case-4 was assumed to be the same fracture zone as that used in Case-3 described above. The hydraulic conductivities used in the fracture zone in this Case are hundred times more than those used in the base case. Figure 5.40 shows the resulting fluid heads from the flow modeling study for this case. The flow in the mine layer in this case is similar to those in Case-1, Case-2 and Case-3. Figure 5.41 and Figure 5.42 show the contaminant concentrations for this case

Figure 5.37: Fluid Head Contours in the Mine Layer for Case-3 (Frvert1)

Figure 5.38 Contaminant Concentrations Contours in the Mine Layer After a Period of 250 Days for Case -3 (frvert1)

Figure 5.39 Contaminant Concentrations Contours in the Mine Layer After a Period of 500 Days for Case -3 (frvert1)

Figure 5.40 Fluid Head Contours in the Mine Layer for Case-4 (frlat1b)

Figure 5.41 Contaminant Concentrations Contours in the Mine Layer After a Period of 250 Days for Case -4 (frvert1b)

Figure 5.42 Contaminant Concentrations Contours in the Mine Layer After a Period of 500 Days for Case -4 (frvert1b)

in the mine layer at the end of 250 days and at the end of 500 days of contaminant transport. The 250-day plume in this case is comparable to the 500-day plumes in the earlier cases.

It can be observed from Figure 5.41 that the contaminant plume here is significantly developed after 250 days and the concentrations are higher than the 250-day plumes of the earlier cases. This is probably due to the high hydraulic conductivities simulating the vertical fracture zone.

5.5.3 Three-Dimensional Fracture Zone

These studies were carried out by assuming a fracture zone which extended both laterally and vertically from the mine i.e. combining the lateral zones described in the first four cases described in the present section (section 5.5). The hydraulic conductivities in the fracture zones in Case-5 (fr3d1) are assumed as ten times the hydraulic conductivities used in the base case. The fluid heads from the flow simulations of this case are shown in Figure 5.43. The fluid head distribution is very similar to those of the lateral and vertical fracture zone cases. The contaminant plumes for this case at the end of 250 days and 500 days are shown in Figure 5.44 and 5.45, respectively. The 250 plume shows a high rate of contaminant transport similar the 250-day plume from Case-4. The concentrations of the two cases are similar too.

The studies involving the effect of fracture zone on the contaminant transport patterns in the Longridge mine seem to indicate a definite trend. The rate at which the contaminants get transported across the mine and reach the drains (which simulate outflow from the mine) appears to increase with an increase in the extent of the fracture zones.

The fracture zones in the mine layer have a greater effect on the contaminant plume propagation than the fracture zones above the mine layer. This may be because the source and sink

Figure 5.43 Fluid Head Contours in the Mine Layer for Case-5 (fr3d1)

Figure 5.44 Contaminant Concentrations Contours in the Mine Layer After a Period of 250 Days for Case -5 (fr3d1)

Figure 5.45 Contaminant Concentrations Contours in the Mine Layer After a Period of 500 Days for Case -5 (fr3d1)

for the contaminant in the models used for the analyses were in the mine layer. However, the three-dimensional fracture zone seems to aid the transport of contaminant more than the other kinds of fracture zones.

Chapter 6 of this report contains a summary of the study described in this report. It also includes the conclusions of from this study and has some recommendations for future research in this area.

CHAPTER 6

SUMMARY AND CONCLUSIONS

Mine subsidence and acid mine drainage (AMD) are two potential problems caused by abandoned underground coal mines. Backfilling of underground mine cavities with a grout is considered as one of the methods to reduce AMD problems. The major objective of this study was to analyze the contaminant transport patterns using a computer model. The study presented in this report deals with the modeling of contaminant (AMD) transport around an underground mine cavity. Also, the computer model was used to study the influence of grouting on the contaminant transport around the mine cavity.

A contaminant transport model requires several input parameters including the solution from a groundwater flow model and some hydro-geologic properties of the aquifer. There are certain factors that limit the scope of the present study dealing with the contaminant transport modeling at the Longridge mine. These factors are: (a) the idealization of the geometry of the mine site and mine configuration, (b) the idealization of the geologic profile at the mine site, and (c) assumptions on the groundwater and material data since the actual data was not available. Therefore, the validity of modeling results depends on the accuracy of the site-specific material properties and groundwater data, which were not available.

In the present study, MODFLOW was chosen as the three-dimensional groundwater flow model. This computer code has been widely used in groundwater flow modeling studies. MT3D was

chosen as the three-dimensional contaminant transport model in this study. Detail descriptions of both these models were presented in Chapter 2 of this report. The site characteristics at the Longridge mine, geologic profiles, idealizations, and assumed geologic and material properties were presented in Chapter 3 of this report. Hydrologic calculations were made on the basis of available rainfall data at the Longridge mine in order to provide input information such as the amount of infiltration to the modeling study. These calculations show that about 20% of the total precipitation infiltrates the mine area leading to AMD production. Chapter 4 contains details on the computer modeling study, selection of three dimensional grids, and numerical procedure used for simulating infiltration and mine outflow in the flow model. Results show that the use of three-layer and five-layer discretizations provide almost similar results, even though the five-layer discretization requires more computational effort. Both these idealizations were used in the contaminant transport modeling study at the Longridge mine.

Chapter 5 contains the results from the contaminant transport simulations. Results show that the contaminant concentrations are the highest in the mine hallways where the groundwater flow gradients are also the highest. It can also be observed from the results that the concentrations are higher in the mine cavity and that the lateral spread of the contaminant plume is relatively small. The results from the simulations of grout backfill configurations in outer hallways show that the contaminant transport occurs largely in the inner hallways and that the lateral spread of the contaminant plume outside of the mine area is limited. The results also show that lateral spread outside the mine area is higher, if the hallways are blocked by grout. The influence of grout backfill height on the contaminant transport was also studied. The results show that the increase of simulated

grout backfill height in the mine cavity causes the contaminant to spread outside of the mine area. This may be due to the restrictions on the longitudinal flow caused by increased backfill height. However, the contaminant concentration is significantly reduced as the backfill height is increased. The grout serves to reduce the production of AMD and restricts its transport in the mine cavity.

The influence of the floor area covered by the grout on the contaminant transport was investigated by varying the mine floor area covered by the grout backfill while keeping the backfill height constant. The results suggest that the transport of the AMD contaminant and its concentration may be effectively lowered by covering the entire floor area of the mine with grout backfill. This method of covering the mine floor with grout backfill probably has a greater impact on the concentration of AMD rather than its transport. The contaminant transport can be changed by increasing the height of grout fill at certain locations of the hallways. These locations would have to be determined by carrying out parametric studies using the computer models.

Influence of cracks and fissures around the mine cavity on the contaminant transport was studied by varying the size of fracture zone and its hydraulic conductivity. The results show that the size of the lateral fracture zone would increase the rate of contaminant propagation. The rate at which the contaminants get transported in the mine appears to increase with an increase in the extent three-dimensional of fracture zones around the mine. The fracture zones in the mine layer have a greater influence on the contaminant plume propagation than the fracture zones above the mine layer.

The study shows that computer models can be used to study the contaminant transport patterns

in abandoned underground mines and the influence of grouting on contaminant transport. However, it should be noted that the computer models require significant site specific data and material properties, which were not available at the Longridge mine. These input data need to be established to have reliable, site-specific predictions. In the absence of site-specific data, the general trends and patterns of contaminant transport around a mine cavity and the influence of different grouting scenarios on contaminant transport can be obtained by using these models.

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