# WEST VIRGINIA UNIVERISTY

FINAL REPORT

# OMEGA MINE INJECTION PROGRAM

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# OMEGA MINE INJECTION PROGRAM

**Final Report, April 2001**

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The work was funded by West Virginia Division of Environmental Protection (WVDEP), Allegheny Energy Supply, Anker Energy, Consol. Inc., U.S. Department of Energy (U.S. DOE), Office of Surface Mining, Reclamation and Enforcement (OSMRE) and the Electric Power Research Institute (EPRI). The author wishes to thank these organizations for their funding and support particularly WVDEP Project Manager, Dave Broschart. Mr. Broschart was instrumental in organizing the financial support for the project and spent many hours in the planning, execution and evaluation phases of this project.

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Through their efforts they greatly improved the report and are not responsible for any remaining deficiencies.

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# EXECUTIVE SUMMARY

The Omega Mine, located 9.6km (six miles) south of Morgantown, West Virginia, closed in the late 1980's. The underground mine workings in the Upper Freeport Coal covered 68 ha (170 acres). The coal and associated rock strata dip to the northwest at 9 percent. Within the mine the Upper Freeport coal varied in elevation from 533m (1750 ft) at the south end to 451m (1480 ft) at the north end. As the lower (down dip) portions of the mine began filling with water, acid mine drainage (AMD) flowed from the coal outcrop into the Cobun Creek watershed. Cobun Creek is utilized for water supply by the City of Morgantown. In July, 1989 an estimated discharge of 57 liters per minute (15 gallons per minute [gpm]) turned 2.4km (1.5 miles) of previously pristine Cobun Creek orange and killed all aquatic life. In response to this seepage, the mine pool in the lowest section of the mine (the North Lobe), was drained by two horizontal borings. The discharges from the horizontal borings and other mine seeps were collected and piped to the mine's AMD treatment plant. A local citizens' group operated the treatment plant with bond forfeiture funds. When the funds were depleted, the West Virginia Department of Environmental Protection (WVDEP) took over operation of the treatment plant in January, 1995. The cost of treatment is $300,000 per year.

In March 1996, an agreement was reached between WVDEP; Monongahela Power Company, a subsidiary of Allegheny Energy Supply; Anker Energy Corporation; Consol, Inc.; United States Office of Surface Mining Reclamation and Enforcement (OSMRE) and the Electric Power Research Institute to contribute funds for reducing AMD from the Omega Mine. The U.S. Department of Energy (USDOE) joined as a project sponsor in the spring of 1998.

Existing water quality data indicated that the majority of the acidity in the AMD at the site came from the North Lobe of the mine. The North Lobe encompasses approximately 10.4 ha (26 acres) of the 68 ha (170 acres) Omega Mine. The agreement dictated that only the North Lobe of the mine would be grouted with coal combustion products (CCP's). The agreement provided that CCP's from Allegheny Energy (fly ash) and Anker (hauler of fluidized bed combustion [FBC] ash from the Morgantown Energy Associates [MEA] power station) would be evaluated for use in the grout mixes.

Although the Omega Mine was a post-1977 deep mine bond forfeiture site, the WVDEP Abandoned Mine Lands (AML) Section took over administration of the design and construction contract. This was due to the AML Section's expertise with subsurface grouting projects (having completed 100 plus subsurface grouting projects to prevent mine subsidence), and the state's disproportionate financial interest in the project. GAI Consultants, Inc. (GAI) was selected as the project designer.

The objectives of the project as identified in discussions between the project sponsors and GAI were to develop a suitable mix of coal combustion products available from two of the project sponsors, Allegheny Energy and Anker, to reduce AMD and prevent subsidence by filling the North Lobe of the Omega Mine. This represents a potential beneficial use of the CCP's which would reduce the amount of material going to disposal sites. The general requirements for such a grouting material are that it possess (a) sufficient fluidity to ensure optimum mine room penetration, (b) the ability to provide physical support to abate surface ground subsidence, and (c) in situ characteristics which reduce drainage (AMD) issuing from the abandoned mine workings.

The Omega Mine Injection Program had four goals:

- To fill the voids in the North Lobe of the Omega Mine to reduce contact of water and air with acid forming material with a secondary requirement that the grout have some alkaline leaching potential to help treat AMD.

- A grout mix which would set to sufficient strength to prevent mine subsidence.

- Use of a mixture of fly ash and FBC materials to demonstrate the synergistic attributes of the combined materials.

- A grout mix that would flow without separation and develop reasonable strength and dimensional stability.

## Grout Mix Design

An extensive laboratory testing program was conducted to evaluate both fly ash and FBC ash or mixtures of the two for injection into an abandoned deep mine to reduce AMD.

The test program indicated that a blend of the two candidate materials provided an acceptable grout mix. The FBC ash had the potential to provide strength to the grout while the fly ash enhanced the fluidity of the grout. The addition of two percent cement provided dimensional stability to the hardened grout.

It was recommended that a grout blend of 49 percent FBC ash and 49 percent fly ash plus two percent cement with enough water added to produce a grout having a flow value of 60 seconds, be used. This mix demonstrated the synergistic attributes of combined fly ash and FBC ash, would flow long distances without separation and develop reasonable strength and dimensional stability. These characteristics provided confidence that the grout would fill 10.4 ha (26 acres) of the mine and encapsulate acid forming materials.

## Mine Grouting

The injection program at the Omega Mine is by far the largest project to date undertaken to reduce acid mine drainage. Almost 61,000cm (80,000 cy) of CCP grout were injected into 10.4 ha (26 acres) of the mine. A grout barrier was formed at the south side of the North Lobe to prevent mine water from flowing down the relatively steep dip (9 percent) into the North Lobe which is the lowest portion of the mine. The mine plan was used to select optimum locations for the injection holes. There was no set hole spacing. Due to the openness of the mine in first mined areas and the steep dip of the mined seam, injection hole spacing in these areas was generally in excess of 30m (100 ft). In second mined areas, hole spacing was reduced to as little as 15m (50 ft) since site exploration (both borings and video camera) indicated that fallen roof material could impede movement of the grout. Due to the openness of some portions of the mine, grout moved laterally up to 457m (1,500 ft) primarily because of the relatively steep dip of the mine. In previous work, CCP grouts had moved over 30m (100 ft) and the flowing grout had moved slowly, like pancake batter (EPRI, 1996). Borehole video views of the grout at it was placed, showed that flow in the Omega Mine was fast and turbulent. The maximum grout take in one hole of 7,812cm (10,218 cy) is an indication of the ability of the grout to flow. The total quantity of the FBC-fly ash mix injected into 156 primary grout holes was 47,185 cm (61,716 cy), an average of 302cm (395.6 cy) per hole.

The injected quantity of 60,500cm (79,130 cy) including drilling resulted in a total cost of $1,946,592 or $24.60 per cy.

## Effectiveness of Filling

The volume of grout injected into the North Lobe, approximately 61,000cm (80,000 cy) is equivalent to 100 percent of the void volume in the first mined area and 75 percent of the mined void volume in the second mined area, where fallen roof material partially filled the mine void and would prevent grout from filling all voids.

Pressure grouting conducted in October 1998, near the end of the injection program, indicated many portions of the mine had been previously filled by grout. Of the 46 holes that were pressure grouted, 18 had takes of only 0.7-1.5cm (1 or 2 cy). The total volume injected under pressure was 995cm (1301.8 cy) an average take of 21.6cm (28.3 cy) per hole. In contrast the average take of the FBC-fly ash grout in 156 primary injection holes was 302cm (395.6 cy) per hole.

Nine exploratory holes were drilled in the North Lobe late in the injection program using a rotary air rig. Eight encountered grout with no loss of air. This indicates the mine was filled at these locations. The other exploration hole encountered a 0.3m (1 ft) void and lost air, but air flow returned while drilling below the void. Subsequent pressure grouting of this hole resulted in a take of 92.5 cm (121 cy).

Four core borings drilled approximately one year after grouting showed good roof contact by the CCP grout. Video camera observations in these holes confirmed good roof contact and showed good distribution of the injected grout. No voids were found.

## AMD Abatement

Reduction in acid mine drainage (AMD) usually infers some sort of water treatment to change the chemistry of the mine water and/or its discharge. Work conducted in recent years in AMD reduction from mines and spoil piles has indicated that the best way to reduce AMD is to prevent the reaction from occurring rather than trying to treat the results.

Filling of the mine voids reduces or eliminates air and/or oxygenated water from reaching the AMD-forming materials thus preventing the generation of AMD. Using a material which will set and develop some strength further limits AMD production by decreasing the permeability of the hardened grout thus limiting air and water movement. Complete filling of the mine workings would theoretically result in limiting AMD formation by creating a barrier between the acid forming materials and air or water.

Testing of four grout core samples recovered from the mine, a minimum of 9-10 months after injection, showed a permeability range of $6.2x10^{-7}$ to $8.9x10^{-8}$ cm/sec. Thus the hardened grout is relatively impervious and with its dimensional stability should encapsulate acid forming materials and greatly reduce future formation of AMD. This does not prevent water and air from moving through fractured rock, containing acid forming minerals, above and below the grouted mine.

The greatly reduced flows from the North Lobe following the grouting in 1998 could not be directly attributed to the grouting due to a drought in 1999. However, precipitation in 2000 was slightly above normal and the flows at the Marshall House and Seeps DEF were consistently lower than pre-grouting flows. Average flows in January-October 2000 at the Marshall House and Seeps DEF were 23.8 liters per minute (6.3 gpm) and 46.5 liters per minute (12.3 gpm), respectively. In the pre-injection period 1993-1998 the average flows were 99 liters per minut (26.3 gpm) and 69 liters per minute (18.3 gpm), respectively.

Anticipated, flow from the mines central pool has increased slightly. In January-October 2000 the average flow was 49.5 liters per minute (13.1 gpm) compared to 48.4 liters per minute (12.8 gpm) in the pre-injection period of 1993-1998.

The quality of water flowing from the North Lobe has not changed significantly. With injection of almost 61,000 cm (80,000 cy) of grout, mostly highly alkaline, the lack of buffering of the AMD is disappointing. Apparently the ground water flows around the filled mine or encapsulated acid forming materials without being greatly affected by the chemistry of the hardened grout.

Two other projects where CCP's were injected into mines producing AMD to improve water quality have not greatly changed the chemistry of the mine effluent.

The reactivation of AMD seepage into the Cobun Creek watershed in March, 1999 appears related to the mine filling on the basis of timing and water quality. This AMD is similar to that flowing from the North Lobe. However, the combined seepage flow is generally low and does not add greatly to total acid load. The source of the poor quality water from the Cobun Creek seeps and the North Lobe is uncertain. A possible source is precipitation moving downward to thin pyritic seams or zones in the mine roof where the water becomes acidic before reaching the grouted mine. However, the monitoring wells installed above the mine in 1999 generally indicate good water quality within a few feet of the mine roof.

In spite of essentially no change in mine water quality, the daily acid load from the Marshall House discharge has been reduced by 75 percent because of the greatly reduced flows. Taking into account the increased flow and unchanged quality of the flow from the central pool the reduction in the daily acid load from the three largest sources of AMD, from the Omega Mine (Marshall House, Seep DEF and Central Pool) is 58 percent.

## Subsidence Control

Injection of CCP grout with the procedures utilized resulted in virtually complete filling of the mine voids is likely to provide excellent subsidence control. The fact that the hardened grout was found to have significant unconfined compressive strength coupled with a low permeability indicates it will provide good roof support while resisting passage of water thus limiting the potential for chemical degradation.

Although no grout strength criteria were specified (since the design priority was flowability to fill the mine), the 28-day strength of cubes formed during the injection program varied from 9,350-22,000 kPa (1356-3192 psi). Seven cubes having an age of 496 to 652 days yielded strengths varying from 1,738-7,847 kPa (252 to 1138 psi) with an average strength of 4,744 kPa (688 psi). Six unconfined compressive strength tests were conducted on core samples recovered of 9-10 months after injection. The test results vary from 6,543 to 13,907 kPa (949 to 2017 psi) with an average for the six tests of 9,653 kPa (1400 psi).

The strength, ability to fill the mine and dimensional stability exhibited by the FBC-fly ash grout indicate it is an excellent material for subsidence control.

## Summary

The injection program met its goal of reducing the contact of water and air with acid forming materials. The secondary goal of buffering the AMD by alkaline leaching did not occur. However, the great amount of acid forming materials that are now encapsulated will not contribute to future AMD. It is anticipated that the materials now contributing to AMD formation will eventually be exhausted and water quality will improve as has been observed in the study of mine discharges from abandoned mines.

# 1

# INTRODUCTION

This report presents the findings of a project in which coal combustion products (CCP's) were injected into the Omega Mine to reduce acid mine drainage and prevent mine subsidence. The Electric Power Research Institute (EPRI) contributed funds to the project as part of its scientific and engineering investigations of the properties and suitability of coal combustion products (CCP's) in coal mine remediation.

## Project Background

The Omega Mine is located along U.S. Route 119, 9.6km (6 miles) south of the City of Morgantown, Monongalia County, West Virginia. The mine is in the Upper Freeport Coal, which is approximately 1.4m (4.5 ft) thick, at depths varying from approximately 21 to 60m (70 to 190 ft). The dip of the seam is to the northwest at approximately 9 percent. The seam was mined by Omega Mining Company, Inc. in the 1980's, in an operation called Omega Mine No. 100 (Omega Mine). Figure 1-1 is a plan of the Omega Mine.

Acid mine drainage (AMD) was noted to be discharging from the Omega Mine site and impacting Owl Creek and other watersheds located geologically downdip of the mine during operation of the mine. (The Owl Creek watershed and these other watersheds lie generally to the west of U.S. Route 119 [See Figure 1-1]). These watersheds were already impacted by AMD from older mine operations in the area. In July, 1989 accumulation of water in the mine after closure resulted in additional AMD discharges into the Cobun Creek watershed north of the mine. The Cobun Creek watershed lies generally east of U.S. Route 119. Cobun Creek flows into a reservoir which provides water to the city of Morgantown. After discovering the AMD discharges north of the mine, horizontal relief drains were installed to lower the water level in the mine in order to reduce the discharges into the Cobun Creek watershed. Collection points were then established to direct the AMD discharge to a central treatment facility. Following bond forfeiture when Omega Mining Company declared bankruptcy, a local citizens group operated the treatment facility. When bond forfeiture funds were exhausted, in early 1995, the West Virginia Division of Environmental Protection (WVDEP) took over treating the AMD from the site. Treatment costs (labor and materials) are approximately $300,000 per year.

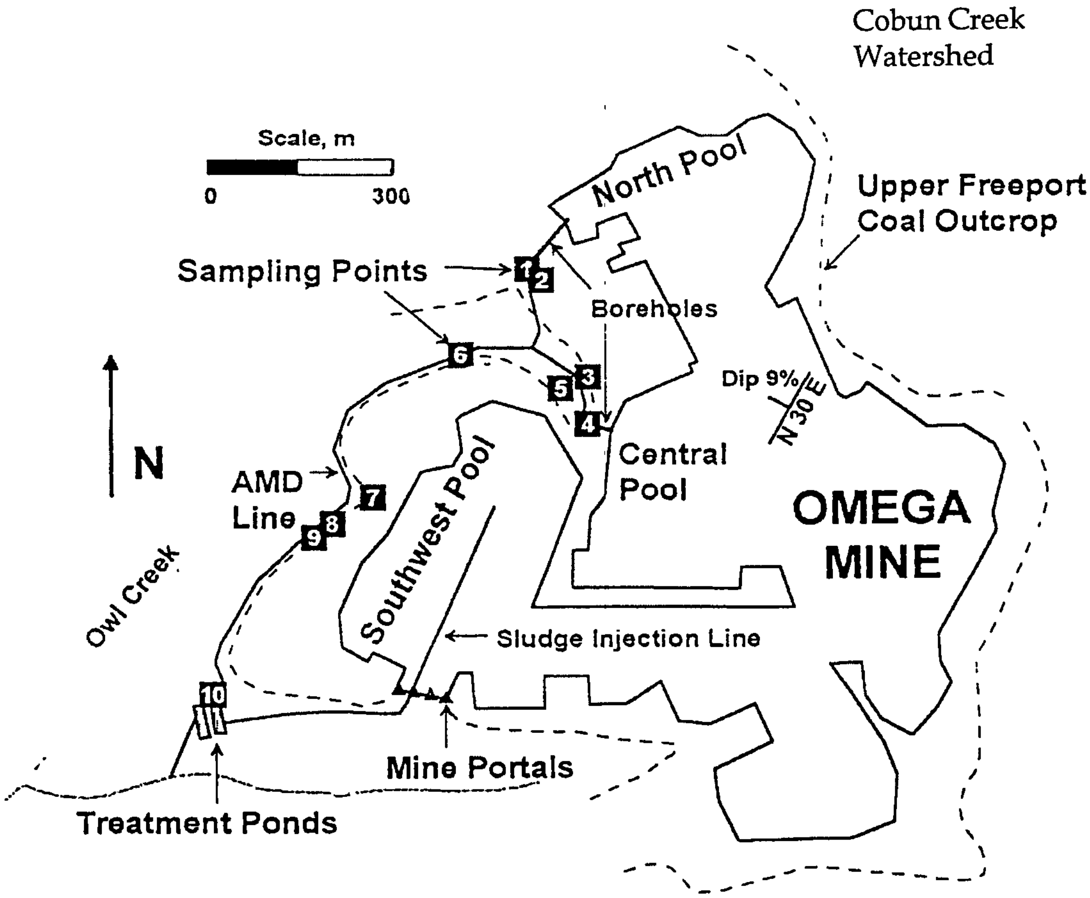



**Figure 1-1**
**Plan of Omega Mine (Aljoe, 1996)[2]**

Concurrent with other activity at the site in the early 1990's, public and private entities completed feasibility investigations for utilizing injection of CCP's to reduce AMD. An agreement was reached in March 1996 between WVDEP; Monongahela Power Company, a subsidiary of Allegheny Energy Supply (Allegheny Energy); Anker Energy Corporation (Anker); Consol, Inc. (Consol); United States Office of Surface Mining Reclamation and Enforcement (OSMRE); and the Electric Power Research Institute (EPRI) to contribute funds for the project. Existing water quality data indicated that the majority of the acidity in the AMD at the site came from the North Lobe of the mine. The North Lobe encompasses approximately 10.4 ha (26 acres) of the 68 ha (170 acres) Omega Mine. The agreement dictated that only the North Lobe of the mine would be grouted with CCP's. The agreement provided that CCP's from Allegheny Energy (fly ash) and Anker (hauler of fluidized bed combustion [FBC] ash from the Morgantown Energy Associates [MEA] power station) would be evaluated for use in the grout mixes. The U. S. Department of Energy (USDOE) joined as a project sponsor in the spring of 1998.

Although the Omega Mine was a post-1977 deep mine bond forfeiture site, the WVDEP Abandoned Mine Lands (AML) Section took over administration of the design and construction contract. This was due to the AML Section's expertise with subsurface grouting projects (having completed 100 plus subsurface grouting projects to prevent mine subsidence), and the state's disproportionate financial interest in the project. GAI Consultants, Inc. (GAI) was selected as the project designer.

## Objectives

The objectives of the project as identified in discussions between WVDEP and GAI were to develop a suitable mix of coal combustion products available from two of the project sponsors, Allegheny Energy and Anker, to reduce AMD and prevent subsidence by filling the North Lobe of the Omega Mine. This represents a potential beneficial use of the coal combustion products which would reduce the amount of material going to disposal sites. The general requirements for such a grouting material are that it possess (a) sufficient fluidity to ensure optimum mine room penetration, (b) the ability to provide physical support to abate surface ground subsidence, and (c) in situ characteristics which reduce drainage (AMD) issuing from the abandoned mine workings.

An additional consideration was that the injected materials should be compatible with the water present in the underground mine workings. This is particularly so with respect to its ability to retain strength for subsidence control, while at the same time retaining chemical integrity against the release of toxic agents (e.g. metals) into the ground water system.

The goals of the mine injection program were as follows:

- To fill the mine voids of the North Lobe of the Omega Mine to reduce contact of water and air with acidic material, with a secondary requirement that the grout have some alkaline leaching potential to help treat AMD.

- A grout mix which when set would have sufficient strength to prevent mine subsidence.

- Use of a mixture of fly ash and FBC material to demonstrate the synergistic attributes of the combined materials.

- The selected mix was to be fluid during injection and develop reasonable strength and dimensional stability.

## Project Tasks

Included in this report are the engineering and testing work conducted by GAI Consultants, Inc. (GAI); water sampling and testing by the USDOE and WVDEP; an initial benthic survey by WVDEP; injection work by the Howard Concrete Pumping Company (Howard); injection monitoring by the WVDEP; and borehole video camera inspection of the exploratory borings by the OSMRE.

GAI's work consisted of:

- Site reconnaissance
- subsurface investigation
- report on pre-injection water quality
- laboratory testing program to select suitable injection mix
- mine injection plan
- technical specifications for the injection program
- periodic monitoring of the injection work
- analysis of pre and post injection water quality
- sampling the injected material and testing for unconfined compressive strength and permeability
- post injection benthic survey
- report on the project including evaluation of the results.

The WVDEP's work consisted of:

- water sampling and testing
- a pre-injection benthic survey
- overall management of the work
- monitoring of the injection work

The USDOE's work consisted of:

- water sampling and testing
- geophysical research

The OSMRE's work consisted of:

- borehole video camera inspection of the mine – pre- and post-injection

## Technology Transfer

As part of EPRI's technology transfer process preliminary or partial accounts of some aspects of this project have been presented at various conferences. A list of these presentations follows:

## Omega Mine Project Papers

"Injection of Alkaline Ashes into Underground coal Mines for Acid Mine Drainage Abatement" William W. Aljoe, 1996, Thirteenth Annual International Pittsburgh Coal Conference Proceedings, The University of Pittsburgh, School of Engineering, Center for Energy Research, September 3-7, 1996.

"Injection of Coal Combustion By-Products into the Omega Mine for the Reduction of Acid Mine Drainage" Terry Moran and Tom Gray, 1996 Annual Joint Fall Meeting of the Central Appalachian Section of SME and the West Virginia Coal Mining Institute in White Sulfur Springs, West Virginia, October 19, 1996.

"Plan for Injection of Coal Combustion By-Products into the Omega Mine for the Reduction of Acid Mine Drainage" Tom Gray, Terry Moran, David Broschart and Gregory Smith, 1997 Annual Meeting of the American Society for Surface Mining and Reclamation (ASSMR) May 10-16, 1997 in Austin, Texas.

"Use of Coal Combustion By-Products to Reduce Acid Mine Drainage at the Omega Mine" Tom Gray, Terry Moran, David Broschart and Gregory Smith, 19th Annual National Abandoned Mine Lands Conference at Canaan Valley, West Virginia, August 18-19, 1997.

"Plan For Injection of Coal Combustion By-Products into the Omega Mine for the Reduction of Acid Mine Drainage" Tom Gray, Terry Moran, David Broschart and Gregory Smith, International Ash Utilization Symposium, Lexington, Kentucky, October 20-22, 1997.

"Plan For Injection of Coal Combustion By-Products into the Omega Mine for the Reduction of Acid Mine Drainage" Terry Moran, Tom Gray, Gregg Smith, and Dave Broschart, West Virginia Surface Mine Drainage Task Force. Morgantown, West Virginia, April 7-8, 1998.

"Injection of Coal Combustion By-Products into the Omega Mine for the Reduction of Acid Mine Drainage" Thomas A. Gray, Terry Moran, D. Broschart and G. Smith, 1998 Annual Meeting of the American Society for Surface Mining and Reclamation (ASSMR) St. Louis, Missouri, May 16-21, 1998.

"Injection of Alkaline Ashes into the Omega Mine for Acid Mine Drainage Abatement," William W. Aljoe, American Coal Ash Association Educational Program for Managers of Coal Combustion Products (CCP's), National Research Center for Coal, Energy, West Virginia University, Morgantown, WV, June 8-12, 1998.

"Plan for Injection of Coal Combustion By-Products into The Omega Mine for the Reduction of Acid Mine Drainage", Thomas A. Gray, Terry Moran, David W. Broschart, and Gregory A. Smith. Fifteenth Annual Meeting of the International Pittsburgh Coal Conference, Pittsburgh, Pennsylvania, September 14-18, 1998.

"The Omega Project" Thomas A. Gray, 26th West Virginia Mining Association Symposium, Charleston, W. Va., January 13-15, 1999.

# 2

# RELATED WORK

This section presents a partial summary of literature on acid mine drainage and fluidized bed combustion (FBC) ash, and the properties and uses of FBC ash related to injection into mines for the reduction of acid mine drainage.

In Appalachia over 16,000 km (10,000 miles) of streams are polluted by coal mining operations (Hill, 1968).[34] Acid mine drainage (AMD) results from the oxidation of pyrite ($FeS_2$). It was estimated that 35% of the acid pollution came from abandoned shaft and drift mines, 25% from abandoned surface mines, and 40% from active mines. Acidic mine drainages characteristically have a low pH, net acidity, high iron, high sulfates and significant amounts of aluminum, manganese, calcium and magnesium. Historically, the major control measures for deep mine discharges are diversions to prevent water from entering the mine, flooding of the mine to prevent oxidation of pyritic material and sealing to prevent air from entering the mine and oxidizing the pyrite (Hill, 1968).[35]

In 1974 Lovell and Gunnett estimated that 17 GL (14.5 billion gallons) of AMD were being produced daily in Pennsylvania.[41]

In 1995 EPA determined streams with fisheries impacted by Acid Mine Drainage in Maryland, Ohio, Pennsylvania, Virginia and West Virginia (Table 2-1). Pennsylvania had the most miles of stream impacted with West Virginia in second place (Faulkner and Skousen, 1998).[28]

**Table 2-1**
**Streams With Fisheries Impacted By Acid Mine Drainage**

| | Stream Lengths Impacted | | |
|---|---|---|---|
| State | No Fish | Some Fish | Total |
| OH | 258 mi<br>415km | 349 mi<br>562km | 607 mi<br>977km |
| PA | 1714 mi<br>2759km | 1525 mi<br>2454km | 3239 mi<br>5212km |
| WV | 488 mi<br>785km | 612 mi<br>985km | 1100 mi<br>1770km |
| VA | 17 mi<br>27km | 0 | 17 mi<br>27km |

| | Stream Lengths Impacted | | |
|---|---|---|---|
| State | No Fish | Some Fish | Total |
| MD | 42 mi<br>68km | 110 mi<br>177km | 152 mi<br>245km |
| Total | 2519mi<br>4053km | 2596 mi<br>4177km | 5115 mi<br>8230km |

Faulkner and Skousen, 1998; EPA, 1995[28,25]

In West Virginia 486 streams were affected by acid mine drainage.

Two counties with large underground mine complexes (Grant and Monongalia) comprised over one-third of the total flow of acid mine drainage in West Virginia. The annual cost of treating mine drainage in West Virginia is estimated to be $60 million. Since 1985, four bond forfeiture sites (including OMEGA) cost over $6.9 million in chemical treatment of acid mine drainage (Faulkner and Skousen, 1998).[28]

Although many coal combustion products have desirable properties for use in manufacturing products and the construction industry only 30 percent of the material produced is utilized (American Coal Ash Association, 2000)[4]. Disposal of these materials requires large areas and continues to increase in cost. Thus disposal or utilization in underground mines has become desirable. Almost ninety percent of the coal mined in the United States is used to generate electricity (Stewart, 1998).[64] In 1997, over 91 million metric tons (100 million short tons) of coal combustion products were generated in the United States (Stewart, 1998).[64] These coal combustion products could be hauled back to the mine supplying the coal for disposal or used to fill nearby abandoned mines. Coal combustion products that develop strength with time or that are alkaline offer advantages in filling mines. They can prevent subsidence, and reduce or prevent acid mine drainage. Thus mine filling can benefit both the utility with a reduction in disposal costs and the coal supplier (Sevim & Unal 1998).[59] Coal combustion products are sometimes mixed to improve the properties of a single product. Often small percentages of cement or lime are added to improve the properties of the coal combustion products (CCP's).

An early evaluation of the feasibility of using flue gas desulfurization (FGD) by-products to backfill abandoned deep coal mines to abate AMD and control subsidence was conducted by Duvel, et al (1979).[21] Some conclusions to the concept feasibility were:

- FGD sludges and fly ash are capable of abating AMD
- AMD abatement will be better assured with complete mine backfilling, but abatement can result with well planned schemes of partial mine backfilling; and
- The leachate will be of only minor significance because of the small quantity that will be produced due to the low permeability of the sludge and the dilution that will occur.

Flue gas desulfurization results in large amounts of solid waste by-products. The large quantity of sulfates and sulfites in these materials react with the calcium hydroxide ($Ca(OH)_2$) present to form calcium sulfo-aluminates, the most important mineral of this group being ettringite. The nature of the mineralogical reactions has a large bearing on the utilization potential of the materials. Reactions controlling the formation, composition and disintegration of ettringite are critical in determining the overall stability and strength of cements and concrete derived by dry FGD materials (Graham and Robl, 1991).[30]

High tensile strength can be obtained from FGD by-products derived from fluidized bed combustion, and their strength can be improved by controlled additions of pulverized coal fly ash and lime. The role of the fly ash is to both contribute to the formation of hydrated phases and to act as a filler, while lime helps to buffer the solution pH. Early ettringite formation leads to good initial strength. This early strength development is further enhanced by dissolution reprecipitation reactions, forming new hydrated phases which preferentially nucleate on fly ash surfaces and continue to grow into available pore space. Non-crystalline precipitates, rich in Al and Si, also contribute to the strength gain. However, the large amounts of anhydrite in the cured materials will cause long-term stability problems due to secondary swelling caused by the delayed hydration of anhydrite to gypsum. The formation of gypsum, which results in a volume increase, causes fracturing and disintegration of the materials upon long-term exposure to moisture (Graham and Robl, 1991).[30]

In an evaluation of the potential of utilizing Atmospheric Fluidized Bed Combustion (AFBC) wastes to produce a low strength concrete for mining related applications, Rose et al., (1985) found the use of AFBC baghouse/cyclone particulate as a supplement to cement resulted in long term strength gains, but not equivalent to the strengths obtained from cement alone.[57] The substitution of AFBC baghouse/cyclone particulate showed moderate strength reduction to levels of 20-40 percent. A 40 percent substitution resulted in drastic strength losses. Expansion characteristics of up to 10-15 percent were noted when AFBC waste was utilized in the mixes.

In a study on using AFBC ash to produce no-cement concrete Bland, et al., (1987) concluded non-prehydrated mixes were applicable for backfilling coal mines where low to moderate compressive strengths and high expansive properties are desired.[10] Expansion can be controlled by the amount of prehydration at the expense of some strength. The material exhibited a resistance to sulfate acid attacks.

Berry et al., (1991) in a study of the cementing action of fluidized bed combustion (FBC) products found that the early cementing action, and approximately 90% of the final strength, is contributed by the formation of calcium suphoaluminate, or ettringite, formed by "sulpho-pozzolanic" reaction of active aluminum in the FBC and coal ash materials in the high pH, high sulfate environment.[8] There is also evidence for a longer-term, though small, contribution from a lime-silica ("silico-pozzolanic") process.

Berry et al., (1991) indicate it will generally be necessary to investigate each FBC/coal ash combination individually.[8] In this respect, no-cement binders differ from Portland cement concrete systems, where generalized proportioning rules can be formulated, based on the assumption that most Portland cements are of similar nature and reactivity. They also caution on the need to consider potential lack of long term stability and durability.

Hassett et al., (1991) indicate ettringite forms and remains fully stable only under highly basic conditions (pH above about 12).[32] When equilibrated at intermediate pH values, sulfate, selenate, and borate ettringite dissolve incongruently into aluminum rich solids and dissolved components. These ettringites dissolve completely between pH values of 1 and 2.

Bland (1994) indicates test cells of FBC material in Canada initially developed strength in excess of 6900 kPa (1000 psi) and the material looked quite stable.[11] Unfortunately, the cells deteriorated with time, losing strength to less than 345kPa (50 psi) and showed visible disruption and deterioration. Coefficients of hydraulic conductivity increased by 2 to 3 orders of magnitude as a result of the deterioration of the fill.

McCarthy et al (1995) report on similar mineralogical transformation in a study of compacted FBC materials.[44]

Hao et al., (1997) in a laboratory study of coal refuse amended with FGD material observed that $Ca\ SO_3$ based FGD strongly inhibited acid production.[33]

Jackson, et al (1993) in a laboratory study of acid mine drainage mitigation using fly ash found fly ash to be effective if adequate alkalinity was present.[37] They cautioned that if fly ash alkalinity is inadequate to balance potential acidity, accelerated leaching of ash bound metals might occur.

In discussing the use of FBC ash to control acid mine drainage on a project in West Virginia, Ziemkiewicz and Head (1993) indicate they expect the FBC ash will fill an acid mine pool with alkaline solid so that ground water will tend to flow around rather than through the mined areas.[67]

Schueck, et al (1993) injected buried piles of tipple refuse with Fluidized Bed Fly Ash grout.[58] They indicate this appeared to be a viable technique to reduce AMD production from small buried pods of pyritic material.

In a greenhouse study of Dry FGD By-Products as amendments for acid mine spoil, Stehouwer et al (1993) found the FGD By-Products were effective in ameliorating acid spoils and had a low potential for creating adverse environmental impacts.[63]

There are four possible mechanisms (two chemical and two physical) by which the injection of coal combustion residues into abandoned or reclaimed surface mine areas may improve the water quality. The chemical mechanisms are related to the addition of alkaline materials, such as coal combustion residues. First, they can neutralize acidic groundwater in the spoil. Second, when the pH of water in the spoil increases, bacterial activity is inhibited and the rate of acid production should decrease. Physical mechanisms prevent the formation of acid by removing one of the essential reactants. Injected coal combustion residues grout may encapsulate pyrite, isolating it from air and water, thus preventing its oxidation. Also, the deposition of the coal combustion residues within coal spoil may reduce its permeability and divert water away from zones of acid production (Kim & Ackman, 1995).[38]

Brant & Ziemkiewicz (1998) studied the effect of various alkaline amendments to control AMD from coal refuse piles.[13] Eleven alkaline materials including 4.19% FBC ash and 12.56% FBC ash were evaluated. The study consisted of constructing refuse piles mixed with the alkaline amendments. The discharge from the pile with 4.19% FBC ash was less acidic than the control pile. Although the discharge from the pile containing the 12.56% FBC ash was acidic it exhibited acidity at least an order of magnitude less than the control pile discharge.

Mixtures of mine tailing and cement have been used as a pumpable paste backfill in hard rock mines (Brackebusch, 1994).[12] The paste is a high density mixture of water and fine solid particles. Consistency as measured by the slump cone test (ASTM C-143) varies from slightly greater than zero up to almost 300 mm (12 inches). The paste can be pumped with concrete pumps over 1.0 km (0.6 mile). Small amounts of cement (3-5%) produce stiff backfill having strengths of 1.5 to 3.5 Mpa (200-500 psi). The lack of particle segregation or size classification of paste is an important difference between conventional dilute tailings slurry deposition. These properties result in a homogenous tailings deposit that solidifies in a reasonable length of time to support vehicle traffic for reclamation purposes. Low permeability of consolidated paste inhibits acid generating potential.

Arioglu et al (1986) report on the strength of lime stabilized mine tailings used as mine backfill.[5]

Aljoe and Hawkins (1993) in reporting on an attempt to neutralize acidic discharge from abandoned underground coal mines by alkaline injection indicate that flooded abandoned mines require substantial exploration to plan an adequate remediation project.[3] In this project chemical neutralization of acidic mine water was the primary objective, no attempt was made to inhibit pyrite oxidation via physical means.

Phelps et al (1985) investigated the burial of toxic mine waste within low permeability packages below the water table.[50] AMD formation can be reduced by limiting the contact of mine waste with oxygen and water. They found encapsulating the mine spoil in clay significantly reduced AMD. Grouting a deep mine with low permeability alkaline material has a similar effect.

The Fossil Energy Coal Combustion Product Utilization Program of the U.S. Department of Energy (DOE) has four short-term goals (Renninger, 1998): [55]

- To characterize waste from coal powered production technologies and ensure safe disposal and utilization practices.
- To develop technologies for solid waste minimization, waste disposal, and underground mine reclamation.
- To perform institutional analysis that provides the framework for regulatory, economic, marketing, and other considerations which influence waste management decisions.

- To coordinate/facilitate information transfer from DOE sponsored or co-funded projects to the private sector.

As these short term goals are achieved, they contribute to the program's long-term objectives of:

- By the year 2010, demonstrate the acceptability of large volume uses of CCP's in such things as surface and underground mine reclamation such that utilization rates for CCP's approach 50%.

- By 2010, complete sampling and characterization of clean coal technology combustion products such that the public and regulatory agencies accept disposal or utilization of CCP's as routine business practices. (Renninger, 1998).[55]

Chugh, et al (1998) summarized some studies related to filling underground voids (see Table 2-2).[19]

**Table 2-2**

**Summary of Studies of Mine Filling**

| Authors and Sources | Backfilling Reason | Comments |
|---|---|---|
| Carlson (1975)[15] | Subsidence Control | Model studies |
| Maser et al (1975)[43] | Subsidence Control | Fly ash-cement mine sealant |
| Whaite and Allen (1975)[65] | Subsidence Control | Slurry backfill |
| Galvin and Wagner (1982)[29] | Enhance extraction | South African coal mines to increase extraction by 8%-12%. Fly ash only |
| Patelunas (1988)[49] | Subsidence Control | High volume use of fly ash in underground void filling. Low solid density mixes. |
| Palarski (1993)[48] | Enhance extraction | In Polish coal mines. Fly ash, tailings, rocks |
| Hollinderbaumer and Kramer (1994)[36] | Subsidence and ground control | Integrated approach in German longwall mines to dispose of incinerator ash |
| Meiers, et al. (1995)[45] | Subsidence Control | Fly ash-scrubber sludge mix |
| Gray et al. (1995)[31] | AMD Control | Disposal of FBC |
| Chugh et al. (1996a)[17] | AMD Control | Fly Ash, Scrubber sludge-based pastes pumped into an abandoned mine in Maryland. |

Chugh and his colleagues observed underground backfilling was done mostly using low solids content slurry and indicated that paste backfills offered economic and environmental advantages (Brackebusch, 1994).[12]

An early demonstration of the ability to inject coal combustion products into mines was conducted by Indianapolis Power and Light (IPC) and EPRI. Seven thousand three hundred metric tons (8,000 tons) of FGD material were injected into a mine at Petersburg, Indiana. The material moved 61m (200 ft) from the injection holes, filled the mine to its roof, developed adequate strengths and created no environmental problems (Meiers, et al, 1995; EPRI, 1996).[45,26]

In a study of pneumatic and hydraulic placement systems into a mine in Illinois an optimum mix for pneumatic placement consisted of 80 percent FBC fly ash and 20 percent spent-bed ash by weight. The optimum mix for hydraulic placement was 55 percent scrubber sludge, 40 percent fly ash and 5 percent lime waste (Chugh, et al, 1996b).[18] It was found that a slump of 203-228mm (8 to 9 inches) could be obtained from grouts with 74%-77% solids. Leaching characteristics of the mixes showed that they were environmentally acceptable. (Chugh, et. al 1998).[19] Grout moved at least 61m (200 ft) from the injection point. Average solids content of grouts pumped was higher than 70% and a pumping rate of 109 metric tons (120 tons) per day was maintained for several days.

In Illinois subsidence is generally caused by weakening of the mine floor. Thus excess water is not desirable and paste like backfills are needed. Chugh, et al., (1998) indicate the individual components of a CCP's-based paste are F-type pulverized coal combustion (PCC) fly ash, bottom ash, sulfate-rich scrubber sludge, fluidized bed combustion fly ash and spent bed ash.[19] The availability of these materials and their production ratios play an important role in designing an appropriate paste. A power plant utilizing scrubber sludge technology will produce fly ash, bottom ash and scrubber sludge. If it is possible to develop a paste using the same proportions of individual components as their production ratios (or ratios of products available after sale of fly ash, bottom ash, and scrubber sludge), then effective utilization of these materials is possible by consuming all the materials destined for surface disposal. But the ratio of available individual materials may not produce a stable paste in terms of bleed off and flowability (or the stability of paste). If F-type fly ash and scrubber sludge are selected as mix components, then a lime-based material is required to activate the pozzolanic reactions in the F-type fly ash. This lime based material can be hydrated lime, FBC fly ash, lime waste, or other materials where free lime is available. Also, sufficient quantity of F type fly ash is required for proper mixing with scrubber sludge.

The fresh grout should have required workability properties, namely flowability and bleed off water. Bleed of a fresh grout should be limited to less than 5% (Chugh, et al., 1998).[19] Chugh, et al (2000) report on their work with paste backfills with a selected dry mix of 53% FBC, 33% mine gob and 14% fly ash.[16]. This mix had a strength of approximately 3500kPa (500 psi) and an elastic modulus of 186,200kPa (27,000 psi) in 28 days. Slump ranged from 228-279mm (9 to 11 inches). More than 8200 metric tons (9,000 tons) of material were injected through two boreholes.

On a project in West Virginia, Ziemkiewicz, et al., (1998) used fluidized bed combustion ash to control acid mine drainage and subsidence at an abandoned coal mine.[68] Mixtures of FBC ash and water were unstable. It was found that 5 percent bentonite provided a stable grout. Access to the mine was available so movement of the grout could be observed. The grout moved 152mm (6 inches) per second on the level mine floor. Seven hundred sixty five cm (1000cy) of grout were placed in the mine. The material flowed about 183m (600 ft) from the injection hole. (Black, et al., 1998.)[9]

In Western Maryland, Mettiki Coal Corporation is injecting flue gas desulfurization (FGD) by products into abandoned portions of an active coal mine for acid mine drainage mitigation. The FGD is mixed with mine water forming a slurry with a 15 percent solids content. Approximately 91,000 metric tons (100,000 tons) of FGD material have been injected (Ashby, 1998a, 1998b).[6,7]

In Ohio, an abandoned mine having a discharge with a pH of 2.8 to 3.0 was selected as a demonstration site by American Electric Power and the Ohio Department of Natural Resources (Mafi, et al, 1998).[42] The FGD material consists of 80 percent calcium sulfite and 20 percent calcium sulfate. A mixture of dry fly ash and the FGD varying from 1.25:1 to 1:1 was mixed with 5 percent quicklime. In the downdip section of the mine the injected material had a slump of 102-152mm (4-6 inches) (ASTM C143). In other portions of the mine a slump of 203-254mm (8-10 inches) was used. Injection began in August 1997 and was completed in January 1998. Monitoring to assess the project effectiveness is ongoing (Mafi, et al, 1998).[42] Lamminen, et al. (2000) in reporting on this project indicates immediately following grout injection significant increases in acidity, iron, aluminum, sulfur and calcium were observed in wells located in the mine voids near where grouting was carried out.[39] Following this initial increase in concentrations, levels of most constituents decreased to near pregrouting levels. Dissolution of the FGD grout material may have contributed to increases in calcium and sulfate concentrations in the drainage waters.

In Maryland, an abandoned coal mine was used to investigate AMD abatement (Rafalko, et al., 1996).[54] The FGD-FBC mixes evaluated (Table 2-3), demonstrated a bleed of less than 2%, slumps in the range of 203-216mm (8 to 8.5 inches), and 28-day compressive strengths ranging from about 4140-4826 (600 to 700 psi). Trial mix testing had shown that increasing the water content increased flowability of these mixes, but also caused a high bleed (i.e., above 3%). When the FBC content was reduced to 20% during trial testing, greater bleed and settlement and lower compressive strength were observed. (Rafalko, et al 1996.)[54]

FGD-lime mixes were also investigated. These mixes generally had better flowability, less bleeding, and higher 28-day strength compared to FGD-FBC mixes. Mixes with 10% lime, by dry weight, had much better strength characteristics than those made with only 5% lime. However, using large quantities of lime poses operation difficulties, and a much greater cost. Mixes with 10% lime and 40% water content, and approximately equal parts fly ash and FGD by-product, had a slump in the range of 203-216mm (8 to 8.5 inches). The compressive strength after 28 days was about 7600-8274kPa (1,100 to 1,200 psi). (Refalko, et al 1996.)[54]

It was possible to formulate either type of mix (FGD-FBC or FGD-lime) to safely achieve suitable workability and strength properties for mine injection. The FGD-FBC mix was selected for the demonstration project over the FGD-lime mix largely because of the lower costs of FBC products compared to lime. Use of the FGD-FBC mix also had an environmental benefit in that the injection materials consist of 100% CCP's that otherwise would require disposal, typically by landfilling. The selected grout mix consisted of 38% fly ash, 32% FGD by-product, and 30% FBC material, plus 38% water, as the optimal formulation. (Rafalko et al, 1996.)[54]

**Table 2-3**
**Comparison of Grout Mixes**

| Grout Mix | Slump | Bleed (%) | 28-Day Strength |
|---|---|---|---|
| 30% fly ash, 40% FGD, 30% FBC plus 38% water | 8.25 in<br>210mm | 1.7 | 627 psi<br>4323 kPa |
| 38% fly ash, 32% FGC, 30% FBC plus 38% water | 8.50 in<br>216mm | 1.8 | 711 psi<br>4902 kPa |
| 50% fly ash, 40% FGC, 10% lime plus 40% water | 8.10 in<br>206mm | 1.0 | 1,095 psi<br>7550 kPa |
| 40% fly ash, 55% FGD, 5% lime plus 38% water | 9.40 in<br>239mm | 1.8 | 548 psi<br>3778 kPa |

**(Rafalko, et al, 1996.)[54]**

The preinjection mine discharge had a pH of 3.0 with high sulfate and dissolved solids content. Approximately 4242 cubic meters (5,600 cubic yards) of grout were injected into dry and inundated sections of the mine. (Rafalko & Petzrick, 1998)[53].

Additional testing with FBC material ages ranging from freshly generated to over four weeks old showed a wide variation in strength. As a result of this testing, the final grout mix design was based on the use of fresh FBC. The mix design consisted of 60% fresh (defined as less than 24 hours old) FBC by-product, 20% FGD by-product, and 20% fly ash. The FBC was conditioned at the plant to contain about 15% moisture, which resulted in about 3% to 5% free lime content. This final design mix yielded 203mm (8 inches) of spread using ASTM PS 28-95, and a 28-day unconfined compressive strength of 3585 kPa (520 psi) as determined using ASTM C 39-94.

The final adjustments to the grout mix were performed in the field during injection. Specifically, the moisture content was increased to 57% on a dry weight basis for the design mix ratio of 60/20/20 for FBC, FGD, and fly ash. The need to increase the water content to achieve the desired workability of the grout was due to: 1) the full-size mixing equipment, which required greater water to provide efficient and uniform mixing; and 2) desire to maximize flowability characteristics to facilitate movement of the grout through the mine voids and collapsed material.

In addition to the geotechnical and workability parameters discussed above, the grout was also tested for metals using the Toxicity Characteristic Leaching procedure (TCLP). The TCLP test for metals was performed on a freshly prepared grout sample with a mix ratio of 30/40/30 for

FBC/FGD/fly ash and 38% water. The only two metals that were detected in the leachate were arsenic and barium at levels of 0. 13 and 0.11 mg/L, respectively. The remaining six TCLP metals were not detected above the method detection limit, and none of the results exceeded their respective regulatory limits for characterization as a hazardous waste. The pH of the mine discharge shows a slowly increasing trend. (Rafalko & Petzrick, 1998).[53]

Rafalko and Petzrick (2000) indicate post injection monitoring of mine discharge water showed an increase in AMD related parameters immediately after grouting.[51] About one year after grout injection; iron, aluminum, total acidity, and trace element concentrations and loadings dropped to levels comparable to or below pre-injection conditions. In the same time period, pH has tended subtly upward. Calcium and sulfate levels have been elevated since injection indicating some grout dissolution. They conclude that the grout has entombed pyritic mine debris and covered pyritic surfaces in the mine, which has reduced the volume of pyrite that would have otherwise been available for acid formation.

Aljoe (1999) indicates the mine's discharge pH remained around 3.0 during and after grout injection, while Ca, Na and K concentrations increased by nearly an order of magnitude.[1]

In September 1997, nine holes were drilled to recover grout material. In general, the grout showed little sign of weathering and displayed good mine roof and floor contact. Grout cores exhibited permeabilities between $1.89 \times 10^{-6}$ and $6.02 \times 10^{-8}$ cm/sec. Grout from one core matched the target compressive strength in the 28 day laboratory test. The other grout cores all had approximately twice the strength achieved in the laboratory after 28 days. (Rafalko and Petzrick (1999)).[52]

In laboratory leaching tests of two FBC materials with sulfuric acid mine drainage it was found that some released elements exceeded drinking water standards. Both arsenic and selenium concentrations in leachates from AMD leaching were much lower than those from sulfuric acid leaching due to metals in AMD causing selenium complexation and co-precipitation (Skousen and Bhumbla, 1998).[60]

Canty and Everett (1998) report on an in situ chemical treatment method to remediate AMD at a coal mine in Oklahoma.[14] The mine water had a pH of 4.3. A CCP slurry 380 metric tons (418 tons) was injected into the mine. During the 15 hours of injection in July 1997, the pH rose to 12.2 and alkalinity increased from 0 to 950 ppm as $CaCO_3$. After 150 days the pH of the mine discharge was above 6.5 and the alkalinity was approximately 100 mg/L as $CaCO_3$.

Ziemkiewicz and Skousen (2000), report on the use of coal combustion products for reclamation.[66] They indicate 23 million metric tons (25 million tons) of FGD solids are produced each year with 9 percent of that total being beneficially used in reclamation. Beneficial CCP applications in coal mines include:

- Neutralization or encapsulation of acid-producing materials;
- Barriers to acid mine drainage formation/transport;

- Alkaline amendment to neutralize acid producing rock;
- Subsidence control in underground mines;
- Filling underground mine voids to control acid drainage;
- Pit filling to reach approximate original contour in surface mines; and
- Soil amendment or substitute.

They conclude:

While each setting and CCP form a unique set of circumstances requiring individual analysis and evaluation, several generalizations can be made.

1. As a mine filling material, CCP's can be used to neutralize acid ground water, encapsulate toxic materials, bring the land surface to approximate original contour, prevent subsidence, and control hydraulic pressure buildup in underground coal mines.

2. CCP-filled areas introduce an alkaline component into the mine fill. By neutralizing acid and metal laden water in the backfill or underground mine, CCP's tend to cause metals to precipitate, lowering the concentrations of nearly all metal ions in solution.

3. In already neutral or alkaline ground water environments, CCP's can exacerbate soil salinity problems if chlorides or sodium are a problem.

4. The extent of positive or negative impacts is a function of the ground water flux through the CCP, its chemistry, and the chemistry of the mine groundwater.

5. Water flux is governed by local hydrology and the permeability of the CCP.

6. Some CCP's can be compacted or formulated as grouts so that they are nearly impermeable to water.

Non-fixated FGD materials contain almost no neutralization potential and are presently not very useful in mine land reclamation. Fixated FGD contains excess alkalinity with low permeability. Fixated FGD materials can be useful in acid mine drainage abatement, subsidence control, and used as a barrier material to encaspulate acidic materials or seal pit floors on surface mines.

Demchak, et al (2000) in a study of 15 West Virginia underground mine discharges over 30 years found AMD showed water chemistry improvements at 14 of the 15 sites.[20] Acidity and iron concentrations were less after 30 years at 14 sites, while aluminum concentrations were less at 13 sites. They conclude that if underground mines are left undisturbed for a long period of time water quality will improve.

# 3

# SITE CONDITIONS

## Topography and Geology

The site is located in the Appalachian Plateaus physiographic province. Erosion has dissected the area so that topography is mainly in slope with a little flat land on hilltops and valley floors.

Within the area underlain by the North Lobe of the Omega Mine, the ground surface varies from a high elevation of 533m (1750 ft) to elevation 472m (1550 ft) as shown in Figure 3-1. Most of the area lies above elevation 488m (1600 ft).

The rock strata in the Appalachian Plateaus province generally exhibit dips no greater than 1 vertical to 100 horizontal. However, the project area is on the flank of the Chestnut Ridge Anticline and the rock strata dip to the northwest at approximately 9 percent. Figure 3-2 is a geologic section located a few miles north of the Omega Mine.

The rock strata are part of the Pennsylvanian age Conemaugh and Allegheny Formations which consist of sandstone, claystone, limestone and coal. (Hennen, et al., 1913)[34]

The Omega Mine is located in the Upper Freeport Coal which marks the top of the 76m (250 ft) thick Allegheny Formation. A 1932 publication on coal in Monongalia County indicates a valuable deposit of Upper Freeport Coal between the Chestnut Ridge Anticline and Monongahela River. The Upper Freeport Coal ranged from 1.2 to 2.4m (4 to 8 ft) in thickness at its outcrop along the western

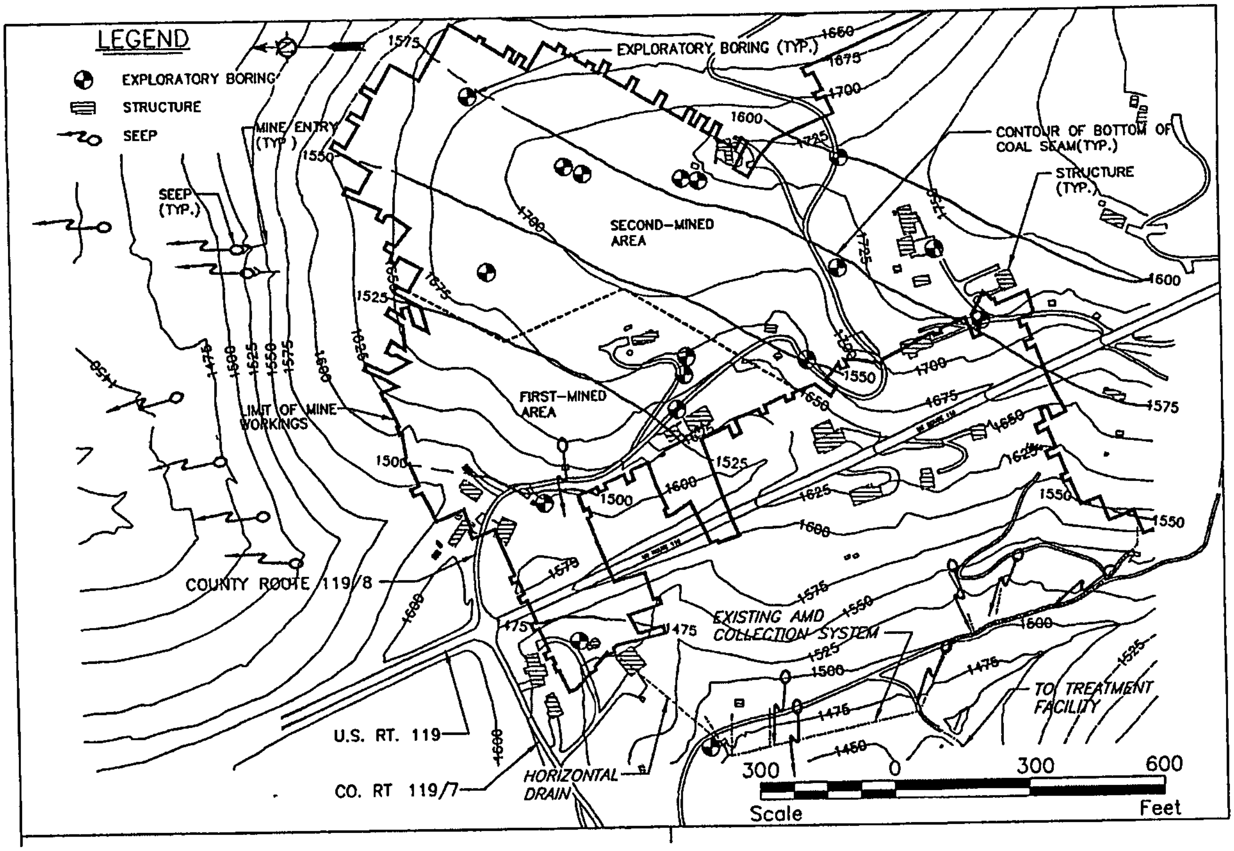



**Figure 3-1**

**North Lobe Omega Mine – Topography and Structural Contours of Base of Upper Freeport Coal**

slope of the Chestnut Ridge Anticline. The coal was reported to thin rapidly westward until 3.2 to 9.6 km (2 to 6 miles) from its outcrop, it was not of commercial thickness and quality (Morris, 1932).[46]

Normally the Upper Freeport Coal consisted of three seams of coal divided by two thin shale partings. The top seam, 0.6 to 0.9m (2-3 ft) in thickness was seldom mined because of its high ash and sulphur content. The middle seam, 0.3 to 0.9m (1-3 ft) thick contained 11 percent ash and 1.5 percent sulphur. The heating value was approximately 13,300 B.T.U. The lower seam rarely exceeded 12 inches in thickness (Morris, 1932).[46]

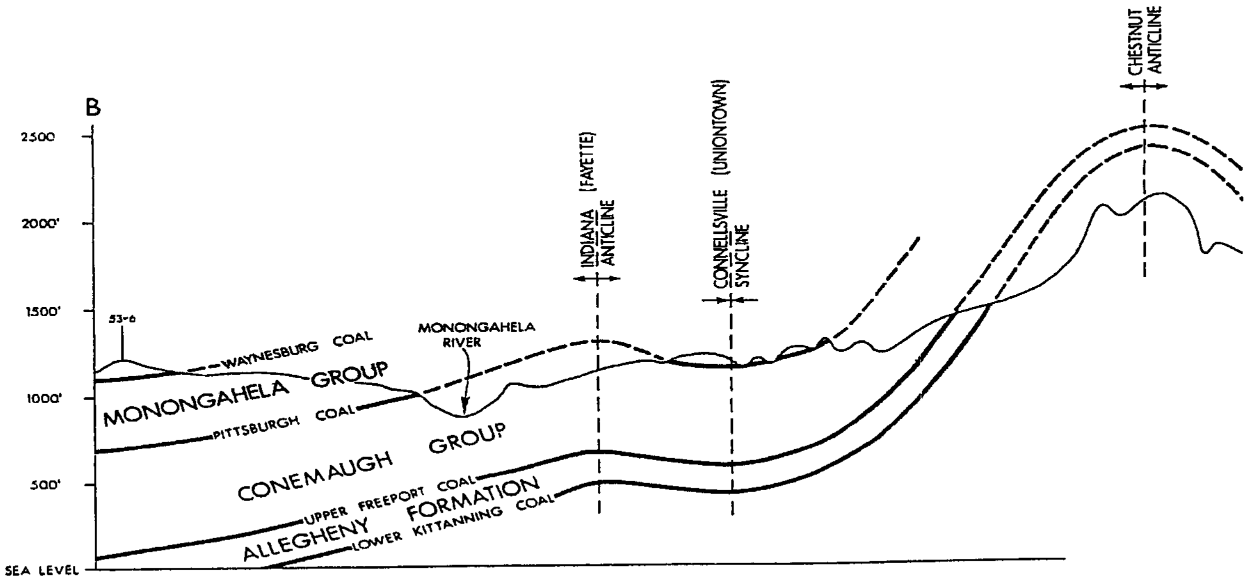



**Figure 3-2**

**Geologic Section (Repine, et al., 1983) [56]**

## Mining

During the 1980's the Omega Mine Company mined 68 ha (170 acres) of the approximately 1.4m (4.5 ft) thick Upper Freeport Coal. Figure 1-1 is a plan of the mine.

Within the mine the Upper Freeport Coal varied in elevation from 533m (1750 ft) at the south end to 451m (1480 ft) at the north end. Maximum depth to the mine was 58m (190 ft). The coal outcropped on the valley walls to the north, east and west. Figure 3-1 shows the structural contours of the Upper Freeport Coal in the North Lobe. Subtracting the elevation of the coal from the corresponding surface contour yields the depth of cover above the mine.

The coal was mined by room and pillar methods. Initial mining extracted approximately 50 percent of the coal and left large pillars of coal in place. Pillar size varied from 5.5x9.5m (18 X 31 ft) to 11.6x17.7 (38 X 58 ft) in plan. In some portions of the mine, second mining occurred in which coal was removed from existing pillars to produce a total extraction of about 60%. In other portions of the mine, the existing pillars were cross cut to produce four small pillars from each large coal pillar, the total extraction in these area was about 72 percent of the coal. Approximately 40 percent of the North Lobe was first mined. Figure 3-3 shows the extraction percentages in the various portions of the mine. Generally 1.2m (4 ft) of coal was mined. In the northern portion of the mine a 0.6 to 0.9m (2 to 3 ft) thick rider seam was immediately above the mined coal. This seam contained pyrite and had a sulfur content of 5 percent. The rider seam was left in place but fell when second mining occurred. Second mining cracked the mine roof, water entered from the mine roof and some subsidence occurred which produced surface cracks up to one foot wide. (J. Laurita, Jr., 1994). [40] This high sulfur rider seam is believed to be the source of the poor quality water flowing from the North Lobe.

Small Upper Freeport Coal mines were apparently operated in the vicinity of the Omega Mine in the 1920's. No maps of these mines are available. Known locations of entries along the coal outcrop near the North Lobe are indicated on Figure 3-1.

## Acid Mine Drainage

When mining ceased in the late 1980's, water began ponding in the Omega Mine. Two mine pools developed, the Central and North Pools. (See Figure 1-1). The mining company left a barrier pillar at least 30 m (100 ft) wide along the down-dip (northwest) edge of the workings to prevent mine drainage. However, this barrier may have been penetrated by several small, unmapped punch mines that allowed water from the Omega Mine to drain through the outcrop.

Also, the mine pool that accumulated behind the barrier in the northern section of the mine began to discharge into the basement of a house on the down-dip (west side) of the workings (Aljoe, 1996).[2]

As water levels increased in the North Pool discharge also occurred from the down-dip (north) side of the mine into Cobun Creek which is utilized for water by the city of Morgantown. An estimated discharge of 57 liters (15 gallons) per minute turned 2.4km (1.5 miles) of previously pristine Cobun Creek orange and killed all aquatic life. The buildup of water in the mine was also a potential source of a blowout or landslide on the steeply-sloping terrain comprising the western mine boundary. To alleviate these problems, horizontal boreholes were drilled through the barrier pillar to intercept the north and central mine pools. The pool levels and flow rates were then regulated by means of manual control valves (Aljoe, 1996).[2]

Water quality analyses of these discharges indicated pH's as low as 2.8, hot acidities as high as 4,430 mg/l, iron readings in excess of 1,400 mg/l, aluminum levels reaching 237 mg/l, and total flows in excess of 380 liters per minute (100 gpm). (Aljoe, 1996).[2]

Following closing of the mine its water treatment plant was operated by a local citizen's group to treat the AMD using bond forfeiture funds. The discharges from the two boreholes and several seeps from punch mines were combined and routed via an underground pipeline to an AMD treatment plant. The treatment consists of anhydrous ammonia for pH elevation, liquid hydrogen peroxide to promote iron oxidation, and two settling ponds for precipitation and storage of metal hydroxide sludges. When sludge production rates exceeded the short-term storage capacity of the settling ponds, sludge was pumped back into the mine via the pipeline (sludge injection line) shown in Figure 1-1. This practice formed a third mine pool in the southwestern portion of the mine. Water decanting from the reinjected sludge emerged along the southwest coal outcrop from fractures and a punch mine and was collected and rerouted to the treatment system (Aljoe, 1996).[2] Injection of sludge into the mine ceased in 1998. The sludge is now permitted to dry in ponds and then hauled to a landfill.

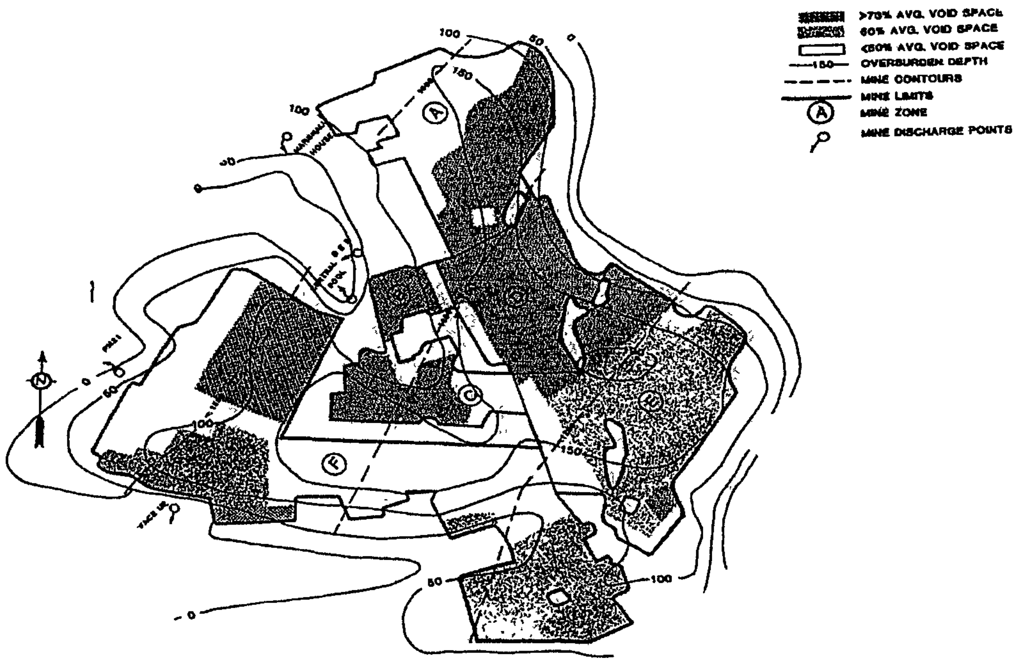



**Figure 3-3**

**Omega Mine – Amount of Extraction**

## Water Sampling

The Omega Mine workings serve as an underdrain for the overlying area. The mine concentrates water from the overlying rock and from updip portions of the mine and discharges through local seeps, springs and the horizontal boreholes installed as part of the AMD treatment system. Many of the seeps were monitored by the U.S. Bureau of Mines (USBM) and WVDEP. Local residences are supplied with public water and ground water use, if any, is probably limited in quantity and not in the immediate vicinity of the North Lobe. There are no sanitary sewers in the project area. Residents have septic tanks.

As the AMD problem developed, both the USBM and the WVDEP began monitoring and water quality testing. Table 3-1 summarizes the data collected (monthly samples and flow measurements) from March 1993 through March 1996. The borehole at the Marshall House into the north mine pool (sampling point 1 on Figure 1-1) was the most contaminated AMD source, accounting for over 50 percent of the contaminant loading at the treatment inlet (sampling point 10). Discharges from several punch mines were combined to form Seeps DEF (sampling point 2); this source flows perennially and constitutes the second greatest source of contamination. The borehole into the central mine pool (sampling point 4) was the third greatest source of contamination; its AMD concentrations were higher than those of Seeps DEF, but it flows for only about seven months of the year (December through June). However, on several occasions during the study, the central pool flow rate was high enough to make it the largest contaminant source. The only important contaminant source from the southwest mine pool was PM 21 (sampling point 9), which emanates from a collapsed punch mine opening. Its AMD concentrations were lower than those of the other three sources in Table 3-1, but its consistently high flow rate made it a significant source of contaminant load. The injection of treatment sludge into the mine was undoubtedly a factor in the high flow rate of PM 21 and the existence of two to four intermittent seeps in the same area (represented by sampling points 7 and 8 in Figure 1-1). Ammonia concentrations of all discharges in the southwest pool area were consistently greater than 100 mg/L. Despite the fact that the water accompanying the injected sludge was alkaline, the alkalinity was lost within the mine voids; i.e., all the southwest pool discharges had low pH with elevated concentrations of dissolved iron and aluminum (Aljoe, 1996).[2]

Sampling points 2, 5 and 6 in Figure 1-1 result from leakage through or around the barrier pillar in the outcrop area. The AMD concentrations and/or low flow rates of these seepages were very low compared with those shown in Table 3-1, but they were also monitored to document any changes that might occur as the result of filling the North Lobe (Aljoe, 1996).[2]

**Table 3-1**
**Median AMD Concentrations and Percentage Load of Omega Discharges (Aljoe, 1996)[2]**

| | | Acidity | | Total Iron | | Aluminum | |
|---|---|---|---|---|---|---|---|
| Sampling Point | I.D. No. (Fig. 3-3) | Conc. (mg/L) | % Total Load | Conc. (mg/L) | % Total Load | Conc. (mg/L) | % Total Load |
| North Pool | 1 | 3960 | 55 | 1307 | 57 | 218 | 48 |
| Seeps DEF | 3 | 2161 | 25 | 590 | 22 | 120 | 25 |
| Central Pool | 4 | 2444 | 16 | 741 | 15 | 145 | 14 |
| PM 21 | 9 | 653 | 12 | 149 | 7 | 42 | 14 |

## Benthic Survey

On August 29, 1994, the West Virginia Division of Environmental Protection, Office of Water Resources and Office of Abandoned Mine Lands and Reclamation initiated a survey of Owl and Booth's Creeks in Monongalia County, West Virginia. The purpose of this study was to determine the current conditions of habitat, benthic macroinvertebrate communities and water chemistry at eight stations located in the headwaters of both streams. The data was intended to have utility as baseline information so future studies could be compared against the data to assist in determining changes in the streams' physical, biological and chemical attributes through time. Based on the results of the samples collected, water quality was lower at all stations sampled when compared to the reference station. Habitat was generally very good at all stations sampled. Only four families of intolerant macroinvertebrates were found at the stations. The remainder of the macroinvertebrates were pollution facultative or tolerant. Nonimpaired biological conditions were found at the reference station and one other station which was borderline with the moderately impaired category. Another station was listed as moderately impaired. The biological conditions at the remainder of the stations were found to be severely impaired. A heavy coating of iron precipitate was found at three of the eight stations sampled. Because of the small number of organisms found in the samples four of the ecological metrics were found to have little utility. However, species richness, the EPT index, the community loss index and the % contribution of the dominant family metric indicated lower water quality at all of the stations when compared to the reference station. Field measurements for pH and conductivity also supported the conclusion that water quality was degraded at all stations when compared to the reference station. (Nowlin, 1994).[47]

# 4

# SITE EVALUATION AND REMEDIATION PLANNING

The State of West Virginia assumed responsibility for water treatment at the Omega mine after all the bond forfeiture funds had been consumed by the treatment effort. However, it was recognized immediately that the existing treatment system could not be maintained in perpetuity, and expanding the treatment facilities was not feasible because of the steep terrain and the population density of the area. Therefore, the WVDEP developed a plan to completely fill the North Lobe of the mine with a grout composed of alkaline fly ashes from local utility sources. Given the contaminant load distribution in Table 3-1, it was hoped that eliminating flow recharge from the North Lobe would result in a 50 to 75 percent reduction in contaminant loading at the site. Although flow from the central pool area might increase, an overall reduced contaminant load was anticipated (Aljoe, 1996).[2]

When Congress eliminated the US Bureau of Mines in 1996, some staff members who were interested in AMD and had sampled water quality at the Omega Mine were transferred to the US Department of Energy (USDOE) where they continued water sampling at the Omega Mine. In the spring of 1998, the USDOE joined the other sponsors of the mine injection project.

GAI Consultants, Inc., which was under contract to the WVDEP for abandoned mine land remediation work, was assigned the responsibility to design an injection plan and develop a suitable mix for AMD remediation and subsidence prevention. Fly ash, which could be supplied by Allegheny Energy and ash from a fluidized bed combustion plant which could be supplied by Anker Energy, were to be evaluated.

## Site Evaluation and Injection Planning

GAI became involved in the Omega Project in July 1994 in support of Allegheny Energy's interest in aiding the reduction of AMD from the Omega Mine. Initial tasks were to determine the existing conditions by talking with knowledgeable persons, studying mine maps and examining the site.

### *Preliminary Information:*

From the interviews it was learned that generally (1.2m (4 ft) of coal was present in the Omega Mine. In the northern portion of the mine a 51-76mm (2 to 3 inch) seam of high sulfur coal (5%) was present in the mine roof. This seam tended to fall from the mine roof. It was not mined but fell in areas of second mining. Little water was present in the mine until second mining occurred. Apparently subsidence occurred with pillar splitting and cracks developed on the ground surface.

After the mine was closed water filled the northern (down dip) portion (North Lobe) of the mine. The water may have risen 24m (80 ft) above the lowest portion of the mine. AMD from this pool discharged from the sandstone about 11m (35 ft) above the level of the coal at a rate estimated to be 57 liters (15 gallons) per minute.

To prevent this flow into the Morgantown water supply, the mine pool was drained into the adjacent Owl Creek watershed by drilling two horizontal borings. Valves were placed on these drain holes to control the flow which is treated before flowing into Owl Creek.

It was believed at least 50 percent of the AMD from the Omega Mine was being generated in the north pool (North Lobe) of the mine, where the roof contained high amounts of sulfur.)

The flow from the mine pool in summer is often less than 114 liters per minute (30 gpm). Since the nearby surface streams were small, water for injection of CCP's into the mine had to be obtained from the Clinton Water Company.

Anker Energy was willing to donate 181,560 metric tons (200,000 tons) of FBC ash to the project from the Morgantown Energy Associates Power Plant. Allegheny Energy Supply planned to contribute fly ash from the Ft. Martin Power Station with a flyash from the Hatfield's Ferry Power Station as an alternate source.

## Initial Planning

In 1994, GAI (then working for Allegheny Energy) determined the amount of mining in 7 sections of the Omega Mine (A through F in Figure 3-3) and estimated the probable volume of voids to be filled (Table 4-1).

The volume of grout estimated to fill Area A (the North Lobe) was approximately 57,341 cubic meters (75,000 cubic yards). The total void volume of the Omega Mine was estimated to be 275,240cm (360,000 cy). These data were generated from examining the mine map. No exploration to determine conditions in the mine had been conducted.

Preliminary costs were estimated for grouting the North Lobe and the entire mine with a fly ash/cement mix and an 80 percent fly ash - 20 percent FBC ash mix. The estimated costs for the North Lobe were $1,427,000 for the fly ash-cement grout and $1,172,000 for the fly ash-FBC grout. For the entire mine the costs were $6,749,000 and $5,522,000, respectively. Figure 3-3 identifies the mine boundaries, mining zones considered separately, and various other data. Table 4-1 contains a summary of estimated quantities for the various portions of the mine.

**Table 4-1**
**Omega Mine Remediation Options**

| Areas | Mined Area (100%) | Total Area (sq. ft) | Void Area (Sq. ft.) | Drilling Centers (ft) | No. of Borings | Avg Depth of Borings (ft) | Drilling (Lineal ft) | Assumed Mining Thickness (ft) | Grout Take (100%) | Grout Vol. (cy)* | Grouting Days (1000 yds/day) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Area A** | | | | | | | | | | | |
| First Mining | 0.50 | 462,172 | 231,086 | 100 | 46 | 125 | 5,750 | 5 | 1 | 42,797 | 43 |
| Second Mining | 0.75 | 573,587 | 430,190 | 50 | 229 | 140 | 32,060 | 4 | 0.5 | 31,868 | 32 |
| Subtotal | | 1,035,759 | 661,276 | | 275 | | 37,810 | | | 74,665 | 75 |
| **Area B** | | | | | | | | | | | |
| First Mining | 0.36 | 29,192 | 10,509 | 100 | 3 | 120 | 360 | 4 | 1 | 1,557 | 2 |
| Second Mining | 0.71 | 137,710 | 97,774 | 50 | 55 | 75 | 4,125 | 4 | 0.5 | 7,243 | 7 |
| Subtotal | | 166,902 | 108,283 | | 58 | | 4,485 | | | 8,800 | 9 |
| **Area C** | | | | | | | | | | | |
| First Mining | 0.41 | 89,001 | 36,490 | 100 | 9 | 120 | 1,080 | 4 | 1 | 5,406 | 5 |
| Second Mining | 0.78 | 367,280 | 286,478 | 50 | 147 | 120 | 17,640 | 4 | 0.5 | 21,222 | 21 |
| Subtotal | | 456,281 | 322,968 | | 156 | | 18,720 | | | 26,628 | 27 |
| **Area D** | | | | | | | | | | | |
| First Mining | 0.45 | 88,322 | 39,745 | 100 | 9 | 150 | 1,350 | 4 | 1 | 5,889 | 6 |
| Second Mining | 0.70 | 596,623 | 417,636 | 50 | 29 | 150 | 35,850 | 4 | 0.5 | 30,938 | 31 |
| Subtotal | | 684,945 | 457,381 | | 248 | | 37,200 | | | 36,827 | 37 |
| **Area E** | | | | | | | | | | | |
| First Mining | 0.47 | 535,868 | 251,858 | 100 | 54 | 160 | 8,640 | 4 | 1 | 37,315 | 37 |
| Second Mining | 0.74 | 48,266 | 35,717 | 50 | 19 | 150 | 2,850 | 4 | 0.5 | 2,646 | 3 |
| | | | | | | | | | | | |

| Areas | Mined Area (100%) | Total Area (sq. ft) | Void Area (Sq. ft.) | Drilling Centers (ft) | No. of Borings | Avg Depth of Borings (ft) | Drilling (Lineal ft) | Assumed Mining Thickness (ft) | Grout Take (100%) | Grout Vol. (cy)* | Grouting Days (1000 yds/day) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Second Mining | 0.60 | 712,303 | 427,382 | 50 | 285 | 160 | 45,600 | 4 | 0.5 | 31,660 | 32 |
| Subtotal | | 1,296,437 | 714,957 | | 358 | | 57,090 | | | 71,621 | 72 |
| **Area F** | | | | | | | | | | | |
| First Mining | 0.44 | 1,111,823 | 489,202 | 100 | 111 | 110 | 12,210 | 4 | 1 | 72,480 | 72 |
| Second Mining | 0.73 | 417,792 | 304,988 | 50 | 167 | 100 | 16,700 | 4 | 0.5 | 22,594 | 23 |
| Second Mining | 0.60 | 1,042,182 | 625,309 | 50 | 417 | 85 | 35,445 | 4 | 0.5 | 46,323 | 46 |
| Subtotal | | 2,571,797 | 1,419,499 | | 695 | | 64,355 | | | 141,397 | 141 |
| Total | | 6,212,121 | 3,684,364 | | 1,790 | | 219,660 | | | 359,938 | 361 |

*To convert cubic yards to cubic meters multiply by 0.76455

Three potential problems were identified with the injection program: 1) redirected mine water flow, 2) induced subsidence and 3) acid contribution of sandstone overburden.

> Redirected mine water flow - The greatest potential for redirecting mine water flow with adverse results could be filling of the mine in Area A. The worst case scenario occurred if mine water and ground water drained down dip into the fractured sandstone overlying Area A, and rose into the overlying sandstone, resulting in seepage discharges into the Cobun Creek watershed. This risk also existed with the complete filling of the mine. Filling Area B would not adversely affect drainage patterns.
>
> Induced subsidence – Drilling and introducing fluid grouts into the mine workings for the purpose of filling the mine voids could disturb existing conditions and trigger subsidence events which would otherwise occur over a longer period of time. This potential existed for each of the proposed options.
>
> Acid Contribution from sandstone overburden - The overlying sandstone could produce acidic water. Filling the mine could result in acidic ground water flowing from the sandstone into both Cobun and Owl Creeks.

Based on the probable costs estimated in 1994, by May 1995 the interested parties had decided to limit injection to the North Lobe since much of the AMD came from the North Lobe. Reducing flows and/or improving water quality from this area was judged most cost effective.

## Final Planning

WVDEP took possession of the Omega Mine site on January 1, 1995. Negotiations continued through 1995 between the project sponsors. The WVDEP agreed to manage and pay $1.375 million of the $2.6 million project. The other project sponsors, Allegheny Energy, Anker, Consol, EPRI and OSMRE contributed the balance. The Omega Mine Project Agreement was finalized at a signing ceremony in Charleston, West Virginia on March 6, 1996. The agreement limited injection activities to the North Lobe of the mine (believed to be where much of the AMD was produced). The agreement also called for CCB's from Allegheny Energy and Anker to be evaluated for use in the remediation injection program.

In January 1996 the WVDEP as the project manager issued a work directive to GAI Consultants to develop an effective low cost plan for the reduction of AMD and to reduce or eliminate the potential for subsidence in the North Lobe of the Omega Mine.

GAI's scope of work included site investigation, laboratory evaluation of grout mixes, preparation of drawings and specifications for the injection work and participation in advisory committee meetings.

Specific tasks were as follows:

- Obtain and review data.
- Interact with the various financial contributors to the project.
- Conduct a detailed site reconnaissance.
- Conduct surveying to locate borings with respect to available mine mapping and locate utilities and "as-built" borings on aerial topographic mapping to be supplied by the WVDEP.
- Conduct a subsurface exploration program including subsurface videotaping.
- Perform hydrogeologic data collection and monitoring.
- Conduct a laboratory testing program in order to design grout mixes.
- Complete preliminary design for review by WVDEP.
- Prepare construction documents.
- Participate in pre-bid and pre-construction meetings.
- Participate in advisory committee meetings.

## Subsurface Exploration

Prior to exploration the project area was examined for depressions, subsidence sinkholes and cracks that would permit precipitation infiltration into the Omega Mine. None were found. Only the storm drains along U.S. Route 119 were identified as possible sources of localized infiltration into the mine.

A subsurface investigation was designed to characterize the mine and provide information that would be used in a selection of grout mixes, design of the grouting program, and assessing the generation of AMD. More specifically, the subsurface investigation had three principal objectives:

> Characterize the voids in first and second mined areas to estimate grout quantities, assess the magnitude and areal extent of roof falls and estimate their effect on grout flow, and evaluate the ability to completely fill the mine with grout.
>
> Assess the continuity of coal on the downdip (west) side of the Omega Mine to identify areas of potential blowouts. Possible reinforcement of these potential blowout areas was to be addressed during design of the grouting plan.
>
> Obtain samples of immediate roof and overburden rock for laboratory testing to help identify acid generation potential.

The subsurface investigation included a drilling program, video camera logging, and laboratory testing. During planning, potential interconnection of punch mines with the Omega Mine workings along the west and east sides of the North Lobe were considered. A combined geophysical and drilling investigation to evaluate the integrity of coal barriers between possible punch mines and the Omega Mine workings was evaluated. The geophysics work involved an in coal seam seismic survey. This part of the investigation was not conducted due to cost and lack of confidence the investigation would yield a determinant answer on barrier integrity.

The exploratory drilling program included three core borings (CB-1 through CB-3) and fourteen air rotary borings. The locations of these borings are shown on Figure 4-1. Boring depths were estimated using the best available topographic and mine maps but, due to scale limitations, were anticipated to vary from expected depths by 3 to 6m (10 to 20 ft). All borings were cased to the top of rock to allow for potential use for monitoring and/or grout injection.

The core borings (CB-1 through CB-3) were intended to penetrate 1.5m (5 ft) below solid coal in the mined seam. The rock core recovered from these borings was used to characterize and describe the rock that overlies the mine, and the rock immediately beneath the mine.

Air rotary borings were to be used to obtain information on mine voids and to provide information on the extent of mining.

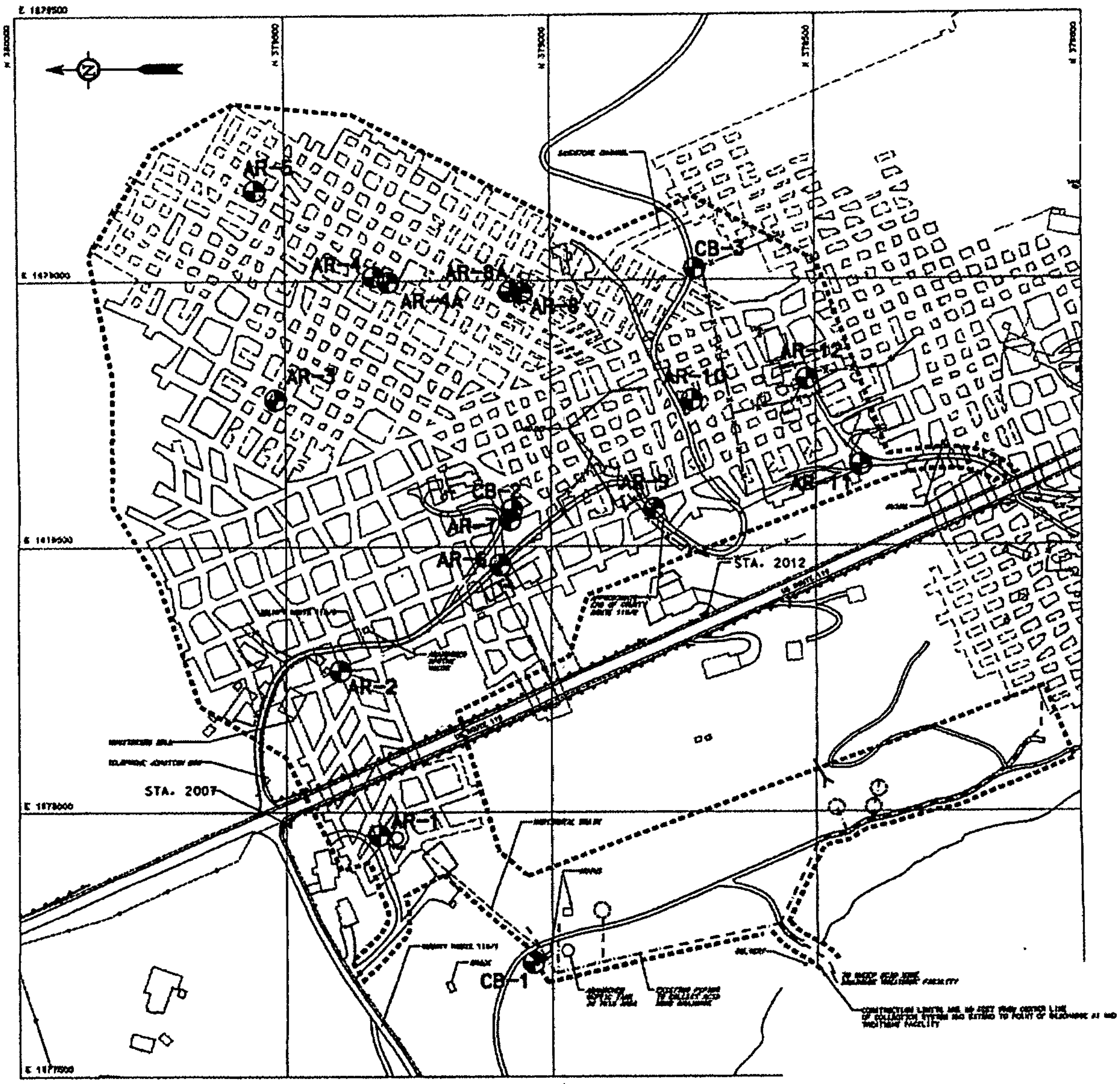



**Figure 4-1**
**Location Plan of Borings**

The air rotary borings were located to intercept mine voids in both "first" or "second" mined areas. First mined areas describe areas that are believed to have been mined once and reportedly have approximately one-half of the original coal remaining. This area was also expected to have relatively few roof falls. In second mined areas, many pillars were split following the initial coal extraction. The additional mining resulted in extraction estimated to be over 70 percent of the original coal. The coal pillars remaining in the second mined areas were reported to provide inadequate roof support possibly resulting in some extensive roof falls.

Assessment of the condition of the mine voids from drilling data and video inspection provided an indication of the extent of roof fall and its potential impact on grout flow. For example, if roof fall is extensive in a portion of the mine, the grouting plan must utilize either a more flowable mix that is not impeded by the debris or a greater density (closer spacing) of injection borings to enhance void filling. The extent and nature of roof fall thus impacts both the grout mix and injection plan design.

Drilling into mine voids depends on accurate mine and surface surveys and vertical boreholes. Minor deviations in survey data or drilling could result in borings terminating in coal pillars. If some air rotary borings targeted to mine voids encountered solid coal, it was planned to redrill after offsetting approximately 3m (10 ft). Three such borings with a total length of 122m (400 ft) of air rotary drilling were budgeted. Similarly, core borings intended for solid coal but encountering mine voids would be offset and redrilled. For budgetary purposes, it was assumed that redrilling of core borings would not be necessary.

Prior to drilling, surveying work was done to relate surface property to the map of the Omega Mine.

The three core borings [54mm] (2 1/8 inches in diameter) and 14 air rotary holes [152mm] (six inches in diameter) were drilled in May, 1996. All of the borings hit their desired targets, either coal pillars or mine voids. Table 4-2 summarizes the data obtained from the borings.

**Table 4-2**
**Omega Mine Complex - Exploration Boring Program**

| | Proposed | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| Boring | Surface Elevation | Mine Floor | Est. Depth (ft) | Surface Elev. | Depth to Top of Coal/ Void (ft) | Depth to Bottom of Coal/ Void (ft) | Coal Void or Gob | Thickness of Coal/Void (ft) | Elev. Mine Floor | Depth Drilled (ft)* |
| CB-1 | 1500 | 1450 | 50 | 1480 | 12.5 | 15.5 | C | 3 | 1464.5 | 23 |
| CB-2 | 1690 | 1540 | 150 | 1660 | 133.2 | 137 | C | 3.8 | 1523 | 143 |
| CB-3 | 1735 | 1600 | 135 | 1730 | 133.4 | 136.2 | C | 2.8 | 1593.8 | 140 |
| AR-1 | 1580 | 1475 | 105 | 1560 | 80 | 87 | V | 7 | 1473 | 98 |
| AR-2 | 1600 | 1495 | 105 | 1600 | 95 | 103 | V | 8 | 1497 | 110 |
| AR-3 | 1690 | 1540 | 150 | 1690 | 147 | 153 | V | 6 | 1537 | 165 |
| AR-4 | 1715 | 1570 | 145 | 1715 | 138 | 144 | C | 6 | 1571 | 145 |
| AR-4A | | | | 1715 | 140 | 142.5 | G | 2.5 | 1572.5 | 153 |

| | Proposed | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| Boring | Surface Elevation | Mine Floor | Est. Depth (ft) | Surface Elev. | Depth to Top of Coal/ Void (ft) | Depth to Bottom of Coal/ Void (ft) | Coal Void or Gob | Thickness of Coal/Void (ft) | Elev. Mine Floor | Depth Drilled (ft)* |
| AR-5 | 1680 | 1570 | 110 | 1680 | 94.5 | 99.5 | C | 5 | 1580.5 | 100 |
| AR-6 | 1600 | 1530 | 70 | 1640 | 105 | 111 | V | 6 | 1529 | 125 |
| AR-7 | 1700 | 1550 | 150 | 1660 | 132 | 139 | V | 7 | 1521 | 145 |
| AR-8 | 1725 | 1585 | 140 | 1725 | 145 | 149 | C | 4 | 1576 | 165 |
| AR-8A | | | | 1725 | 145.5 | 149.5 | V | 4 | 1575.5 | 165 |
| AR-9 | 1650 | 1540 | 110 | 1680 | 114 | 120 | V | 6 | 1560 | 125 |
| AR-10 | 1720 | 1570 | 150 | 1710 | 137 | 142.5 | V | 5.5 | 1567.5 | 145 |
| AR-11 | 1740 | 1580 | 160 | 1730 | 143 | 147 | C | 4 | 1583 | 152 |
| AR-12 | 1740 | 1600 | 140 | 1740 | 148 | 149 | G | 1 | 1591 | 163.5 |
| | TOTALS | | 1870 | | | | | | | 2262.5 |

* To convert feet to meters multiply by 0.3048

Eight coal pillars were evaluated to determine their ability to support the mine overburden. In a few cases two pillar heights were considered to examine the impact of fallen roof coal which would increase pillar height. The sizes of the pillars were determined from the mine map (Figure 4-1). A strength of 6210 kPa (900 psi) was assumed for the coal. As indicated in Table 4-3 the Factor of Safety of these pillars varied from 0.8 to 5.7. Pillars with a factor of safety greater than 1.5 are generally considered to have long term load carrying capacity. This analysis did not consider failure by pillar punching into the mine floor.

**Table 4-3**
**Coal Pillar -- Factor of Safety**

| Pillar No. | Width(ft) | Length (ft) | Area ($ft^2$) | Assumed Height (ft) | Tributary Area ($ft^2$) | Overburden Thickness (ft) | Factor of Safety |
|---|---|---|---|---|---|---|---|
| 1 | 18 | 31 | 558 | 4.3 | 1662.5 | 150 | 2.2 |
| 1 | 18 | 31 | 558 | 6.3 | 1662.5 | 150 | 1.9 |
| 2 | 10 | 29 | 290 | 4.3 | 1228.5 | 150 | 1.2 |
| 3 | 38 | 58 | 2204 | 4.5 | 4060 | 150 | 5.7 |

| Pillar No. | Width(ft) | Length (ft) | Area ($ft^2$) | Assumed Height (ft) | Tributary Area ($ft^2$) | Overburden Thickness (ft) | Factor of Safety |
|---|---|---|---|---|---|---|---|
| 4 | 13 | 18 | 234 | 4.4 | 737.5 | 180 | 1.5 |
| 5 | 7 | 35 | 245 | 4.6 | 1298.5 | 150 | 0.8 |
| 6 | 9 | 35 | 315 | 4.6 | 1431 | 150 | 1.06 |
| 6 | 9 | 35 | 315 | 6.6 | 1431 | 150 | 0.95 |
| 7 | 15 | 35 | 525 | 4.6 | 1802 | 150 | 1.7 |
| 8 | 15 | 18 | 270 | 4.2 | 1072.5 | 100 | 2.4 |
| 8 | 15 | 18 | 270 | 6.2 | 1072.5 | 100 | 1.9 |

To convert feet to meters multiply by 0.3048.

## Evaluation of Acid Mine Drainage Formation

To evaluate the potential for the formation of Acid Mine Drainage, acid/base accounting tests were performed on coal samples from the exploratory borings, as well as on rock samples from the overburden (from the first two distinct strata over the mine: a sandstone and a silty sandstone) and mine floor to help evaluate the acid generating potential of the strata above and below the mine. In the core samples the rider coal was not distinguishable from the remainder of the Upper Freeport Coal. Laboratory procedures used were in accordance with published procedures (Sobek, 1978).[62] Samples were obtained from discrete intervals of strata in two core borings. The percent total sulfur and neutralization potential (NP) were measured for each sample. The maximum potential acidity (MPa) and net neutralization (NNP) were then estimated for each sample. All were expressed in equivalent tons $CaCo_3$ per 907 metric tons (one thousand tons) of material. The MPa was calculated by multiplying the percent total sulfur by 31.25, a conversion factor that assumes all sulfur contributes to acidity (this possibly overestimates the MPa). The NNP was then estimated by subtracting the MPa from the NP. In all samples except one, the MPa was greater than the NP. Review of the NNP values listed in Table 4-4 indicates that the AMD potential of the Upper Freeport Coal is much greater than surrounding strata as indicated by the coal's NNP values of approximately 10 to 100 times the adjacent strata NNP values.

Thus, based on estimated NNP's, the strata adjacent to the mine (where water may flow after mine filling) has much less potential to contribute AMD than the mined Upper Freeport Coal.

**Table 4-4**
**Acid/Base Accounting Results**

| Sample ID | Strata | Neutralization Potential (NP) tons CaCO3 equiv/thousand tons material | Total Sulfur % | Max. Potential Acidity (MPA) tons CaCO3 equiv/thousand tons material (calculated from % total sulfur) | Net Neutralization Potential (NNP) tons CaCO3 equiv/thousand tons material |
|---|---|---|---|---|---|
| CB-2-1 | Sandstone | 3.5 | 0.13 | 4.1 | -0.6 |
| CB-2-2 | Silty Sandstone | 1.0 | 0.39 | 12.2 | -11.2 |
| CB-2-3 | Upper Freeport Coal | <1 | 12.19 | 380 | -380 |
| CB-2-4 | Shale (underclay) | 2.5 | 1.29 | 40.3 | -37.8 |
| CB-3-1 | Sandstone | 7.0 | 0.09 | 2.8 | 4.2 |
| CB-3-2 | Silty Sandstone | 2.0 | 0.63 | 19.7 | -17.7 |
| CB-3-3 | Upper Freeport Coal | <1 | 5.32 | 170 | -170 |
| CB-3-4 | Shale (underclay) | 3.8 | 0.16 | 5.0 | -1.2 |

## Borehole Camera Investigation

The purpose of the borehole investigation of the mine voids was to evaluate conditions for the flow of the injected material. Only the air rotary holes were examined with the camera. The investigation was conducted by OSMRE which used its camera specially designed for observations in abandoned coal mines. The borehole camera work was conducted on 12 and 13 June, 1996.

Ten air rotary boreholes into mine voids were inspected by video camera and 4 video tapes were recorded. The locations of the air rotary boreholes are shown on Figure 4-1. Brief observations were as follows:

12 June

- AR-2, good roof and visibility, 1.4-1.5m (4.5-5 ft) void
- AR-9, some roof fall and gob, timbers visible, 1.2m (4 ft) void
- AR-10, extensive roof fall, crushed timbers, 0.6-0.9m (2-3 ft) void

- AR-9A, extensive roof fall, crushed timbers, 0.6-0.9m (2-3 ft) void, borehole inclined 6 degrees;

- AR-4A, close to pillars, timbers in good shape, some roof fall, void appears 0.7-0.9m (2.5-3 ft) high.

13 June

- AR-1, only boring with water at mine level, drilling may have loosened a block of roof-- only 1 ft void visible and this is above mine level. Could see sheared roof bolts.

- AR-7, good roof and visibility, stoppings visible, 1.2-1.5m (4-5 ft) void,

- AR-12, boring cap is 76-102mm (3-4 inches) below grade of driveway, extensive roof fall--recorded void may be in roof rock above actual mine level, 0.3-0.5m (1-1.5 ft) void, visibility no more than 1.5-3.0m (5-10 ft).

- AR-3, most open 2nd mined area, timbers in good condition, void appears to be 0.9-1.2m (3-4 ft), borehole inclined 5 degrees.

- AR-6, borehole cap 152mm (6 inches) below regraded road surface, good roof, 1.2±m (4± ft) void; camera problems occurred and could not be fully corrected, camera rotation was accomplished by hand twisting cable at the surface, bearings to objects viewed not available.

The first mined areas (holes AR-1, -2, -6, -7 and -9) were generally open and in good condition with little roof fall. Pillars were intact, roof bolts common and no or few timber posts. The mine floor was covered with several feet of loose gob and the entries could be clearly viewed. The original mine void was only closed by a few feet from floor heave or slight spalling of the immediate roof. First mined areas will take significant amounts of grout and will present little in the way of obstructions other than mine stoppings.

The second mined areas (holes AR-3, -4A, -8A and -10) revealed numerous timber posts, some split or bent, and rock bolts were hanging from the broken roof. Large rock fragments were on the mine floor and the pillars were spalling or crushed. Views of pillars or caved entries were usually blocked by roof fall material. The mine void was closed 50 percent or more from the roof fall materials and swelling of the mine floor. In second mined areas roof fall material will obstruct grout flow through smaller openings. There may be areas where voids are not connected due to roof collapse.

In general, the mine appears relatively open with little or no stress in the overburden rock. Occasional high angle fractures and open horizontal fractures are indicative of the overburden rock closing into the void. No zones indicative of subsidence reaching the surface were observed. Long continuous entries with little roof fall and incomplete closure of the mine voids should provide excellent conduits for the injection of fly ash type materials. Minor restrictions to grout injection may occur in the areas that were observed as secondary mining. Water levels were observed near the floor of the mine or in the shallow gob with the exception of hole AR-1. In this hole the water level was at 21m (69.1 ft) depth from the surface or about 3m (10 ft) above the mine floor. (Ehler, 1996).[22] Table 4-5 summarizes the observations from each hole in the order that they were viewed.

**Table 4-5**
**Borehole Camera Observations 6/12-13/1996, Omega Mine**

| Hole Number | Tape No.; Video Run Time | Depth Mine Void Viewed (ft)* | Total Depth Viewed (ft)* | Mine Level Conditions | Observations/ Conclusions |
|---|---|---|---|---|---|
| AR-2 Axial | Tape 1 - 0:00:00 to 0:07:20 | 92.6-98.8 | 98.8 | Open Void No water level | CSG to 10.7; High angle fracture at 52.6. bottom hole covered w/broken rock fragments |
| AR-2 Radial Zoom | Tape 1 - 0:07:20 to 0:55:32 | 94.4 – 97.8 | 98.8 | Open Void 1st Mining | Numerous hanging roof bolts, gob on floor; Stoppings at brgs 169° and 310° about 25' distant. |
| AR-9 Axial | Tape 1 -- 0:55:32 to 1:05:05 | 114.2 - 120.0 | 120.0 | Open Void Water Level at 120 | CSG to 12.4; Bottom covered with broken rock fragments w/ water filling hole at btm of void. |
| AR-9 Radial Zoom | Tape 1 -- 1:05:05 to 1:46:36 | Same | 120.0 | Open Void 1st Mining | Bottom covered with large rock slabs; few timber posts & bolts. Open room sloping down at 235° |
| AR-10 Axial | Tape 2 -- 0:00:00 to 0:12:14 | 134.4 to 136.5 | 141.9 | Open Void 2nd Mining; water level at 141.0 | CSG to 12.0 ft. Partially collapsed mine void with 5' of gob on floor. |
| AR-10 Radial Zoom | Tape 2 -- 0:12:14 to 0:40:58 | 134.4 to 136.5 | 136.9 | Narrow void 2nd Mining; Gob covered floor. | Numerous timbers some leaning & splintered; no bolts viewed; no clear view of entries or pillars; possible entry 245° & crushing pillar 260°. Mine level about 50 percent collapsed. |
| AR-8A Axial | Tape 2 -- 0:40:58 to 0:50:24 | 144.1 to 145.8 | 149.0 | Open Void 2nd Mining; no water level. | CSG to 12.3 ft. Partially collapsed mine void with 3' of broken rock gob on floor. Zone broken 52-59'. |
| AR-8A Radial Zoom | Tape 2 -- 0:50:24 to 1:06:35 | 144.1 to 145.6 | 145.6 | Narrow void 2nd Mining; Gob covered floor | Numerous timbers some leaning & splintered; hanging bolt w/ plate viewed; no clear view of entries or pillars; Mine level about 50% collapsed. |
| AR-4A Axial | Tape 2 -- 1:06:35 to 1:15:27 | 139.2 to 142.0 | 150.9 | Open Void 2nd Mining; water level at 146.7 | CSG to 10.0 ft Partially collapsed mine void; drilled through broken rock, gob on floor. Occasional horizontal fracture from 57.0 - 72.0 |

| Hole Number | Tape No.; Video Run Time | Depth Mine Void Viewed (ft)* | Total Depth Viewed (ft)* | Mine Level Conditions | Observations/ Conclusions |
|---|---|---|---|---|---|
| AR-4A Radial Zoom | Tape 2 -- 1:15:27 to 1:33:45 | 139.2 to 142.0 | 140.7 | Open void; 2nd Mining; Gob covered floor | Numerous timbers leaning & splintered; few rock bolts w/ plate; possible entry at brg. 270°; crushed pillar from brg. 300° – 360° and intact pillar 180° -230°; Mine level about 30% collapsed. |
| AR-1 Axial | Tape 3 -- 0:00:00 to 0:11:07 | 76.2 to 78.2 | 81.1 | Open Void; 1st Mining; Water level at 69.1 | CSG to 13.0 ft. High angle fractures at 36.0, 39.9. Floor of mine not viewed. |
| AR-1 Radial Zoom | Tape 3 -- 0:11:07 to 0:33:03 | 76.7 to 77.8 | 77.8 | Open void; Can't view through water | Minor void or open fracture at 56.6. Broken bolt or artifact at brg. 119° and 218°. Possible faint view of timbers and intact pillars. |
| AR-7 Axial | Tape 3 -- 0:33:03 to 0:43:45 | 131.0 to 135.8 | 136.8 | Open Void; 1st mining; Water level at 136.3 | CSG. 10 11.8 ft High angle fractures at 79.3. Floor of mine covered w/rock fragments water in hole through floor. |
| AR-7 Radial Zoom | Tape 3 -- 0:43:45 to 1:21:38 | 132.0 to 135.8 | 135.8 | Open void; 1st mining; some gob on floor. | Clear open mine void; High angel fracture 79.4 to 81.0 dipping 80°. Intact pillars, bolts and block stoppings. Stoppings: Brg. 163° w/cribbing; Brg. 335° 1/2 wall into entry; brg 49° w/drain pipe at 35 ft distance. |
| AR-12 Axial | Taps 3 -- 1:21:38 to 1:23:52 | 146.7 to 148.7 | 161.0 | Collapsed Void, Water level 161.0 | CSG depth not recorded; bottom of void to 158.0 broken rock. |
| AR-12 Radial Zoom | Tape 3 -- 1:21:38 to 1:34:43 | same | 148.5 | Void in roof | Narrow void; no pillars or entries observed. Void with very large slabs of rock; widest dimension at 7.5 ft brg. 240°. |
| AR-3 Axial | Tape 4 -- 0:00:00 to 0:08:58 | 145.2 to 147.7 | 153.0 | Open Void, No water level. | CSG to 11.4 Gob on floor drilled through to total depth |
| AR-3 Radial Zoom | Tape 4 -- 0:08:58 to 0:30:27 | 146.8 to 147.7 | 147.7 | Open Void; 2nd mining; Gob covered floor | Numerous timbers leaning & splintered; few rock bolts w/plate on top of gob. Marks in pillar from mining machine; spalling pillar from brg. 316° –235° and intact pillar 350° -070° at 25' distant; Mine level about 30% collapsed but roof appears smooth. |

| Hole Number | Tape No.; Video Run Time | Depth Mine Void Viewed (ft)* | Total Depth Viewed (ft)* | Mine Level Conditions | Observations/ Conclusions |
|---|---|---|---|---|---|
| AR-6 Axial | Tape 4 -- 0:30:57 to 0:49:35 | 104.1 to 108.5 | 109.8 | Open Void; 1st mining; water level at 109.8 | CSG to 12.9 ft. Floor of mine covered w/rock fragments water in hole through floor. |
| AR-6 Radial . Zoom | Tape 4 -- 0:30:57 to 0:49:35 | 105.3 to 108.5 | 105.8 | Open Void; 1st mining | clear open mine void; Intact pillars, open entries near intersection no hanging bolts and no timbers. Floor of mine covered w/shallow gob rock fragments. Camera malfunctioned at VRT 0:41:38 finish w/manual rotation and no brgs. |

(Ehler, 1996)[22]
*To convert feet to meters multiply by 0.3048

Note all depths are approximate as measured in feet from ground level. Depths may differ from the drilling depths. Bearing angles (brg) were measured clockwise by degrees great circle from magnetic North (Ehler, 1996).[22]

# 5

# GROUT MIX DESIGN

The WVDEP in its work directive to GAI indicated that the materials to be considered for injection into the mine would be coal combustion products (CCP) consisting of fly ash and fluidized bed combustion (FBC) ash or mixtures of the two. Fly ash would be provided by Allegheny Energy. Anker Energy, the contracted disposer for Morgantown Energy Associates (MEA), would provide the FBC ash.

The Omega Project Agreement, signed March 6, 1996, further stated that it is the intent of WVDEP and the Contributors (Anker Energy; Monongahela Power Company; Consol, Inc.; Electric Power Research Institute; and the Office of Surface Mining) "to develop new technology to reduce or eliminate AMD and subsidence" through the development of "the most appropriate and best suited grout mixture(s) and placement thereof" with GAI making "a reasonable attempt to incorporate the use of fluidized bed ash into the Project if and where technologically feasible."

## Project Goals

The goals of the grouting program, as identified in discussions between the project sponsors and GAI, were as follows:

- The primary requirement of the grout is to fill the mine voids of the north lobe of the Omega Mine to reduce contact of water and air with acidic material, with a secondary requirement that the grout have some alkaline leaching potential to help treat AMD.
- The set-up grout mix must have sufficient strength to allow it to prevent mine subsidence.
- Use of a mixture of fly ash and FBC material is preferred so that the project will demonstrate the synergistic attributes of the combined materials.

To meet these goals in an economical manner requires utilizing as few injection holes as possible. To accomplish this the grout must be fluid, have stability and develop reasonable strength and dimensional stability.

## Laboratory Program

A laboratory testing program was designed to evaluate potential grout components and to select and evaluate a grout mix(es) of suitable character for the injection program. Data from the subsurface investigation were utilized to develop material parameters and the laboratory program to facilitate the selection of the grout mix(es) that would best meet the requirements of the overall project goals.

The laboratory testing program was initiated with the objective of identifying a grout mixture(s) within 60 days of GAI commencing work. This time frame was selected so the preferred grout(s) could then be further tested prior to beginning the injection program. This required the application of prior knowledge and experience to achieve the objective and precluded an extensive test program that evaluated small, incremental changes in mixture components.

## Laboratory Methods

The laboratory testing was conducted in two phases: a preliminary material screening phase and a mix design and product evaluation phase. The second phase, the mix design and product evaluation, was iterative in that an initial series of tests was conducted with blends of ash at three cement contents. After a review of initial results, refined blends were evaluated. Finally, four selected blends were tested under simulated field conditions of water quality and temperature to evaluate the blends for field applications. The four blends were:

1. Fort Martin fly ash,

2. Harrison fly ash,

3. FBC, and

4. A combination of the above, including FBC.

The laboratory test program evaluated the various grout mixtures by using the following principal criteria:

- **Flowability**, determined by flow cone measurements of approximately 60 seconds, indicating that the grout would flow in the mine without separating into solid and liquid portions, and could be expected to flow into small voids.

- **Set time**, determined to be approximately 2 days for final set, indicating that the grout would remain in place and not flow to other portions of the mine, causing loss of roof contact, when injection at that location ceased.

- **Dimensional stability** indicating that, after placement and final set involving approximately $\pm 1$ percent shrink/swell, the grout mix would have no tendencies to shrink or swell in the long term.

- **Unconfined compressive strength** indicating the grout has the integrity to resist overburden pressures; strength development under simulated mine conditions was used to determine what degree of cementing of the grout product was occurring and to compare the various grout mixtures. Strength was, in this case, a secondary consideration in the grout selection process and, therefore, no specific strength criterion was established for the grout.

An additional consideration in the design of the grout(s) was that the contractor might not have refined mixture control devices. It was therefore important that the mix be readily achievable with unsophisticated equipment, be easily monitored by the WVDEP, and be relatively insensitive to minor variations in mix components.

A portion of the testing program evaluated the grout mixtures in simulated mine conditions. That is, the grout was cured at 10 ° C (50° F) in air and in mine water so that set time, dimensional stability, and strength could be evaluated and predicted for field conditions.

Permeability testing was conducted to determine if the grout would reduce the potential for water and air to contact acidic material. TCLP leaching analysis was conducted to evaluate possible leachate production from the grout.

## Laboratory Testing

The laboratory testing program was designed to address a number of issues regarding the use and mixture of CCP materials as grout. The characteristics of both the coal combustion fly ashes and FBC material are widely variable. From experience it was known that certain high loss-on-ignition fly ashes may not set under mine conditions (lower temperatures, on the order of 10 ° C (50° F), but develop satisfactory strength in the laboratory. Similarly, it had been reported that some FBC materials may "flash" set, have poor fluid and pumping characteristics, and have mineral instability that manifests itself in dimensional instability of the hardened material (EPRI, 1988).[27] These concerns were addressed through an evaluation of the following characteristics of both the fluid and hardened grouts:

- Pumpability/flowability of the fluid grout;
- Dimensional stability of the hardened grout mix;
- Strength when cured at mine temperatures;
- Permeability of hardened grout samples;
- Stability and strength development when cured in mine water;
- Use of CCP materials individually and as blend components to achieve suitable grout mixtures; and
- Identification of methods, suitable for field use, to characterize changes in CCP materials that might require modification of the mixtures.

Representative samples of potential injection materials were obtained from the MEA Morgantown Power Station and from the Allegheny Energy Ft. Martin and Harrison Generating

Stations. MEA provided samples of the FBC material proposed for the grouting project. Both dry and conditioned samples of FBC fly ash and bottom ash were obtained. Allegheny Energy provided dry fly ash from the Ft. Martin Generating Station and conditioned fly ash from the Harrison Generating Station.

Information from previous testing conducted by GAI and others was reviewed to plan the test program. Previous experience indicated that the Allegheny Energy fly ashes would require the addition of a stabilizing agent to solidify. The MEA FBC material, having self-cementing qualities was investigated for use as a stabilizing agent in addition to use as bulk fill material. Because FBC materials have the potential to be unstable due to formation of transient minerals that degrade with time, creating a dimensionally unstable material that loses strength as it undergoes the mineral changes, a main thrust of the laboratory program was to evaluate and tailor the CCP grout blends to preclude deterioration from internal and external influences while taking full advantage of the unique qualities of the candidate materials to produce a relatively strong, stable grout.

The variability of the candidate materials was evaluated by focusing the test program to ascertain that the products being tested were representative of the typical materials that would be used in the grouting program. To accomplish this, several source samplings were conducted over the duration of the test program. The materials used in the program are discussed below.

### *Materials*

#### Fly Ash

Pulverized coal combustion fly ash used in the test program was acquired from the Allegheny Energy Ft. Martin and Harrison Generating Stations from storage silos in May of 1996. The testing for loss-on-ignition, specific gravity, and gradation indicated that the fly ash was representative of the fly ash typically produced at the stations; therefore, no further sampling of these sources was conducted. Characteristics of these fly ashes are presented in Table 5-1.

#### FBC Material

FBC ash samples were obtained from the MEA Power Station in Morgantown over the period of the test program. A more extensive sampling program was conducted due to the potential for varying blends of FBC fly ash (FA) and bottom ash (BA) to be off loaded from the system at any time. The limits and general average product ratios of FBC fly ash to bottom ash were determined and the test program was tailored to investigate the potential impact this variability might have on the performance of the grout blends. Over the period of the test program, the FBC showed characteristics similar to those presented in Table 5-1.

**Table 5-1**
**Laboratory Test Summary Coal Combustion Product Properties**

| Materials | Moisture Content (%) | SIEVE ANALYSIS Gravel (%) | Sand (%) | Fines (%) | pH | LOI (%) |
|---|---|---|---|---|---|---|
| MEA PRODUCTS: | | | | | | |
| FLY ASH | 0.01 | 0.0 | 16.9 | 83.1 | 12.4 | 2.00 |
| BOTTOM ASH | 0.02 | 18.0 | 80.5 | 1.5 | 12.3 | 0.24 |
| 25%FA:75%BA | 0.02 | 9.9 | 65.1 | 25.0 | 12.4 | 1.13 |
| 50%FA:50%BA | 0.02 | 5.9 | 46.2 | 47.9 | 12.5 | 1.55 |
| 75%FA:25%BA | 0.03 | 1.7 | 26.1 | 72.2 | 12.5 | 2.11 |
| PLANT BLEND | 12.70 | 3.2 | 59.6 | 37.2 | 12.3 | 8.86 |
| FT MARTIN PRODUCTS: | | | | | | |
| FLY ASH | 0.11 | 0.0 | 16.3 | 83.7 | 11.5 | 4.11 |
| HARRISON PRODUCTS: | | | | | | |
| FLY ASH | 12.40 | 0.2 | 9.1 | 90.7 | 10.7 | 5.67 |

### Cement

Type I cement meeting the requirements of ASTM C150 was investigated for use as a stabilizing agent due to its general availability.

### Lime

Type N lime meeting the requirements of ASTM C207 was investigated for use as a stabilizing agent.

### Water

Distilled water was used in the preparation of the various grout blends investigated in the test program. Mix water available at the site was anticipated to be a public water supply, based on:

- Presumed access difficulty and lack of availability of Omega Mine drainage.

- Lack of nearby large-scale surface waters.
- Preliminary assumption that Clinton Water Authority public water supply could be used, provided some capital improvements were made to their system.

### Mine Water.

The mine water used in the test program was obtained from the WVDEP water treatment facilities at the subject site. Mine water was used to evaluate the properties of grout cured in the mine.

## *Physical and Engineering Test Procedures*

A brief description of the test procedures and the rationale for performing the tests follows:

### Sample Preparation.

Blends of coal combustion fly ash, FBC ash and stabilizing agents were prepared and adjusted with the addition of water to a uniform fluid consistency. Each batch of grout was classified based on stabilizing agent content calculated as a percentage of the dry ingredients (fly ash or FBC and stabilizing agent) by weight. The percentages of stabilizing agent investigated were 0, 2, 5, 10 and 15 percent for fly ash grouts and 0, 2, and 5 for FBC grouts. Additionally, the FBC product, in ratios of 25FA:75BA, 50FA:50BA and 75FA:25BA, was used as a stabilizing agent in combination with the Allegheny Energy fly ash with the further addition of 0 and 2 percent cement.

Each batch was mixed with a rotary paddle mixer. After mixing, the specific gravity of the grout was determined by mud balance measurements in accordance with ASTM D4380. A flow cone was used to measure fluidity in accordance with ASTM C939 procedures. To provide fluid grout and maintain a uniform consistency from batch to batch for the blends, flow values for all candidate grouts were adjusted through the manipulation of mix water quantities so that a flow value of approximately 60 seconds was obtained. Once the desired consistency was achieved, the mixtures were placed in 76mm (3-inch) diameter by 152mm (6-inch) high cylinder molds and cured under appropriate conditions to model desired exposure scenarios. Initially, to identify candidate blends of materials for use as grouts, the samples were cured in a moist cure room at a controlled temperature of 21 ° C± 2% (72° ± 3° F) and 100 percent humidity. During the curing period, the initial and final time of set was determined by procedures of ASTM C191.

Additional samples of candidate blends identified by the initial testing for time-of-set, dimensional stability, and strength development with age were manufactured and cured at both 21 ° ± 2 ° C (72° ± 3° F) and at 10 ° C (50° F) in air and inundated in mine water to monitor grout behavior at temperatures and conditions representative of the mine environment. After curing for time periods of 3, 7, 28, 90, or 120 days, individual cylindrical specimens were removed from their molds, measured, and weighed to determine density and yield (volume). The specimens were tested for unconfined compressive strength according to ASTM C39 procedures.

## Loss On Ignition.

The loss-on-ignition (LOI) test value is a measure of the amount of chemically bound moisture, carbon, and other volatiles remaining in the CCP's after combustion. Typically, the carbon content of the material is within a percent of the LOI value obtained. From experience, it has been determined that LOI has an influence on the amount of cement necessary to solidify CCP grouts. Generally, the higher the LOI, the more stabilizing agent will be required. Therefore, the LOI values obtained in this test program were not only used as a measure of the variability of the materials but also to provide an indication of the amount of stabilizing agent that would be required to achieve the desired grout characteristics.

According to ASTM C618 procedures, the LOI was determined for a representative, one gram sample that had been dried to a constant weight at a temperature of 55 ° C. The sample was fired in a muffle furnace at 750 ± 50 ° C to a constant weight. The LOI value is presented as a percent that is obtained by dividing the weight loss in the muffle furnace by the dry weight of the sample and multiplying the quotient by 100.

## pH.

The pH of the slurry grouts is measured to provide an indication of the grout's potential to set. Additionally, the pH of the component CCP's is measured to provide a rough estimate of the amount of stabilizing agent required to increase the pH to create a suitable environment for the hydration of the calcium products in the blends. Experience has shown that, typically, fly ashes having a low pH will require more alkaline stabilizer than higher pH fly ashes.

## Moisture Content.

The moisture that is of consequence in a material in engineering applications is present in the voids between the solid particles. Consequently, the moisture content test for engineering purposes is performed in a manor that measures the amount of water present in the void spaces and precautions are taken not to remove water that may be bound to minerals comprising the solid constituents of the solid/void mass. In this application, the moisture content is determined by dividing the weight of the "free" water present by the "dry" weight of the solid constituents and then multiplying by 100. This calculation can produce moisture contents in excess of 100 percent.

The moisture content of the fly ash or FBC was determined using ASTM approved methods for microwave and conventional oven drying. Microwave procedures, according to ASTM D4643 methods, were used as a rapid means of determining moisture contents to expedite sample preparation for testing. Conventional oven drying procedures, according to ASTM D2216 methods, were used for moisture content documentation. The conventional oven drying tests were performed at a temperature of 55° C as suggested in Note 5 of the ASTM standard. The lower temperature is used in this case due to the possibility of gypsum or volatile components being present in the FBC which have the potential to influence the moisture content determinations. In addition, the lower temperatures were used for the hydrated grout samples also. The measurements were obtained from appropriate test samples to determine the moisture contents at which tests for density and strength were conducted.

### Slurry Specific Gravity.

The slurry specific gravity was measured to determine the solids content of the grout. It can be used as one of the field control techniques for monitoring mix consistency. The ASTM Test Method D4380 is a modified method from the American Petroleum Institute's Recommended Practice 13B. The apparatus, the mud balance, is essentially a balance that has a fixed volume container and sliding weight balance. By measuring the weight of a fixed volume of grout, the fluid density can be determined by comparing the grout weight to the weight of water.

### Fluidity.

The fluidity is a measure of the flowability of the grout. This characteristic is determined by measuring the ability of the grout to pass an orifice, and thus, is a means of predicting pumpability and/or void filling capabilities under either static or pressure heads. The fluidity is determined according to procedures of ASTM C939 using a flow cone to determine the time required for a specified volume of grout to flow through a standard sized orifice (often defined as the time of efflux). The cone is initially filled with a standard volume of grout which is permitted to flow from the cone. The time required for the standard volume to efflux the cone is defined as the flow cone value. Flow cone values can be used in the field to control grout mixing so that a uniformly consistent grout product is produced.

The fluidity was monitored in this test program to provide a means of blending the various grouts being investigated to one consistency. This was achieved by limiting flow values to on the order of 60 seconds. From previous experience, this value has provided grout that has had satisfactory pumping and fluid characteristics. Additionally, the fluidity of the grout was monitored as a function of time to determine whether there was a tendency of the FBC material to "flash" set.

### Solid Unit Weight.

The solid unit weight of the cured grout specimens was determined to document the total and dry density of the solidified materials. To determine the solid unit weight, each specimen was physically measured and weighed prior to testing for unconfined strength. After the unconfined strength was determined, the moisture content of the specimen was determined according to conventional oven-drying techniques. From a standard geotechnical calculation, the total and dry densities were determined.

### Yield.

From the measurements for determining the solid unit weight, the yield of the different mixes was determined. The yield is used to determine the volume the fluid grout will occupy once it has hardened. By comparing the height of the specimens when first formed in the cylinder mold and height after curing, a percent of the initial volume is determined.

As a general rule, for a given composition of dry materials, increasing water content will increase flowability; however, beyond a certain point, the yield will begin to diminish due to a larger volume of excess water bleeding out of the material. Typically, mixes will flow when the water content is several percent above that required for saturation of the dry material. If additional water is added, the material remains uniformly mixed while flowing; however, once the fluid grout is in-place, the solids settle out of suspension prior to hardening and the excess water bleeds off, providing less solid volume for initial fluid grout volume.

### Dimensional Stability.

The dimensional stability of the hardened grout was monitored using a free swell testing procedure. This procedure allows the material to either expand or contract without restriction. The investigation of dimensional stability was important in this program because of the affects of undesirable mineralogical changes within the solidified grout that are manifested in a volume change of the grout. Thus, the dimensional stability of the hardened grouts was monitored after the materials had achieved a set and over a prolonged period of time.

Typically, by controlling cementitious material content and moisture and/or using admixtures, dimensional stability can be achieved. Therefore, the test results for dimensional stability were used in this program to assist in the selection of blended FBC and fly ashes with stabilizing agents.

### Time of Set.

The time of set is of interest for predicting the time frame in which the grout will remain fluid and when the grout will have developed a sufficient set to remain where it has settled. This information is critical to project planning in that the flowability of the grout influences the spacing requirements for the injection holes.

ASTM C191 procedure for conventional concrete was used to define time-of-set. The procedure for conventional concrete defines initial set as the time after contact of water and cement required to achieve a needle penetration resistance of 3450 kPa (500psi) and the final set is defined as the elapsed time required to achieve a needle penetration resistance of 27,580 kPa (4000 psi). Revisions to these required strengths are made because predicted unconfined strengths of the grout were not expected to exceed 3450kPa (500 psi) for the most part. Final set was taken as the point of zero penetration. This was difficult to define in some cases where softer grout slurry remained on top of a harder grout mass.

### Unconfined Compressive Strength.

The unconfined compressive strength is used as a measure of a material's ability to support loads without the benefit of lateral confinement. The unconfined compressive strength is a measure of the cohesiveness of the material. Comparisons of the unconfined strength developed as a function of time were used to predict product behavior and to identify those blends that had potential to be used for CCP grout.

ASTM C39 procedures were followed to perform the unconfined compression testing. Three-inch diameter by 152mm (6 in) high specimens were prepared by pouring the mixes into the cylinders in one application. Slight agitation of the molds by gently tapping the sides was performed to achieve some minimal amount of initial consolidation of the fluid grout in the molds. The prepared specimens were covered with polyethylene bags and placed in a moist-cure environment for curing periods from 0 to 120 days. The specimens were removed from the curing environment and tested in unconfined compression.

### Permeability

Permeability testing was conducted according to procedures of U.S. Army Corps of Engineers Publication EM110-2-1906 on samples that had cured for at least 90 days to evaluate the rate of flow through the hardened grout.[24]

### TCLP Leaching.

Leachate generation from cured grout was evaluated by the EPA's TCLP procedures. These tests were conducted to evaluate the potential impact of the grout on the chemical quality of water that may filter through the solidified grout.

## Laboratory Test Results

Summaries of the testing of the Ft. Martin and Harrison fly ashes, the MEA FBC by-products and blends of these components with and without stabilizing agents are provided in Tables 5-1 through 5-11 inclusive. Table 5-1 includes a summary of the as-received moisture contents, gradation, LOI, and pH of the by-products. Table 5-2 summarizes the moisture content, gradation, pH, and LOI of the CCP and cement contents that were tested. Tables 5-3, 5-4, and 5-5 include cement content, flow cone values, mud balance specific gravity, final time of set, pH and dimensional change of the grout blends produced for testing. The dimensional change presented in Tables 5-3, 5-4, and 5-5 includes the volume change of the hardened grout specimens after 90-days exposure in the free swell test. Tables 5-6, 5-7 and 5-8 provide a summary of laboratory testing to monitor the hardened grout moisture content, dry density and unconfined compressive strength development as a function of time and curing conditions. Table 5-9 summarizes 90 day permeability data for FBC material alone and for a blend of FBC ash with Ft. Martin fly ash. Table 5-10 summarizes TCLP results for FBC material alone and blended with Ft. Martin and Harrison products. Table 5-11 presents the results of testing of fluidity as a function of time.

Figure 5-1 portrays the change in specific gravity of an FBC slurry as the percentage of FBC fly ash varies in comparison to FBC bottom ash. Figures 5-2 through 5-28 illustrate dimensional stability, time of set and strength development with time. The figures illustrate the performance of comparative materials so that they facilitate the rational selection of grout blends based on the materials ability to provide superior performance in comparison to the other materials tested. Figure 5-29 presents data regarding grout fluidity over time. Figure 5-29 graphically portrays the physical changes the grout would undergo once mixed but if placement is delayed and/or the period of time the grout would be mobile in the mine. Figures 5-30 through 5-35 present laboratory permeability test results.

## LOI

The LOI values for the as received by-products are included in Table 5-1. The LOI values for the Ft. Martin and Harrison fly ashes were 4.1 and 5.7, respectively, indicating that they would be well suited for use in the grout blends. The LOI for the conditioned MEA by-product was 8.9 percent whereas the LOI values for the dry MEA by-products are on the order of 1 to 2 percent. This indicates that, due to the hydration of calcium oxide products in the FBC, the LOI for the conditioned FBC is a function of the content of hydrated products and carbon.

## pH

The pH values for the as-received by-products are included in Table 5-1 and for the grout blends in Tables 5-2 through 5-5. The Ft. Martin and Harrison fly ashes are alkaline with pH values of 11.5 and 10.7, respectively, thus indicating that they should have the potential to be stabilized with a reasonable amount of stabilizing agent. It has been observed that strength development is somewhat retarded when pH values are less than 11.0. Therefore, as a means of expediting the targeting of grout mixes, a target pH for the grout slurry would be on the order of 11.5. The MEA by-products have pH values in excess of 12.3. In addition, pH values indicate that the grout mixes would have some alkaline leaching potential.

**Table 5-2**

**Laboratory Test Summary Coal Combustion Product Properties of Laboratory Blended Materials**

| PRODUCT | CEMENT CONTENT (%) | MOISTURE CONTENT (%) | SIEVE ANALYSIS GRAVEL (%) | SAND (%) | FINES (%) | pH | LOI (%) |
|---|---|---|---|---|---|---|---|
| | | | | | | | |
| FT MARTIN PRODUCTS: | | | | | | | |
| FLY ASH | 5 | 0.11 | 0.0 | 11.3 | 88.7 | 12.1 | NT |
| FLY ASH | 5(LIME) | 0.08 | 0.0 | 13.6 | 86.4 | 12.4 | " |
| FLY ASH + MEA25FA:75BA | 0 | <0.01 | NT | NT | NT | 11.9 | " |
| FLY ASH + MEA50FA:50BA | 0 | <0.01 | " | " | " | 11.9 | " |
| FLY ASH + MEA75FA:25BA | 0 | <0.01 | " | " | " | 11.9 | " |
| FLY ASH + MEA25FA:75BA | 2 | 0.04 | " | " | " | 12.5 | " |
| FLY ASH + MEA50FA:50BA | 2 | 0.04 | " | " | " | 12.5 | " |
| FLY ASH + MEA75FA:25BA | 2 | <0.01 | " | " | " | 12.6 | " |
| | | | | | | | |
| HARRISON PRODUCTS: | | | | | | | |
| FLY ASH | 5 | 10.83 | 0.2 | 11.0 | 88.8 | 11.9 | NT |
| FLY ASH | 5(LIME) | 10.51 | 0.0 | 14.2 | 85.8 | 12.3 | " |
| FLY ASH + MEA25FA:75BA | 0 | 4.43 | NT | NT | NT | 12.3 | " |
| FLY ASH + MEA50FA:50BA | 0 | 3.99 | " | " | " | 12.3 | " |
| FLY ASH + MEA75FA:25BA | 0 | 5.82 | " | " | " | 12.3 | " |
| FLY ASH + MEA25FA:75BA | 2 | 4.05 | " | " | " | 12.3 | " |
| FLY ASH + MEA50FA:50BA | 2 | 4.14 | " | " | " | 12.3 | " |
| FLY ASH + MEA75FA:25BA | 2 | 4.40 | " | " | " | 12.4 | " |
| NOTES: NT = NOT TESTED | | | | | | | |

**Table 5-3**
**Laboratory Test Summary Slurry Grout Properties MEA Product Grout**

| PRODUCT | CEMENT CONTENT (%) | FLOW CONE (SEC) | SLURRY SPECIFIC GRAVITY | FINAL TIME OF SET (HRS) | pH | DIMENSIONAL CHANGE (%) |
|---|---|---|---|---|---|---|
| | | | | | | |
| FLY ASH | 0 | 58-62 | 1.54 | 91 | 12.4 | 0.9 |
| | | | | | | |
| BOTTOM ASH | 0 | 58-62 | 1.92 | 139 | 12.3 | NT |
| | | | | | | |
| 25%FA:75%BA | 0 | 57-64 | 1.86 | 191 | 12.4 | 1.8 |
| | | | | | | |
| 50%FA:50%BA | 0 | 59-64 | 1.76 | 188 | 12.5 | 1.3 |
| | | | | | | |
| 75%FA:25%BA | 0 | 57-65 | 1.60 | 186 | 12.5 | 0.8 |
| | | | | | | |
| PLANT BLEND | 0 | 58-62 | 1.69 | 74 | 12.3 | 1.8 |
| | | | | | | |
| PLANT BLEND (2ND) | 0 | 54-61 | 1.68 | 126 | NT | NT |
| | | | | | | |
| PLANT BLEND (2ND) | 2 | 56-60 | 1.68 | 78 | " | " |
| | | | | | | |
| PLANT BLEND (2ND) | 5 | 55-63 | 1.69 | 95 | " | " |
| | | | | | | |
| PLANT BLEND (2ND) ** | 5 | 57-59 | 1.69 | 70 | " | " |

NOTES:
NT = NOT TESTED
2ND = SECOND BATCH MIXED WITH RESAMPLED FBC FOR VERIFICATION TESTING.
** USING MINE WATER

**Table 5-4**
**Laboratory Test Summary Slurry Grout Properties Ft. Martin Product Grout**

| PRODUCT | CEMENT CONTENT (%) | FLOW CONE (SEC) | SLURRY SPECIFIC GRAVITY | FINAL TIME OF SET (HRS) | pH | DIMENSIONAL CHANGE (%) |
|---|---|---|---|---|---|---|
| FLY ASH | 0 | 56-65 | 1.59 | NT | 11.5 | NT |
| FLY ASH | 5 | 57-62 | 1.58 | 98 | 12.1 | 0.1 |
| FLY ASH | 5(LIME) | 56-63 | 1.55 | NT | 12.4 | 0.1 |
| FLY ASH | 10 | 55-61 | 1.68 | 51 | NT | NT |
| FLY ASH | 15 | 59-64 | 1.68 | 43 | " | " |
| FLY ASH + MEA25FA:75BA | 0 | 56-65 | 1.77 | 149 | 11.9 | -0.1 |
| FLY ASH + MEA50FA:50BA | 0 | 52-60 | 1.68 | 146 | 11.9 | 0.2 |
| FLY ASH + MEA75FA:25BA | 0 | 57-60 | 1.60 | 144 | 11.9 | 0.4 |
| FLY ASH + MEA25FA:75BA | 2 | 56-64 | 1.71 | 163 | 12.5 | 0.0 |
| FLY ASH + MEA50FA:50BA | 2 | 55-60 | 1.67 | 161 | 12.5 | 0.3 |
| FLY ASH + MEA75FA:25BA | 2 | 55-61 | 1.62 | 160 | 12.6 | 0.1 |
| FLY ASH + MEA50FA:50BA (2ND) | 0 | 57-65 | 1.67 | 143 | NT | NT |
| FLY ASH + MEA50FA:50BA (2ND) | 2 | 55-63 | 1.67 | 142 | " | " |

NOTES:

NT = NOT TESTED
2ND = SECOND BATCH MIXED WITH RESAMPLED FBC FOR VERIFICATION TESTING

**Table 5-5**
**Laboratory Test Summary Slurry Grout Properties Harrison Product Grout**

| PRODUCT | CEMENT CONTENT (%) | FLOW CONE (SEC) | SLURRY SPECIFIC GRAVITY | FINAL TIME OF SET (HRS) | pH | DIMENSIONAL CHANGE (%) |
|---|---|---|---|---|---|---|
| FLY ASH | 0 | 55-64 | 1.69 | NT | 10.7 | NT |
| FLY ASH | 5 | 59-62 | 1.69 | " | 11.9 | 0.0 |
| FLY ASH | 5(LIME) | 55-65 | 1.66 | " | 12.3 | NT |
| FLY ASH | 10 | 55-66 | 1.70 | 91 | NT | " |
| FLY ASH | 15 | 55-65 | 1.71 | 94 | " | " |
| FLY ASH + MEA25FA:75BA | 0 | 55-63 | 1.82 | 119 | 12.3 | 3.0 |
| FLY ASH + MEA50FA:50BA | 0 | 56-57 | 1.73 | 116 | 12.3 | 2.6 |
| FLY ASH + MEA75FA:25BA | 0 | 56-61 | 1.66 | 115 | 12.3 | 3.2 |
| FLY ASH + MEA25FA:75BA | 2 | 56-60 | 1.79 | 54 | 12.3 | -0.1 |
| FLY ASH + MEA50FA:50BA | 2 | 55-62 | 1.72 | 45 | 12.3 | 0.5 |
| FLY ASH + MEA75FA:25BA | 2 | 57-60 | 1.67 | 44 | 12.4 | -0.3 |
| FLY ASH + MEA50FA:50BA (2ND) | 0 | 56-65 | 1.72 | 118 | NT | NT |
| FLY ASH + MEA50FA:50BA (2ND) | 2 | 55-62 | 1.72 | 114 | NT | NT |

NOTES:

NT = NOT TESTED
2ND = SECOND BATCH MIXED WITH RESAMPLED FBC FOR VERIFICATION TESTING

**Table 5-6**
**Laboratory Test Summary Grout Property Averages MEA Product Grout**

| PRODUCT | CEMENT CONTENT (%) | CURING CONDITIONS (DAYS/TEMP) | MOISTURE CONTENT (%) | DRY DENSITY (PCF) | UNCONFINED STRENGTH (PSI)* |
|---|---|---|---|---|---|
| FLY ASH | 0 | 3/72F | 60.5 | 63.4 | 10 |
| | | 7/72F | 66.4 | 58.9 | 75 |
| | | 28/72F | 59.6 | 60.7 | 238 |
| | | 90/72F | 33.5 | 71.8 | 200 |
| | | 7/50F(2ND) | 85.3 | 49.3 | 10 |
| BOTTOM ASH | 0 | 3/72F | N | N | N |
| | | 7/72F | N | N | N |
| | | 28/72F | 28.4 | 89.4 | 31 |
| | | 90/72F | 15.0 | 100.4 | 269 |
| 25%FA:75%BA | 0 | 3/72F | 30.8 | 88.3 | 17 |
| | | 7/72F | 29.4 | 90.2 | 66 |
| | | 28/72F | 18.6 | 97.0 | 516 |
| | | 90/72F | 11.1 | 100.5 | 666 |
| 50%FA:50%BA | 0 | 3/72F | 44.3 | 74.9 | 13 |
| | | 7/72F | 38.3 | 77.3 | 139 |
| | | 28/72F | 18.8 | 88.9 | 666 |
| | | 90/72F | 10.7 | 96.3 | 989 |
| 75%FA:25%BA | 0 | 3/72F | 58.7 | 63.7 | 9 |
| | | 7/72F | 50.3 | 65.4 | 147 |
| | | 28/72F | 35.8 | 70.9 | 394 |
| | | 90/72F | 28.4 | 75.8 | 456 |
| PLANT BLEND | 0 | 3/72F | N | N | N |
| | | 7/72F | 52.5 | 68.2 | 44 |
| | | 28/72F | 44.9 | 71.9 | 115 |
| | | 90/72F | 22.7 | 83.8 | 213 |
| | | 7/50F(2ND) | 82.3 | 49.3 | 10 |
| | | 7/72F(2ND) | 91.0 | 47.6 | 11 |
| | | 28/50F(2ND) | 55.6 | 54.9 | 24 |
| | | 28/72F(2ND) | 70.7 | 53.0 | 34 |
| | | 90/50F(2ND) | 48.3 | 54.6 | 50 |
| | | 90/72F(2ND) | 67.9 | 54.8 | 38 |
| PLANT BLEND | 2 | 7/50F(2ND) | 80.4 | 52.1 | 4 |
| | | 7/72F(2ND) | 78.2 | 98.3 | 11 |
| | | 28/50F(2ND) | 75.2 | 51.2 | 8 |
| | | 28/72F(2ND) | 76.1 | 49.8 | 34 |
| | | 90/50F(2ND) | 66.6 | 45.9 | 81 |
| | | 90/72F(2ND) | 98.1 | 88.9 | 41 |
| PLANT BLEND | 5 | 7/50F(2ND) | 58.0 | 58.8 | 8 |
| | | 7/50F(2ND) ** | 31.6 | 83.0 | 102 |
| | | 7/72F(2ND) | 60.6 | 57.2 | 14 |
| | | 28/50F(2ND) | 59.7 | 56.4 | 12 |
| | | 28/50F(2ND) ** | 29.0 | 81.5 | 698 |
| | | 28/72F(2ND) | 40.5 | 75.5 | 65 |
| | | 90/50F(2ND) | 40.8 | 51.3 | 121 |
| | | 90/72F(2ND) | 57.8 | 59.2 | 113 |

NOTES:
N = MATERIAL DID NOT GAIN SUFFICIENT STRENGTH TO BE REMOVED FROM THE MOLDS AND TESTED.
2ND = SECOND BATCH MIXED WITH RESAMPLED FBC FOR VERIFICATION TESTING.
** USING MINE WATER

*To convert strength in pounds per square inch to kilopascals multiply by 6.895

**Table 5-7**
**Laboratory Test Summary Grout Property Averages* Ft. Martin Product Grout**

| PRODUCT | CEMENT CONTENT (%) | CURING CONDITIONS (DAYS/TEMP) | MOISTURE CONTENT (%) | DRY DENSITY (PCF) | UNCONFINED STRENGTH (PSI)+ |
|---|---|---|---|---|---|
| | | | | | |
| FLY ASH | 0 | 3/72F | 45.3 | 69.4 | 5 |
| | | 7/72F | 45.8 | 70.5 | 8 |
| | | 28/72F | 44.1 | 70.0 | 20 |
| | | 90/72F | 34.5 | 70.4 | 17 |
| | | | | | |
| FLY ASH | 5 | 3/72F | 46.2 | 68.2 | 41 |
| | | 7/72F | 47.6 | 66.3 | 46 |
| | | 28/72F | 43.2 | 65.8 | 54 |
| | | 90/72F | 41.1 | 66.3 | 78 |
| | | | | | |
| FLY ASH | 5(LIME) | 3/72F | 50.1 | 66.7 | 8 |
| | | 7/72F | 51.4 | 65.3 | 6 |
| | | 28/72F | 49.2 | 66.0 | 14 |
| | | 90/72F | 49.4 | 67.0 | 23 |
| | | | | | |
| FLY ASH | 10 | 7/50F | 37.3 | 77.3 | 29 |
| | | 7/72F | 33.7 | 79.4 | 112 |
| | | 28/50F | 28.5 | 80.7 | 93 |
| | | 28/72F | 28.4 | 81.2 | 164 |
| | | 90/50F | 26.4 | 81.1 | 129 |
| | | 90/72F | 32.3 | 80.3 | 227 |
| | | | | | |
| FLY ASH | 15 | 7/50F | 34.8 | 79.7 | 56 |
| | | 7/72F | 36.2 | 77.3 | 121 |
| | | 28/50F | 29.2 | 79.8 | 129 |
| | | 28/72F | 32.3 | 78.1 | 163 |
| | | 90/50F | 25.2 | 80.6 | 174 |
| | | 90/72F | 27.5 | 81.8 | 349 |

+To convert strength in pounds per quare inch to kilopascals multiply by 6.895

**Table 5-7 (Continued)**

| PRODUCT | CEMENT CONTENT (%) | CURING CONDITIONS (DAYS/TEMP) | MOISTURE CONTENT (%) | DRY DENSITY (PCF) | UNCONFINED STRENGTH (PSI)+ |
|---|---|---|---|---|---|
| | | | | | |
| FLY ASH + MEA25FA:75BA | 0 | 3/72F | 27.7 | 85.6 | 53 |
| | | 7/72F | 37.9 | 75.5 | 137 |
| | | 28/72F | 22.2 | 87.1 | 790 |
| | | 90/72F | 16.1 | 92.0 | 1336 |
| | | 7/72F(2ND) | 37.9 | 75.5 | 120 |
| | | | | | |
| FLY ASH + MEA50FA:50BA | 0 | 3/72F | 33.0 | 79.3 | 38 |
| | | 7/72F | 41.0 | 72.6 | 114 |
| | | 28/72F | 28.8 | 79.6 | 514 |
| | | 90/72F | 27.6 | 81.7 | 651 |
| | | 7/50F(2ND) | 37.0 | 74.8 | 28 |
| | | 7/72F(2ND) | 41.0 | 72.6 | 110 |
| | | 7/72F(2ND) | 32.5 | 77.5 | 113 |
| | | 28/50F(2ND) | 31.6 | 77.8 | 419 |
| | | 28/72F(2ND) | 26.3 | 80.7 | 437 |
| | | 90/50F(2ND) | 12.0 | 77.8 | 531 |
| | | 90/72F(2ND) | 18.8 | 87.0 | 698 |
| | | 3/50F(3RD) | 45.3 | 72.1 | 11 |
| | | 3/50F(3RD) ** | 48.1 | 70.3 | 9 |
| | | 3/72F(3RD) | 45.9 | 71.5 | 29 |
| | | 7/50F(3RD) | 46.1 | 70.4 | 15 |
| | | 7/50F(3RD) ** | 45.3 | 71.8 | 14 |
| | | 7/72F(3RD) | 45.4 | 70.5 | 82 |
| | | 28/50F(3RD) | 29.5 | 75.2 | 69 |
| | | 28/50F(3RD) ** | 38.8 | 73.8 | 64 |
| | | 28/72F(3RD) | 39.3 | 73.8 | 355 |
| | | | | | |
| FLY ASH + MEA75FA:25BA | 0 | 3/72F | 42.4 | 70.9 | 34 |
| | | 7/72F | 47.8 | 67.4 | 103 |
| | | 28/72F | 40.7 | 69.9 | 293 |
| | | 90/72F | 42.0 | 70.7 | 395 |
| | | 7/72F(2ND) | 47.8 | 67.4 | 101 |

+To convert strength in pounds per square inch to kilopascals multiply by 6.895

| Central Pool<br>Combined State (815 HI Manhole) & Federal (Loc. ID #8) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| 1993 | | | 2928.5 | 3305.0 | 2712.0 | 2436.5 | | | | | | 2900.0 |
| 1994 | 2989.0 | | | | 2514.0 | 2410.0 | 1937.0 | | | | | |
| 1995 | 1302.0 | 2385.0 | 2371.0 | 2302.0 | 1921.0 | 2509.0 | | | | | | |
| 1996 | 2522.5 | 3881.5 | 4317.5 | 3460.7 | 2965.0 | 3053.7 | 1419.5 | 2966.0 | 4030.0 | 2281.0 | 1594.5 | 2933.0 |
| 1997 | 2403.0 | 1763.0 | 2944.5 | 2050.0 | 2584.5 | 1830.0 | | | | | | 1690.0 |
| 1998 | 2747.0 | 2620.0 | 2126.0 | 2725.5 | 3240.0 | 1330.0 | 2050.0 | 1653.5 | 761.5 | | | |
| Average | 2392.7 | 2662.4 | 2937.5 | 2768.6 | 2656.1 | 2261.5 | 1802.2 | 2309.8 | 2395.8 | 2281.0 | 1594.5 | 2507.7 |
| 1999 | 2240.0 | 3141.5 | 2897.3 | 2509.0 | 3660.0 | | | | | | | |
| 2000 | 1000.0 | 745.0 | 4020.0 | 2870.0 | 3030.0 | 1890.0 | 2740.0 | 4650.0 | | | | |
| Average | 1620.0 | 1943.3 | 3458.6 | 2689.5 | 3345.0 | 1890.0 | 2740.0 | 4650.0 | | | | |

| Central Pool<br>Pre- and Post-Grouting Average Net Acidity | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| Pre-1999 | 2392.7 | 2662.4 | 2937.5 | 2768.6 | 2656.1 | 2261.5 | 1802.2 | 2309.8 | 2395.8 | 2281.0 | 1594.5 | 2507.7 |
| 1999 to date | 1620.0 | 1943.3 | 3458.6 | 2689.5 | 3345.0 | 1890.0 | 2740.0 | 4650.0 | | | | |

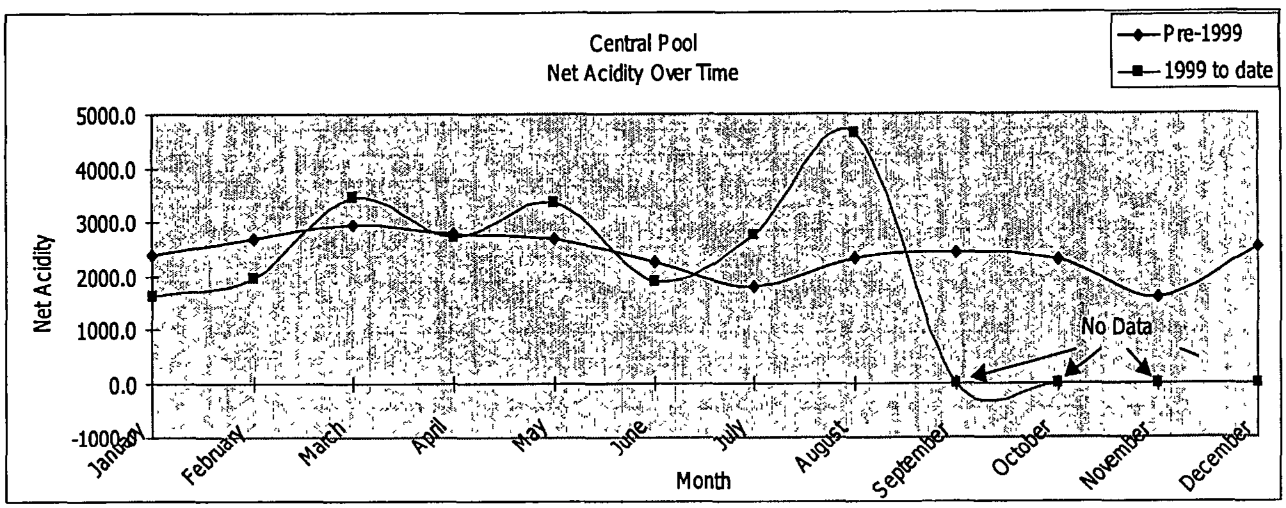



**Figure 7-11**
**Central Pool – Net Acidity Over Time**

**Table 5-8**

**Laboratory Test Summary Grout Property Averages Harrison Product Grout**

| PRODUCT | CEMENT CONTENT (%) | CURING CONDITIONS (DAYS/TEMP) | MOISTURE CONTENT (%) | DRY DENSITY (PCF) | UNCONFINED STRENGTH (PSI)* |
|---|---|---|---|---|---|
| | | | | | |
| FLY ASH | 0 | 3/72F | N | N | N |
| | | 7/72F | 37.1 | 82.8 | 11 |
| | | 28/72F | 39.0 | 76.8 | 10 |
| | | 90/72F | 35.7 | 83.0 | 6 |
| | | | | | |
| FLY ASH | 5 | 3/72F | 42.8 | 72.3 | 22 |
| | | 7/72F | 41.2 | 73.6 | 32 |
| | | 28/72F | 48.1 | 74.2 | 48 |
| | | 90/72F | 29.9 | 80.7 | 239 |
| | | | | | |
| FLY ASH | 5(LIME) | 3/72F | 44.1 | 74.8 | 8 |
| | | 7/72F | 46.2 | 73.9 | 3 |
| | | 28/72F | 42.3 | 73.8 | 18 |
| | | 90/72F | 40.3 | 76.3 | 109 |
| | | | | | |
| FLY ASH | 10 | 7/50F | 39.4 | 76.3 | 51 |
| | | 7/72F | 43.6 | 71.9 | 45 |
| | | 28/50F | 30.1 | 75.7 | 66 |
| | | 28/72F | 35.6 | 73.2 | 94 |
| | | 90/50F | 17.9 | 38.4 | 70 |
| | | 90/72F | 40.5 | 71.4 | 121 |
| | | | | | |
| FLY ASH | 15 | 7/50F | 40.4 | 75.1 | 92 |
| | | 7/72F | 39.6 | 94.6 | 117 |
| | | 28/50F | 29.0 | 74.8 | 102 |
| | | 28/72F | 32.9 | 76.8 | 225 |
| | | 90/50F | 20.6 | 78.3 | 142 |
| | | 90/72F | 23.8 | 88.4 | 364 |

*To convert strength in pounds per square inch to kilopascals multiply by 6.895

**Table 5-8 (Continued)**

| BY-PRODUCT | CEMENT CONTENT (%) | CURING CONDITIONS (DAYS/TEMP) | MOISTURE CONTENT (%) | DRY DENSITY (PCF) | UNCONFINED STRENGTH (PSI)* |
|---|---|---|---|---|---|
| | | | | | |
| FLY ASH | 0 | 3/72F | N | N | N |
| | | 7/72F | 37.1 | 82.8 | 11 |
| | | 28/72F | 39.0 | 76.8 | 10 |
| | | 90/72F | 35.7 | 83.0 | 6 |
| | | | | | |
| FLY ASH | 5 | 3/72F | 42.8 | 72.3 | 22 |
| | | 7/72F | 41.2 | 73.6 | 32 |
| | | 28/72F | 48.1 | 74.2 | 48 |
| | | 90/72F | 29.9 | 80.7 | 239 |
| | | | | | |
| FLY ASH | 5(LIME) | 3/72F | 44.1 | 74.8 | 8 |
| | | 7/72F | 46.2 | 73.9 | 3 |
| | | 28/72F | 42.3 | 73.8 | 18 |
| | | 90/72F | 40.3 | 76.3 | 109 |
| | | | | | |
| FLY ASH | 10 | 7/50F | 39.4 | 76.3 | 51 |
| | | 7/72F | 43.6 | 71.9 | 45 |
| | | 28/50F | 30.1 | 75.7 | 66 |
| | | 28/72F | 35.6 | 73.2 | 94 |
| | | 90/50F | 17.9 | 38.4 | 70 |

*To convert strength in pounds per square inch to kilopascals multiply by 6.895

**Table 5-9**
**Laboratory Test Summary 90-Day Permeability Averages* Plant Blends and Laboratory Blends**

| PRODUCT | CEMENT CONTENT (%) | CURING CONDITIONS (DAYS/TEMP) | PERMEABILITY (CM/SEC) |
|---|---|---|---|
| | | | |
| MEA BY-PRODUCTS: | | | |
| | | | |
| PLANT BLEND (2ND) | 0 | 96/50F | $6.94 \times 10^{-5}$ |
| | | | |
| PLANT BLEND (2ND) | 0 | 96/50F + | $7.31 \times 10^{-5}$ |
| | | | |
| PLANT BLEND (2ND) | 0 | 96/72F | $5.27 \times 10^{-5}$ |
| | | | |
| | | | |
| FT MARTIN BY-PRODUCTS: | | | |
| | | | |
| FLY ASH + MEA50FA:50BA (2ND) | 2 | 92/50F | $9.28 \times 10^{-6}$ |
| | | | |
| FLY ASH + MEA50FA:50BA (2ND) | 2 | 92/50F + | $9.44 \times 10^{-6}$ |
| | | | |
| FLY ASH + MEA50FA:50BA (2ND) | 2 | 92/72F | $9.45 \times 10^{-6}$ |

NOTES:

2ND = SECOND BATCH MIXED WITH RESAMPLED FBC FOR VERIFICATION TESTING.
* AVERAGE OF TWO SAMPLES UNLESS INDICATED OTHERWISE.
+ ONLY ONE SAMPLE TESTED

**Table 5-10**
**Laboratory Test Summary 90-Day Toxicity Characteristic Leaching Procedure (TLCP) Results Plant Blends and Laboratory Blends**

| PRODUCT | TCLP PARAMETERS (mg/L) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | As | Ba | Cd | Ca | Cr | Fe | Pb | Mg | Mn | Hg | Se | Ag | SO4 -2 |
| MEA BY-PRODUCTS: | | | | | | | | | | | | | |
| PLANT BLEND (2ND) | 0.002 | 4.92 | 0.053 | 1840 | 0.07 | 0.26 | 0.9 | 70.1 | 0.11 | <0.0002 | <0.002 | 0.02 | 771 |
| FT MARTIN BY-PRODUCTS: | | | | | | | | | | | | | |
| FLY ASH + MEA50FA:50BA 2% CEMENT CONTENT (2ND) | 0.003 | 4.96 | 0.071 | 1860 | 0.07 | 0.29 | 0.9 | 78.4 | 1.12 | <0.0002 | 0.002 | 0.03 | 783 |
| HARRISON BY-PRODUCTS: | | | | | | | | | | | | | |
| FLY ASH + MEA50FA:50BA 2% CEMENT CONTENT (2ND) | <0.002 | 5.03 | 0.06 | 1950 | 0.05 | 0.23 | 0.8 | 74.2 | 1.03 | <0.0002 | <0.002 | 0.05 | 756 |

NOTES:

2ND = SECOND BATCH MIXED WITH RESAMPLED FBC FOR VERIFICATION TESTING.

**Table 5-11**
**Laboratory Test Summary Suggested Grout Mix Grout Flow and Angle of Repose**

| TIME AFTER MIXING (MIN.) | FLOW VALUE (SEC.) | INITIAL ANGLE OF REPOSE (%) | ELAPSED TIME, INITIAL TO FINAL ANGLE OF REPOSE (MIN.) | FINAL ANGLE OF REPOSE (%) |
|---|---|---|---|---|
| 0 | 58 | 0 | - | 0 |
| 30 | 72 | 0 | - | 0 |
| 60 | 91 | 0 | - | 0 |
| 120 | 138 | 15 | 5 | 7 |
| 180 | - | 24 | 10 | 12.6 |

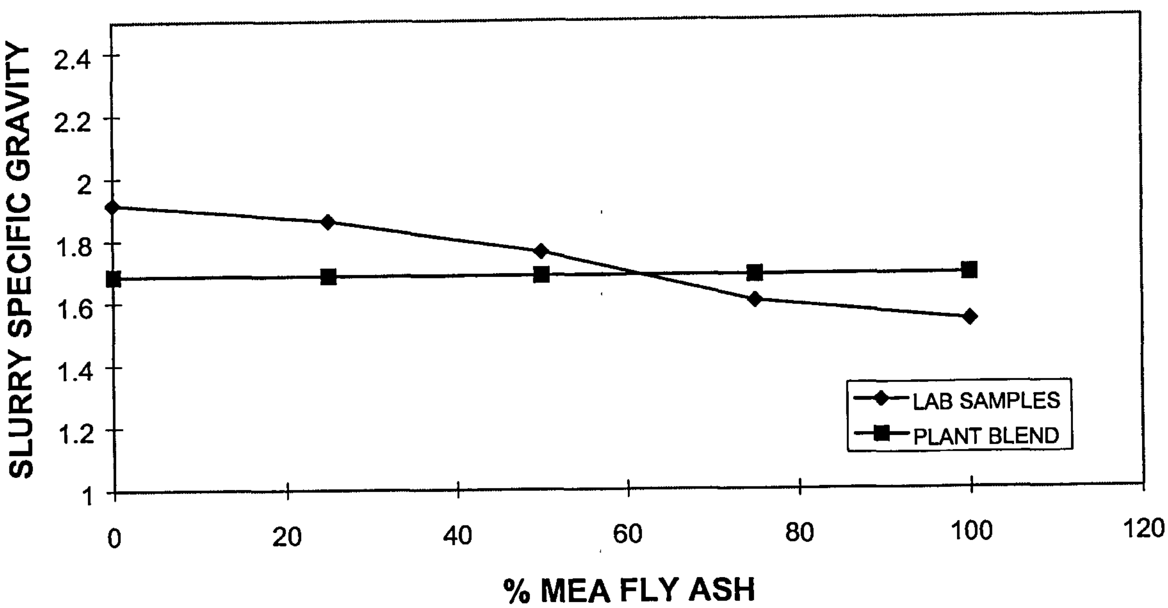



**Figure 5-1**
**Slurry Specific Gravity as % MEA Fly Ash**

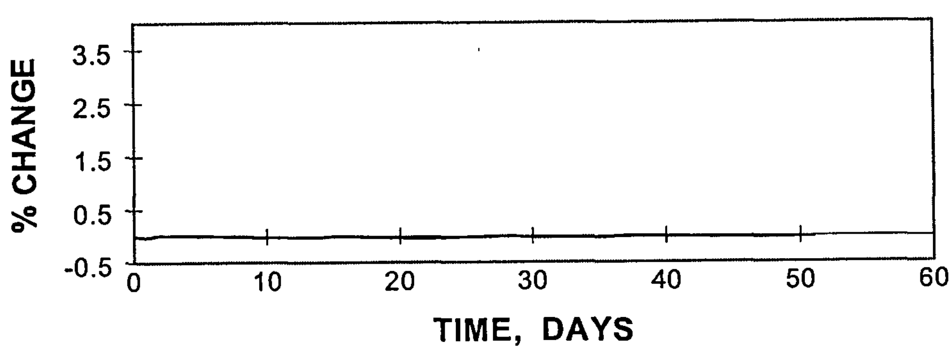



**Figure 5-2**
**Dimensional Stability, Harrison FA + 5% Cement**

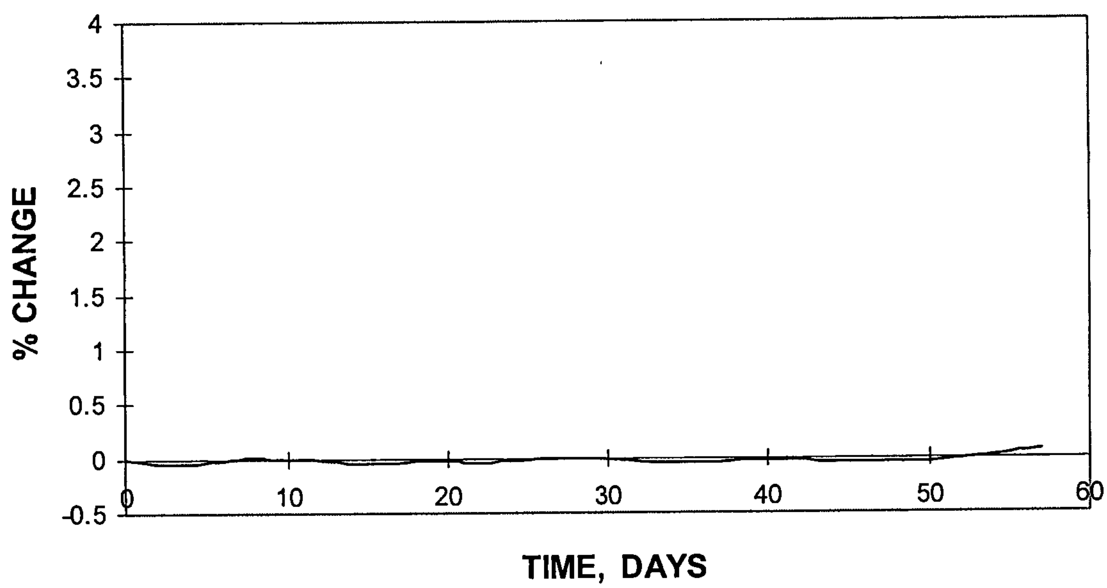



**Figure 5-3**
**Dimensional Stability Ft. Martin FA +5% Lime**

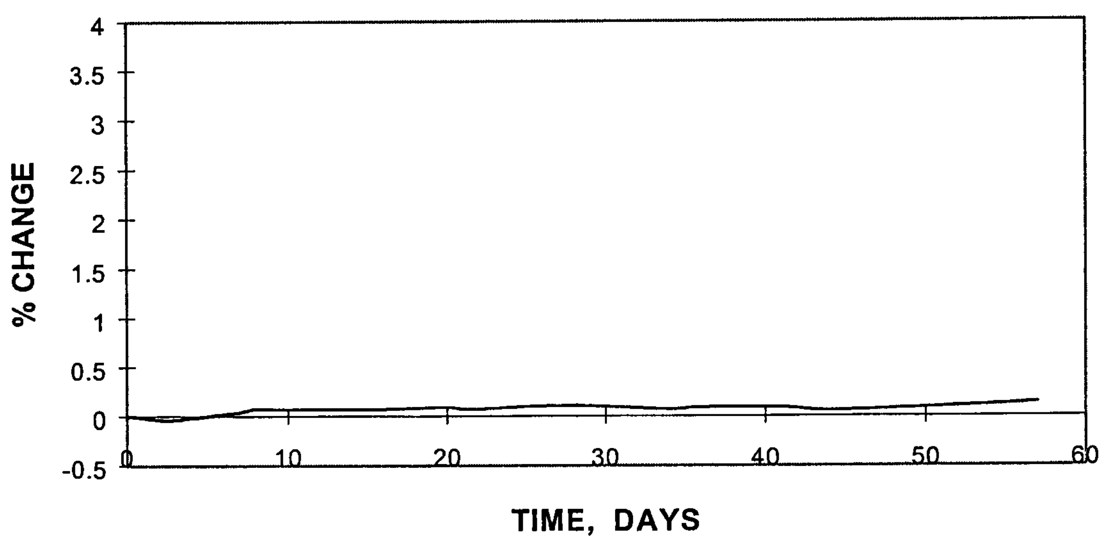



**Figure 5-4**
**Dimensional Stability Ft. Martin FA +5% Cement**

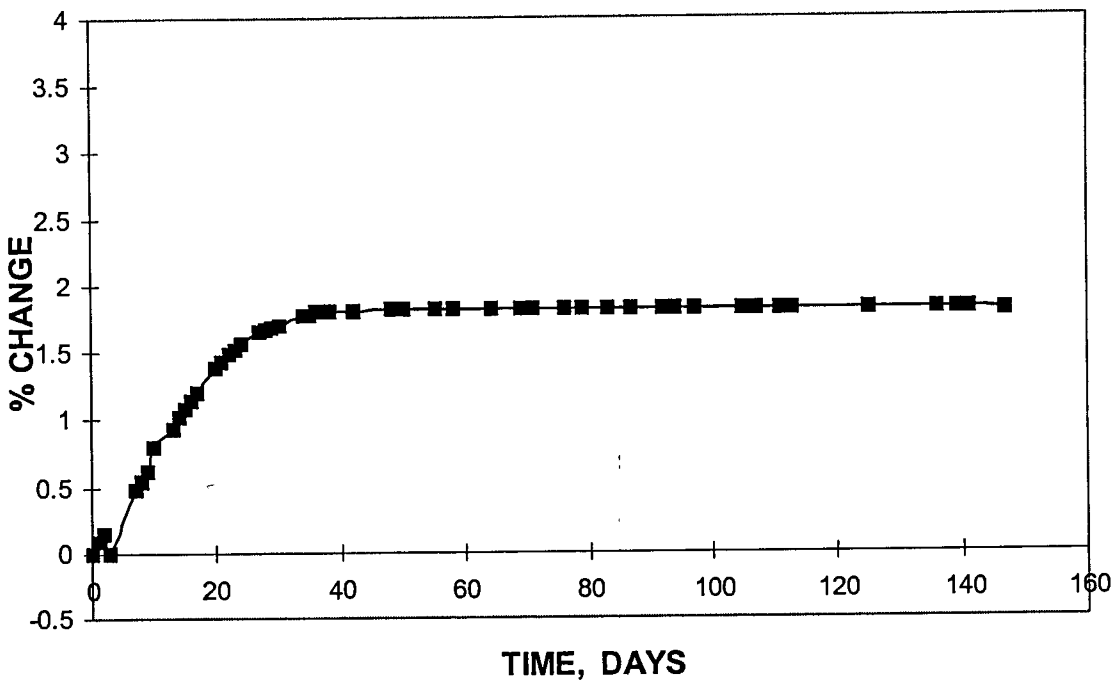



**Figure 5-5**
**Dimensional Stability 100% MEA Conditioned Plant Bend**

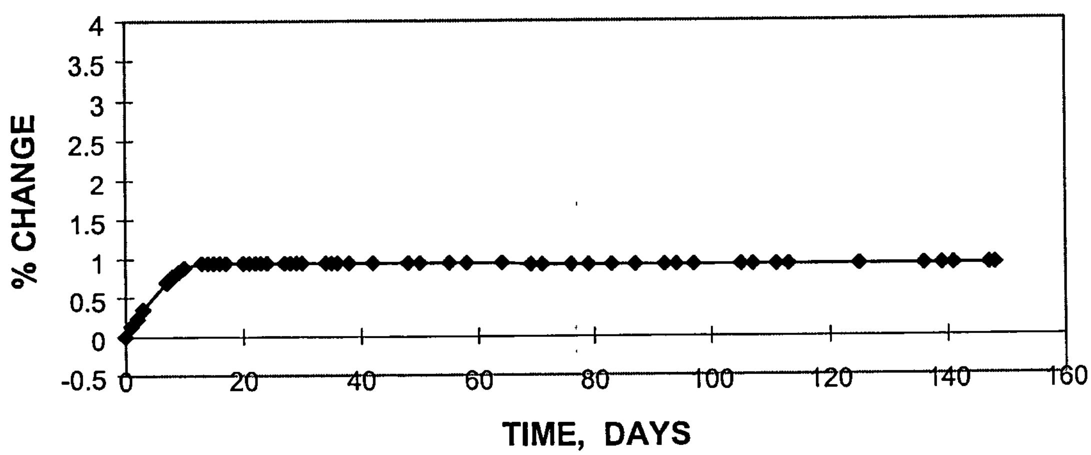



**Figure 5-6**
**Dimensional Stability 100% MEA FA**

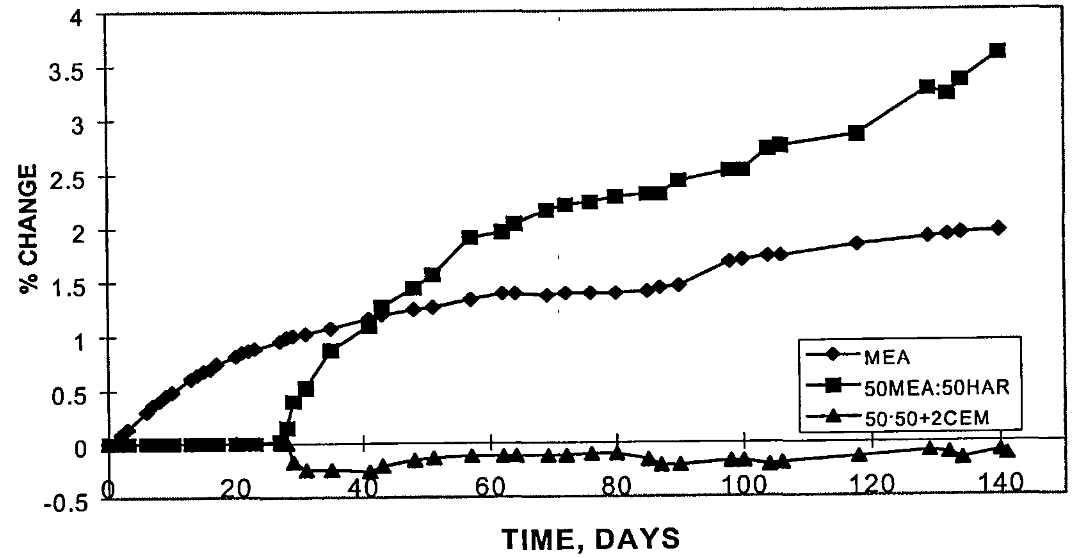



**Figure 5-7**
**Dimensional Stability 25 MEAFA:75 MEABA+Harrison FA**

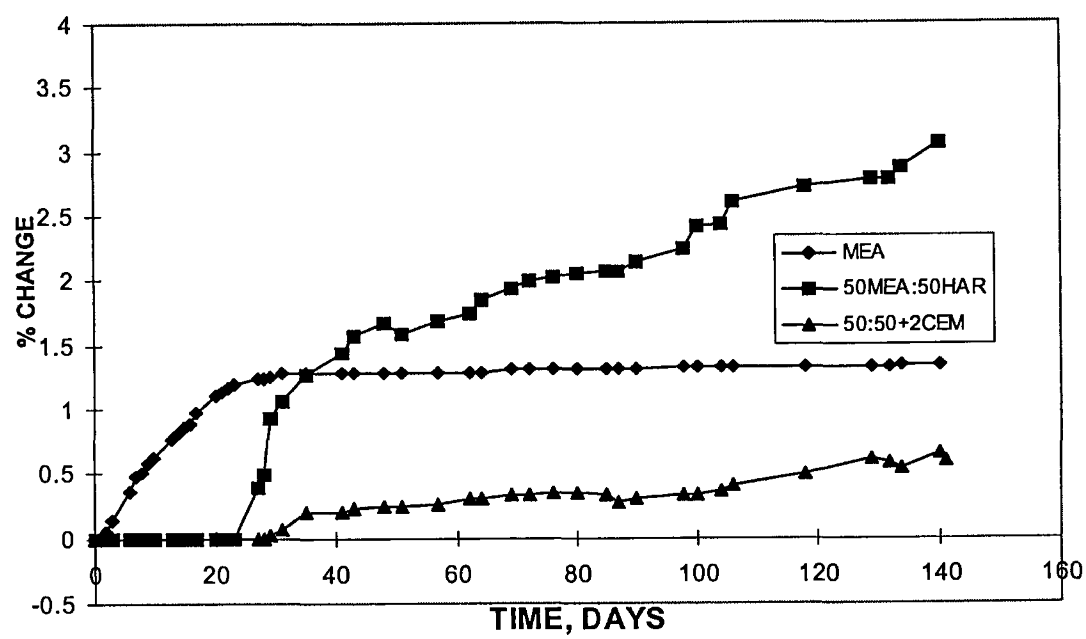



**Figure 5-8**
**Dimensional Stability 50MEAFA:50MEABA+Harrison FA**

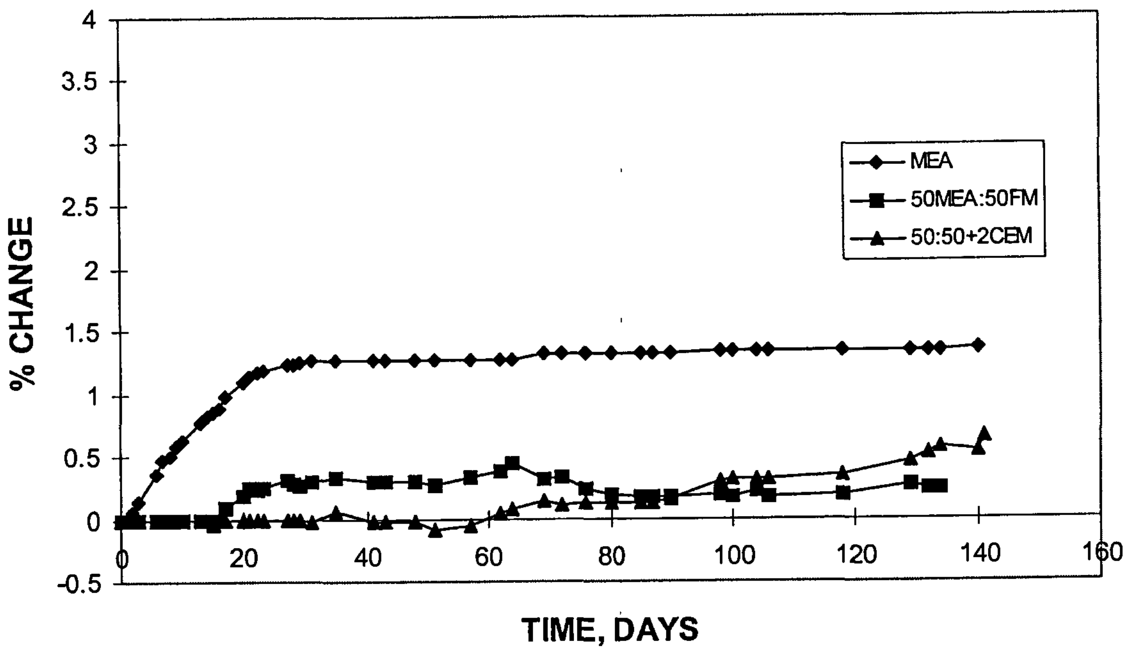



**Figure 5-11**
**Dimensional Stability 50MEAFA:50MEABA+Ft. Martin FA**

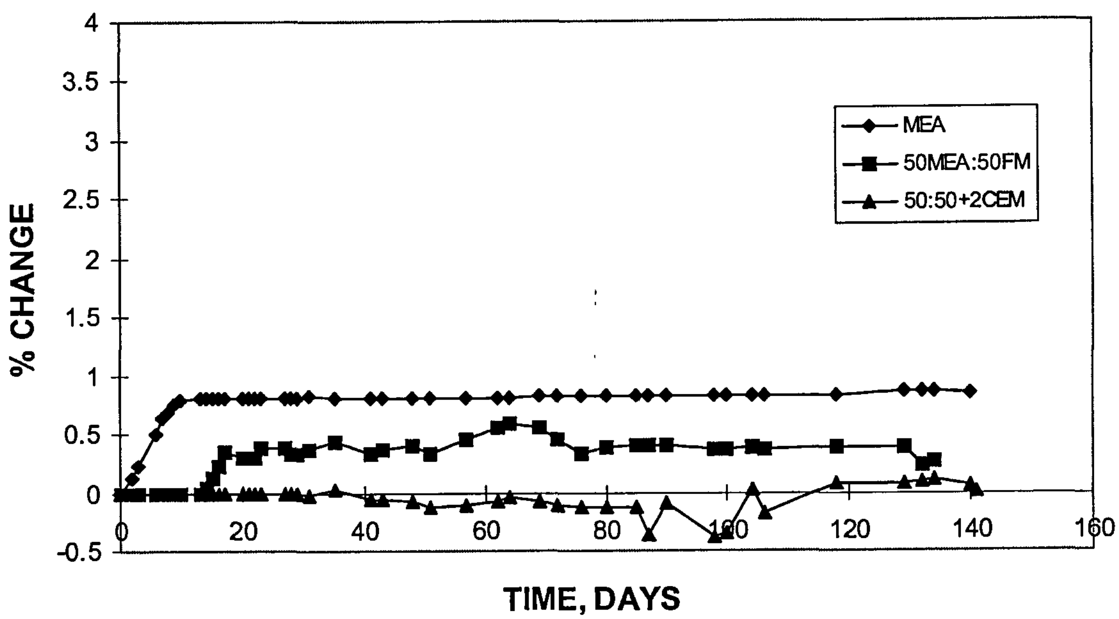



**Figure 5-12**
**Dimensional Stability 75MEAFA:25MEABA+Ft. Martin FA**

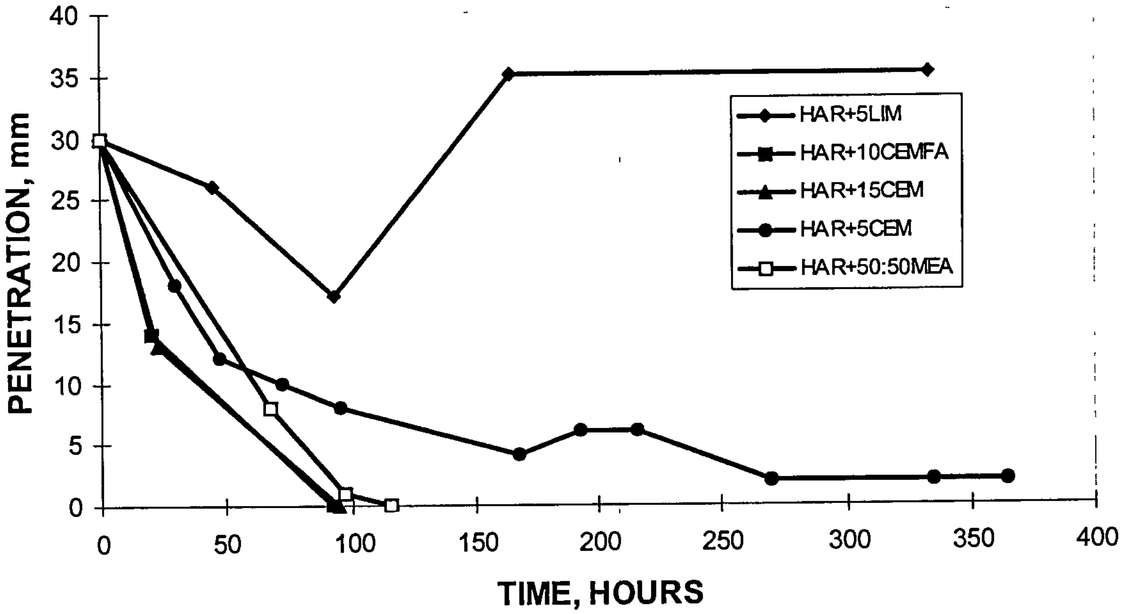



**Figure 5-13**
**Time of Set Harrison FA**

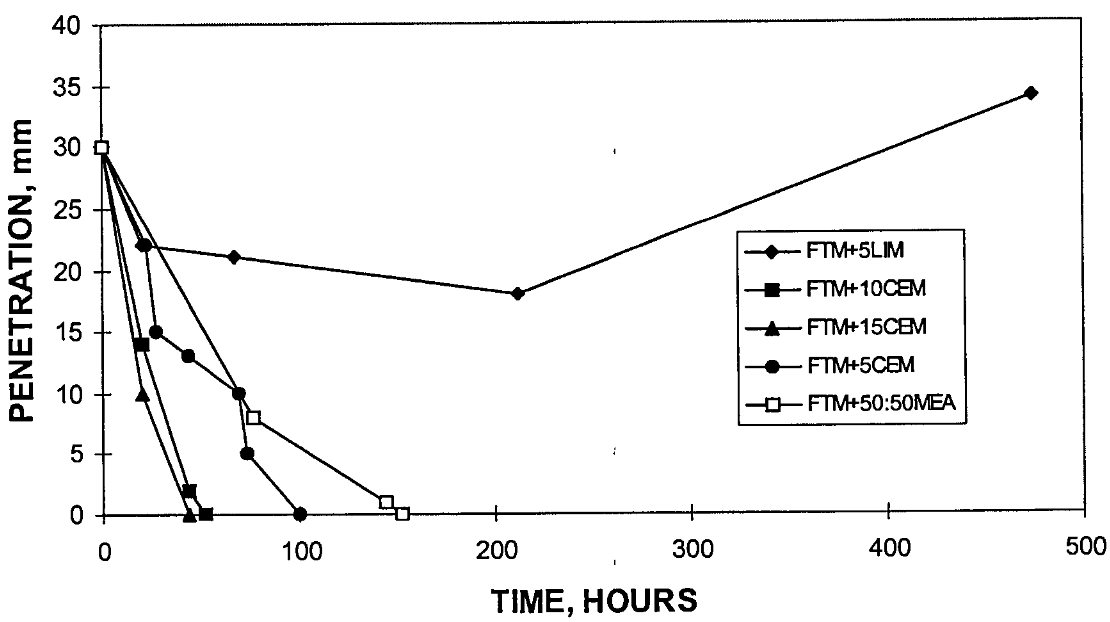



**Figure 5-14**
**Time of Set Ft. Martin FA**

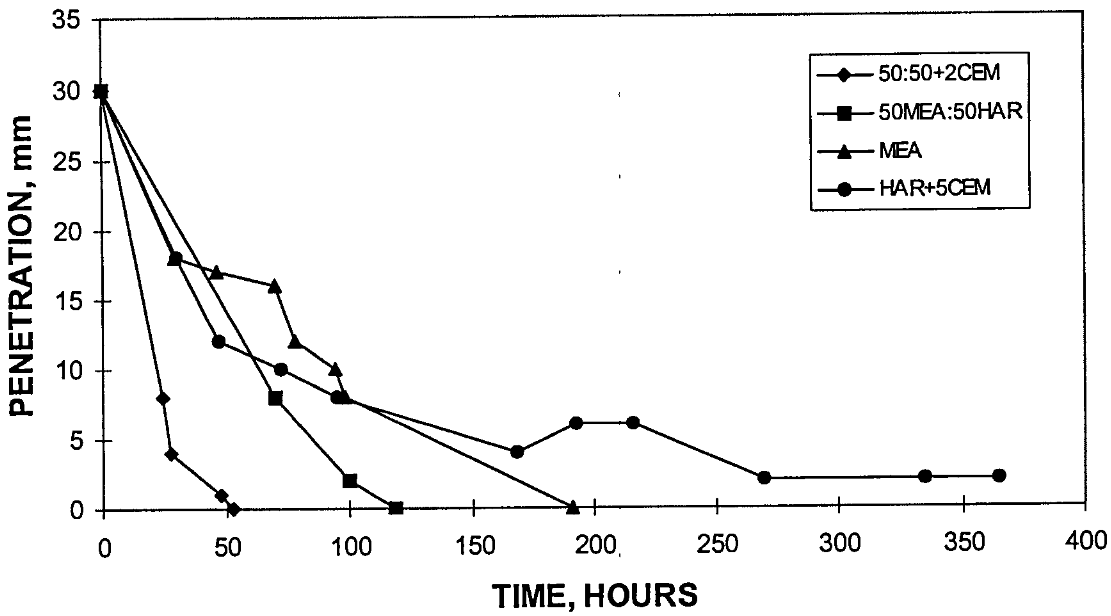



**Figure 5-15**
**Time of Set 25MEAFA:75MEABA+Harrison FA**

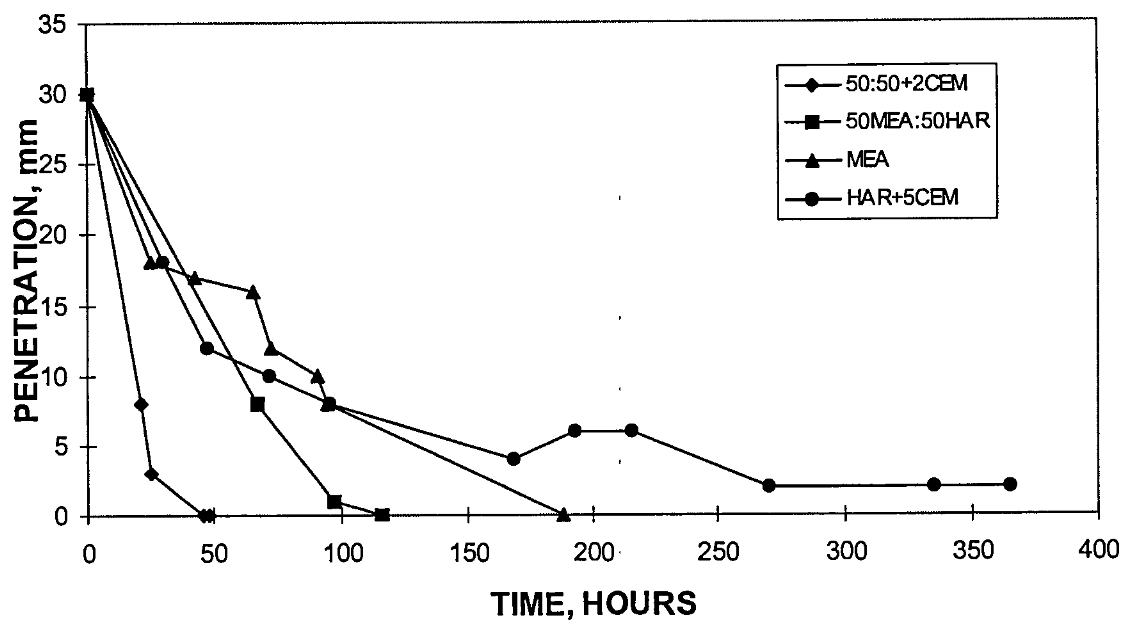



**Figure 5-16**
**Time of Set 50MEAFA:50MEABA+Harrison FA**

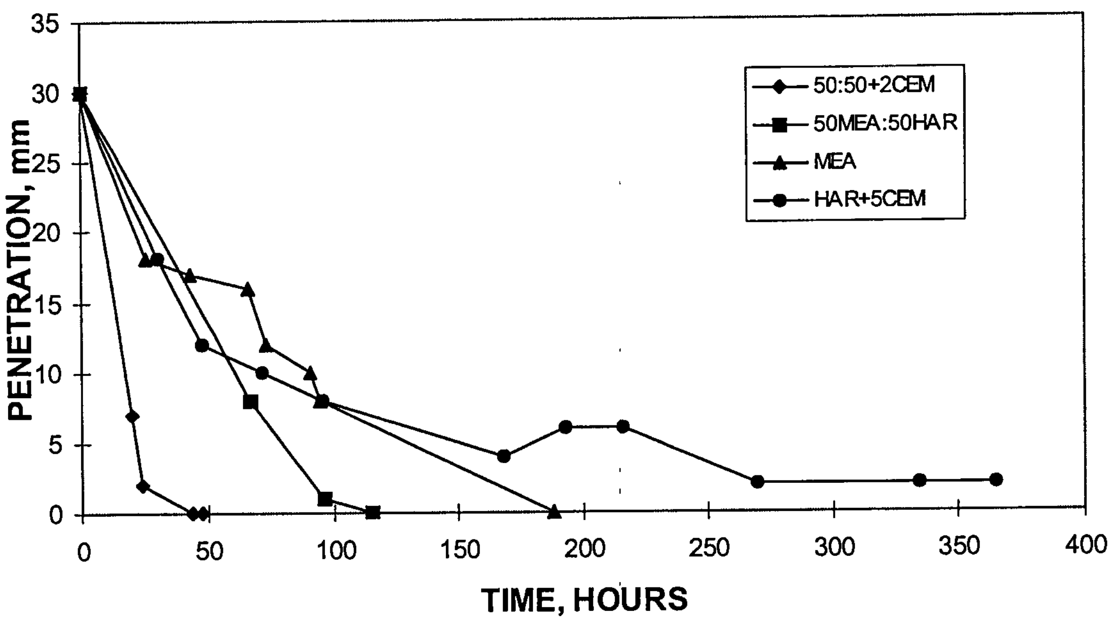



**Figure 5-17**
**Time of Set 75MEAFA:25MEABA+Harrison FA**

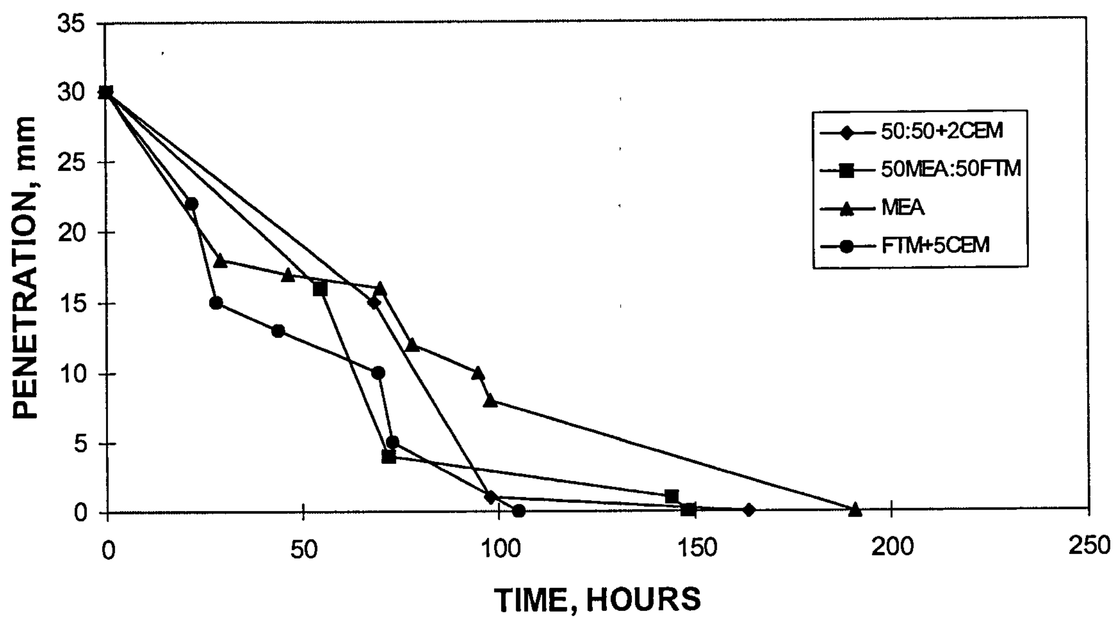



**Figure 5-18**
**Time of Set 25MEAFA:75MEABA+Ft. Martin FA**

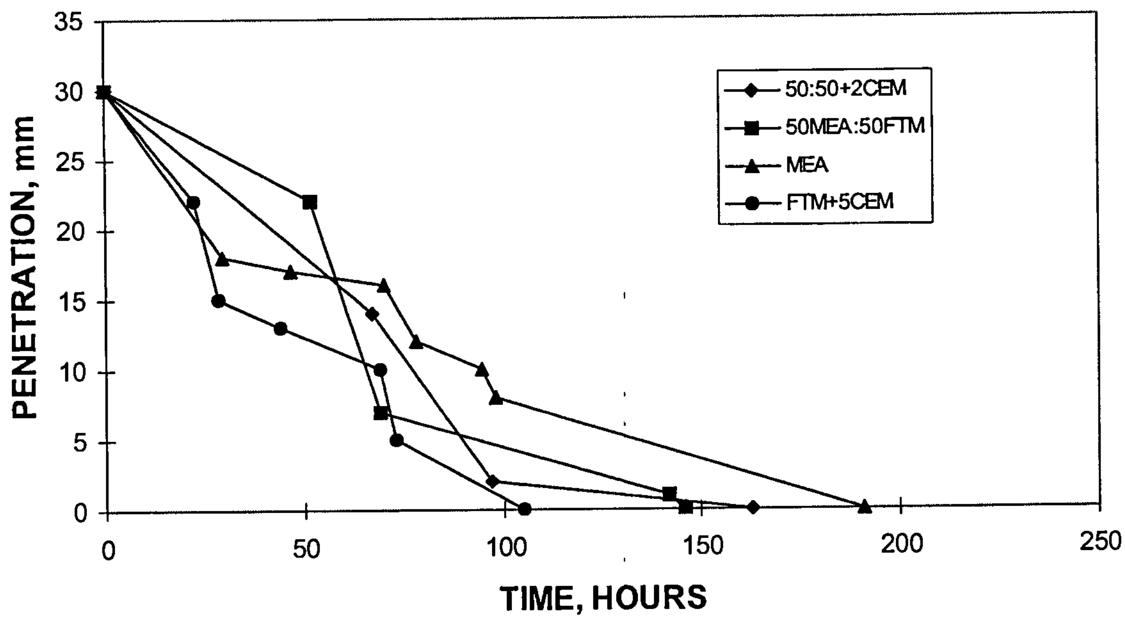



**Figure 5-19**
**Time of Set 50MEAFA:50MEABA+Ft. Martin FA**

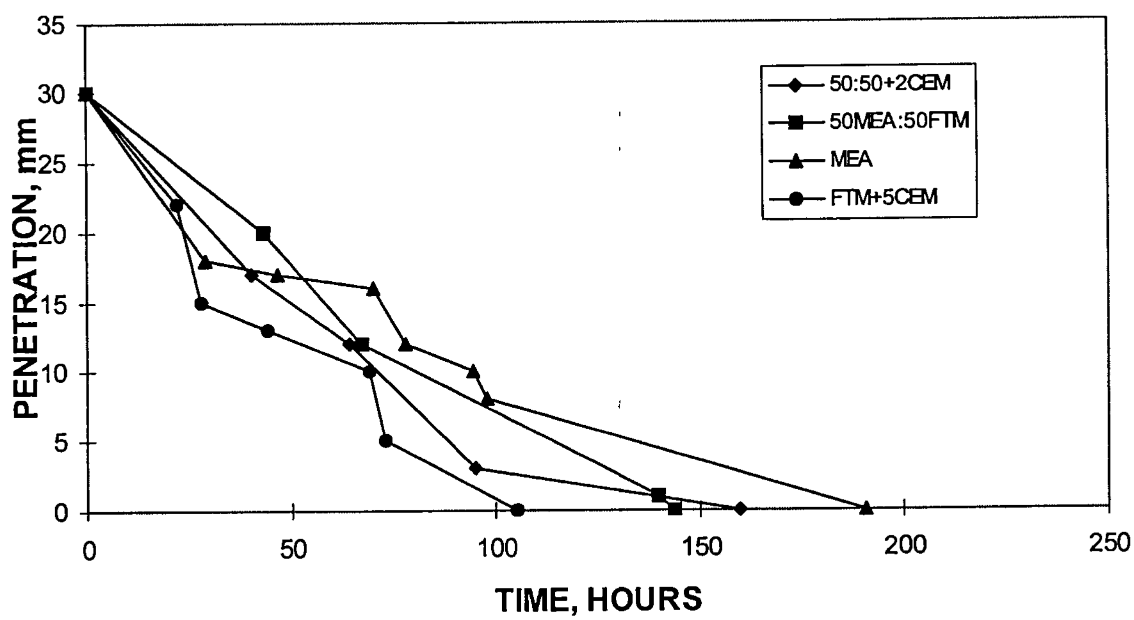



**Figure 5-20**
**Time of Set 75MEAFA:25MEABA+Ft. Martin FA**

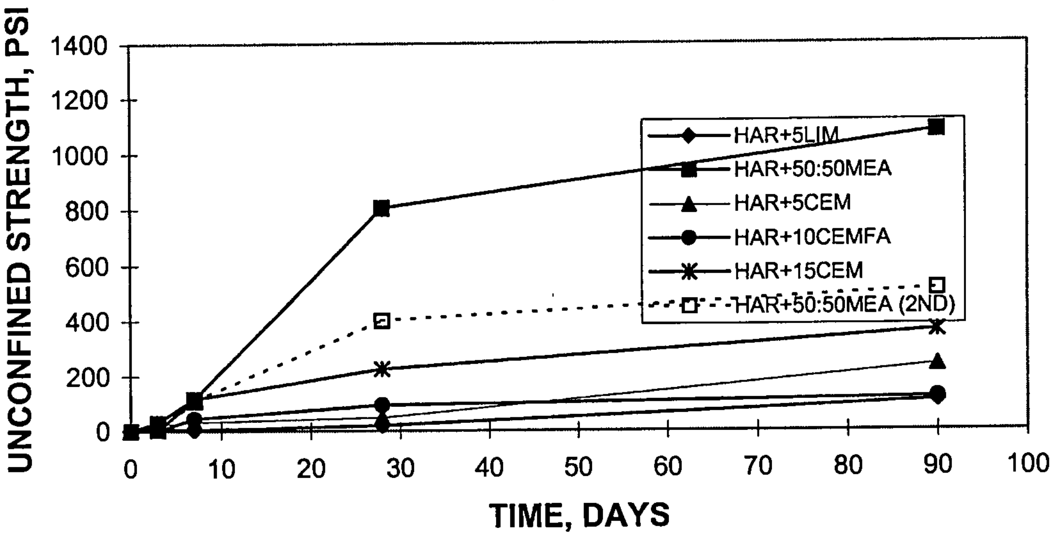



**Figure 5-21**
**Strength Development w/Time Harrison FA**
**Note: 2nd – Second Batch Mixed with Resampled FBC for Verification Testing**
**To convert strength in pounds per square inch to kilopascals multiply by 6.895**

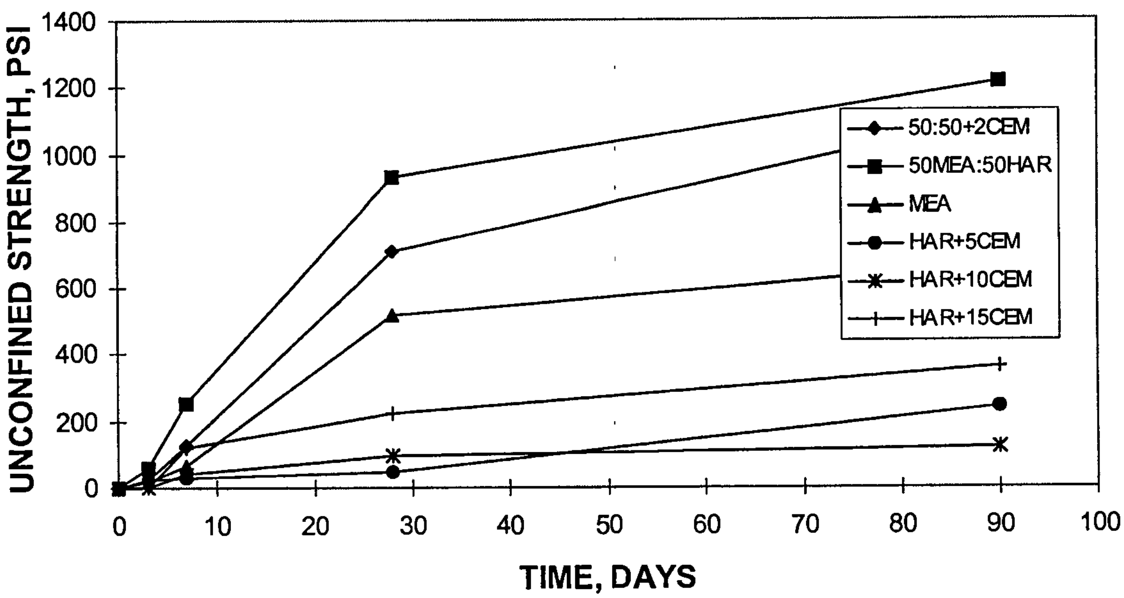



**Figure 5-22**
**Strength Development w/Time 25MEAFA:75MEABA+Harrison FA**
**To convert strength in pounds per square inch to kilopascals multiply by 6.895**

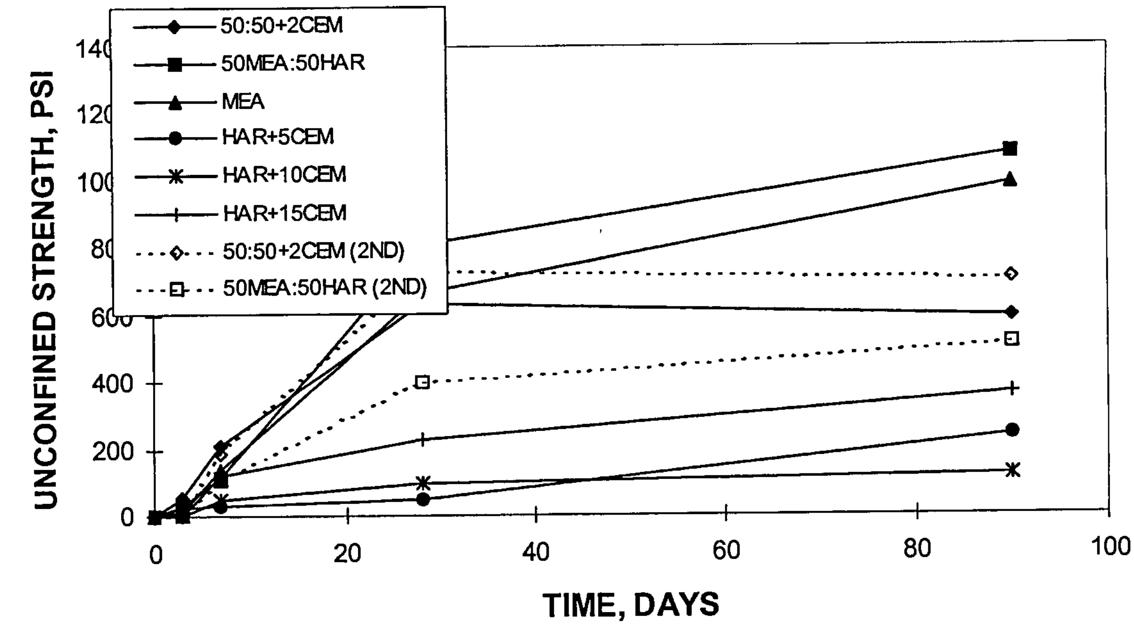



**Figure 5-23**
**Strength Development w/Time 50MEAFA:50MEABA+Harrison FA**
**Note: 2nd – Second Batch Mixed with Resampled FBC for Verification Testing**
**To convert strength in pounds per square inch to kilopascals multiply by 6.895**

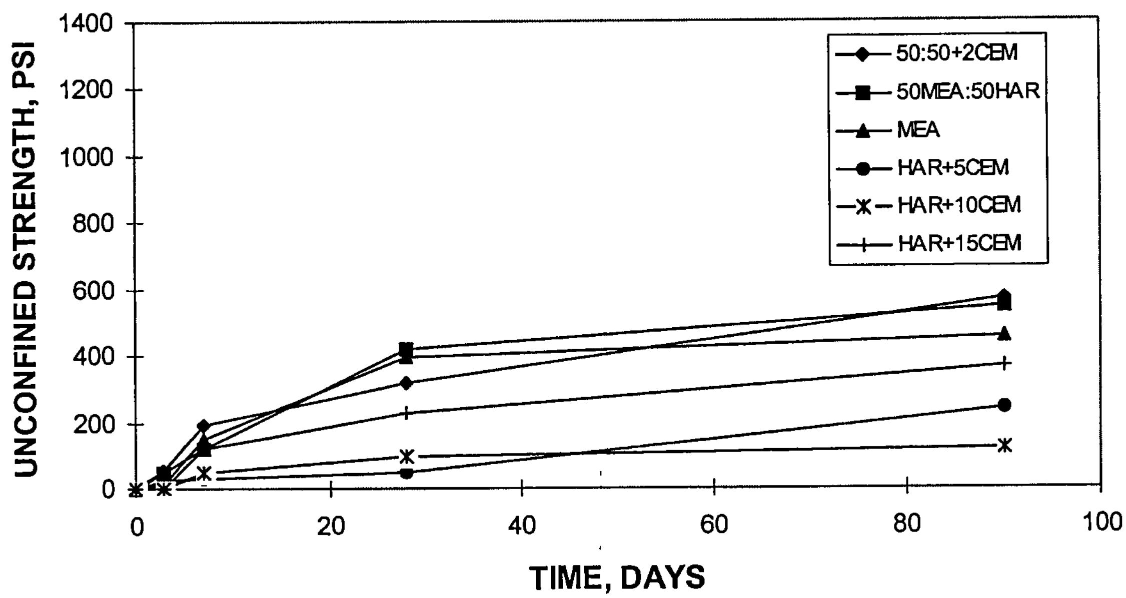



**Figure 5-24**
**Strength Development w/Time 75MEAFA:25MEABA+Harrison FA**
**To convert strength in pounds per square inch to kilopascals multiply by 6.895**

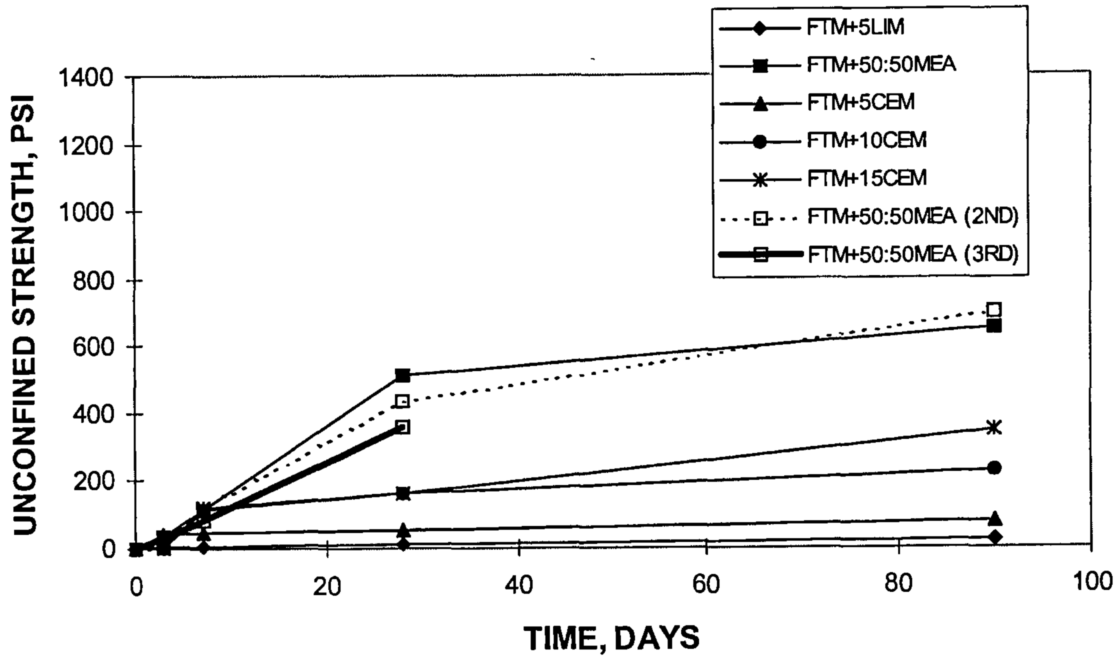



**Figure 5-25**
**Strength Development w/Time Ft. Martin, FA**
**Note: 2nd – Second Batch Mixed with Resampled FBC for Verification Testing**
**3rd – Third Batch Mixed with FBC that Mellowed for Three Days**
**To convert strength in pounds per square inch to kilopascals multiply by 6.895**

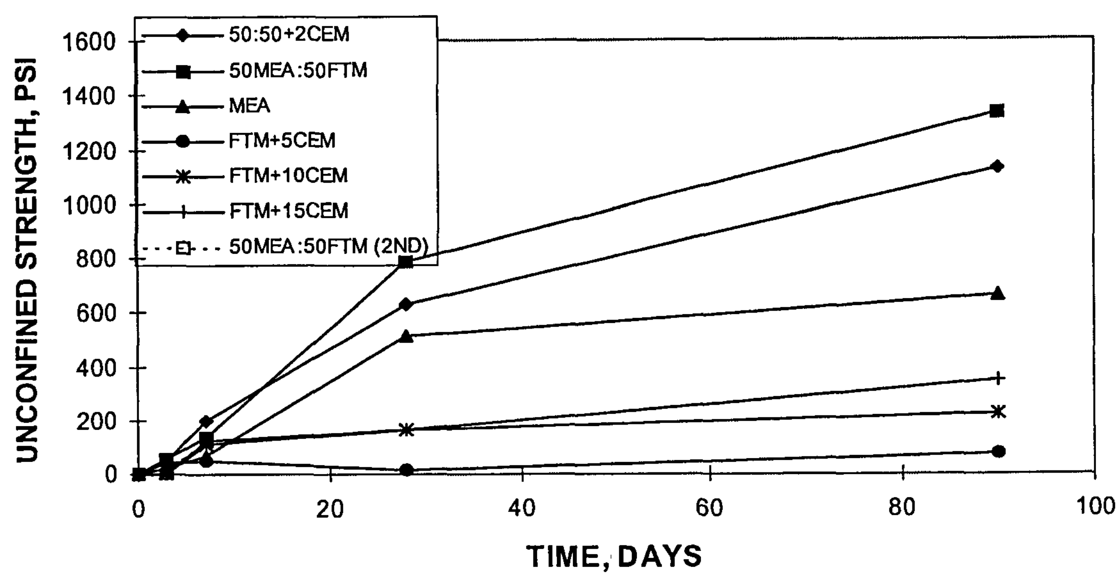



**Figure 5-26**
**Strength Development w/Time 25MEAFA:75MEABA+Ft. Martin FA**
**Note: 2nd – Second Batch Mixed with Resampled FBC for Verification Testing**
**To convert strength in pounds per square inch to kilopascals multiply by 6.895**

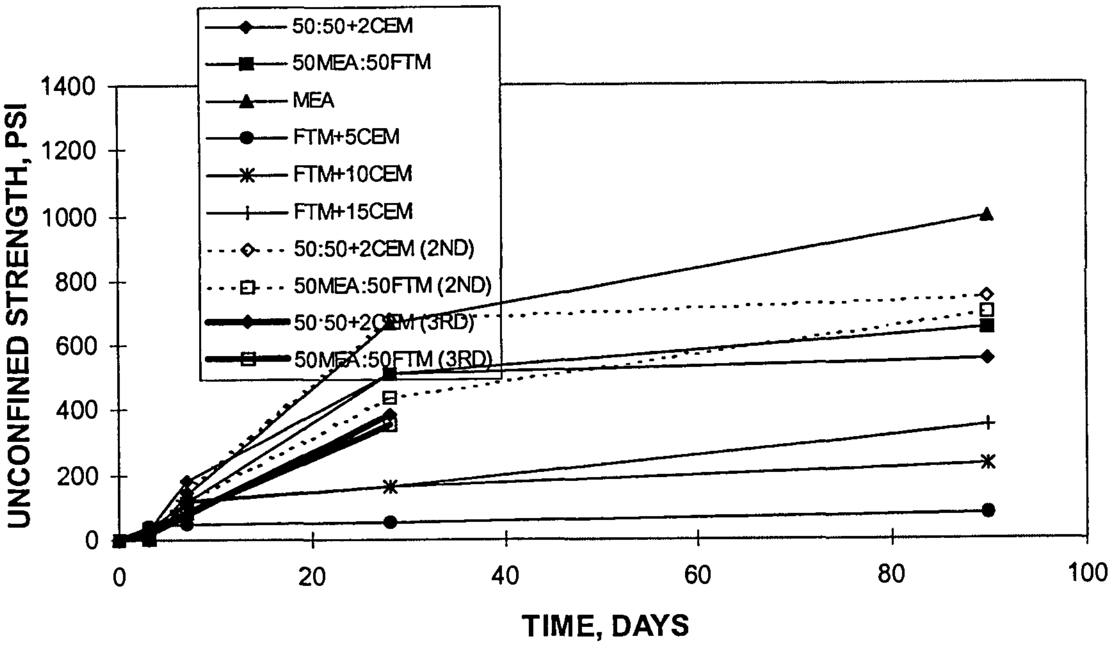



**Figure 5-27**
**Strength Development w/Time 50MEAFA:50MEABA+Ft. Martin, FA**
**Note: 2nd – Second Batch Mixed with Resampled FBC for Verification Testing**
**3rd – Third Batch Mixed with FBC that Mellowed for Three Days**
**To convert strength in pounds per square inch to kilopascals multiply by 6.895**

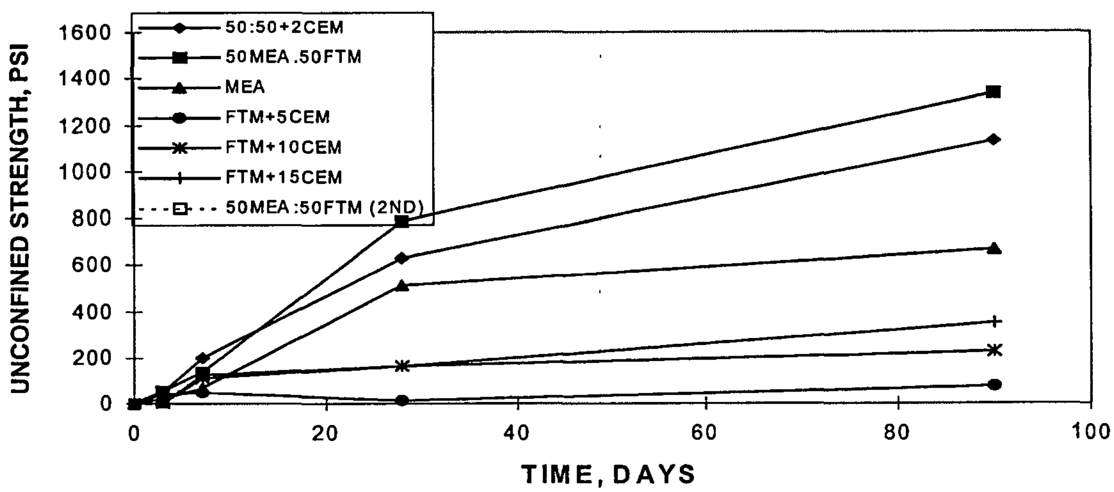



**Figure 5-28**
**Strength Development w/Time 75MEAFA:25MEABA+Ft. Martin, FA**
**Note: 2nd – Second Batch Mixed with Resampled FBC for Verification Testing**
**To convert strength in pounds per square inch to kilopascals multiply by 6.895**

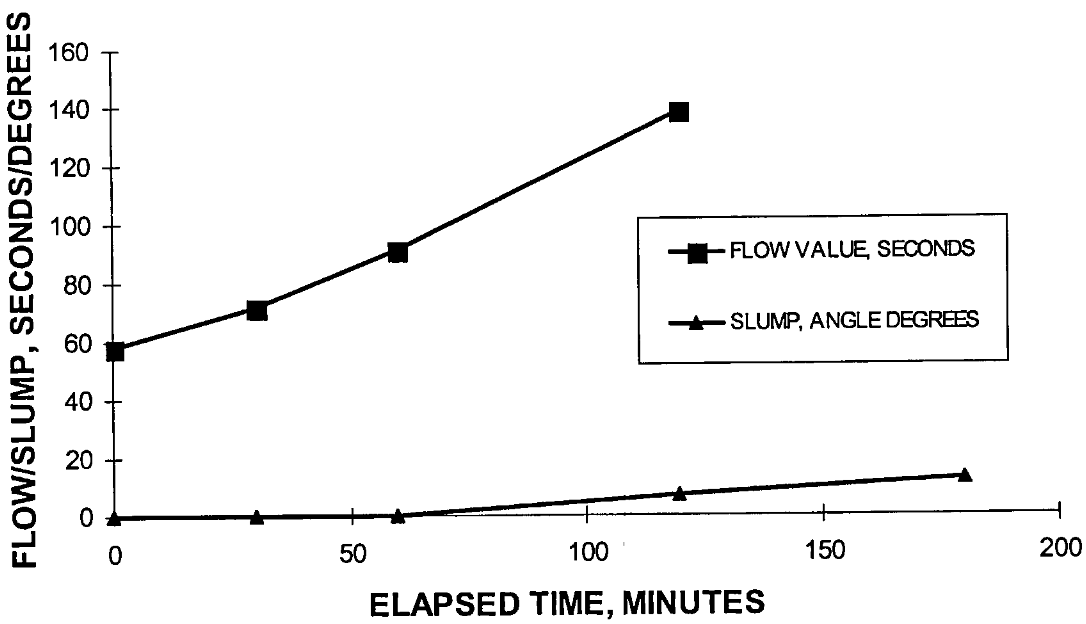



**Figure 5-29**
**Grout Fluidity with Time Selected Blend, Ft. Martin +MEA ASH 2% Cement**

## Slurry Specific Gravity

The slurry specific gravity ranged from 1.55 to 1.82 for the pulverized coal combustion fly ash grout slurries and 1.54 to 1.92 for the MEA FBC grout slurries. The grout slurries produced with the Harrison fly ash are slightly heavier than those produced with the Ft. Martin fly ashes.

One of the concerns of the investigation was the potential variability of the products. One potential cause is due to the MEA fly ash (FA) and bottom ash (BA) being randomly blended when off loaded from the storage silos (identified as plant blend); therefore, the test program investigated the impact variability may have on the performance of the grout. It was determined that the variability of the FBC FA/BA blends could be defined by measuring the slurry specific gravity produced with the material and comparing those values to values measured from laboratory controlled blend ratios. Figure 5-1 presents slurry specific gravity as a function of the percent of MEA fly ash in the blend and illustrates that the conditioned plant blends are generally at a 60 FA:40 BA ratio. To investigate the impact variability might have on the properties of the grout, the test program evaluated blends that represented the limiting bounds of 25FA:75BA and 75FA:25BA and a product, at a ratio of 50FA:50BA, that tended toward typical plant blend and the average of the limiting conditions. Laboratory test results indicate that variation in the FBC material FA:BA ratio has very little impact on grout specific gravity prepared with coal combustion fly ash and cement.

## Unit Weight

The solid dry unit weight for the various hardened grouts is presented in Tables 5-6, 5-7 and 5-8. The unit weights of the hardened grouts were determined by physical measurement. The dry unit weights range from approximately 10,212 $N/m^3$ (65 pounds per cubic foot [pcf]) to 15,710$N/m^3$ 100 [pcf] as the ratios of the various components were varied. Generally, as the grouts containing only the MEA FBC product aged, the dry unit weight of the product increased. This trend was not as evident for the grouts containing only Allegheny Energy fly ash and cement and/or lime stabilizers.

## Yield

The results of the measurements to determine yield show that on average, the yield for the grout slurries is on the order of 98 percent while some of the cement stabilized FBC slurries provided 100 percent yields. It is important to note that these yields were obtained for grout slurries having controlled moisture contents. The flow cone rate for the grouts was maintained at 60 seconds.

## Dimensional Stability

The dimensional stability characteristics of hardened grout specimens are provided in Tables 5-3, 5-4 and 5-5 and illustrated by Figures 5-2 through 5-12 inclusive. As illustrated by the relatively small changes in height indicated in Figures 5-2, 5-3 and 5-4, the grouts prepared with either Ft. Martin or Harrison fly ashes and cement are dimensionally stable. For the most part, after an initial expansion on the order of 2 percent, the grouts prepared from FBC by-products without other additives reached an equilibrium and remained at a constant volume (Figures 5-5 and 5-6). In most cases based strictly on physical performance, an expansion of 2 percent of a product for mine backfilling would be acceptable based on the variable mine conditions anticipated. However, because as initially discussed, detrimental mineral instability is manifest by volume change, it was taken as an indication that there was a potential for deleterious mineralogical forms to have been produced. The testing program was extended to determine if stable blends could be produced. Figures 5-7 through 5-12 illustrate the findings of this extended investigation. The figures demonstrate that when fly ash from the candidate sources was added to the FBC, variable results were obtained. The Ft. Martin fly ash reduced the tendency for the hardened grout to expand while adding the Harrison fly ash tended to increase the expansion. However, as illustrated by Figures 5-7 through 5-12, the addition of 2 percent cement by weight to the blends of FBC ash and the fly ashes provided a grout that exhibited dimensional changes of approximately ± 0.5 percent. Cement was selected over lime since its reaction is less sensitive to mine temperature and lime could combine with sulfur to form expansive and/or unstable minerals.

## Time of Set

Tables 5-3, 5-4 and 5-5 provide a summary of the data generated regarding final time of set. Figures 5-13 through 5-20 illustrate the time of set for various blends of materials. The initial time of set for the fly ash slurries with cement and/or lime stabilizer and the FBC product was on the order of several days. Blending the fly ashes with the FBC products had a tendency to shorten the time of set. The addition of the 2 percent cement to a 50:50 blend of either fly ash with FBC by-product produces a grout that will achieve an initial set within several hours and a final set within approximately two days. This testing indicates that within the first several hours the grout slurry would remain fluid (mobile) and, because the grout would be setting, the lateral loads it would impose on containment in outcrop areas would be diminishing. The final set at approximately 48 hours should be sufficient that a significant out flow from a "blow out" at a coal outcrop would not occur.

To verify the above conclusions regarding fluidity of the grout as a function of time, the fluidity of the grout was measured over a three-hour period. As summarized in Table 5-11 and illustrated by Figure 5-29, initially the grout was fluid approaching an exposure of 120 minutes. At the 120 minute mark, the fluid grout began behaving as a plastic material similar to hydraulic concrete. At this point, upon initial egress from the flow cone, an angle of repose for the material was on the order of 15 degrees. However, after 5 minutes from the time of egress, the material slumped so that the angle of repose had diminished to 7 degrees, indicating that the material was still fluid. After 180 minutes from the time of mixing, the angle of repose was 24 degrees upon initial egress from the flow cone and after 10 minutes, diminished to 12.7 degrees, again indicating that the grout still remained highly plastic at this point.

The implication of this testing is that, should pumping of mixed grout be postponed, the material will still remain sufficiently fluid for periods up to 2 to 3 hours to disperse within the mine. This indicates that the "flash" set of other FBC materials is not inherent to the MEA FBC product as blended and tested in this laboratory program.

## Unconfined Compressive Strength

Tables 5-6, 5-7 and 5-8 provide a summary of the unconfined compression test results for blends of fly ash and FBC products. The relationships of unconfined compressive strength as a function of age are graphically presented in Figures 5-21 through 5-28, inclusive. As indicated in the tables and figures, all the blends exhibited the tendency to gain strength with age. Also illustrated is the tendency of the materials to exhibit synergism when blended. That is, the blends of FBC product and the fly ashes tend to have higher strengths than the products separately combined with stabilizing agent. As a point of interest, it appears that over the period of time of the test program, the blends of FBC and fly ash without the addition of cement produce grout that is stronger than, but not as dimensionally stable as, with the addition of cement.

Unconfined compression testing of specimens cured at temperatures representative of mine conditions indicates that, for all the blends, some retardation of strength development occurs in comparison to curing at room temperature.

Data summarized in Tables 5-6 and 5-7 were developed from samples prepared with both laboratory blended and conditioned FBC materials and from samples of plant blended and condition FBC materials. The test results presented in Tables 5-6 and 5-7, include mellowed

FBC ash, that is, FBC ash that has been moistened and allowed to sit for three days prior to mixing with Ft. Martin fly ash and 2 percent cement. The three-day time period was selected to reflect anticipated field conditions of on-site storage. The plant blend samples summarized in Table 5-6 were prepared beyond the three-day mellowing period to evaluate the effects of prolonged on-site storage. These test results demonstrate that the FBC material can be stored for use in the grout blend. To minimize the potential of difficulties the contractor may have handling aggregated FBC, it was suggested that a three-day use window be established.

## Permeability

The data summarized in Table 5-9 indicate that grout composed of FBC material alone has a permeability in the range of 5 to 7 x $10^{-5}$ cm/sec. Laboratory permeability test results are also presented in Figures 30 through 35. Grout mixed with Ft. Martin fly ash, FBC ash, and two percent cement has a somewhat lower permeability of approximately 9 x $10^{-6}$ cm/sec. Permeability testing did not include samples mixed with Harrison ash because it had been withdrawn as a candidate material. This permeability indicates that water would flow through hardened grout at a rate of approximately 2.7m (9 ft) per year. Curing at mine temperatures appeared to have no effect on permeability.

## TCLP Leaching

The data summarized in Table 5-10 indicate that grouts composed of FBC material alone and blended with Ft. Martin and Harrison fly ash yield comparable leaching results. Prior testing experience indicates that coal combustion fly ash grouts typically yield very low concentrations of metals, therefore, these materials were not tested in this program. The low concentrations of metals leached from the grouts, in conjunction with the low permeability values described above, indicate that leaching is not likely to affect groundwater.

## Summary

The test program indicated that by blending the candidate materials an acceptable grout mix could be provided. The MEA ash had the potential to provide strength to the grout while the coal combustion fly ash enhanced the fluidity of the grout. The addition of 2 percent cement provided dimensional stability to the hardened grout product. Figure 5-36 shows a photograph of an expanded FBC ash and the recommended stabilized grout mix. It was recommended that a grout blend of 49 percent MEA ash + 49 percent Ft. Martin coal combustion fly ash + 2 percent cement with enough water added to produce a grout having a flow value of 60 seconds be used. The proposed grout was estimated to require approximately 380 liters (100 gallons) of water per cubic yard of grout.

# PERMEABILITY TEST REPORT

**TEST DATA:**

Specimen Height (cm): 7.75
Specimen Diameter (cm): 7.70
Dry Unit Weight (pcf): 54.1
Moisture Before Test (%): 43.0
Moisture After Test (%): 66.4

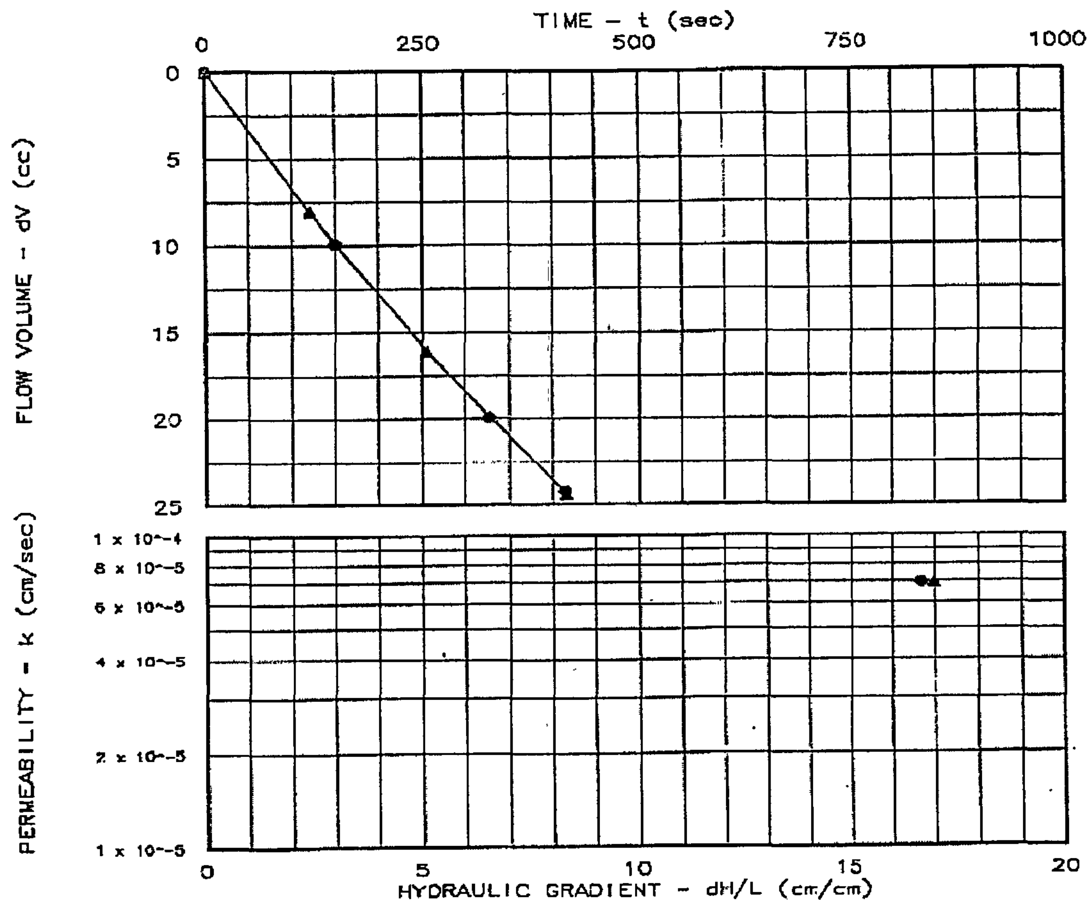
| Run Number: | 1 ● | 2 ▲ |
|---|---|---|
| Cell Pressure (psi): | 65.0 | 65.0 |
| Test Pressure(psi): | 61.0 | 61.0 |
| Back Pressure(psi): | 59.2 | 59.1 |
| Diff. Head (psi): | 1.8 | 1.9 |
| Flow Rate (cc/sec): | 5.42 x 10^-2 | 5.51 x 10^-2 |
| Perm. (cm/sec): | 5.94 x 10^-5 | 5.94 x 10^-5 |

**SAMPLE DATA:**

Sample Identification: MEA-MORGANTOWN CONDITIONED ASH 96 DAY CURE
Visual Description: SPECIMEN CURED @ 50F

Remarks: FINAL B-VALUE 95

Maximum Dry Density (pcf): NA
Optimum Moisture Content (%): NA
ASTM(NA)
Percent Compaction:
Permeameter type: FLEX
Sample type: REMOLD




Project: WVDEP
Location: OMEGA MINES
Date: 30OCT96

Project No.: 93-198-53
File No.: 160
Lab No.:
Tested by: DJN
Checked by: DDK
Test: CH - Constant head

PERMEABILITY TEST REPORT

GAI Consultants, Inc.

FIGURE 30

**Figure 5-30**
**Permeability Test Report, 100% MEA Conditioned Ash, Cured at 50°F**

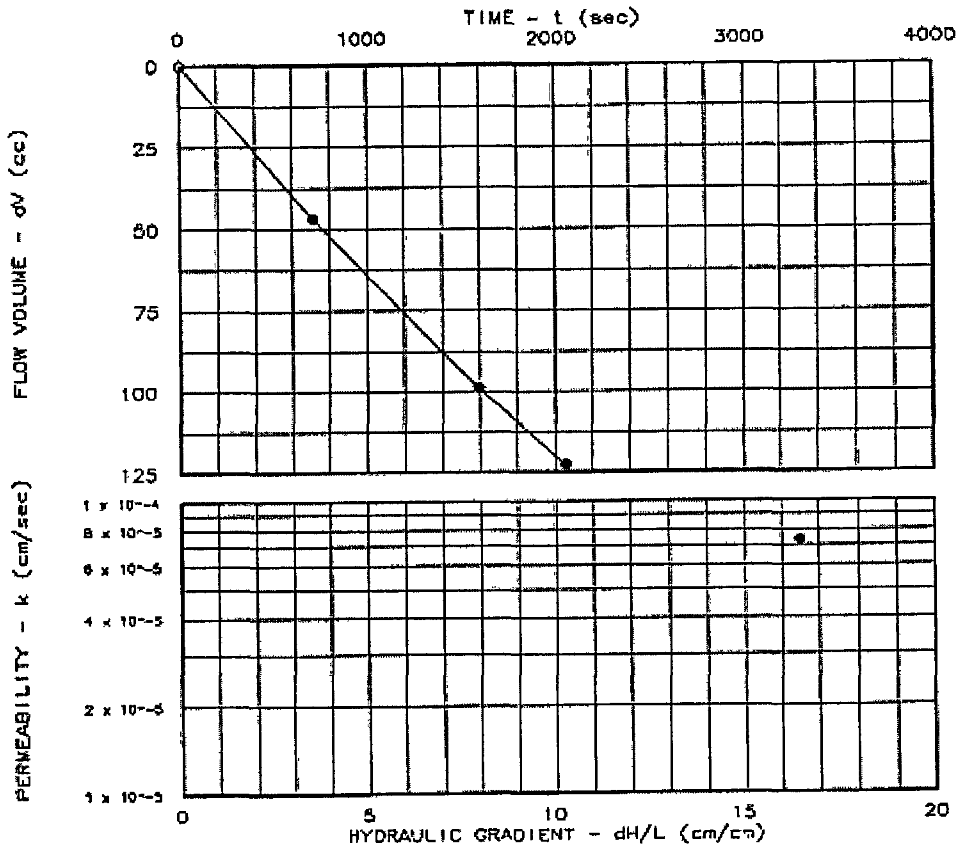
# PERMEABILITY TEST REPORT

TEST DATA:
Specimen Height (cm): 7.75
Specimen Diameter (cm): 7.70
Dry Unit Weight (pcf): 54.1
Moisture Before Test (%): 43.0
Moisture After Test (%): 66.4
Run Number: 1 ● 2 ▲
Cell Pressure (psi): 65.0
Test Pressure(psi): 61.0
Back Pressure(psi): 59.2
Diff. Head (psi): 1.8
Flow Rate (cc/sec): 5.64 x 10^-2
Perm. (cm/sec): 7.31 x 10^-5

SAMPLE DATA:
Sample Identification: MEA-MORGANTOWN
CONDITIONED ASH 96 DAY CURE
Visual Description: SPECIMEN CURED @ 50F

Remarks: FINAL B-VALUE 95

Maximum Dry Density (pcf): NA
Optimum Moisture Content (%): NA
ASTM(NA)
Percent Compaction:
Permeameter type: FLEX
Sample type: REMOLD




Project: WVDEP
Location: OMEGA MINES
Date: 30OCT96

PERMEABILITY TEST REPORT
GAI Consultants, Inc.

Project No.: 93-198-53
File No.: 161
Lab No.:
Tested by: DJN
Checked by: DDK 12Nov96
Test: CH - Constant head

FIGURE 31

**Figure 5-31**
**Permeability Test Report, 100% MEA Conditioned Ash, Cured at 50°F**

# PERMEABILITY TEST REPORT

**TEST DATA:**

Specimen Height (cm): 7.92
Specimen Diameter (cm): 7.66
Dry Unit Weight (pcf): 56.9
Moisture Before Test (%): 66.1
Moisture After Test (%): 61.9

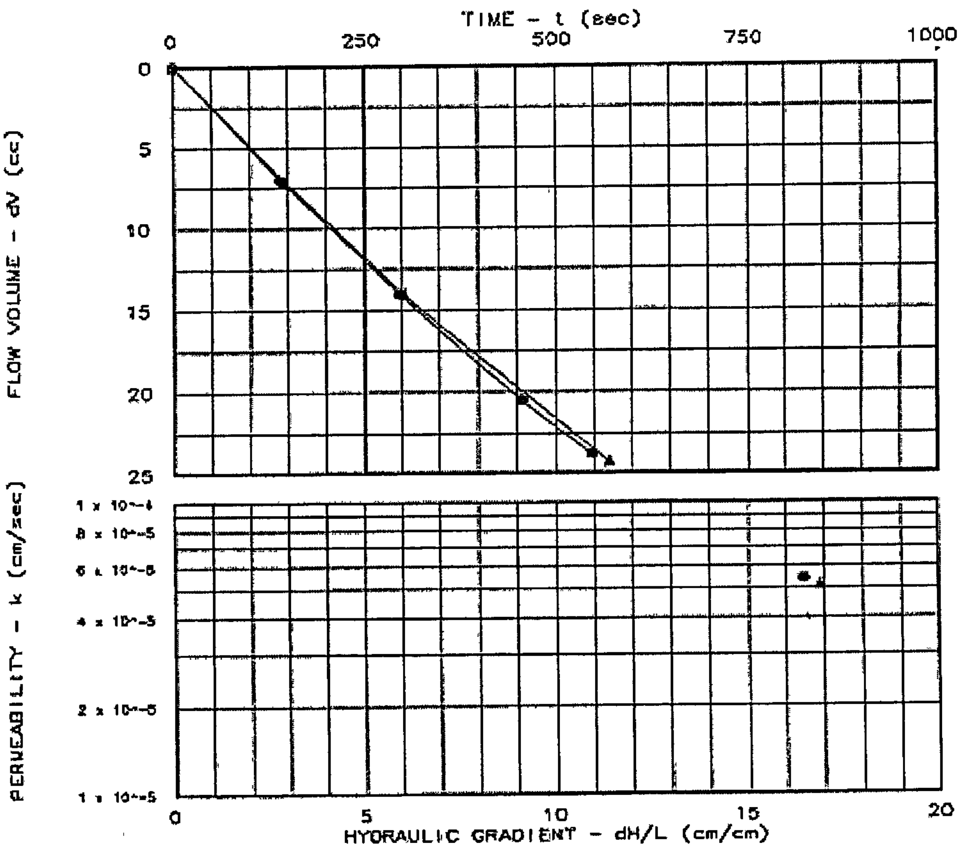
| Run Number: | 1 ● | 2 ▲ |
|---|---|---|
| Cell Pressure (psi): | 65.0 | 65.0 |
| Test Pressure(psi): | 61.0 | 61.0 |
| Back Pressure(psi): | 59.1 | 59.1 |
| Diff. Head (psi): | 1.9 | 1.9 |
| Flow Rate (cc/sec): | $4.13 \times 10^{-2}$ | $4.02 \times 10^{-2}$ |
| Perm. (cm/sec): | $5.40 \times 10^{-5}$ | $5.13 \times 10^{-5}$ |

**SAMPLE DATA:**

Sample Identification: MEA-MORGANTOWN CONDITIONED ASH 96 DAY CURE
Visual Description: SPECIMEN CURED @ 72F

Remarks: FINAL B-VALUE 99

Maximum Dry Density (pcf): NA
Optimum Moisture Content (%): NA
ASTM(NA)
Percent Compaction:
Permeameter type: FLEX
Sample type: REMOLD




Project: WVDEP
Location: OMEGA MINES
Date: 30OCT96

PERMEABILITY TEST REPORT
GAI Consultants, Inc.

Project No.: 93-198-53
File No.: 162
Lab No.:
Tested by: DJH
Checked by: DDE 12Nov96
Test: CH - Constant head

FIGURE 32

**Figure 5-32**
**Permeability Test Report, 100% MEA Conditioned Ash, Cured at 72°F**

# PERMEABILITY TEST REPORT

**TEST DATA:**

Specimen Height (cm): 7.62
Specimen Diameter (cm): 7.75
Dry Unit Weight (pcf): 73.9
Moisture Before Test (%): 21.9
Moisture After Test (%): 38.3

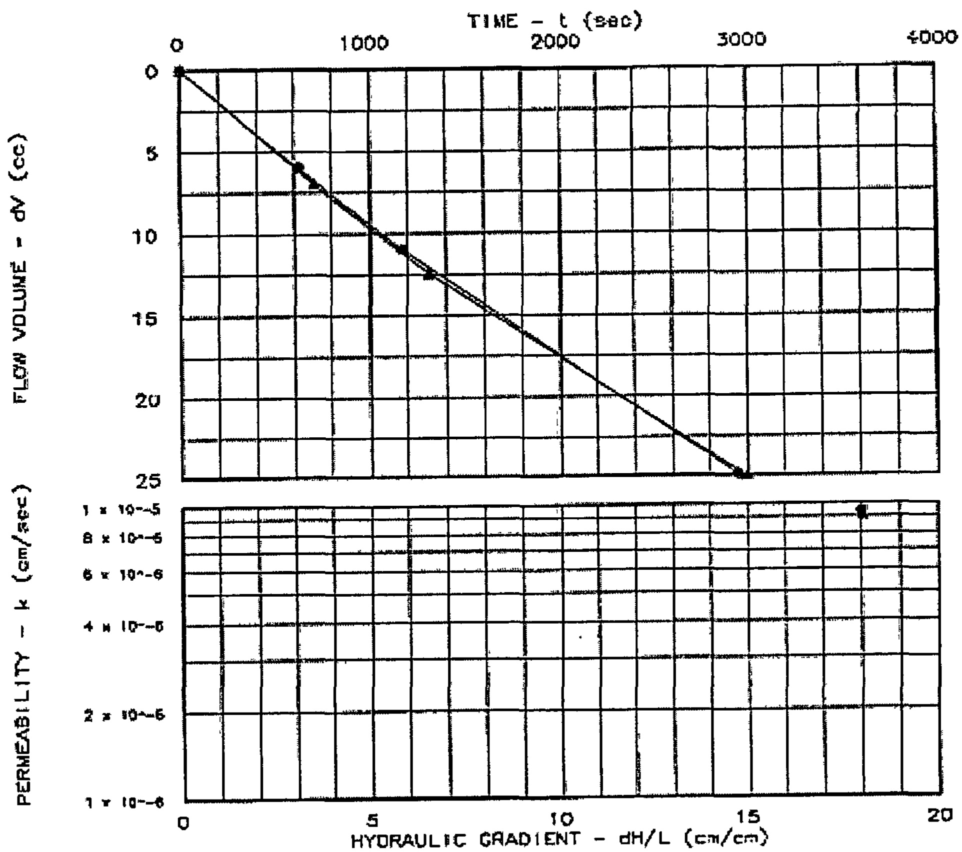
| Run Number: | 1 ● | 2 ▲ |
|---|---|---|
| Cell Pressure (psi): | 65.0 | 65.0 |
| Test Pressure(psi): | 61.0 | 61.0 |
| Back Pressure(psi): | 59.0 | 59.0 |
| Diff. Head (psi): | 2.0 | 2.0 |
| Flow Rate (cc/sec): | 8.06 x 10^-3 | 7.78 x 10^-3 |
| Perm. (cm/sec): | 9.45 x 10^-6 | 9.10 x 10^-6 |

**SAMPLE DATA:**

Sample Identification: FT.MARTIN+MEA 50:50
+2% CEMENT 92 DAY CURE
Visual Description: SPECIMEN CURED @ 50F

Remarks: FINAL B-VALUE 65

Maximum Dry Density (pcf): NA
Optimum Moisture Content (%): NA
ASTM(NA)
Percent Compaction:
Permeometer type: FLEX
Sample type: REMOLD




| | |
|---|---|
| Project: WVDEP<br>Location: OMEGA MINES<br>Date: 30OCT96 | Project No.: 93-198-53<br>File No.: 164<br>Lab No.:<br>Tested by: DJN |
| PERMEABILITY TEST REPORT<br>GAI Consultants, Inc. | Checked by: [illegible]<br>Test: CH - Constant head |

FIGURE 33

**Figure 5-33**
**Permeability Test Report, Suggested Blend, Ft. Martin+ MEA Ash +2% Cement, Cured at 50°F**

# PERMEABILITY TEST REPORT

**TEST DATA:**

Specimen Height (cm): 7.62
Specimen Diameter (cm): 7.75
Dry Unit Weight (pcf): 73.9
Moisture Before Test (%): 21.9
Moisture After Test (%): 38.3

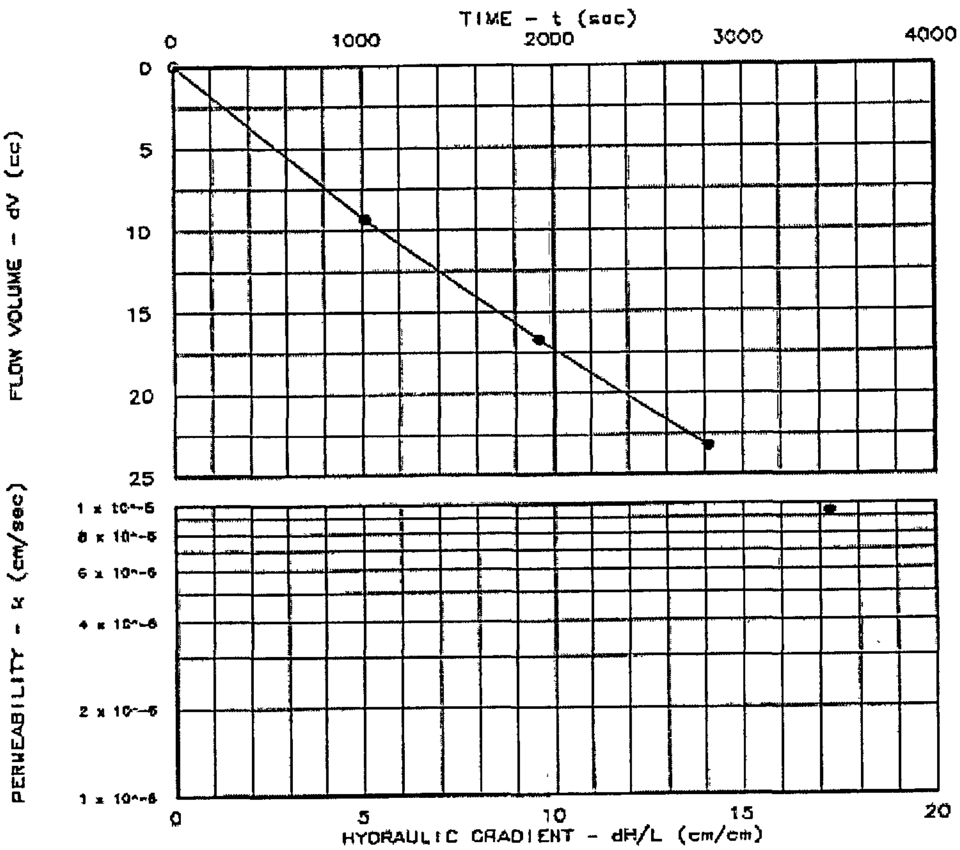
| Run Number: | 1 ● | 2 ▲ |
|---|---|---|
| Cell Pressure (psi): | 65.0 | |
| Test Pressure(psi): | 61.0 | |
| Back Pressure(psi): | 59.1 | |
| Diff. Head (psi): | 1.9 | |
| Flow Rate (cc/sec): | $7.72 \times 10^{-3}$ | |
| Perm. (cm/sec): | $9.44 \times 10^{-6}$ | |

**SAMPLE DATA:**

Sample Identification: FT.MARTIN+MEA 50:50 +2% CEMENT 92 DAY CURE
Visual Description: SPECIMEN CURED @ 50F

Remarks: FINAL B-VALUE 88

Maximum Dry Density (pcf): NA
Optimum Moisture Content (%): NA
ASTM(NA)
Percent Compaction:
Permeameter type: FLEX
Sample type: REMOLD




Project: WVDEP
Location: OMEGA MINES
Date: 30OCT96

PERMEABILITY TEST REPORT
GAI Consultants, Inc.

Project No.: 93-198-53
File No.: 165
Lab No.:
Tested by: DJN
Checked by: DD

Test: CH - Constant head

FIGURE 34

**Figure 5-34**
**Permeability Test Report, Suggested Blend, 49% Ft. Martin + 49% MEA Ash +2% Cement, Cured at 50°F**

# PERMEABILITY TEST REPORT

**TEST DATA:**

Specimen Height (cm): 7.92
Specimen Diameter (cm): 7.71
Dry Unit Weight (pcf): 76.5
Moisture Before Test (%): 35.7
Moisture After Test (%): 35.5

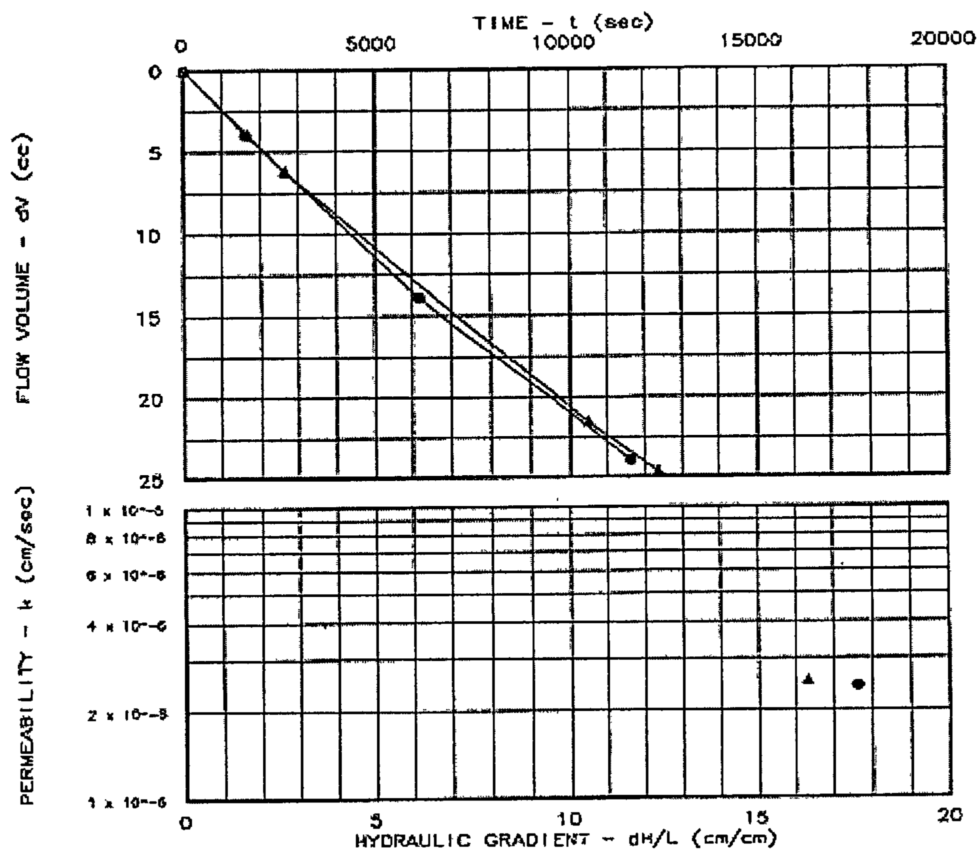
| | Run 1 | Run 2 |
|---|---|---|
| Run Number: | 1 ◆ | 2 ▲ |
| Cell Pressure (psi): | 65.0 | 65.0 |
| Test Pressure(psi): | 61.0 | 61.0 |
| Back Pressure(psi): | 59.0 | 59.2 |
| Diff. Head (psi): | 2.0 | 1.8 |
| Flow Rate (cc/sec): | 1.99 x 10^-3 | 1.91 x 10^-3 |
| Perm. (cm/sec): | 2.40 x 10^-6 | 2.30 x 10^-6 |

**SAMPLE DATA:**

Sample Identification: FT.MARTIN+MEA 50:50 +2% CEMENT 92 DAY CURE
Visual Description: SPECIMEN CURED @ 72F

Remarks: FINAL B-VALUE 100

Maximum Dry Density (pcf): NA
Optimum Moisture Content (%): NA
ASTM(NA)
Percent Compaction:
Permeameter type: FLEX
Sample type: REMOLD




| | |
|---|---|
| Project: WVDEP | Project No.: 93-198-53 |
| Location: OMEGA MINES | File No.: 163 |
| Date: 30OCT96 | Lab No.: |
| PERMEABILITY TEST REPORT | Tested by: DJN |
| GAI Consultants, Inc. | Checked by: [illegible] |
| | Test: CH - Constant head |

FIGURE 35

**Figure 5-35**
**Permeability Test Report, Suggested Blend, Ft. Martin + MEA Ash +2% Cement, Cured at 72°F**


**Figure 5-36**
**Expanded FBC Ash and Recommended Grout Mix**

The suggested grout mix demonstrated the following characteristics in the laboratory:

- Flow without separating into solid and liquid portions;
- A set time of approximately 2 days (so that it will remain in place following injection);
- Dimensional stability throughout the test period;
- Low permeability;
- Little potential to leach metals to groundwater; and
- Relatively insensitive to variations in mix components.

These characteristics meet the primary project goals for the grout to fill mine voids in the North Lobe of the Omega Mine to reduce AMD generation and have sufficient strength to prevent mine subsidence. In addition, the grout has some alkaline leaching potential and utilizes a mixture of fly ash and FBC material. Furthermore, the grout will be easy to mix in the field with typical construction equipment and be easy to monitor by the WVDEP.

# 6

# INJECTION PROGRAM

The goals of the injection program were:

1. To reduce the contact of water and air with acid forming material, with a secondary requirement that the grout have some alkaline leaching potential to help treat AMD that may come in contact with the hardened grout;

2. To achieve a grout mix of sufficient strength to prevent mine subsidence in the North Lobe; and

3. To use a mixture of materials supplied by Allegheny Energy and Anker Energy.

- The principal criteria used by GAI to meet these goals and thus develop a suitable grout mix were:

- Flowability -- set at 60 seconds as determined by flow cone measurements. For coal combustion product (CCP) grouts this 60 second value typically produces a self-leveling grout that is readily pumpable and will flow in the mine without separating into solid and liquid portions.

- Set time -- within approximately 2 days, indicating that the grout will take an initial set and remain in place. This criteria was included so that grout in filled portions of the mine would be immobile and not subsequently flow to open portions of the mine, causing loss of roof contact.

- Dimensional stability -- indicating that, after placement and set involving approximately ± 1 percent shrink/swell, the grout mix would have no tendencies to shrink or swell in the long term. This criterion was adopted because some FBC ashes have shown swelling characteristics that have resulted in long-term loss of strength and increase in permeability.

Strength was not a primary criterion but was used to indicate the ability of the grout to form a stable, durable mass within the mine to prevent future mine subsidence. Laboratory testing indicated the long term strength of the recommended blend (49 percent fly ash, 49 percent FBC ash, and 2 percent cement) would be in excess of 3,450 kPa (500 psi).

An additional consideration in the design of the grout mixture was that the likely grout injection contractors would have different types of mixing equipment. Thus, the mix design had to be readily achievable with various types of mixing and placement equipment, easily monitored by the WVDEP, and relatively insensitive to minor variations in mixture quality control.

The recommended grout mixture of equal parts of fly ash and FBC ash takes advantage of the available free lime in the FBC ash for developing the strength of the grout while the fly ash increases the flowability/pumpability of the mixture. The addition of 2 percent cement was for dimensional stability of the grout regardless of minor variations in the ashes, or the inevitable variations in mix preparation during injection operations.

The proposed grout was estimated to require approximately 100 gallons of water per cubic yard of grout. This water usage was established so that the grout was capable of achieving the desired flow cone values without creating "bleed water" that would have to be removed from the mine.

## Project Specifications

In developing the specifications, previous experience with FBC grouts and the initial laboratory results indicated that one mix could be used. This led to an elimination of barrier grouts which would have had different properties from the self-leveling grout to be used for bulk fill. Self-leveling grouts can flow long distances without segregation and the injection plan took advantage of this with widely spaced injection holes to reduce costs. High injection pressures were specified to aid in moving the grout throughout the mine.

The injection plan, described by the specifications, differed substantially from past mine stabilization projects in that: 1) the grout was to flow long distances without segregation and 2) total filling was necessary to reduce acid mine drainage (AMD). Typical mine stabilization specifications require the contractor to provide a grout mix with a minimum unconfined compressive strength in a certain number of days. This requirement allows the contractor to select the ash supply source while assuring a minimum strength is achieved.

For the Omega Project, the overall concern of the work was flowability and grout stability while achieving some nominal strength. GAI considered not providing a strength criteria in the specifications since one of the principal criteria would be to meet the flow cone requirement. Furthermore, the contractor would be provided with the required major components of the grout (i.e., the fly ash and FBC) as well as the proportions in which they were required to be mixed. Therefore, the Contractor had no control over the mix except for the water added to achieve a 60 second flow.

Allegheny Energy indicated they would deliver ash by triaxle trucks at a maximum rate of 544 to 816 metric tons (600 to 900 tons) per day. Anker Energy planned to use bottom dump trucks for delivery of FBC ash at a maximum rate of 453 – 544 metric tons (500 to 600 tons) per day.

Anker Energy and Allegheny Energy were notified that the specifications would require the contractor be capable of injecting grout at a minimum rate of 76 cubic meters (cm) (100 cy) per hour. For planning purposes, a weekly injection rate of 3,060 cm (4,000 cy) was anticipated based on an assumed 8-hour workday (5 days per week) averaging 76 cm (100 cy) per hour. At this rate of injection, Allegheny Energy and Anker Energy would be responsible for delivering approximately 1,814 metric tons (2,000 tons) per week or 363 metric tons (400 tons) per day to maintain steady production. However, they were advised that the contractor might be allowed to work longer hours or be more productive than the minimum rate and thus might require higher supply rates.

Vertical injection holes were proposed (Figure 6-1). The mine plan was used to select optimum locations for the injection holes. There was no set hole spacing. However, it was assumed the injected grout would flow over 30m (100 ft) in more open mine entries (i.e. - generally areas of first mining) and 15m (50 ft) in entries where broken material (i.e. - generally areas of second mining) could reduce the ability of grout to flow.

Injection of materials was to commence with the barrier injection holes at the south end of the project site. This would block the inflow of water from updip portions of the mine. The injection program would then proceed downdip, beginning with the second-mined areas. The next area to be grouted was the first-mined area to the east of U.S. Route 119. The final area to be grouted would be the area west of U.S. Route 119 in the vicinity of the horizontal drains. The sequence of injection was intended to allow water to drain from the mine during filling. The WVDEP had the flexibility of adjusting the grouting sequence depending on the flow of grout. If water (mine pool) was encountered, the specifications directed the contractor to convey it (via pumping to other locations in the mine or to the existing collection system) to the flows being treated by the WVDEP.

A barrier was not proposed for the downdip portions of the mine workings. This decision was based on the lower probability of completely filling the mine voids with a stiffer mix, along with the increased cost of the additional injection holes that would be required. Seeps (existing or new) adjacent to the site were to be monitored on a regular basis for indications of grout leaking from the mine. The cleanup of any blowout was specified to be an incidental cost to the contractor. The grout plan was developed to minimize the effect of blowouts by requiring the monitoring of the coal outcrop and giving the WVDEP the ability to alter the grout sequence if blowouts occurred.

Costs for the CCP materials delivered to the site were determined in advance by the WVDEP. The specifications then required the contractor to reimburse Allegheny Energy and Anker Energy directly for supplying the materials at these rates.

The cost of the FBC ash delivered to the site was $1.50 per ton. Fly ash obtained from silos and delivered to the site was $1.50 per ton. If material needs required loading from the plant stockpile the cost increased to $2.78 per ton. The increase was the result of Allegheny Energy already having paid for placement of the fly ash in the stockpile. Thus, Allegheny Energy did not have an avoided cost. The contractor was not permitted to apply a surcharge on these material rates when billing WVDEP.

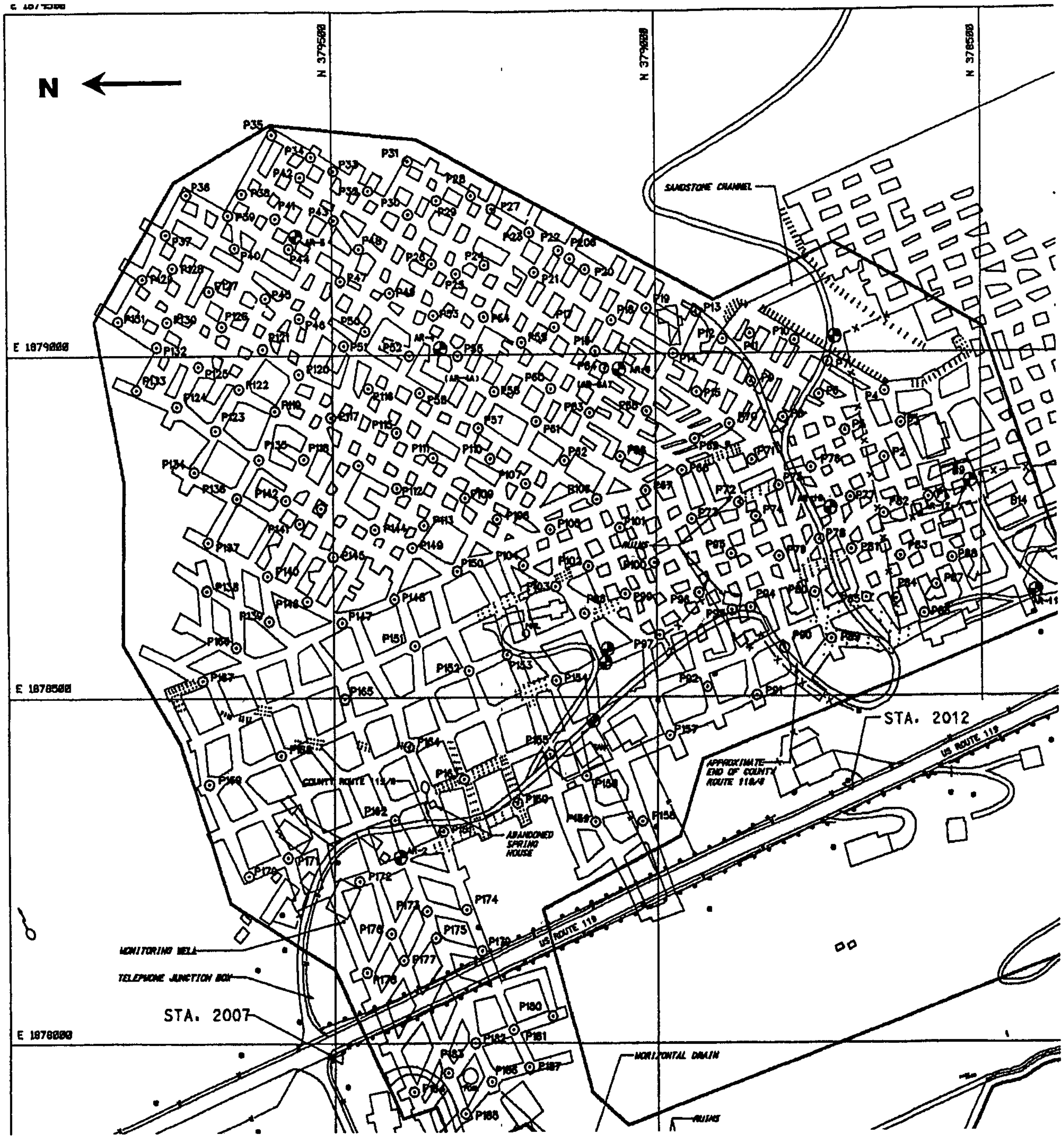



**FIGURE 6-1**

### Plan Location of Injection Holes

Injection hole drilling was permitted using either standard rotary type or percussion type equipment. Dust control devices and gauges to indicate downhole drilling pressures were required. The specified injection hole diameter was 152mm (6 inches).

The Contractors stockpiling area was to be sufficient to store a volume of CCP materials for 3 days of work. The FBC ash storage area was to be subdivided into 3 piles, each containing the materials delivered on a single day. FBC ash older than 3 calendar days from the date shipped could not be utilized for grout as the available free lime would start to set-up.

Water used in grouting was required to be obtained from existing lines of the Clinton Water Association (CWA). The rate of withdrawal from the CWA system was limited to 190 gallons per minute. To provide this rate of water, the project specifications included constructing an upgraded booster station for the CWA.

The primary grout mix was permitted to vary between a 55:45 fly ash to FBC ash to a 55:45 FBC Ash to fly ash with 2 percent cement by dry weight measurement.

The specified grout mix for injection was a ratio of 1:1 fly ash to FBC ash with 2 percent cement by dry weight. Approximately 378 liters (100 gallons) of water per cubic yard was projected to achieve the desired flowability. A flow cone value of 60 seconds ± 10 seconds was specified (ASTM C939).

If additional cement were requested, provisions were made to pay for it on a per ton used basis. If additional water were added to the mix, the cost was to be considered incidental.

If, during injection operations, CCP material changes or shortages occurred, such as an unplanned shutdown of a supply source, the contractor was to use a back-up grout mix with the approval of the WVDEP. It was intended that the use of such back-up mixes be limited to 1 or 2 days at a time. Two back-up mixes were to be developed by the contractor (one with fly ash/cement and one with FBC ash/cement) which would meet the flowability requirements. The back-up mixes were to have minimum 3-day strengths of 206 kPa (30 psi) [cured at 22° C (72° F) and 100% humidity], and minimum 28-day strengths of 1,379 kPa (200 psi) [cured at 22° C (72°F) and 100% humidity]. In addition, the FBC ash/cement back-up mix was to have a dimensional stability of less than ±1% shrink/swell measured for not less than 90 days as determined by free swell testing.

Back-up mixes were to utilize ash from Allegheny Energy's Fort Martin power station and FBC ash from Anker Energy.

The mixing and pumping equipment was to provide for continuous measurement of the following parameters:

1. grout pressure at the injection holes;
2. pumping rate of grout;
3. cumulative volume of grout pumped; and
4. cumulative volume of water used for mixing.

The contractor was required to use an on-site batch mixing facility. Grout particles were limited to no larger than a No. 16 U.S. Standard Screen. This was done to prevent cementitious material from clumping and not contributing to the strength and uniformity of the grout.

The grout pump(s) were to be capable of developing sufficient pressure to overcome all head and friction losses, and still attain 3,450 kPa (500 psi) at the top of the injection hole when pumping at the maximum rate.

The tremie pipe was to be metal or plastic and stiff enough to maintain the pipe tip below the level of CCP grout during placement, and strong enount , and strong enough to withstand pressures. The tremie pipe was to have a minimum diameter of 76mm (3 inches).

The contractor was to perform periodic soundings, in surrounding holes during CCP grout injection operations to determine the levels of injected materials in the holes.

The CCP grout was to be injected continuously into a hole until the injection pressure reached 3,450 kPa (500 psi) at the top of the injection hole.

Nearby seeps and portals were to be monitored by the Contractor during injection activities for evidence of blow-outs.

The contractor was to flush an injection hole and grout equipment with water whenever grouting activities at an injection hole were temporarily suspended.

The specifications required the contractor to have borehole video equipment available to view conditions at mine level prior to injection and to evaluate the effectiveness of filling the mine voids.

## Field Work

The grouting contractor started their fieldwork on January 18, 1998. Initial tasks included surveying to layout the injection holes and construction of a road for project access. The drilling rig and air compressor were mobilized on January 26, 1998. Drilling started on January 27, 1998 with the barrier holes at the south end of the North Lobe. The contractor, with WVDEP approval, decided to drill all the injection holes prior to injection of grout into the North Lobe. Drilling of the planned barrier and primary injection holes was completed on May 1, 1998. The 19 barrier holes (B1-B17 and B1b and B14b) and 188 primary injection holes (P1 – P187 and P20b) totaled 8,430m (27, 656 ft) (Figure 6-1). Each hole was videotaped by the contractor to record conditions at mine level to assist them and the WVDEP inspector in performing the injection work.

Survey control, with respect to the mine map, was good in that only 15 of the initial 207 injection holes encountered coal pillars. Thirty holes encountered mine gob/broken rock while 162 holes encountered an open void. Only one of the holes in the first mined area encountered gob/broken rock. All other holes in the first mined area encountered a mine void. In the second mined area a large percentage of the holes also encountered voids.

The contractor set-up the grout plant on the hilltop in the central portion of the site in the vicinity of injection holes P-66, P-67 and P-68. The maximum length of piping required to reach the most distant injection hole was slightly less than 300m (1,000 ft).

FBC ash was delivered to the site in bottom dump trailers to control dust since the delivery vehicles traveled through the city of Morgantown, West Virginia. Fly ash delivery vehicles did not travel through Morgantown. These vehicles were tri-axel trucks covered with tarpaulins. The fly ash was dumped directly into the stockpile area. The bottom dump trailers delivered FBC ash into a hopper underlain by a conveyor which moved the ash to the FBC stockpile. A Caterpillar IT 28B front end loader with a digital weight bucket moved the materials into bins from which another conveyor moved the materials into a mixing tank where water and cement were added. The cement was stored in a silo above the mixing tank. A screw type mixer moved the grout to the hopper of a pump truck which delivered the grout into the distribution pipe. Figure 6-2 is a plan of the grout plant. Schedule 40 PVC pipe with a diameter of 102mm (4in) was used to distribute the grout to the injection holes.

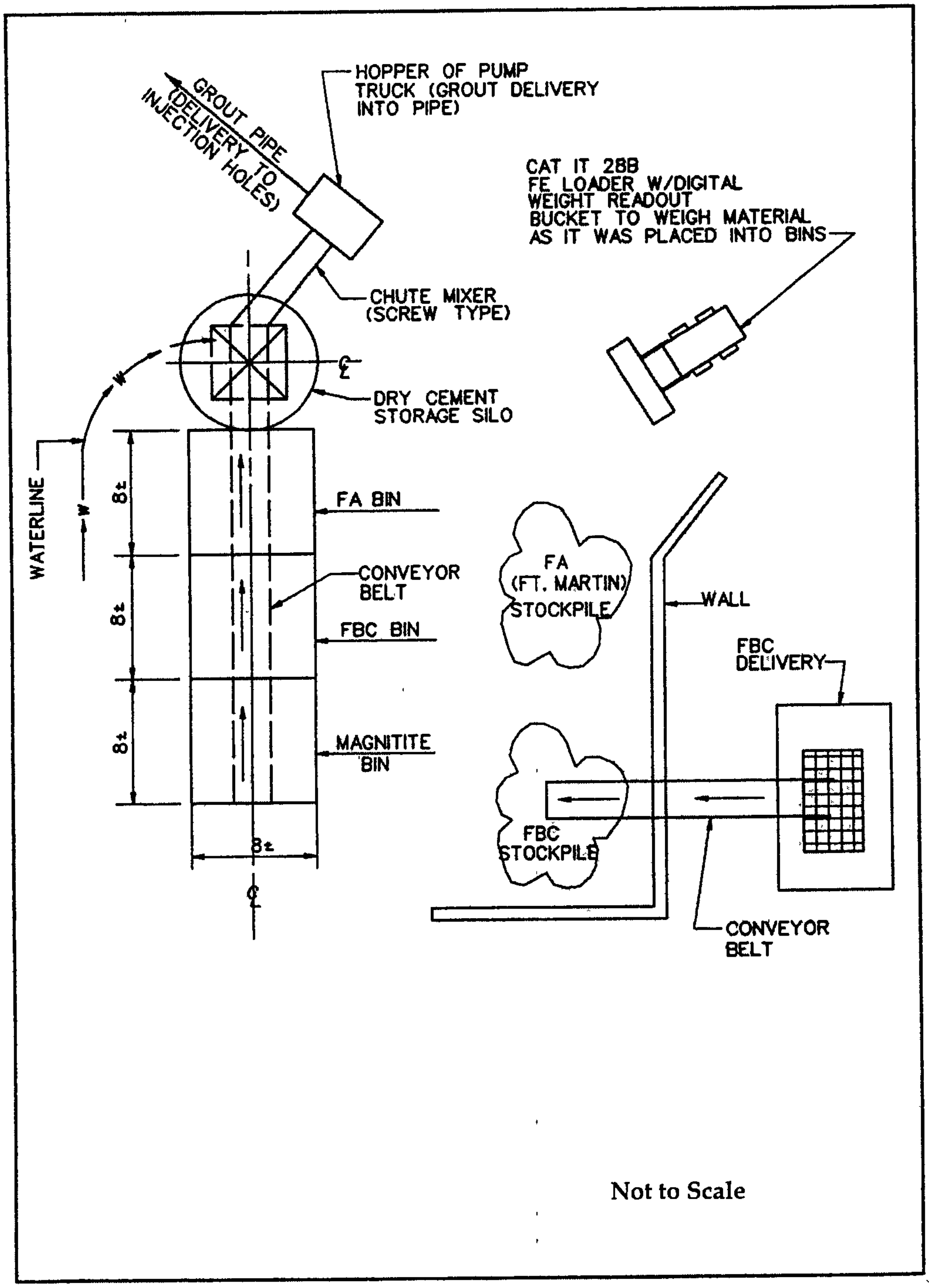



**Figure 6-2**

**Plan of the Grout Plant**

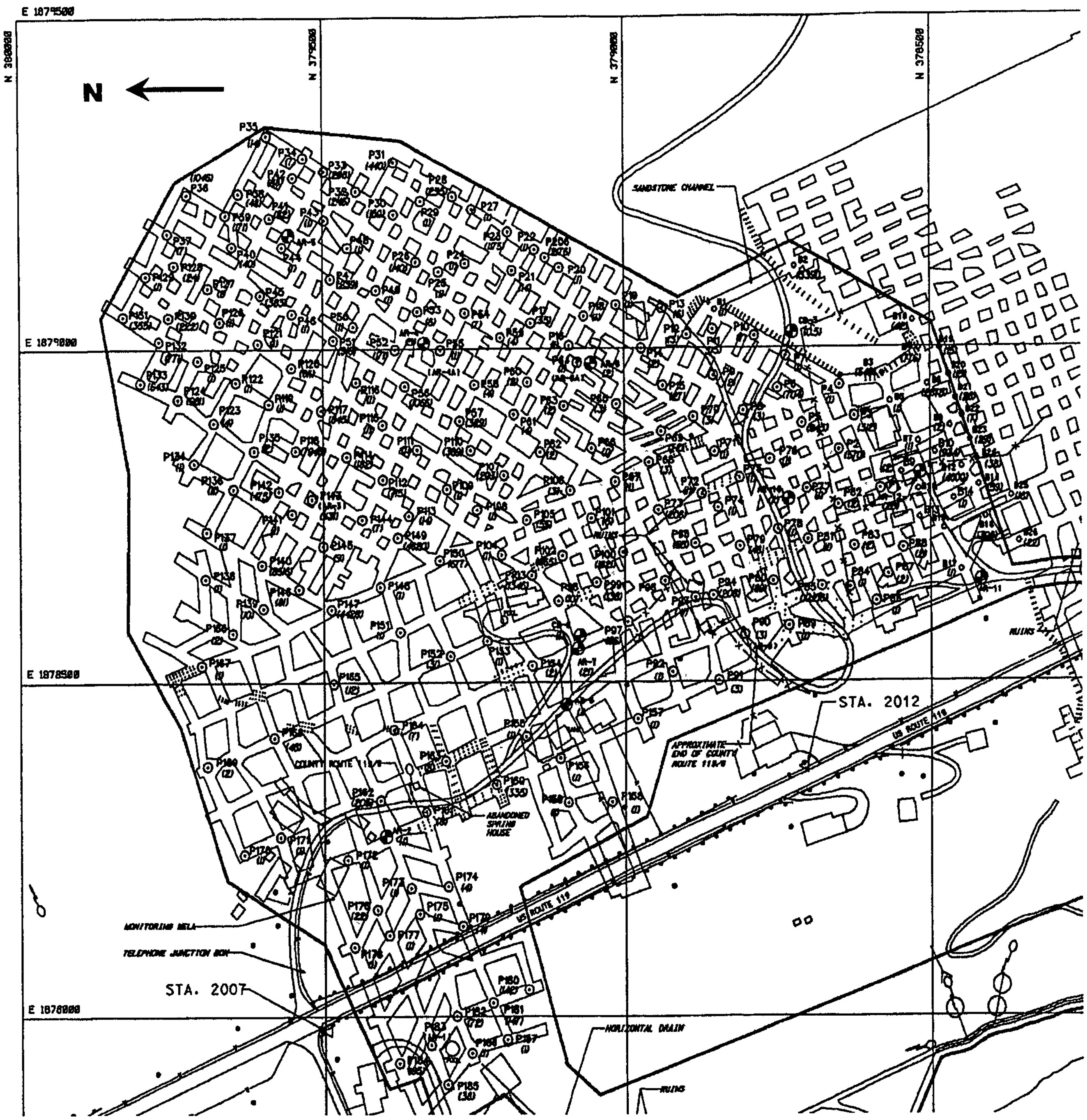
Grouting started on May 11, 1998 in the barrier holes at the south side of the North Lobe and continued until November 4, 1998. Figure 6-3 shows the injection holes and the quantity of grout injected (take) in each hole. Grout takes varied from 0.76cm (1cy) in some holes to 7,812 cm (10,218 cy) in P-85. Although this hole is approximately 396m (1,300 ft) from the Marshall House drains, it is believed that some grout reached and blocked the drains. The drains at the Marshall House were blocked twice during the injection program. One of these blockages occurred on August 3, 1998 while injecting into P-118. This hole had a take of 6074 cm (7,944 cy) and is located 457m (1,500 ft) from the Marshall House horizontal drain. A number of injection holes in the second mined area had very large takes. The holes with the largest takes are indicated in Table 6-2. It is believed that some of the grout injected in these holes flowed downdip into the first mined area.

**Table 6-1**

**Injection Holes with Large Takes**

| Hole | Take (cy)* |
|---|---|
| P-85 | 10,218 |
| P-140 | 8,516 |
| P-118 | 7,944 |
| P-149 | 4,620 |
| P-147 | 4,428 |
| P-150 | 1,686 |
| P-143 | 1,131 |

*To convert cubic yards to cubic meters multiply by 0.76455




## Figure 6-3

## Injection Hole Locations and Grout Injected into Each Hole

Takes in many of the holes in the first mined area are small in comparison to their mined void space indicating grout had flowed into these voids from other injection locations

In August WVDEP decided to add nine barrier holes (B18-B26) at the south side of the North Lobe to provide assurance that the barrier was preventing water flow from updip portions of the mine from entering the North Lobe. These holes were located south of the initial barrier holes.

In October when FBC ash was not available, WVDEP decided to pressure grout selected holes using a mix consisting of 1, 036 Kg (2,285 pounds) of Ft. Martin fly ash, 136 Kg (300 pounds) of cement and 265 liters (70 gallons) of water. Packers were set at the top of rock. Pressure grouting was conducted in 46 holes between October 5 and October 29, 1998. The takes using pressure grouting varied from 0.76 to 256cm (1 to 335 cy) as shown in Table 6-3.

**Table 6-2**

**Pressure Grouting Takes**

| Hole | Pressure Grout Take (cy)* | Hole | Pressure Grout Take (cy)* |
|---|---|---|---|
| P97 | 86.2 | P179 | 4 |
| P92 | 1 | P174 | 4.3 |
| P157 | 1 | P184 | 95 |
| AR7 | 21 | AR1 | 56.8 |
| P154 | 2 | P181 | 146.6 |
| P153 | 1 | P182 | 72 |
| P160 | 335.2 | P185 | 38.3 |
| P161 | 5 | P186 | 2 |
| P156 | 1 | P187 | 1 |
| P159 | 1 | P180 | 142.1 |
| P158 | 1 | B18 | 42 |
| P162 | 204.5 | B19 | 15 |
| AR2 | 1 | B20 | 15 |
| P171 | 1 | B21 | 52.3 |
| P168 | 48 | B22 | 29.3 |
| P169 | 2 | B23 | 29 |
| P170 | 1 | B24 | 36 |
| P172 | 1 | B27 | 15 |

| Hole | Pressure Grout Take (cy)* | Hole | Pressure Grout Take (cy)* |
|---|---|---|---|
| P178 | 1 | B25 | 16 |
| P177 | 1 | B26 | 20 |
| P176 | 21.7 | E11 | 4 |
| P173 | 1 | E11a | 4 |
| P175 | 1 | E4 | 121 |

On October 29, 1998 the Contractor returned to using the contract grout mix.
*To convert cubic yards to cubic meters multiply by 0.76455.

In October, drilling equipment was remobilized for exploratory holes to evaluate the extent of mine filling and the barrier at the south side of the North Lobe. Holes E1-E6 and E8 – E17 were drilled in late October.

Holes E1 through E6 and E8 through E10 were drilled to evaluate the mine filling. E7 was not drilled since it was located in a septic field. Holes E11 through E17 were drilled south of the barrier holes. The locations of these holes are shown in Figure 6-4. Table 6-4 presents the findings in Holes E1-E6 and E8-E18.

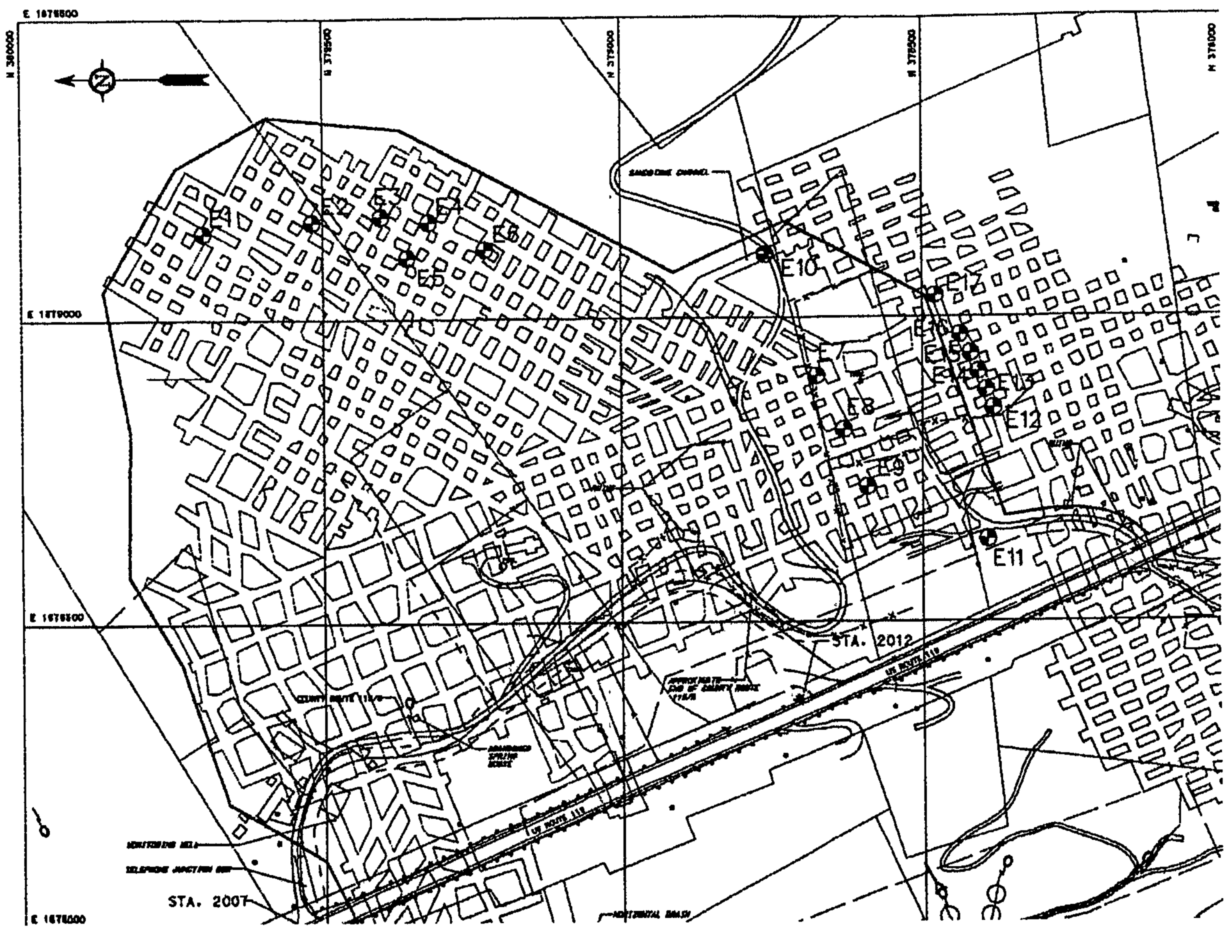



**Figure 6-4**

**Plan Location of E-Holes**

**Table 6-3**

**Conditions Found at Mine Level**

| Hole | Conditions at Mine Level |
|---|---|
| E1 | Grout, Shale and Coal – No air loss |
| E2 | Grout, Shale and Coal -- No Air Loss |
| E3 | Grout, Shale and Coal -- No Air Loss |
| E4 | 50% Air loss at depth of 105 – 106 ft. – Air returned at 106 ft. |
| E5 | Grout, Shale and Coal -- No air loss |

| Hole | Conditions at Mine Level |
|---|---|
| E6 | Grout, Shale and Coal – No air loss |
| E8 | Grout, Shale and Coal -- No air loss |
| E9 | Grout, Coal and Shale – No air loss |
| E10 | Coal, Grout and Shale – No air loss |
| E-11 | Coal |
| E-11a | Coal |
| E-12 | Caved and Broken – 100% air loss |
| E-13 | Caved and Broken – 100% air loss |
| E-14 | Caved and Broken 100% air loss |
| E-15 | Caved and Broken – 100% air loss |
| E-16 | Caved and Broken – 100% air loss |
| E-17 | Caved and Broken – 75% air loss |
| E-18 | Grout, Shale and Coal – No air loss |

*To convert feet to meters multiply by 0.3048

Of the nine borings drilled into the grouted North Lobe (Holes E-1 – E-6 and E-8 – E-10) all but one (E-4) encountered grout at mine level with no loss of air which indicates the mine is filled with grout at these locations. In Hole E-4 there was a 50% air loss between a depth of 32.0-32.3m (105-106 ft). Below 33m (106 ft) the air flow returned. This indicates a small mine void. E-4 was pressure grouted with a take of 93cm (121 cy). In the holes drilled south of the barrier (E-11 and E-11a-E-18) a coal pillar was encountered in E-11 and E-11a. Caved and broken conditions with loss of air was found in holes E-12 through E-17. This is typical of an open mine void. Hole E-18 encountered grout placed in the barrier with no loss of air. Holes E-11 through E-16 were filled with the FBC grout mix. The quantities of grout injected in these holes are shown in Table 6-5.

Hole E-17 was later pressure tested with water with no loss – thus it was not grouted.

Table 6-4

Grout Takes in Holes E11 – E16

| Hole | Take (c.y.)* |
|---|---|
| E11 | 4 |
| E11a | 4 |
| E12 | 212.2 |
| E13 | 67 |
| E14 | 719 |
| E15 | 242 |
| E16 | 307.8 |

*To convert cubic yards to cubic meters multiply by 0.76455

Two hundred thirty one holes (17 barrier, 183 primary injection and 31 extra holes) totaling 9,332m (30,617 ft) were drilled during the injection program. Table 6-6 summarizes the depths of all holes.

Grouting was completed on November 4, 1998. The 156 primary injection holes, in which the FBC-fly ash mix was used, had a total take of 47,185cm (61,716 cubic yards), an average of 302cm (395.6 cy) per hole.

The 31 primary injection holes which were filled by pressure grouting with fly ash and cement had a total take of 978cm (1,280 cubic yards), an average of 32.6cm (42.6 cy) per hole. The total quantity of grout (the FBC-fly ash or the fly ash-cement mixes) in all holes including barrier and exploratory holes was 60,499cm (79,130 cy). The grout takes in each hole are summarized in Table 6-7. The total cost for grouting was $1,946,592 or $24.60 per cy.

Samples of grout were collected throughout the injection program. Cubes of approximately 48mm (1.9 inches) were formed and cured. Tests were generally conducted at three, seven and 28-days. Twenty-eight day strengths ranged from 9,350-22,009 kPa (1,356 – 3,192 psi).

**Table 6-5**

**Drilling Footages – Actual Depths/Depth to Base of Mine (Depth to Base of Mine = Actual Depth – 2ft)**

BARRIER BORINGS :

| Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| B-1 | 143 | 141 | B-7 | 161 | 159 | B-13 | 161 | 159 | P-155/ AR-4a | 150 | 148 |
| B-2 | 113 | 111 | B-8 | 162 | 160 | B-14 | 160 | 158 | P-64/ AR-8a | 152 | 150 |
| B-3 | 160 | 158 | B-9 | 162 | 160 | B-15 | 160 | 158 | P-143/ AR-3 | 155 | 153 |
| B-4 | 154 | 152 | B-10 | 161 | 159 | B-16 | 159 | 157 | P-183/ AR-1 | 87 | 85 |
| B-5 | 159 | 157 | B-11 | 162 | 160 | B-17 | 158 | 156 | | | |
| B-6 | 161 | 159 | B-12 | 160 | 158 | | | | | | |
| Sum = | 890 | 878 | Sum = | 968 | 956 | Sum = | 798 | 788 | Sum = | 544 | 536 |

TOTALS :

Number of Borings = 17

Actual Lengths Drilled = 2656 ft.

Actual Length to Base of Mine = 2622 ft.

Estimated Length to Base of Mine = 2525 ft

Average Depth to Base of Mine:

Actual 154.2 ft.

Estimate = 148.5 ft (increase 5.7 ft)

To convert feet to meters multiply by 0.3048

Primary Borings:

| Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| P-1 | 162 | 160 | P-48 | 115 | 113 | P-95 | 137 | 135 | P-142 | 160 | 158 |
| P-2 | 155 | 153 | P-49 | 137 | 135 | P-96 | 132 | 130 (Existing) | P-143/ AR-3 | 0 | 0 |
| | | | | | | | | | | | |

| Primary Borings: (Continued) | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) |
| P-3 | 159 | 157 | P-50 | 146 | 144 | P-97 | 130 | 128 | P-144 | 157 | 155 |
| P-4 | 156 | 154 | P-51 | 148 | 146 | P-98 | 155 | 153 | P-145 | 155 | 153 |
| P-5 | 145 | 143 | P-52 | 150 | 148 | P-99 | 150 | 148 | P-146 | 149 | 147 |
| P-6 | 142 | 140 | P-53 | 142 | 140 | P-100 | 149 | 147 | P-147 | 152 | 150 |
| P-7 | 140 | 138 | P-54 | 143 | 141 | P-101 | 157 | 155 | P-148 | 160 | 158 |
| P-8 | 143 | 141 (Existing) | P-55/ AR-4a | 0 | 0 | P-102 | 158 | 156 | P-149 | 160 | 158 |
| P-9 | 149 | 147 | P-56 | 152 | 150 | P-103 | 158 | 156 | P-150 | 162 | 160 |
| P-10 | 141 | 139 | P-57 | 155 | 153 | P-104 | 160 | 158 | P-151 | 160 | 158 |
| P-11 | 145 | 143 | P-58 | 152 | 150 | P-105 | 161 | 159 | P-152 | 158 | 156 |
| P-12 | 149 | 147 | P-59 | 147 | 145 | P-106 | 161 | 159 | P-153 | 159 | 157 |
| P-13 | 140 | 138 | P-60 | 156 | 154 | P-107 | 161 | 159 | P-154 | 146 | 144 |
| P-14 | 152 | 150 | P-61 | 157 | 155 | P-108 | 161 | 159 | P-155 | 116 | 114 |
| P-15 | 152 | 150 | P-62 | 163 | 161 | P-109 | 160 | 158 | P-156 | 117 | 115 |
| P-16 | 149 | 147 | P-63 | 155 | 153 | P-110 | 158 | 156 | P-157 | 102 | 100 |
| P-17 | 145 | 143 (Existing) | P-64/ AR-8a | 0 | 0 | P-111 | 158 | 156 | P-158 | 90 | 88 |
| P-18 | 148 | 146 | P-65 | 156 | 154 | P-112 | 154 | 152 | P-159 | 98 | 96 |
| P-19 | 146 | 144 | P-66 | 155 | 153 | P-113 | 160 | 158 | P-160 | 121 | 119 |
| P-20 | 126 | 124 | P-67 | 158 | 156 | P-114 | 157 | 155 | P-161 | 115 | 113 |
| P-21 | 127 | 125 | P-68 | 155 | 153 | P-115 | 155 | 153 | P-162 | 106 | 104 |
| P-22 | 112 | 110 | P-69 | 158 | 156 | P-116 | 152 | 150 | P-163 | 149 | 147 |

| Primary Borings: (Continued) | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) |
| P-23 | 102 | 100 | P-70 | 151 | 149 | P-117 | 158 | 156 | P-164 | 150 | 148 |
| P-24 | 120 | 118 | P-71 | 148 | 146 | P-118 | 154 | 152 | P-165 | 141 | 139 |
| P-25 | 127 | 125 | P-72 | 146 | 144 | P-119 | 144 | 142 | P-166 | 139 | 137 |
| P-26 | 118 | 116 | P-73 | 148 | 146 | P-120 | 147 | 145 | P-167 | 133 | 131 |
| P-27 | 92 | 90 | P-74 | 143 | 141 | P-121 | 139 | 137 | P-168 | 109 | 107 |
| P-28 | 88 | 86 | P-75 | 143 | 141 | P-122 | 136 | 134 | P-169 | 105 | 103 |
| P-29 | 92 | 90 | P-76 | 141 | 139 | P-123 | 136 | 134 | P-170 | 97 | 95 |
| P-30 | 98 | 96 | P-77 | 148 | 146 | P-124 | 111 | 109 | P-171 | 104 | 102 |
| P-31 | 85 | 83 | P-78 | 147 | 145 | P-125 | 116 | 114 | P-172 | 103 | 101 |
| P-32 | 96 | 94 | P-79 | 134 | 132 | P-126 | 116 | 114 | P-173 | 97 | 95 |
| P-33 | 91 | 89 | P-80 | 146 | 144 | P-127 | 102 | 100 | P-174 | 102 | 100 |
| P-34 | 88 | 86 | P-81 | 154 | 152 | P-128 | 91 | 89 | P-175 | 99 | 97 |
| P-35 | 82 | 80 | P-82 | 159 | 157 | P-129 | 89 | 87 | P-176 | 99 | 97 |
| P-36 | 85 | 83 | P-83 | 161 | 159 | P-130 | 99 | 97 | P-177 | 97 | 95 |
| P-37 | 87 | 85 | P-84 | 162 | 160 | P-131 | 92 | 90 | P-178 | 101 | 99 |
| P-38 | 94 | 92 | P-85 | 155 | 153 | P-132 | 98 | 96 | P-179 | 100 | 98 |
| P-39 | 94 | 92 | P-86 | 161 | 159 | P-133 | 98 | 96 | P-180 | 80 | 78 |
| P-40 | 100 | 98 | P-87 | 162 | 160 | P-134 | 134 | 132 | P-181 | 92 | 90 |
| P-41 | 99 | 97 | P-88 | 162 | 160 | P-135 | 150 | 148 | P-182 | 91 | 89 |
| P-42 | 89 | 87 | P-89 | 140 | 138 | P-136 | 149 | 147 (Existing) | P-183/ AR-1 | 0 | 0 |

| | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| P-43 | 104 | 102 | P-90 | 127 | 125 | P-137 | 143 | 141 | P-184 | 99 | 97 |
| P-44 | 109 | 107 | P-91 | 115 | 113 | P-138 | 143 | 141 | P-185 | 87 | 85 |
| P-45 | 126 | 124 | P-92 | 105 | 103 | P-139 | 144 | 142 | P-186 | 86 | 84 |
| P-46 | 141 | 139 | P-93 | 114 | 112 | P-140 | 150 | 148 | P-187 | 82 | 80 |
| P-47 | 132 | 130 | P-94 | 115 | 113 | P-141 | 158 | 156 | | | |
| Sum = | 5722 | 5628 | Sum = | 6587 | 6497 | Sum = | 6581 | 6487 | Sum = | 5285 | 5197 |

TOTALS :

Number of Borings = 183 (excluding existing borings)

Actual Length Drilled= 24175 ft.

Actual Length to Base of Mine= 23809 ft.

Estimated Length to Base of Mine = 23300 ft.

**To convert feet to meters multiply by 0.3048**

Average Depth to Base of Mine:

Actual = 130.1 ft

Estimate = 127.3 ft (change 2.8 ft)

**Extra and Previously Drilled Borings:**

| Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) | Hole No. | Actual Depth (ft) | Depth to Mine (ft) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| B-1b | 144 | 142 (Existing) | AR-2 | 0 | 0 | E-7 | 0 | 0 | CB-3 | 138 | 136 |
| B-14b | 161 | 159 (Existing) | AR-4 | 0 | 0 | E-8 | 162 | 160 | AR-2 | 105 | 103 |
| B-18 | 158 | 156 (Existing) | AR-7 | 0 | 0 | E-9 | 167 | 165 | AR-4 | 146 | 144 |
| B-19 | 159 | 157 (Existing) | AR-10 | 0 | 0 | E-10 | 120 | 118 | AR-7 | 141 | 139 |
| B-20 | 161 | 159 (Existing) | AR-12 | 0 | 0 | E-11 | 156 | 154 | AR-10 | 145 | 143 |
| B-21 | 162 | 160 | P-20b | 120 | 118 | E-11b | 155 | 153 | AR-12 | 151 | 149 |
| B-22 | 162 | 160 | E-1 | 102 | 100 | E-12 | 162 | 160 | SUM = | 826 | 814 |

| Extra and Previously Drilled Borings: | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **Hole No.** | **Actual Depth (ft)** | **Depth to Mine (ft)** | **Hole No.** | **Actual Depth (ft)** | **Depth to Mine (ft)** | **Hole No.** | **Actual Depth (ft)** | **Depth to Mine (ft)** | **Hole No.** | **Actual Depth (ft)** | **Depth to Mine (ft)** |
| B-23 | 162 | 160 | E-2 | 116 | 114 | E-13 | 161 | 159 | | | |
| B-24 | 163 | 161 | E-3 | 111 | 109 | E-14 | 162 | 160 | | | |
| B-25 | 159 | 157 | E-4 | 116 | 114 | E-15 | 162 | 160 | | | |
| B-26 | 159 | 157 | E-5 | 132 | 130 | E-16 | 160 | 158 | | | |
| CB-3 | 0 | 0 (Existing) | E-6 | 130 | 128 | E-17 | 155 | 153 | | | |
| | | | E-19 | 159 | 157 | E-18 | 155 | 153 | | | |
| Sum = | 1750 | 1728 | Sum = | 986 | 970 | Sum = | 1877 | 1853 | | | |

TOTALS

Number of Borings = 31

Actual Length Drilled = 3786 ft

Actual Length to Base of Mine = 3738 ft

Average Depth to Base of Mine:

Actual = 120.6 ft

Estimate = 0.0 ft

TOTAL DRILLING FOOTAGES : ( lin. ft. )

| | | | | |
|---|---|---|---|---|
| Barrier | 2,622 | | | |
| Primary | 24,175 | | | |
| Extra | 3,786 | | | |
| Sum = | 30,583 L.F. | Estimate = 26233 L.F. | Increase : 4350 L.F. | % Inc. : 16.58 |

To convert feet to meters multiply by 0.3048

## Table 6-6

## Grout Takes

| BARRIER INJECTION HOLES : B-1 THROUGH B-26 | | | | PRIMARY INJECTION HOLES : P-1 THROUGH P-92 | | | | PRIMARY INJECTION HOLES : P-93 THROUGH P-187 | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date |
| B-1 | 1 | 6/16/98 | 6/16/98 | P-1 | 20 | 6/3/98 | 6/3/98 | P-93 | 41 | 9/30/98 | 9/30/98 |
| B-1b | 1 | 6/16/98 | 6/16/98 | P-2 | 509.7 | 6/4/98 | 6/8/98 | P-94 | 208 | 9/30/98 | 9/30/98 |
| CB-3 | 0.5 | 6/11/98 | 6/11/98 | P-3 | 311.9 | 6/3/98 | 6/4/98 | P-95 | 63.1 | 9/29/98 | 9/30/98 |
| B-2 | 3390.9 | 6/9/98 | 6/16/98 | P-4 | 1 | 5/26/98 | 5/26/98 | P-96 | 1 | X | |
| B-3 | 548.3 | 5/11/98 | 5/14/98 | P-5 | 842.5 | 6/8/98 | 6/10/98 | P-97 | 0 | PG | |
| B-4 | 776.1 | 5/12/98 | 6/1/98 | P-6 | 703.5 | 6/10/98 | 6/10/98 | AR-7 | 0 | PG | |
| B-5 | 1 | 5/26/98 | 5/26/98 | P-7 | 1 | 6/11/98 | 6/11/98 | CB-2 | 1 | X | |
| B-6 | 2878.4 | 5/13/98 | 6/1/98 | P-8 | 3 | 10/1/98 | 10/1/98 | P-98 | 10 | 9/28/98 | 9/28/98 |
| B-7 | 1 | 6/2/98 | 6/2/98 | P-9 | 2 | 10/1/98 | 10/1/98 | P-99 | 137.8 | 9/24/98 | 9/28/98 |
| B-8 | 2 | 6/2/98 | 6/2/98 | P-10 | 7 | 6/16/98 | 6/16/98 | P-100 | 1620.7 | 8/4/98 | 9/28/98 |
| B-9 | 2 | 6/3/98 | 6/3/98 | P-11 | 15 | 6/16/98 | 6/16/98 | P-101 | 5 | 10/1/98 | 10/1/98 |
| B-10 | 933.5 | 6/1/98 | 6/3/98 | P-12 | 3 | 10/1/98 | 10/1/98 | P-102 | 655 | 9/24/98 | 9/24/98 |
| AR-12 | 218 | 6/2/98 | 6/3/98 | P-13 | 6 | 10/1/98 | 10/1/98 | P-103 | 1335.2 | 9/22/98 | 9/24/98 |
| B-11 | 1 | 6/2/98 | 6/2/98 | P-14 | 2 | 9/29/98 | 9/29/98 | P-104 | 1 | 9/22/98 | 9/22/98 |
| B-12 | 3999.7 | 5/20/98 | 6/2/98 | P-15 | 27 | 9/30/98 | 9/30/98 | P-105 | 55 | 9/24/98 | 9/24/98 |
| B-13 | 1 | 6/3/98 | 6/3/98 | P-16 | 1 | 9/29/98 | 9/29/98 | P-106 | 3 | 10/1/98 | 10/1/98 |
| B-14 | 1 | 6/2/98 | 6/2/98 | P-17 | 35 | 7/2/98 | 10/1/98 | P-107 | 203 | 7/14/98 | 10/1/98 |
| B-14b | 59 | 6/3/98 | 6/3/98 | AR-8 | 2 | 9/29/98 | 9/29/98 | P-108 | 1 | 9/22/98 | 9/22/98 |
| B-15 | 1 | 6/3/98 | 6/3/98 | P-18 | 1 | 9/29/98 | 9/29/98 | P-109 | 1 | 7/1/98 | 7/1/98 |

| BARRIER INJECTION HOLES : B-1 THROUGH B-26 | | | | PRIMARY INJECTION HOLES : P-1 THROUGH P-92 | | | | PRIMARY INJECTION HOLES : P-93 THROUGH P-187 | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date |
| B-14 | 1 | 6/2/98 | 6/2/98 | P-17 | 35 | 7/2/98 | 10/1/98 | P-107 | 203 | 7/14/98 | 10/1/98 |
| B-14b | 59 | 6/3/98 | 6/3/98 | AR-8 | 2 | 9/29/98 | 9/29/98 | P-108 | 1 | 9/22/98 | 9/22/98 |
| B-15 | 1 | 6/3/98 | 6/3/98 | P-18 | 1 | 9/29/98 | 9/29/98 | P-109 | 1 | 7/1/98 | 7/1/98 |
| B-16 | 1307.9 | 6/2/98 | 6/3/98 | P-19 | 1 | 9/29/98 | 9/29/98 | P-110 | 369 | 7/15/98 | 7/15/98 |
| B-17 | 1 | 6/3/98 | 6/3/98 | P-20 | 1 | 6/24/98 | 6/24/98 | P-111 | 1 | 7/1/98 | 7/1/98 |
| B-18 | 42 | 10/26/98 | 10/26/98 PG | P-20b | 2175.8 | 6/22/98 | 6/24/98 | P-112 | 714.9 | 7/15/98 | 7/16/98 |
| B-19 | 15 | 10/26/98 | 10/26/98 PG | P-21 | 14 | 7/2/98 | 7/2/98 | P-113 | 14 | 9/14/98 | 9/14/98 |
| B-20 | 33.3 | 10/26/98 | 10/27/98 PG | P-22 | 1 | 6/24/98 | 6/24/98 | P-114 | 182 | 7/16/98 | 7/16/98 |
| B-21 | 49 | 10/26/98 | 10/26/98 Mix | P-23 | 1173.3 | 6/24/98 | 7/2/98 | P-115 | 1 | 7/1/98 | 7/1/98 |
| B-22 | 36.3 | 10/26/98 | 11/3/98 Mix | P-24 | 1 | 6/30/98 | 6/30/98 | P-116 | 1 | 7/1/98 | 7/1/98 |
| B-23 | 36 | 10/27/98 | 10/27/98 Mix | P-25 | 1 | 6/30/98 | 6/30/98 | P-117 | 844.6 | 7/16/98 | 7/20/98 |
| B-24 | 18 | 10/27/98 | 11/3/98 Mix | P-26 | 140 | 7/14/98 | 7/14/98 | P-118 | 7944.5 | 7/20/98 | 8/6/98 |
| B-25 | 21 | 10/27/98 | 10/27/98 Mix | P-27 | 1 | 7/2/98 | 7/2/98 | P-119 | 1 | 8/11/98 | 8/11/98 |
| B-26 | 17 | 10/27/98 | 11/3/98 Mix | P-28 | 235.3 | 7/2/98 | 7/2/98 | P-120 | 611 | 8/6/98 | 8/6/98 |
| B27/A R11? | 15 | 10/27/98 | PG,HCP 10/27/98 | P-29 | 1 | 7/14/98 | 7/14/98 | P-121 | 1 | 8/11/98 | 8/11/98 |
| Sum = | 14392.9 | C.Y. | | P-30 | 150 | 7/13/98 | 7/13/98 | P-122 | 1 | 8/11/98 | 8/11/98 |
| | | | | P-31 | 440 | 7/13/98 | 7/13/98 | P-123 | 4 | 8/11/98 | 8/11/98 |

| | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | P-32 | 246 | 7/13/98 | 7/13/98 | P-124 | 981.2 | 7/9/98 | 7/13/98 |
| EXPLORATORY HOLES E-1 THROUGH E-18 | | | | PRIMARY INJECTION HOLES : P-1 THROUGH P-92 | | | | PRIMARY INJECTION HOLES : P-93 THROUGH P-187 | | | |
| Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date |
| | | | | P-33 | 297.6 | 7/13/98 | 7/14/98 | P-125 | 1 | 7/9/98 | 7/9/98 |
| | | | | P-34 | 7 | 7/14/98 | 7/14/98 | P-126 | 1 | 8/11/98 | 8/11/98 |
| | | | | P-35 | 14 | 7/14/98 | 7/14/98 | P-127 | 1 | 7/9/98 | 7/9/98 |
| E-1 | 1 | 10/27/98 | 10/27/98 | P-36 | 1045.5 | 7/1/98 | 7/6/98 | P-128 | 24 | 7/8/98 | 7/8/98 |
| E-2 | 1 | 10/27/98 | 10/27/98 | P-37 | 7 | 7/6/98 | 7/6/98 | P-129 | 1 | 7/6/98 | 7/6/98 |
| E-3 | 1 | 10/27/98 | 10/27/98 | P-38 | 48 | 7/14/98 | 7/14/98 | P-130 | 222 | 7/8/98 | 7/8/98 |
| E-4 | 121 | 10/28/98 | 10/28/98PG | P-39 | 171 | 7/14/98 | 7/14/98 | P-131 | 354.6 | 7/6/98 | 7/7/98 |
| E-5 | 1 | 10/27/98 | 10/27/98 | P-40 | 40 | 7/1/98 | 7/1/98 | P-132 | 771.3 | 7/8/98 | 7/8/98 |
| E-6 | 1 | 10/27/98 | 10/27/98 | P-41 | 82 | 7/14/98 | 7/14/98 | P-133 | 643 | 7/7/98 | 7/13/98 |
| E-7 | 1 | Not Drilled | | P-42 | 48 | 7/14/98 | 7/14/98 | P-134 | 1 | 9/22/98 | 9/22/98 |
| E-8 | 4 | 11/3/98 | 11/3/98 | P-43 | 1 | 7/14/98 | 7/14/98 | P-135 | 1.5 | 8/11/98 | 8/11/98 |
| E-9 | 4 | 11/3/98 | 11/3/98 | P-44 | 1 | 7/14/98 | 7/14/98 | P-136 | 11 | 8/10/98 | 8/10/98 |
| E-10 | 1 | 10/27/98 | 10/27/98 | AR-5 | 1 | 7/14/98 | 7/14/98 | P-137 | 1 | 8/10/98 | 8/10/98 |
| E-11 | 2 | 10/27/98 | 10/27/98 | P-45 | 383 | 7/1/98 | 7/1/98 | P-138 | 1 | 8/10/98 | 8/10/98 |
| E-11a | 2 | 10/27/98 | 10/27/98 | P-46 | 1 | 7/7/98 | 7/7/98 | P-139 | 10 | 9/14/98 | 9/14/98 |
| E-12 | 238.2 | 10/29/98 | 11/2/98 | P-47 | 239 | 7/1/98 | 7/1/98 | P-140 | 8515.5 | 8/18/98 | 9/14/98 |
| E-13 | 67 | 11/2/98 | 11/2/98 | P-48 | 1 | 7/14/98 | 7/14/98 | P-141 | 1 | 8/6/98 | 8/6/98 |
| E-14 | 719 | 11/2/98 | 11/2/98 | P-49 | 1 | 7/1/98 | 7/1/98 | P-142 | 475 | 7/23/98 | 7/23/98 |
| E-15 | 242 | 11/2/98 | 11/3/98 | P-50 | 1 | 7/7/98 | 7/7/98 | P-143/AR-3 | 1130.5 | 7/22/98 | 7/23/98 |

| EXPLORATORY HOLES E-1 THROUGH E-18 | | | | PRIMARY INJECTION HOLES : P-1 THROUGH P-92 | | | | PRIMARY INJECTION HOLES : P-93 THROUGH P-187 | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date |
| E-14 | 719 | 11/2/98 | 11/2/98 | P-49 | 1 | 7/1/98 | 7/1/98 | P-142 | 475 | 7/23/98 | 7/23/98 |
| E-15 | 242 | 11/2/98 | 11/3/98 | P-50 | 1 | 7/7/98 | 7/7/98 | P-143/AR-3 | 1130.5 | 7/22/98 | 7/23/98 |
| E-16 | 307.8 | 11/3/98 | 11/3/98 | P-51 | 918.1 | 6/30/98 | 7/1/98 | P-144 | 7 | 8/6/98 | 8/6/98 |
| E-17 | 1 | 10/27/98 | 10/27/98 | P-52 | 171 | 6/30/98 | 6/30/98 | P-145 | 9 | 8/6/98 | 8/6/98 |
| E-18 | 1 | 10/27/98 | 10/27/98 | P-53 | 6 | 7/1/98 | 10/1/98 | P-146 | 81 | 8/6/98 | 8/6/98 |
| E-19 | 1 | 10/27/98 | 10/27/98 | P-54 | 7 | 6/30/98 | 10/1/98 | P-147 | 4428 | 8/6/98 | 8/18/98 |
| Sum = | 1716 | C.Y. | | AR-4 | 5 | 10/1/98 | 10/1/98 | P-148 | 1 | 9/22/98 | 9/22/98 |
| | | | | P-55/AR4a | 7 | 7/1/98 | 10/1/98 | P-149 | 4619.9 | 9/14/98 | 9/21/98 |
| PRESSURE GROUT (PG) ALL HOLES | | | | PRIMARY INJECTION HOLES : P-1 THROUGH P-92 | | | | PRIMARY INJECTION HOLES : P-93 THROUGH P-187 | | | |
| Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date |
| | | | | P-56 | 1065.7 | 6/25/98 | 6/30/98 | P-150 | 1686.2 | 9/21/98 | |
| P-92 | 1 | 10/6/98 | 10/6/98 | P-57 | 329 | 6/25/98 | 6/25/98 | P-151 | 1 | 9/22/98 | 9/22/98 |
| P-97 | 86.2 | 10/5/96 | 10/6/98 | P-58 | 4 | 6/25/98 | 6/25/98 | P-152 | 31 | 9/22/98 | 9/22/98 |
| AR-7 | 21 | 10/6/98 | 10/6/98 | P-59 | 4 | 6/30/98 | 10/1/98 | P-153 | 0 | PG | |
| P-153 | 1 | 10/6/98 | 10/6/98 | P-60 | 2 | 9/29/98 | 9/29/98 | P-154 | 0 | PG | |
| P-154 | 2 | 10/6/98 | 10/6/98 | P-61 | 4 | 10/1/98 | 10/1/98 | P-155AR-6 | 1 | X | |
| P-156 | 1 | 10/12/98 | 10/12/98 | P-62 | 2 | 9/30/98 | 9/30/98 | P-156 | 0 | PG | |
| P-157 | 1 | 10/6/98 | 10/6/98 | P-63 | 2 | 9/30/98 | 9/30/98 | P-157 | 0 | PG | |

| PRESSURE GROUT (PG) ALL HOLES | | | | PRIMARY INJECTION HOLES : P-1 THROUGH P-92 | | | | PRIMARY INJECTION HOLES : P-93 THROUGH P-187 | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date |
| P-158 | 1 | 10/12/98 | 10/12/98 | P64/AR8a | 1 | 9/29/98 | 9/29/98 | P-158 | 0 | PG | |
| P-159 | 1 | 10/12/98 | 10/12/98 | P-65 | 3.2 | 10/1/98 | 10/1/98 | P-159 | 0 | PG | |
| P-160 | 335.2 | 10/6/98 | 10/12/98 | P-66 | 1 | 9/29/98 | 9/29/98 | P-160 | 0 | PG | |
| P-161 | 5 | 10/12/98 | 10/14/98 | P-67 | 1 | 9/29/98 | 9/29/98 | P-161 | 0 | PG | |
| P-162 | 204.5 | 10/12/98 | 10/13/98 | P-68 | 3 | 10/1/98 | 10/1/98 | P-162 | 0 | PG | |
| AR-2 | 1 | 10/14/98 | 10/14/98 | P-69 | 72 | 9/30/98 | 9/30/98 | P-163 | 6 | 9/22/98 | 9/22/98 |
| P-168 | 48 | 10/14/98 | 10/14/98 | P-70 | 3 | 10/1/98 | 10/1/98 | P-164 | 7 | 9/22/98 | 9/22/98 |
| P-169 | 2 | 10/14/98 | 10/14/98 | P-71 | 1 | 8/27/98 | 8/27/98 | P-165 | 12 | 9/14/98 | 9/14/98 |
| P-170 | 1 | 10/14/98 | 10/14/98 | P-72 | 1 | X | | P-166 | 2 | 8/10/98 | 8/10/98 |
| P-171 | 1 | 10/14/98 | 10/14/98 | P-73 | 205.8 | 9/28/98 | 9/29/98 | P-167 | 1 | 8/10/98 | 8/10/98 |
| P-172 | 1 | 10/14/98 | 10/14/98 | P-74 | 1 | 8/27/98 | 8/27/98 | P-168 | 0 | PG | |
| P-173 | 1 | 10/14/98 | 10/14/98 | P-75 | 1 | 8/27/98 | 8/27/98 | P-169 | 0 | PG | |
| P-174 | 4.3 | 10/15/98 | 10/15/98 | P-76 | 1 | 10/1/98 | 10/1/98 | P-170 | 0 | PG | |
| P-175 | 1 | 10/15/98 | 10/15/98 | P-77 | 1 | 6/10/98 | 6/10/98 | P-171 | 0 | PG | |
| P-176 | 21.7 | 10/14/98 | 10/15/98 | AR-10 | 7 | 6/16/98 | 6/16/98 | P-172 | 0 | PG | |
| P-177 | 1 | 10/14/98 | 10/14/98 | P-78 | 1 | 8/27/98 | 8/27/98 | P-173 | 0 | PG | |
| P-178 | 1 | 10/14/98 | 10/14/98 | P-79 | 46 | 9/29/98 | 9/29/98 | P-174 | 0 | PG | |
| P-179 | 4 | 10/15/98 | 10/15/98 | P-80 | 67 | 9/29/98 | 9/29/98 | P-175 | 0 | PG | |
| P-180 | 142.2 | 10/22/98 | 10/26/98 | P-81 | 1 | 6/17/98 | 6/17/98 | P-176 | 0 | PG | |
| P-181 | 146.6 | 10/19/98 | 10/20/98 | P-82 | 2 | 6/10/98 | 6/10/98 | P-177 | 0 | PG | |
| | | | | | | | | | | | |

| PRESSURE GROUT (PG) ALL HOLES | | | | PRIMARY INJECTION HOLES : P-1 THROUGH P-92 | | | | PRIMARY INJECTION HOLES : P-93 THROUGH P-187 | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date | Hole No. | Grout Take C.Y. | Start Date | Finish Date |
| P-182 | 72 | 10/21/98 | 10/21/98 | P-83 | 2 | 6/22/98 | 6/22/98 | P-178 | 0 | PG | |
| P-183/ AR-1 | 56.8 | 10/19/98 | 10/20/98 | P-84 | 1 | 6/17/98 | 6/17/98 | P-179 | 0 | PG | |
| P-184 | 95 | 10/19/98 | 10/19/98 | P-85 | 10218.3 | 6/16/98 | 9/3/98 | P-180 | 0 | PG | |
| P-185 | 38.3 | 10/21/98 | 10/22/98 | P-86 | 1 | 6/17/98 | 6/17/98 | P-181 | 0 | PG | |
| P-186 | 2 | 10/26/98 | 10/26/98 | P-87 | 2 | 6/3/98 | 6/3/98 | P-182 | 0 | PG | |
| P-187 | 1 | 10/22/98 | 10/22/98 | P-88 | 2 | 6/3/98 | 6/3/98 | P-183 | 0 | PG | |
| Sum = | 1301.8 | C.Y. | | P-89 | 1 | X | | P-184 | 0 | PG | |
| | | | | P-90/AR-9 | 3 | 9/30/98 | 9/30/98 | P-185 | 0 | PG | |
| | | | | P-91 | 3 | 9/30/98 | 9/30/98 | P-186 | 0 | PG | |
| | | | | P-92 | 0 | PG | | P-187 | 0 | PG | |
| | | | | Sum = | 22644.2 | C.Y. | | Sum = | 39072.5 | C.Y. | |
| | | | | | | | | | | | |

TOTAL OF ALL GROUT PLACED = 79,130 cy

To convert cubic yards to cubic meters multiply by 0.76455

# 7

# PROJECT PERFORMANCE

## Core Borings

In August 1999, four core borings identified as OCB-1, OCB-2, OCB-3, and OCB-4 (locations shown on Figure 7-1) were used to recover samples of the hardened grout. In order to reduce costs, the holes were drilled by an air rotary drill rig to within a few feet of the mine roof. Core boring was then utilized to sample the grout within the mine interval.

Two holes were drilled in the first mined area and two in the second mined area. Logs of the recovered cores are presented below.

**Table 7-1**
**Logs of Core Borings to Sample Hardened Grout**

| Boring | Thickness and Description (ft)* |
|---|---|
| OCB-1 | ~3.0 Mine Gob<br>1.4 claystone<br>1.9 sandy shale |
| OCB-2 | No recovery |
| OCB-3 | 0.2 shale<br>4.9 grout<br>3.1 mine gob<br>1.6 shale grading into sandstone |
| OCB-4 | 2.5 sandstone and sandy shale<br>2.5 grout<br>Coal<br>4.8 siltstone |

*To convert feet to meters multiply by 0.3048.

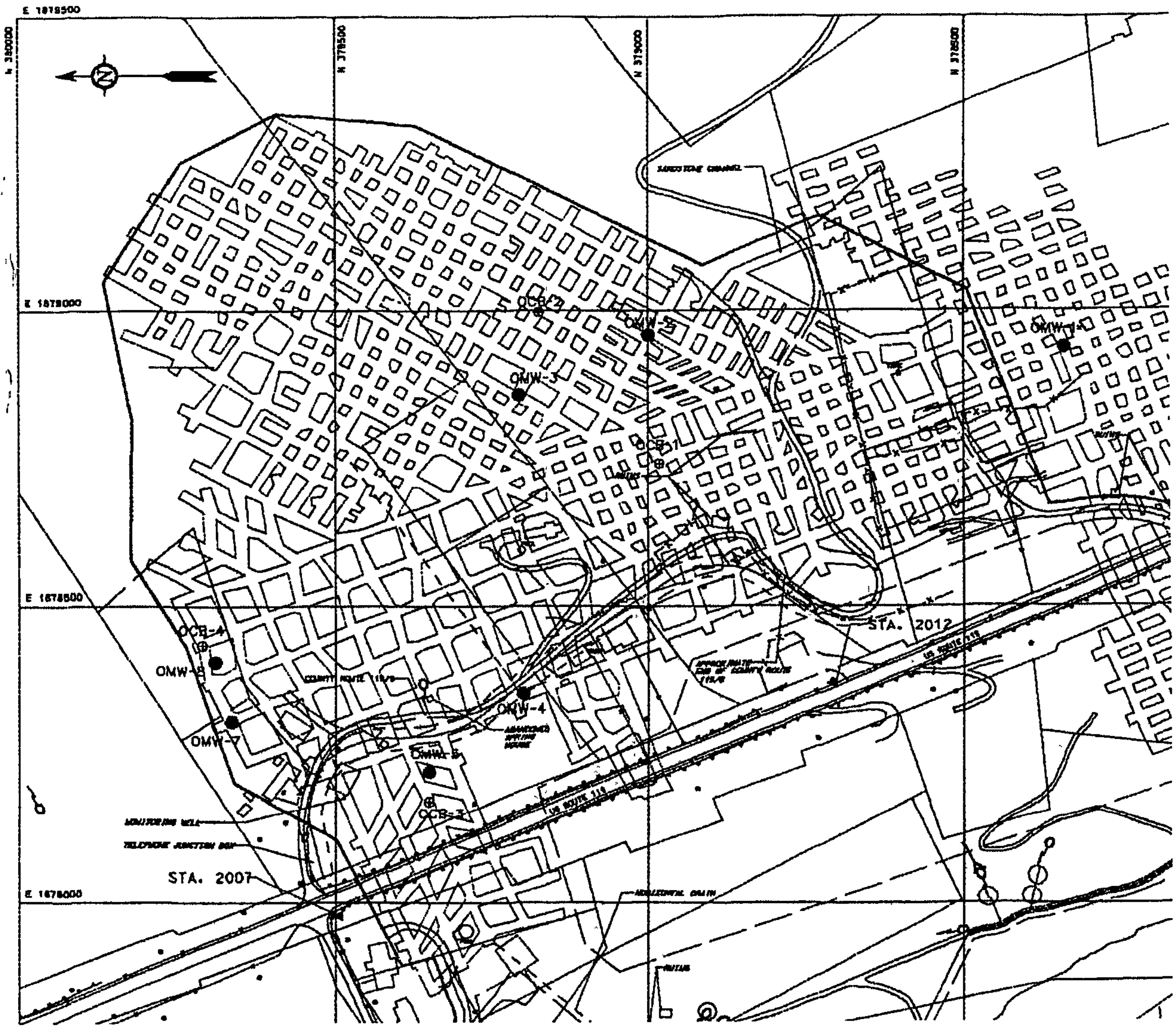



**Figure 7-1**

**Plan of Core Borings to Sample Hardened Grout and Monitoring Wells**

Cores of grout suitable for strength and permeability testing were obtained from OCB3 and OCB4. Six unconfined compressive strength tests were conducted and four permeability tests (Table 7-2). Dry unit weights were obtained on 7 of the test samples. Dry unit weights of the grout core samples ranged from 11,531 to 16,055 $N/m^3$ (73.4 to 102.2 pounds per cubic foot). Unconfined compressive strengths varied from 6,543 to 13,907 kPa (949 to 2,017 psi). The average strength of the 6 samples was 9,653 kPa (1,400 psi). Due to the long distances which grout moved from some injection holes the time of the injection of the grout obtained from the cores is unknown. However, on average the grout had been in the mine about 1 year with a minimum time in the mine of 9-10 months. Permeability of the 4 grout core samples (Table 7-2) varied from $6.2 \times 10^{-7}$ to $8.9 \times 10^{-8}$ cm/sec.

Table 7-2

**Omega Grout Core Test Results**

| Boring | Sample | Unit Weight $\gamma$(pcf) | Unit Weight $\gamma_d$(pcf) | Unconfined Compression $q_u$ (psi) | Hydraulic Conductivity K (cm/sec) | Location within Run Recovery |
|---|---|---|---|---|---|---|
| OCB-3 | 1 | 105.6 | 85.0 | 1548 | --- | 1.1' below top of run and 0.8' below shale |
| OCB-3 | 2 | 101.8 | 73.4 | 949 | --- | 2.7' below top of run and 2.3' below shale |
| OCB-3 | 3 | 104.8 | 74.3 | 1298 | --- | 3.6' below top of run and 3.3 below shale |
| OCB-3 | 4 | 104.0 | ---* | 1361 | --- | 5.0' below top of run and at top of lower shale |
| OCB-3 | 1 | 91.8 | 76.7 | --- | $6.2\times10^{-7}$ | 1.5' below top of run and 1.3' below shale |
| OCB-3 | 2 | 90.1 | 74.7 | --- | $9.9\times10^{-7}$ | 4.7 below top of run and 0.5 above shale/coal |
| OCB-4 | 1 | 95.7 | ---* | 2017 | --- | 3.2 below top of run and 0.7 below sandstone |
| OCB-4 | 2 | 93.3 | ---* | 1229 | --- | 3.7' below top of run and 1.4' above weathered shale |
| OCB-4 | 1 | 95.5 | 85.2 | --- | $1.1\times10^{-7}$ | 3.85 below top of run and 1.45 below rock |
| OCB-4 | 2 | 109.8 | 102.2 | --- | $8.9\times10^{-8}$ | 4.95 below top of run and at top of weathered shale |

*Insufficient material trimmed from sample for moisture determination.
To convert unit weight in pounds per cubic foot to Newtons per cubic meter multiply by 157.1
To convert pounds per square inch to kilopascals multiply by 6.895.
To convert feet to meters multiply by 0.3048.

At completion of the injection program untested cube samples were obtained from Howard. In March, 2000 seven cubes having an age (curing time) of 496 to 652 days were tested for strength. Strengths varied from 1,738 to 7,846 kPa (252 to 1,138 psi) with an average strength of 4,744 kPa (688 psi) [Table 7-3].

**Table 7-3**

**Grout Cube Strength Test Results**

| Sample No | AM | PM2 | AM1 | PM2 | AM2 | AMII | --- |
|---|---|---|---|---|---|---|---|
| Date Molded | 5/19/98 | 6/1/98 | 6/30/98 | 7/27/98 | 8/31/98 | 9/28/98 | 10/22/98 |
| Date Tested | 3/1/00 | 3/1/00 | 3/1/00 | 3/1/00 | 3/1/00 | 3/1/00 | 3/1/00 |
| Curing Time (Days) | 652 | 639 | 610 | 583 | 548 | 520 | 496 |
| Weight (gms) | 172.71 | 209.85 | 203.94 | 195.55 | 233.03 | 222.09 | 213.31 |
| Length (in) | 1.9435 | 2.0290 | 2.0025 | 1.9825 | 1.9980 | 1.9905 | 1.9855 |
| Width (in) | 1.9865 | 1.9920 | 1.8890 | 1.9645 | 1.9605 | 1.9510 | 1.8525 |
| Height (in) | 2.0185 | 1.9875 | 1.9900 | 1.9920 | 1.9980 | 2.0030 | 1.9700 |
| Area (sq. in) | 3.86 | 4.04 | 3.78 | 3.89 | 3.92 | 3.88 | 3.68 |
| Density (pcf) | 84.4 | 99.5 | 103.2 | 96.0 | 113.4 | 108.8 | 112.2 |
| Strength (psi) | 252 | 1096 | 845 | 268 | 1057 | 1138 | 661 |

*To convert psi to kilopascals multiply by 6.895.

## Borehole Video Observations

The Office of Surface Mining Reclamation and Enforcement (OSMRE) utilized a borehole video camera on September 8, 1999 to examine the four core borings as well as a hole drilled into the mine updip of the grout barrier (this hole is designated OMW-1R).

Mr. William Ehler's report (September, 1999) contained the following information:[23]

"Four holes were cored through the CCP grout and we observed the contact of the mine roof. These were borings OCB-1 through –4. These holes were drilled at a 152mm (6-inch) diameter to a level about 3m (10 ft) above the mine floor where the holes were cored. The overburden observed in these holes had little or no fracturing above the immediate mine roof. All holes except OCB-3 could be clearly observed. Hole OCB-3 had a water level at 24.7m (80.9 ft) and was blocked below 27.4m (90.0 ft). The cloudy water left from drilling prevented clear views of the grout/mine roof contact. The other three holes showed a clear contact of the CCP grout and mine roof." Table 7-4 presents Mr. Ehler's observations of these holes.

Table 7-4

**Borehole Video Observations**

| Hole No. | Total Depth Observed (ft)* | Water Level (ft)* | Depth to Mine Roof (top of void) (ft)* | Depth to Top of Grout (ft)* | Comments |
|---|---|---|---|---|---|
| OCB-1 | 149.1 | 146.9 | 136.5 | 137.2 | Grout below minor void, becomes finer grained with depth; minor voids near floor @ 145.8 ft |
| OCB-2 | 142.6 | Dry | 139.0 | 139.6 | 4" core hole; cuttings and soft grout @ TD. Good roof contact w/small voids in roof rock @ 22° bearing |
| OCB-3 | 90.9 | 80.9 | | | Cuttings and fine clay @ TD, water too cloudy from clay particles to observe borehole side walls |
| OCB-4 | 103.5 | 102.3 | 95.1 | 95.1 | Mine roof fractured 94.6-95.1 ft; Grout 95.1-100.4 ft, slightly washed out; Gob w/grout 100.4-102.0 ft |

TD = Total Depth (from Ehler, 1999) [23]
To convert feet to meters multiply by 0.3048.

"Another hole, OMW-1R was drilled up dip of the grout placement, where we observed a mine void partially filled with collapsed roof fall and channels of grout. An open void was observed from 45.9 – 46.9m (150.7-153.9 ft). The base of the hole was dry and covered with drill cuttings and mine gob. The mine entry was trending east-west, with a crushed and spalling pillar visible to the south. A channel created by flowing grout was observed at a bearing of 269°. We also observed grout solidified near its angle of repose (5-10°) at a bearing of 347°. Numerous wood posts, in place and some broken, were observed at various bearings and distances from the camera.

Conclusions: Borings OCB-1 through -4 showed good roof contact with the CCB grout but did not flow much into the fractured, immediate overburden. Small discontinuous voids and open fractures along the roof probably indicates that material contacted the roof but was washed away during the core drilling. Grout with coarse-grained rock fragments observed in OCB-1 indicated a mixing of floor materials with the grout as it traveled from the injection hole. The observations from OMW-1R showed that grout flowed at a relatively low slump through and around the mine entries to the north.

The lack of water in all of the holes indicates little flow of the mine water through the grout. Water standing in OCB-3 appeared to be drill water and showed no relative movement. Observation in OCB-1 of small voids on the mine floor and the absence of grout indicates that material may be a conduit of water. Water was observed at the mine floor interval in OCB-1 and-4." (Ehler, 1999)[23]

The observations showed good roof contact and distribution of the injected grout. There appeared to be little water within the grouted mine workings. The following provides more specific information regarding the borings:

- Boring OCB-2 was dry.
- Boring OCB-1 and OCB-4 contained 0.67 (2.2 ft) and 0.37 (1.2 ft) of water, respectively.
- Boring OCB-3 contained water at the mine level which was thought to be from the drilling operation. No water flow was noted in the boring.
- An open void was present at mine level in OMW-1R. No water was noted in the boring.

## Monitoring Wells

In March 1999 following a large snow melt, very poor quality AMD discharged into the Cobun Creek watershed from several springs located to the north of the North Lobe. The total maximum flow to Cobun Creek was reported to be 114 liters per minute (30 gpm). The flow dropped to approximately 4 liters per minute (1 gpm) following the melt period.

Following this event, the WVDEP designed and constructed a water collection system and pump station to convey the AMD drainage out of the Cobun Creek watershed to the existing AMD treatment operation.

To obtain additional information regarding the groundwater conditions, six monitoring wells (MWs) were installed at the locations shown on Figure 7-1. The six MWs identified as OMW-2, OMW-3, OMW-4, OMW-5, OMW-7, and OMW-8, were installed above the mine workings of the North Lobe. An additional MW identified as OMW-1R was drilled into the mine workings just updip of the grouted North Lobe to evaluate whether groundwater was being impounded by the grouted mine workings.

The MWs were installed by drilling a 152mm (6-inch) diameter rotary boring, and cased in rock. With the exception of the OMW-1R, 51mm (two-inch) diameter PVC pipe was placed in the wells with slotted screens, aggregate and bentonite seals to isolate the lower 3m (10 ft) of the well from the overlying strata.

Static water levels in the MWs were recorded in August and September 1999, and in February, May, August and November, 2000. The data collected are presented in Table 7-5. The sampling which was to have been quarterly was delayed due to a drought.

### Table 7-5

### Groundwater Levels in Monitoring Wells

| Monitoring Well | Top Elevation (ft) | Well Depth (ft) | Water Levels (feet) | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | 8-19-99 | 8-20-99 | 8-24-99 | 8-26-99 | 9-1-99 | 9-8-99 | 2-15-00 | 5-17-00 | 8-30-00 | 11-30-00 |
| OMW2 | 1724 8 | 148 | 147 | 147 | Dry | 144 | 144 | 141 | 140 | 140 | Dry | Dry |
| OMW3 | 1710.9 | 150.5 | 61 | 140 | -- | Dry | Dry | Dry | Dry | Dry | Dry | Dry |
| OMW4 | 1627.2 | 105 | 22 | 80 | -- | 72 | 89 | 91 | 92.5 | 93.5 | 103 | * |
| OMW5 | 1587.9 | 87 | -- | Dry | Dry | Dry | Dry | Dry | Dry | 80 | 85 | * |
| OMW7 | 1572.1 | 73 | -- | 43 | 44 | 68 | 69 | 67 | 68 | 58 | Blocked | Blocked |
| OMW8 | 1601.2 | 92 | 31 | 49 | 52 | 76 | 79 | 84 | 89 | 82 | Dry | ** |
| OMW-1R | 1762.7 | 162 | -- | --- | --- | Dry | Dry | Dry | Dry | Dry | Dry | *** |

*Blocked by Ice **Destroyed by site grading ***Could not find in snow.
To convert feet to meters multiply by 0.3048.

The six monitoring wells above the North Lobe are located just above the mine roof. They were installed to determine water levels and water quality in the rock strata immediately above the Omega Mine. Little water was found. Once the wells were in place a few weeks, water levels on September 9, 1999 ranged from dry in OMW-3 and OMW-5 to 4.2m (14 ft) above the well bottom in OMW-4 (Table 7-5). These low water levels were initially thought to be a result of drought conditions. Leakage into the underlying mine workings may also be the reason for the small amount of water in the wells.

OMW-1R drilled into the mine south of the North Lobe barrier has been dry since installed. It had been thought the mine water might be ponding updip of the North Lobe and flowing down dip above the mine roof to produce the seepage from the coal outcrop above Cobun Creek. Ground water samples for analytical testing were obtained at various times from OMW-2, OMW-4, OMW-7, and OMW-8. The analytical results are presented in Table 7-6. They indicate that the pH of the ground water above the mine is generally greater than 7.0 (neutral) and therefore slightly alkaline. OMW-5 is the only well that has been consistently acidic. None of the water from the monitoring wells exhibits acidic properties comparable to the AMD seeps above Cobun Creek or the flow from the Marshall House drains.

## Benthic Survey

A post-injection benthic survey was conducted on September 7, 1000 in the area previously assessed by the WVDEP in August 1994. However, due to the drought conditions the streams attributes could not be adequately assessed or compared to previous data.

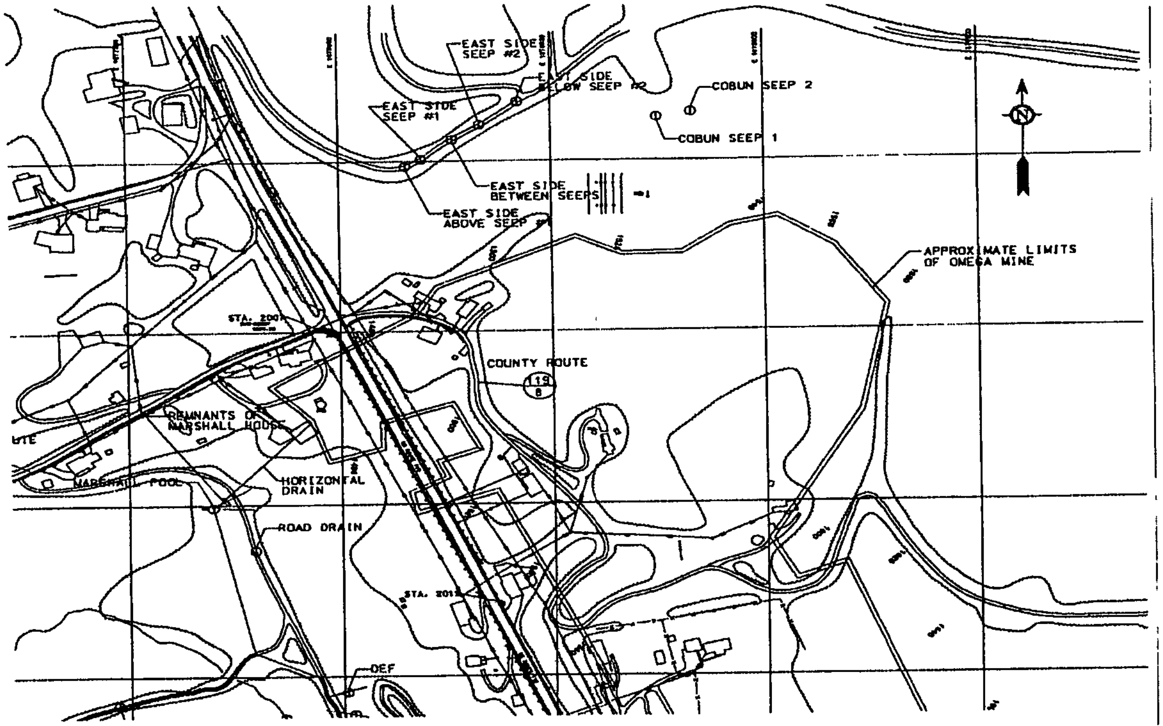
## Mine Discharges -- Quantity and Quality

Acid mine drainage (AMD) discharges from multiple points around the Omega mine into the Owl Creek and Cobun Creek watersheds. Both the West Virginia DEP and the U.S. DOE have monitored the flow and chemistry of various discharges since February 1993 until the present (October 2000). Most of the AMD is collected and treated by the WVDEP. Twenty-three separate monitoring points were established and monitored on a relatively consistent basis. Not all of these monitoring points were monitored by both agencies or for the entire monitoring time frame.

Seven of these 23 monitoring points were located north of the North Lobe in the Cobun Creek watershed (Figure 7-2). Flows were generally small and often insufficient for water chemistry analysis. However, when measured acidity sometimes exceeded 10,000 parts per million (ppm) and on June 2, 1999 peaked at 31,800 ppm with a flow of 15 liters per minute (4 gpm). With installation of the collection system and pump station these seven monitoring points were abandoned in September, 1999. Flows and water quality are now measured only for the combined collected seeps. Figure 7-3 shows the flow and corresponding acidity for this combined sample in the Cobun Creek watershed in the last half of 2000. As indicated, the maximum flow was 22.7 liters per minute (6 gpm) [Figure 7-3]. The acidity is extremely variable ranging from 500 to 4500 ppm.

From a practical standpoint only four of the 23 monitoring points can be considered major discharges. These are the Marshall House where two horizontal borings approximately 91m (300 ft) in length drain the North Lobe of the Omega Mine, Seeps DEF which include discharges from several punch mines, a borehole into the central mine pool and PM21 which emanates from a collapsed punch mine opening that is believed to drain the southwest portion of the Omega Mine. The locations of these four major discharges are shown on Figure 1-1. These four discharges account for over 95 percent of the AMD treated by the WVDEP at the Omega Mine.

The flow from the Marshall House was by far the greatest AMD source, accounting for over 50 percent of the contaminant load of the Omega Mine water treated by the WVDEP. Seeps DEF flowed continuously and were the second greatest source of AMD. The borehole into the central mine pool was the third largest source of AMD; concentrations were higher than those of Seeps DEF, but it did not flow throughout the year. However, on several occasions the central pool flow rate was high enough to make it the largest source of AMD. PM21's AMD concentrations are lower than the other three major discharges but its consistent high flow rate made it a significant source of contaminant load. The injection of sludge from the AMD treatment system into the southwest mine pool was likely a factor in the high flow rate. It is interesting to note that




## Figure 7-2

## Monitoring Point Locations

## Table 7-6

## Monitoring Well Water Analyses

| | | | | | | | | | | | | ANALABS |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | OMW-2 | OMW-4 | | | | OMW-5 | | OMW-7 | OMW-8 | | | |
| | 5/17/00 | 8/26/99 | 2/16/00 | 5/17/00 | 8/30/00 | 5/17/00 | 8/30/00 | 8/26/99 | 8/26/99 | 2/16/00 | 5/17/00 | 2/16/00 |
| pH | 7 3 | 7 2 | 7 5 | 7 4 | 7 8 | 5 3 | 3 6 | 7 1 | 7 3 | 7 2 | 4 0 | 7 1 |
| Hot Acid | <1mg/lCaCO3 | <1mg/lCaCO3 | <1mg/lCaCO3 | <1mg/lCaCO3 | <1mg/lCaCO3 | 145mg/lCaCO3 | 997mg/lCaCO3 | <1mg/lCaCO3 | <1mg/lCaCO3 | <1mg/lCaCO3 | 202 mg/lCaCO3 | <0 20mg/l |
| Min Acid | 0mg/lCaCO3 | 0mg/lCaCO3 | 0mg/lCaCO3 | 0 mg/lCaCO3 | 0 mg/lCaCO3 | 0mg/lCaCO3 | 516mg/lCaCO3 | 0mg/lCaCO3 | 0 mg/lCaCO3 | 0mg/lCaCO3 | <1mg/lCaCO3 | — |
| ALK | 161 mg/lCaCO3 | 103 mg/lCaCO3 | 180 mg/lCaCO3 | 204 mg/lCaCO3 | 210 mg/lCaCO3 | 19mg/lCaCO3 | <1mg/lCaCO3 | 289mg/lCaCO3 | 95 mg/lCaCO3 | 227 mg/lCaCO3 | <1mg/lCaCO3 | 167 mg/l |
| Fe | 1 02 ppm | 9 10 ppm | — | 1 17 ppm | 5 13 ppm | 584 ppm | 598 ppm | 1 57 ppm | 7 83 ppm | — | 651 ppm | — |
| Fe-d | < 05 ppm | 1 03 ppm | 40 ppm | 45 ppm | < 05 ppm | 576 ppm | 169 ppm | 1 06 ppm | 3 56 ppm | 5 13 ppm | 562 ppm | 5 03 mg/l |
| Mn | 10 ppm | 30 ppm | 38 ppm | 58 ppm | 60 ppm | 11 9 ppm | 11 2 ppm | 82 ppm | .24 ppm | 5 87 ppm | 17 5 ppm | 2 97 mg/l |
| TDS | 576 ppm | 124 ppm | 134 ppm | 430 ppm | 438 ppm | 4650 ppm | — | 230 ppm | 127 ppm | 212 ppm | 4650 ppm | 241 mg/l |
| SO4 | 216 ppm | 13 ppm | 74 ppm | 143 ppm | 148 ppm | 3130 ppm | 2400 ppm | 92 ppm | 12 ppm | 2 ppm | 4080 ppm | 5 0 mg/l |
| Cl- | 4 4 ppm | 11 0 ppm | 14 9 ppm | 9 2 ppm | 8 7 ppm | 18 6 ppm | 14 6 ppm | 6 8 ppm | 3 0 ppm | 3 7 ppm | 9 2 ppm | 2 98 mg/l |
| Br | < 10 ppm | 50 ppm | < 10 ppm | < 10 ppm | 14 ppm | < 10 ppm | 18 ppm | < 10 ppm | 50 ppm | < 10 ppm | 2 05 ppm | — |
| NH3-N | .27 ppm | < 06 ppm | 26 ppm | < 06 ppm | < 06 ppm | 40 ppm | 62 ppm | < 06 ppm | 06 ppm | 17 ppm | 81 ppm | — |
| Fe+2 | 40 ppm | 1 42 ppm | 18 ppm | 12 ppm | 4 34 ppm | 576 ppm | 563 ppm | 9 22* ppm | 1 42 ppm | 5 02 ppm | 598 ppm | 5 03 mg/l |
| Fe+3 | 62 ppm | 7 68 ppm | 22 ppm | 46 ppm | 79 ppm | 8 00 ppm | 35 0 ppm | *** | 6 41 ppm | 85 ppm | 53 0 ppm | 85 12 mg/l |
| Sb | — | < 005 ppm | — | — | — | — | — | < 005 ppm | < 005 ppm | — | — | — |
| Al | 51 ppm | 4 70 ppm | 1 50 ppm | 32 ppm | 57 ppm | 1 69 ppm | 10 3 ppm | 1 37 ppm | 5 15 ppm | 64 2 ppm | 64 5 ppm | 10 24 mg/l |
| Ca | 78 3 ppm | 42 6 ppm | 56 1 ppm | 91 9 ppm | 99 1 ppm | 537 ppm | 635 ppm | 70 3 ppm | 36 7 ppm | 109 ppm | 370 ppm | 76 80 mg/l |
| Mg | 19 7 ppm | 8 05 ppm | 12 1 ppm | 19 1 ppm | 21 1 ppm | 147 ppm | 161 ppm | 9 06 ppm | 6 33 ppm | 28 7 ppm | 177 ppm | 17 10 mg/l |
| K | 4 18 ppm | 7 06 ppm | 4 16 ppm | 5 42 ppm | 4 75 ppm | 17 2 ppm | 17 8 ppm | 8 35 ppm | 4 92 ppm | 11 1 ppm | 22 6 ppm | 18 48 mg/l |
| Na | 72 4 ppm | 3 46 ppm | 43 9 ppm | 33 4 ppm | 29 0 ppm | 37 7 ppm | 32 0 ppm | 5 92 ppm | 2 28 ppm | 5 76 ppm | 36 0 ppm | — |
| As | < 001 ppm | 001 ppm | 002 ppm | < 001 ppm | 005 ppm | 0 001 | 007 ppm | < 001 ppm | < 001 ppm | 018 ppm | 009 ppm | 005 mg/l |
| Ba | < 10 ppm | 25 ppm | 19 ppm | 31 ppm | 31 ppm | < 10 ppm | 12 ppm | < 10 ppm | < 10 ppm | 1 14 ppm | .23 ppm | 0 630 mg/l |
| Be | < 005 ppm | < 005 ppm | < 005 ppm | < 005 ppm | < 005 ppm | < 005 ppm | < 005 ppm | < 005 ppm | < 005 ppm | <0 005 ppm | < 005 ppm | 0 004 mg/l |
| Cd | < 005 ppm | < 005 ppm | < 005 ppm | < 005 ppm | < 005 ppm | 103 ppm | 154 ppm | < 005 ppm | < 005 ppm | 026 ppm | 121 ppm | 0 001 mg/l |
| Co | < 10 ppm | < 10 ppm | < 10 ppm | < 10 ppm | < 10 ppm | 0 3 | < 10 ppm | < 10 ppm | < 10 ppm | < 10 ppm | 59 ppm | — |
| Cr | < 05 ppm | < 05 ppm | < 05 ppm | < 05 ppm | < 05 ppm | < 05 ppm | < 05 ppm | < 05 ppm | < 05 ppm | 08 ppm | < 05 ppm | 0 013 mg/l |
| Cu | < 02 ppm | < 02 ppm | < 02 ppm | < 02 ppm | < 02 ppm | < 02 ppm | 37 ppm | < 02 ppm | < 02 ppm | 06 ppm | 09 ppm | 0 002 mg/l |
| Pb | 028 ppm | 0 012 | 0 003 | 014 ppm | 014 ppm | 002 ppm | 020 ppm | 002 ppm | 010 ppm | 089 ppm | 019 ppm | 023 mg/l |
| Ni | < 04 ppm | < 04 ppm | < 04 ppm | < 04 ppm | < 04 ppm | 72 ppm | 86 ppm | < 04 ppm | < 04 ppm | 10 ppm | 1 14 ppm | 0 016 mg/l |
| Se | < 002 ppm | < 002 ppm | < 002 ppm | < 002 ppm | < 002 ppm | < 002 ppm | < 002 ppm | < 002 ppm | < 002 ppm | < 002 ppm | < 002 ppm | 0 005 mg/l |
| Zn | 040 ppm | 0 09 | 019 ppm | 040 ppm | 040 ppm | 1 06 ppm | 3 78 ppm | < 005 ppm | 019 ppm | 325 ppm | 3 82 ppm | 0 020 mg/l |

All of the above tests were conducted by Sturm Environmental Services, except for a duplicate sample of OMW-8 from 2-16-00 conducted by Analabs, Inc

*Due to high solids content in sample, we were unable to accurately determine ferrous iron

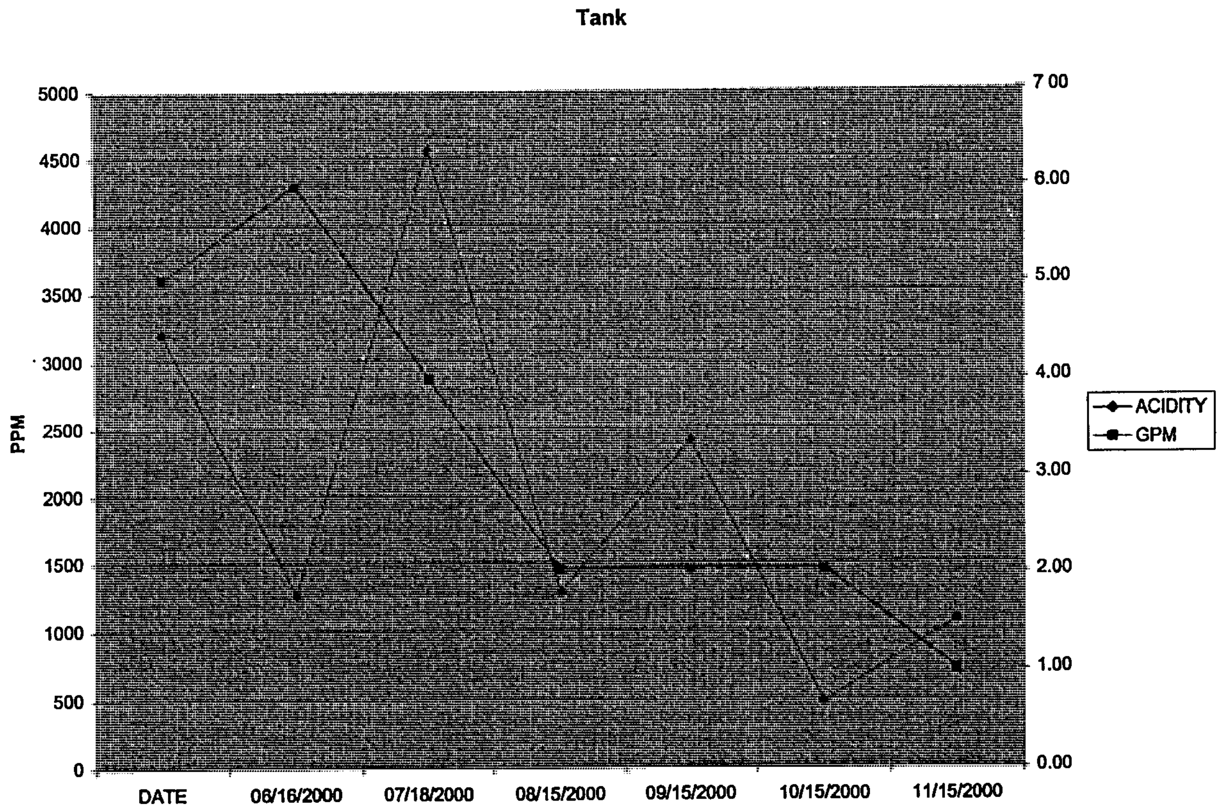



**Figure 7-3**

**Flow and Acidity for Combined Collected Seeps Above Cobun Creek (from WVDEP)**

the injected water and sludge were alkaline, but the discharge from PM21 had a low pH with elevated concentrations of dissolved iron and aluminum. Table 7-7 summarizes the median flows and AMD parameters of the four major discharges and of the intake of the AMD treatment plant prior to grouting. Table 7-8 shows the percentages of flow, acidity, total Fe and Al to the total treated by the WVDEP.

**Table – 7-7**
**Omega Mine Site: Pre-Grouting Flow and Water Quality**

| Median AMD Parameters Flow in gpm; pH in std. Units; others in mg/L | | | | | | |
|---|---|---|---|---|---|---|
| **Source** | **Flow** | **PH** | **Acidity** | **Total Fe** | **Al** | **SO4** |
| Marshall | 17 | 2.74 | 3689 | 1238 | 215 | 4672 |
| DEF | 15 | 2.72 | 1808 | 477 | 107 | 2398 |
| Central | 13 | 2.79 | 2250 | 689 | 139 | 3020 |
| PM21 | 22 | 3.01 | 644 | 139 | 42 | 2472 |

| Median AMD Parameters Flow in gpm; pH in std. Units; others in mg/L | | | | | | |
|---|---|---|---|---|---|---|
| Source | Flow | PH | Acidity | Total Fe | Al | SO4 |
| Treatment | 70 | 2.83 | 1746 | 549 | 108 | 3061 |

(unpublished data obtained from U.S. DOE)

**Table –7-8**
**Omega Mine Site: Pre-Grouting Contribution of Major Sources to Total AMD Load**

| Source | Flow | Acidity | Total Fe | Al |
|---|---|---|---|---|
| Marshall | 28% | 55% | 60% | 52% |
| DEF | 23% | 21% | 19% | 22% |
| Central | 14% | 17% | 18% | 17% |
| PM21 | 32% | 12% | 8% | 13% |

(unpublished data obtained from U.S. DOE)

It was anticipated that filling the north lobe with alkaline grout would result in a 50 percent or greater reduction in AMD loading even though flows from updip sections of the mine would increase at other discharge points.

Table 7-9 presents average daily mass loadings of select parameters for the pre-injection period.

**Table –7-9**
**Average Daily Mass Loadings of Select Parameters, Pre-Injection Water Quality Data**

| Location | Avg. Flow gpm | Acidity kg/Day | Fe Total kg/Day | SO4 kg/Day | Al kg/Day | Mn kg/Day |
|---|---|---|---|---|---|---|
| Marshall House | 22.6 | 596.3 | 185.9 | 707.2 | 32.2 | 0.7 |
| DEF | 20.9 | 197.4 | 56.1 | 270.1 | 11.6 | 0.3 |
| Central Pool | 9.2 | 192.8 | 58.2 | 249.8 | 11.3 | 0.3 |
| PM21 | 28.1 | 94.0 | 18.7 | 365.2 | 6.3 | 0.7 |
| Treatment | 92.6 | 1066.9 | 333.8 | 1667.0 | 63.3 | 2.3 |
| CobunCreek | 4311.8 | 270.3 | 3.6 | 431.9 | 4.0 | 1.7 |

*Indicates WVDEP monitoring point downstream of the Omega Mine.

- Loading values expressed as kg/day

- Average Mass Loadings are based on parameter loadings calculated for each specific monitoring event
- Acidity loadings expressed as kg/day of $CaCO_3$ equivalent (WVDEP monitoring points = Hot Acidity; USDOE monitoring points = Net Acidity)

Figures 7-4 through 7-7 show flows at the Marshall House, Seeps DEF, Central Pool and PM21 from 1993 through October 2000, respectively. The monthly flows from 1993 and 1998 are averaged as are the 1999 and 2000 monthly flows. Injection of alkaline grout into the North Lobe occurred from May 1998 until November 1998. Thus, the average flows can be compared for the pre-injection (1993-1998) and post-injection (1999-2000) periods. Comparisons may be impacted by the drought in much of 1999. Figure 7-8, which shows the flow of Cobun Creek for the same time periods, indicates the effects of the drought where grouting of the North Lobe is not a factor. Flows in Cobun Creek prior to 1999 (Figure 7-8) are consistently greater than in 1999 and 2000.

In computing average values zero values were included. However, for blank cells within the figures it was assumed monitoring was not conducted and these time periods were not included in averaging the data.

| Marshall House Combined State & Federal Data | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| 1993 | | 39.6 | 20.5 | 66.6 | 57.7 | 17.0 | 19.0 | 18.6 | 8.8 | 6.4 | 7.6 | 10.2 |
| 1994 | 27.9 | | | | 49.3 | 27.9 | 12.8 | 11.5 | 6.3 | 8.2 | 8.2 | 7.6 |
| 1995 | 7.7 | 10.0 | 10.0 | 11.3 | 36.2 | 36.8 | 36.8 | 10.7 | 7.5 | 6.1 | 5.6 | |
| 1996 | 11.2 | 23.4 | 64.3 | | 56.4 | 31.0 | 15.1 | 19.2 | | 17.1 | 32.8 | 57.7 |
| 1997 | 13.3 | 9.5 | 65.2 | | 40.6 | | 17.7 | | 9.5 | | 7.1 | |
| 1998 | 23.4 | 53.7 | 78.1 | 63.5 | 52.4 | 71.2 | [illegible] | 35.6 | 16.9 | 3.9 | [illegible] | 2.1 |
| Average | 16.7 | 27.2 | 47.6 | 47.1 | 48.7 | 36.8 | 24.8 | 19.1 | 9.8 | 8.3 | 10.7 | 19.4 |
| 1999 | [illegible] | 7.1 | 9.0 | 7.9 | 6.5 | 4.4 | 3.6 | 4.0 | [illegible] | [illegible] | 1.8 | [illegible] |
| 2000 | 3.0 | [illegible] | 6.0 | [illegible] | [illegible] | [illegible] | 6.0 | 7.0 | 4.0 | [illegible] | | |
| Average | 4.9 | 6.0 | 7.5 | 8.4 | 6.7 | 5.2 | 5.8 | 5.5 | 3.6 | 2.4 | 1.8 | 7.2 |

| Marshall House Average Flows Pre- and Post-Grouting | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| Pre-1999 | 16.7 | 27.2 | 47.6 | 47.1 | 48.7 | 36.8 | 24.8 | 19.1 | 9.8 | 8.3 | 10.7 | 19.4 |
| 1999 to Date | 4.9 | 6.0 | 7.5 | 8.4 | 6.7 | 5.2 | 5.8 | 5.5 | 3.6 | 2.4 | 1.8 | 7.2 |

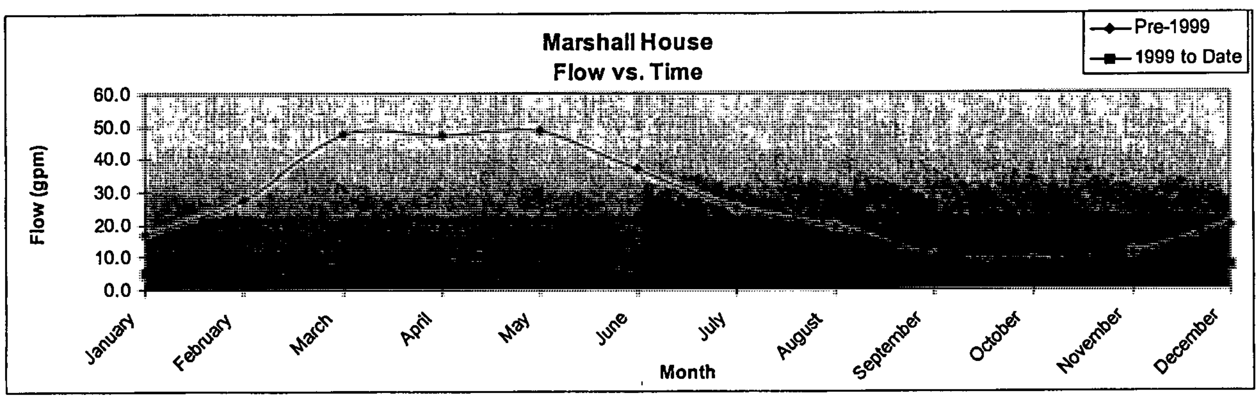



**Figure 7-4**

**Marshall House – Flow vs Time**

| DEF<br>Combined State (DEF39 Punch Mine) & Federal Data (DEF, Loc. ID#6) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| 1993.0 | | | 53.2 | 60.5 | 36.4 | 24.4 | 7.5 | 6.2 | 4.2 | 7.4 | 9.7 | 31.1 |
| 1994.0 | 75.4 | | | | 32.6 | 20.7 | 9.9 | 8.6 | 13.0 | 9.9 | 4.5 | 17.0 |
| 1995.0 | 15.4 | 10.6 | 13.2 | 8.9 | 14.4 | 12.3 | 7.1 | 6.4 | 3.6 | 3.1 | 13.2 | |
| 1996.0 | 19.3 | 31.3 | 32.6 | 16.0 | 45.3 | 17.2 | 15.1 | 9.3 | 12.3 | 15.8 | 16.3 | 20.5 |
| 1997.0 | 15.4 | 13.2 | 18.9 | 19.7 | 22.3 | 17.9 | 10.7 | 9.0 | 45.3 | 7.6 | 13.4 | 12.5 |
| 1998.0 | 26.7 | 25.6 | 29.1 | 26.4 | 18.3 | 11.9 | 12.5 | 9.9 | 8.3 | 6.2 | 1.3 | 2.1 |
| Average | 30.4 | 20.2 | 29.4 | 26.3 | 28.2 | 17.4 | 10.5 | 8.2 | 14.4 | 8.4 | 9.8 | 16.7 |
| 1999.0 | 39.9 | 17.4 | 21.0 | 11.9 | 7.5 | 5.1 | 4.0 | 3.1 | 1.8 | 1.8 | 1.8 | 5.8 |
| 2000.0 | 5.0 | 6.0 | 20.0 | 32.0 | 13.0 | 12.0 | 15.0 | 9.0 | 5.0 | 6.0 | | |
| Average | 22.4 | 11.7 | 20.5 | 21.9 | 10.2 | 8.6 | 10.0 | 6.1 | 3.4 | 3.9 | 1.8 | 5.8 |

| DEF<br>Average Flows Pre-and Post-Grouting | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| Pre-1999 | 30.4 | 20.2 | 29.4 | 26.3 | 28.2 | 17.4 | 10.5 | 8.2 | 14.4 | 8.4 | 9.8 | 16.7 |
| 1999 to Date | 22.4 | 11.7 | 20.5 | 21.9 | 10.2 | 8.6 | 10.0 | 6.1 | 3.4 | 3.9 | 1.8 | 5.8 |

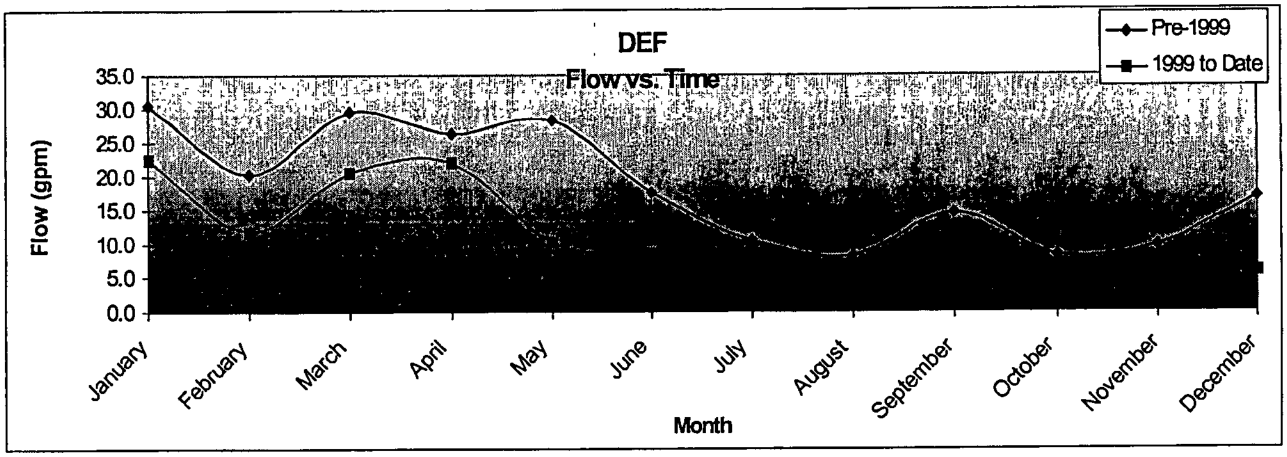



**Figure 7-5**
**DEF – Flow vs. Time**

| Central Pool<br>Combined State (815 H Manhole) & Federal (Loc. ID#8) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| 1993.0 | | | 11.0 | 10.7 | 17.0 | 10.1 | | | | | | 11.4 |
| 1994.0 | 10.3 | | | | 7.6 | 4.9 | 0.9 | | | | | |
| 1995.0 | 10.7 | 18.4 | 23.5 | 9.7 | 17.4 | 20.2 | | | | | | |
| 1996.0 | 21.0 | 13.3 | 30.5 | 14.9 | 24.2 | 23.9 | 12.0 | 13.0 | 18.8 | 18.0 | 28.1 | 36.5 |
| 1997.0 | 15.9 | 7.6 | 4.4 | 13.9 | 20.1 | 9.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 16.6 |
| 1998.0 | 26.7 | 29.8 | 42.4 | 27.5 | 31.5 | 9.0 | 15.7 | 0.4 | 0.3 | 0.0 | 0.0 | 0.0 |
| Average | 16.9 | 17.3 | 22.4 | 15.3 | 19.6 | 12.8 | 7.1 | 4.4 | 6.4 | 6.0 | 9.4 | 16.1 |
| 1999.0 | 19.7 | 23.0 | 38.6 | 22.7 | 8.1 | 0.0 | 0.0 | | 0.0 | 0.0 | 0.0 | 0.0 |
| 2000.0 | 2.0 | 0.9 | 22.0 | 39.0 | 14.0 | 23.0 | 20.0 | 10.0 | 0.0 | 0.0 | | |
| Average | 10.9 | 11.9 | 30.3 | 30.9 | 11.0 | 11.5 | 10.0 | 10.0 | 0.0 | 0.0 | 0.0 | 0.0 |

| Central Pool<br>Average Flows Pre- and Post-Grouting | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| Pre-1999 | 16.9 | 17.3 | 22.4 | 15.3 | 19.6 | 12.8 | 7.1 | 4.4 | 6.4 | 6.0 | 9.4 | 16.1 |
| 1999 to Date | 10.9 | 11.9 | 30.3 | 30.9 | 11.0 | 11.5 | 10.0 | 10.0 | 0.0 | 0.0 | 0.0 | 0.0 |

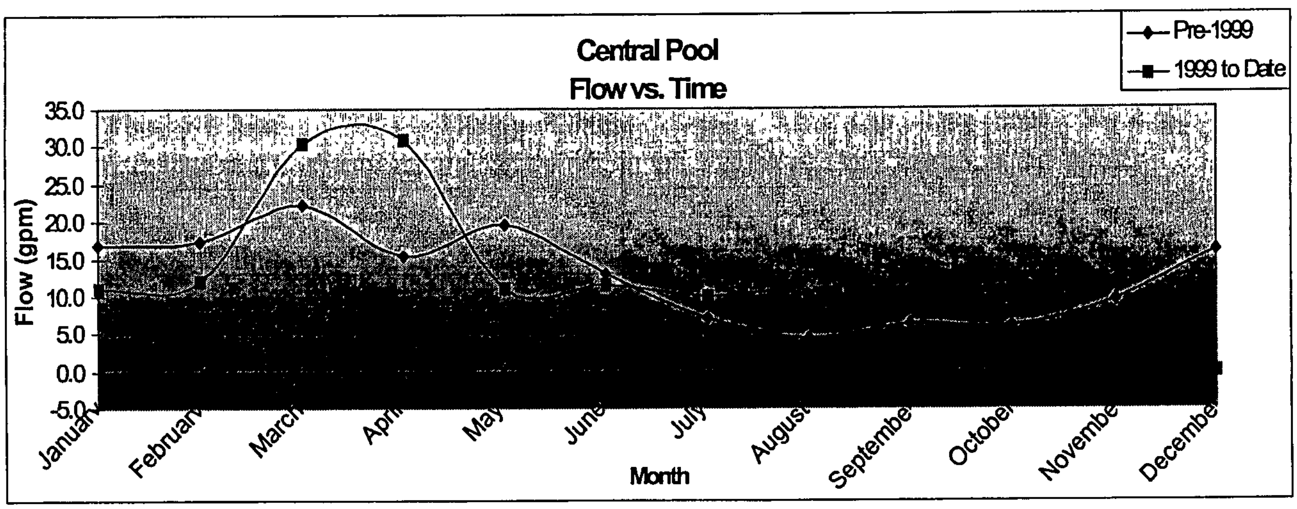



**Figure 7-6**
**Central Pool – Flow vs Time**

| Punch Mine 21 (PM 21) Combined State (Punch Mine 21) & Federal (PM 21 Loc. ID #18) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| 1993.0 | | | 115.6 | 48.2 | 31 6 | 27.4 | 16 7 | 18.0 | 2.9 | 20.2 | 25 4 | 17.2 |
| 1994.0 | 84 6 | | | | 14.7 | 29.0 | 36 5 | 19.5 | 21.4 | 15.7 | 8.6 | 15.8 |
| 1995.0 | 19.6 | 10 6 | 9 9 | 36 6 | 25.4 | 16.1 | | 11.5 | 9.8 | 7 3 | 10 0 | |
| 1996.0 | 11.7 | 21.9 | 94 7 | 32.3 | 50 6 | 26.4 | 21.3 | 22.2 | 20 2 | 117.0 | 29 8 | 39 8 |
| 1997.0 | 26.5 | 22.3 | 61 2 | 17 9 | 56.1 | 13 9 | 9.1 | 6 7 | 6.3 | 2.7 | 27.1 | 13.9 |
| 1998.0 | 104.7 | 21 5 | 42.9 | 27.5 | 21.2 | 18.3 | 17 9 | 13.4 | 10.8 | 5.6 | 3 6 | 3.2 |
| Average | 49.4 | 19 1 | 64.9 | 32.5 | 33.3 | 21.9 | 20.3 | 15.2 | 11.9 | 28.1 | 17.4 | 18.0 |
| 1999.0 | 26 9 | 25.3 | 29.7 | 24.3 | 13 8 | 9 0 | 4.0 | 3 1 | 1 8 | 1 8 | 0 9 | 7 2 |
| 2000.0 | 8.0 | 6 0 | 30 0 | 46.0 | 25 0 | 23 0 | 25.0 | 19 0 | 10 0 | 7 0 | | |
| Average | 17 5 | 15.7 | 29.8 | 35.2 | 21.9 | 16.0 | 14.5 | 11.1 | 5.9 | 4.4 | 0 9 | 7.2 |
| Punch Mine 21 (PM 21) Average Flows Pre- and Post-Grouting | | | | | | | | | | | | |
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| Pre-1999 | 49.4 | 19.1 | 64 9 | 32.5 | 33.3 | 21.9 | 20.3 | 15.2 | 11.9 | 28.1 | 17.4 | 18.0 |
| 1999 to Date | 17.5 | 15.7 | 29 8 | 35.2 | 21.9 | 16.0 | 14.5 | 11.1 | 5.9 | 4.4 | 0 9 | 7.2 |

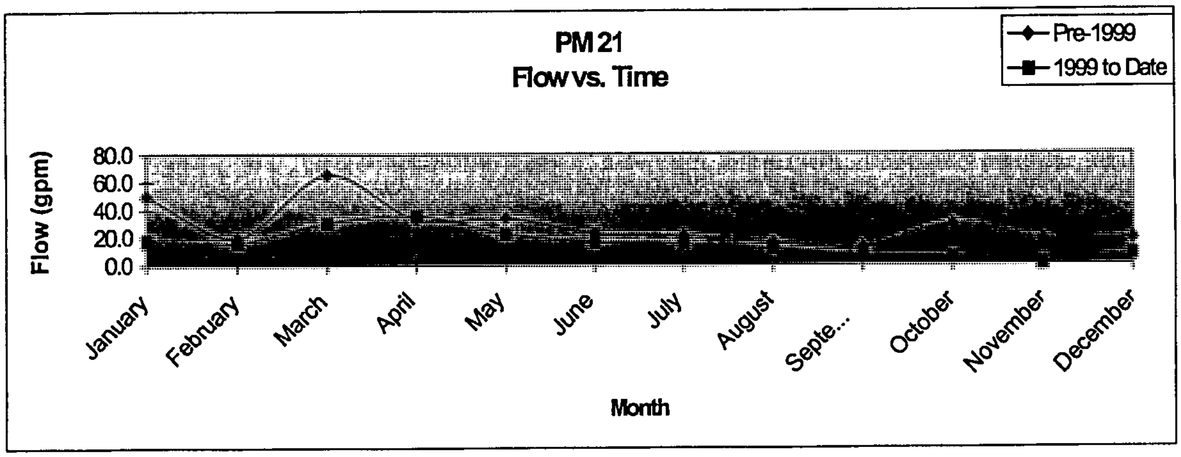



**Figure 7-7**
**PM21 – Flow vs Time**

| Cobun Creek DEP Location # 15 (Cobun Creek at Bridge) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| 1993.0 | | | | | | | | | | | | |
| 1994.0 | | | | | | | | | | | | |
| 1995.0 | | | | | | | | | | | | |
| 1996 0 | | | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 1997.0 | [illegible] | | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 1998.0 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| Average | 5738.9 | 4990.7 | 5822.1 | 4192.2 | 6820.4 | 4658.0 | 1613.6 | 4400.1 | 1214.3 | 11912.8 | 3826.2 | 3659.9 |
| 1999.0 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 2000.0 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | | |
| Average | 4494.0 | 4744.4 | 2900.0 | 2921.5 | 3596.7 | 1903.6 | 1705.1 | 1104.0 | 600.0 | 440.1 | 134.9 | 2800.0 |

| Cobun Creek DEP Location # 15 (Cobun Creek at Bridge) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| Pre-1999 | 5738.9 | 4990.7 | 5822.1 | 4192.2 | 6820.4 | 4658.0 | 1613.6 | 4400.1 | 1214.3 | 11912.8 | 3826.2 | 3659.9 |
| 1999 to Date | 4494.0 | 4744.4 | 2900.0 | 2921.5 | 3596.7 | 1903.6 | 1705.1 | 1104.0 | 600.0 | 440.1 | 134.9 | 2800.0 |

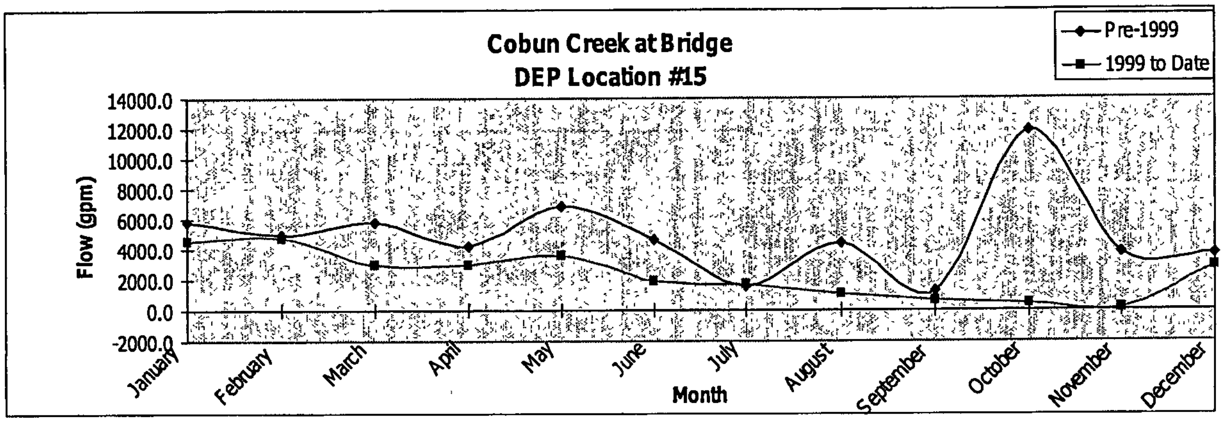



**Figure 7-8**

**Cobun Creek at Bridge – Flow vs Time**

Average pre-injection flows at the Marshall House in the Spring (March-May) were typically 181 liters per minute (48 gpm) while post-injection average flows do not exceed 31.7 liters per minute (8.4 gpm) and are generally less than 26 liters per minute (7 gpm) (Figure 7-4). A greatly reduced flow was anticipated since the Marshall House flow is from the now filled North Lobe. Seeps DEF (Figure 7-5) also show reduced average flows in the post injection period. This could be the result of the drought, but the North Lobe may also be a source of some of the flow from Seeps DEF. If it is, reduced post injection flows would be expected. The central pool post injection average flows in part exceed pre-injection average flows (Figure 7-6). Greater flows were anticipated since water flowing down dip through the mine to the North Lobe should be diverted to the central pool. PM 21 (Figure 7-7) exhibits lower average flows in the post injection period. This is not unexpected since the southwest portion of the Omega Mine is not likely to gain water diverted from filling the North Lobe. Also, reinjection of sludge from the WVDEP treatment plant into the southwest portion of the mine ceased late in 1998.

Figures 7-9 through 7-13 show the net acidity at the Marshall House, Seeps DEF, Central Pool, PM21 and Cobun Creek for the same pre- (1993-1998) and post- (1999-2000) injection periods. The data are averages as described above for the flows. Average net acidity of the discharge at the Marshall House (Figure 7-9) is generally greater in the post injection period. The peak in the May post injection data is due to net a acidity value of 47,000 mg/l in May 1999. The peaks present in August and November of the post- injection period may be related to very small flows. Figure 7-14 shows both flow and acidity from May 1998 through 2000 for the Marshall House. This is the same data presented on Figures 7-4 and 7-9. There is no obvious relationship between flow and acidity. Figure 7-15 shows flow and acidity at the WVDEP treatment plant inlet for the same time period. In this case there is a clear relationship between low flow and high acidity and vice versa. The unchanged or increased net acidity at the Marshall House is difficult to explain in view of the large quantity of alkaline grout injected into the North Lobe.

The average net acidities at Seeps DEF show no significant difference for pre- and post- injection between January and July (Figure 7-10). The peaks in August and October during the post-

**Marshall Pool Net Acidity**
**Combined State (Marshall House) and Federal (DOE Location #1) Data**

| Month | January | February | March | April | May | June | July | August | September | October | November | December |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1993 | | 6450 0 | 6173 5 | 6897 7 | 5244 0 | 4846 7 | 4632 5 | 3603 0 | 3388 5 | 3557 0 | 3371 0 | 4446 0 |
| 1994 | 6466 0 | | | | 3927 0 | 3843 0 | 3699 0 | 3353 0 | 3872 0 | 3825 0 | 3496 0 | 3535 0 |
| 1995 | 3689 0 | 3993 0 | 4214 0 | 5103 0 | 4265 0 | 4725 0 | 4130 0 | 3627 0 | 3502 0 | 3200 0 | 3054 0 | |
| 1996 | 3856 0 | 6941 0 | 5343 0 | 3685 0 | | 3460 0 | 3222.0 | 3345 0 | | 3070 0 | 3499 0 | 3628.0 |
| 1997 | 2153 0 | 3424 0 | 3625.0 | | 2672 0 | | 2799.0 | | 2375.0 | | 2235.0 | |
| 1998 | 3170 0 | 4451.0 | 3430 0 | 2859 0 | 2507 5 | 2420.0 | 3970 0 | 2751 3 | 4902 5 | 7955.0 | 4930 0 | 4590 0 |
| Average | 3866.8 | 5051 8 | 4557.1 | 4636 2 | 3723 1 | 3858.9 | 3742 1 | 3335 9 | 3608.0 | 4321 4 | 3430 8 | 4049 8 |
| 1999 | 9710 0 | 6785 0 | 9293.5 | 10613.3 | 47000 0 | 7220 0 | 4830 0 | 17300 0 | 6310 0 | 9770 0 | 10100 0 | 4920 0 |
| 2000 | 4610 0 | 3830 0 | 1890 0 | 5890 0 | 4850 0 | 4010 0 | 4610 0 | 5070 0 | 3160 0 | 1480 0 | | |
| Average | 7160.0 | 5307.5 | 5591.8 | 8251.7 | 25925.0 | 5615.0 | 4720.0 | 11185 0 | 4735 0 | 5625 0 | 10100 0 | 4920 0 |

**Marshall Pool**
**Pre- and Post-Grouting Average Net Acidity**

| Month | January | February | March | April | May | June | July | August | September | October | November | December |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Pre-1999 | 3866 8 | 5051.8 | 4557.1 | 4636 2 | 3723.1 | 3858.9 | 3742 1 | 3335 9 | 3608 0 | 4321 4 | 3430 8 | 4049 8 |
| 1999 to date | 7160 0 | 5307 5 | 5591.8 | 8251.7 | 25925.0 | 5615 0 | 4720 0 | 11185.0 | 4735 0 | 5625 0 | 10100 0 | 4920.0 |

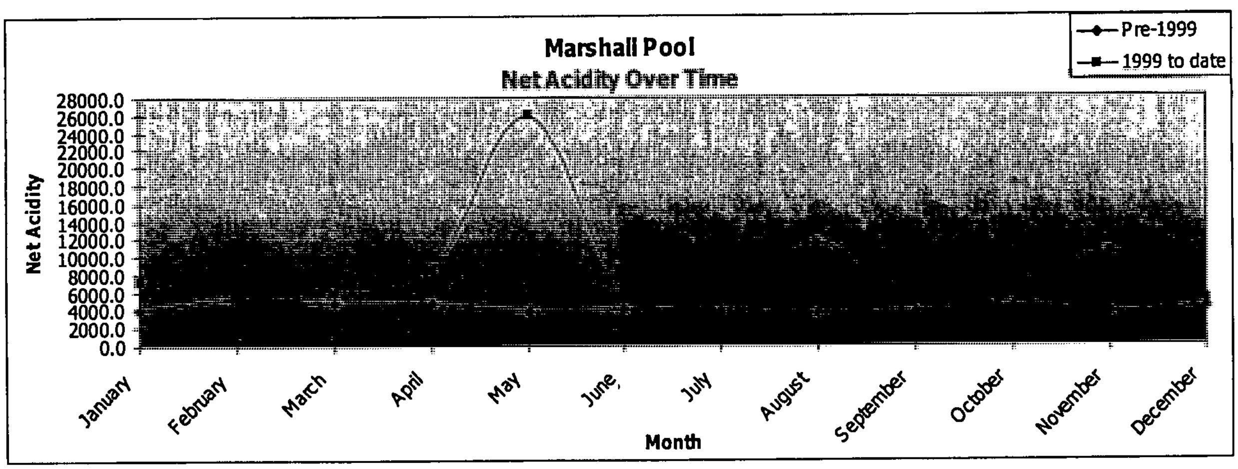



**Figure 7-9**
**Marshall Pool – Net Acidity Over Time**

| DEF Combined State (DEF39 Punch Mine) & Federal Data (DEF, Loc. ID #5) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| 1993 | | | 1692.0 | 2314.0 | 2471.0 | 2407.7 | 2442.5 | 2624.0 | 2622.0 | 2117 0 | 1830 0 | 1424.5 |
| 1994 | 727.0 | | | | 2389 0 | 2312.0 | 2343.0 | 1885.0 | 1696.0 | 2548.0 | 2348.0 | 1172.0 |
| 1995 | 1941.0 | 1616.0 | 1314 0 | 1895.0 | 140 0 | 819.7 | 2280 0 | 2651.0 | 2658.0 | 2424.0 | 1469.0 | |
| 1996 | 1359.0 | 3009.5 | 1741.5 | 1970 0 | 1485 0 | 1744.7 | 1872.7 | 2123.0 | 3375 0 | 1098.3 | 822.0 | 781.5 |
| 1997 | 995.4 | 988.0 | 1963 0 | 2340 0 | 3760.5 | 1710 0 | 2050 5 | 2300 0 | 1556.5 | 2160 0 | 1534 0 | 1470 0 |
| 1998 | 1535.5 | 991.0 | 935.6 | 1931.2 | 1902.5 | 1405.0 | 1650 0 | 2108.5 | 1998.5 | 2180.0 | 2120 0 | 1985.0 |
| Average | 1311 6 | 1651.1 | 1529 2 | 2090.0 | 2024.7 | 1733.2 | 2106 4 | 2281.9 | 2317.7 | 2087 9 | 1687.2 | 1366.6 |
| 1999 | 537 0 | 1565.0 | 951 8 | 1616.5 | 2010 0 | 1840 0 | 2140 0 | 7480 0 | 2700 0 | 8900.0 | 2020 0 | 1380 0 |
| 2000 | 1690 0 | 2020 0 | 435 0 | 806 0 | 1680 0 | 1260 0 | 1400 0 | 48 0 | 2130 0 | 2230.0 | | |
| Average | 1113 5 | 1792.5 | 693.4 | 1211.3 | 1845.0 | 1550.0 | 1770.0 | 3764.0 | 2415.0 | 5565.0 | 2020 0 | 1380.0 |

| DEF Pre- and Post-Grouting Average Net Acidity | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| Pre-1999 | 1311.6 | 1651.1 | 1529 2 | 2090 0 | 2024 7 | 1733.2 | 2106.4 | 2281.9 | 2317.7 | 2087.9 | 1687.2 | 1366.6 |
| 1999 to date | 1113.5 | 1792.5 | 693.4 | 1211.3 | 1845.0 | 1550 0 | 1770 0 | 3764.0 | 2415 0 | 5565 0 | 2020.0 | 1380.0 |

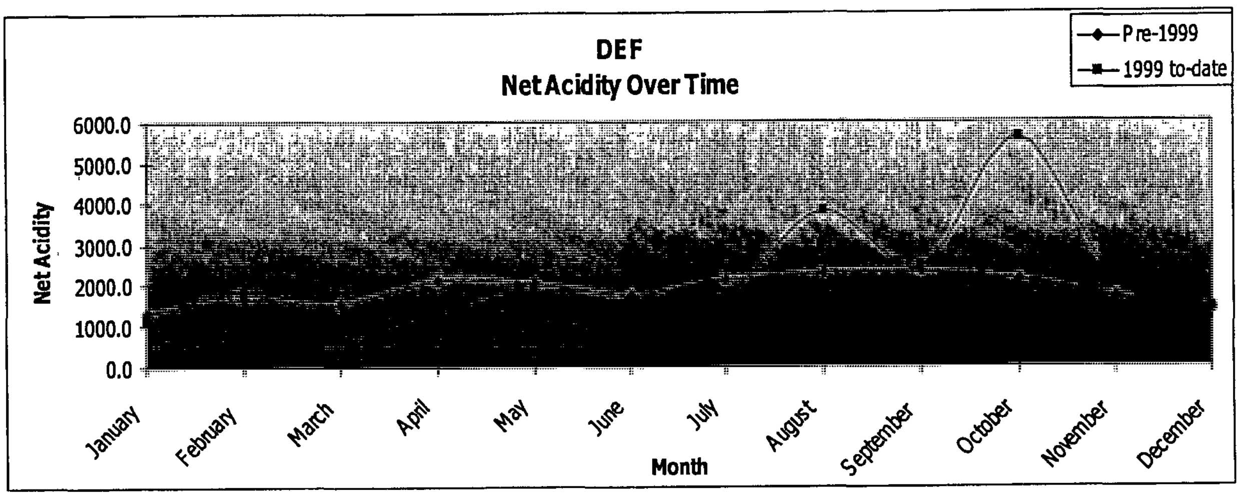



**Figure 7-10**
**DEF – Net Acidity Over Time**

| Central Pool<br>Combined State (815 HI Manhole) & Federal (Loc. ID #8) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| 1993 | | | 2928.5 | 3305.0 | 2712.0 | 2436.5 | | | | | | 2900.0 |
| 1994 | 2989.0 | | | | 2514.0 | 2410.0 | 1937.0 | | | | | |
| 1995 | 1302.0 | 2385.0 | 2371.0 | 2302.0 | 1921.0 | 2509.0 | | | | | | |
| 1996 | 2522.5 | 3881.5 | 4317.5 | 3460.7 | 2965.0 | 3053.7 | 1419.5 | 2966.0 | 4030.0 | 2281.0 | 1594.5 | 2933.0 |
| 1997 | 2403.0 | 1763.0 | 2944.5 | 2050.0 | 2584.5 | 1830.0 | | | | | | 1690.0 |
| 1998 | 2747.0 | 2620.0 | 2126.0 | 2725.5 | 3240.0 | 1330.0 | 2050.0 | 1653.5 | 761.5 | | | |
| Average | 2392.7 | 2662.4 | 2937.5 | 2768.6 | 2656.1 | 2261.5 | 1802.2 | 2309.8 | 2395.8 | 2281.0 | 1594.5 | 2507.7 |
| 1999 | 2240.0 | 3141.5 | 2897.3 | 2509.0 | 3660.0 | | | | | | | |
| 2000 | 1000.0 | 745.0 | 4020.0 | 2870.0 | 3030.0 | 1890.0 | 2740.0 | 4650.0 | | | | |
| Average | 1620.0 | 1943.3 | 3458.6 | 2689.5 | 3345.0 | 1890.0 | 2740.0 | 4650.0 | | | | |

| Central Pool<br>Pre- and Post-Grouting Average Net Acidity | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| Pre-1999 | 2392.7 | 2662.4 | 2937.5 | 2768.6 | 2656.1 | 2261.5 | 1802.2 | 2309.8 | 2395.8 | 2281.0 | 1594.5 | 2507.7 |
| 1999 to date | 1620.0 | 1943.3 | 3458.6 | 2689.5 | 3345.0 | 1890.0 | 2740.0 | 4650.0 | | | | |

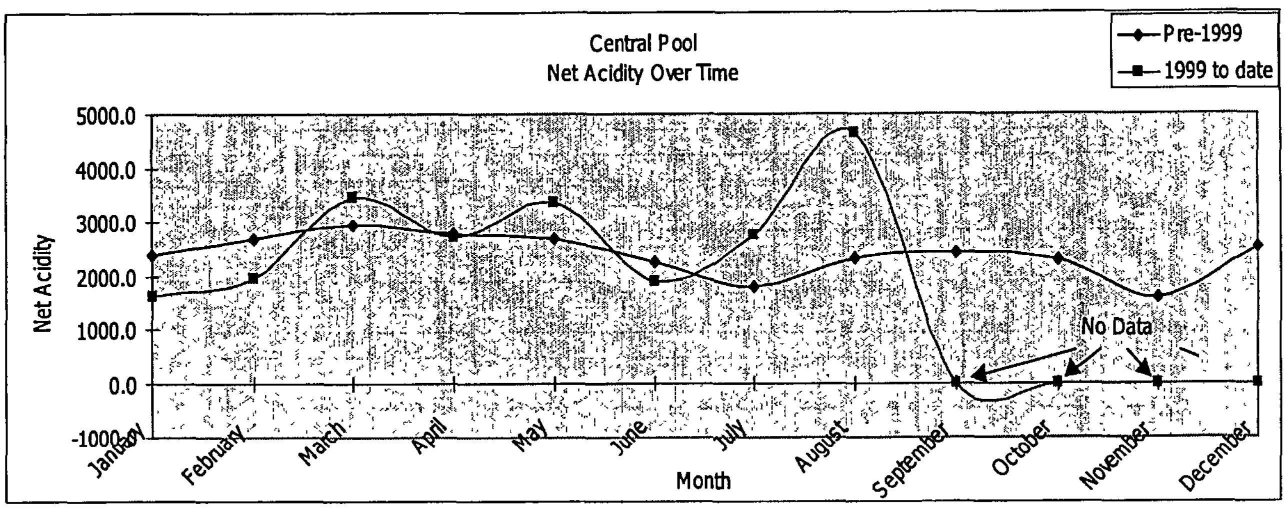



**Figure 7-11**
**Central Pool – Net Acidity Over Time**

| Punch Mine 21 (PM21) Combined State (Punch Mine 21) & Federal (PM21 Loc. ID #18) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| 1993 | | 573.0 | 493.0 | 554.3 | 862.0 | 853.7 | 896.5 | 827.0 | 776.0 | 604.5 | 569.0 | 472.0 |
| 1994 | 508.0 | | | | 533.0 | 754.0 | 782.0 | 644.0 | 530.0 | 721.0 | 696.0 | 656.0 |
| 1995 | 705.0 | 748.0 | 596.0 | 644.0 | 663.0 | 570.0 | 653.0 | 741.0 | 759.0 | 753.0 | 613.0 | |
| 1996 | 772.0 | 759.0 | 448.0 | 597.5 | 350.0 | 698.7 | 757.7 | 846.7 | 734.0 | 763.1 | 592.5 | 524.5 |
| 1997 | 326.5 | 467.0 | 407.0 | 482.0 | 548.6 | 651.0 | 846.4 | 964.0 | 1257.5 | 1530.0 | 707.0 | 1350.0 |
| 1998 | 685.8 | 807.0 | 662.8 | 1187.2 | 822.0 | 807.0 | 1330.0 | 1184.0 | 1081.5 | 1570.0 | 1500.0 | 1245.0 |
| Average | 599.5 | 670.8 | 521.4 | 693.0 | 629.8 | 722.4 | 877.6 | 867.8 | 856.3 | 990.3 | 779.6 | 849.5 |
| 1999 | 539.0 | 1410.0 | 1622.0 | 1848.0 | 2970.0 | 1340.0 | 1400.0 | 2610.0 | 1470.0 | 1550.0 | 1730.0 | 1680.0 |
| 2000 | 1870.0 | 1770.0 | 1390.0 | 1180.0 | 1500.0 | 2130.0 | 1330.0 | 5990.0 | 1600.0 | 1620.0 | | |
| Average | 1204.5 | 1590.0 | 1506.0 | 1514.0 | 2235.0 | 1735.0 | 1365.0 | 4300.0 | 1535.0 | 1585.0 | 1730.0 | 1680.0 |

| Punch Mine 21 (PM21) Pre- and Post-Grouting Average Net Acidity | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| Pre-1999 | 599.5 | 670.8 | 521.4 | 693.0 | 629.8 | 722.4 | 877.6 | 867.8 | 856.3 | 990.3 | 779.6 | 849.5 |
| 1999 to date | 1204.5 | 1590.0 | 1506.0 | 1514.0 | 2235.0 | 1735.0 | 1365.0 | 4300.0 | 1535.0 | 1585.0 | 1730.0 | 1680.0 |

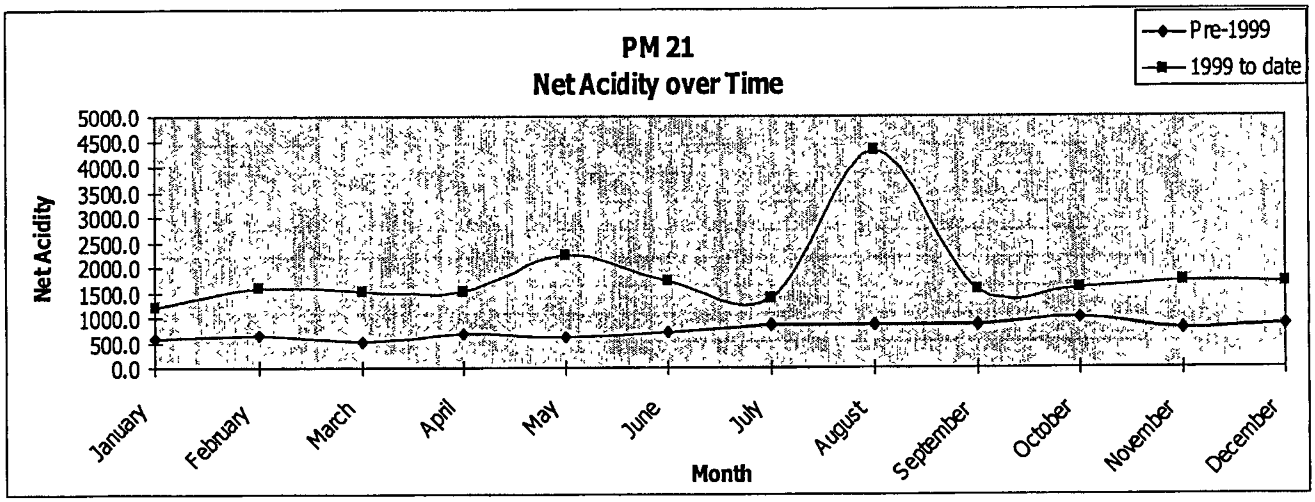



**Figure 7-12**
**PM-21 – Net Acidity Over Time**

| Cobun Creek DEP Location # 15 (Cobun Creek at Bridge) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| 1993 | | | | | | | | | | | | |
| 1994 | | | | | | | | | | | | |
| 1995 | | | | | | | | | | | | |
| 1996 | | | 19 5 | 5 5 | 7 0 | 11 5 | 5 0 | 15 5 | 12 0 | 7 0 | 9 0 | 15 0 |
| 1997 | 2 0 | | 25 6 | 20 0 | 16 0 | 0 0 | 10 0 | 2 0 | 12 0 | 0 0 | 0 0 | 4 0 |
| 1998 | 4 0 | 0 0 | 1 0 | 5 0 | 5 0 | 0 0 | 0 6 | 3 0 | 0 6 | 0 0 | 0 0 | 4 0 |
| Average | 3 0 | 0 0 | 16 2 | 10 2 | 10.0 | 3 8 | 8 0 | 6 8 | 8 0 | 2.3 | 3 0 | 7.7 |
| 1999 | 17 0 | 0 0 | 15 8 | 13 3 | 2 5 | 1 8 | 0 0 | 0 0 | | 22 0 | 3 0 | 0 0 |
| 2000 | 2 0 | 2 0 | 1 0 | 21 0 | 2 0 | 2 0 | 1 0 | 0 0 | 0 0 | 0 0 | | |
| Average | 9 5 | 1 0 | 8 4 | 17.1 | 2.3 | 1.9 | 0 5 | 0.0 | | 11 0 | 3 0 | 0 0 |

| Cobun Creek DEP Location # 15 (Cobun Creek at Bridge) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| Pre-1999 | 3 0 | 0.0 | 16 2 | 10 2 | 10.0 | 3 8 | 8 0 | 6 8 | 8 0 | 2.3 | 3 0 | 7.7 |
| 1999 to Date | 9 5 | 1 0 | 8 4 | 17.1 | 2.3 | 1 9 | 0.5 | 0 0 | | 11 0 | 3 0 | 0 0 |

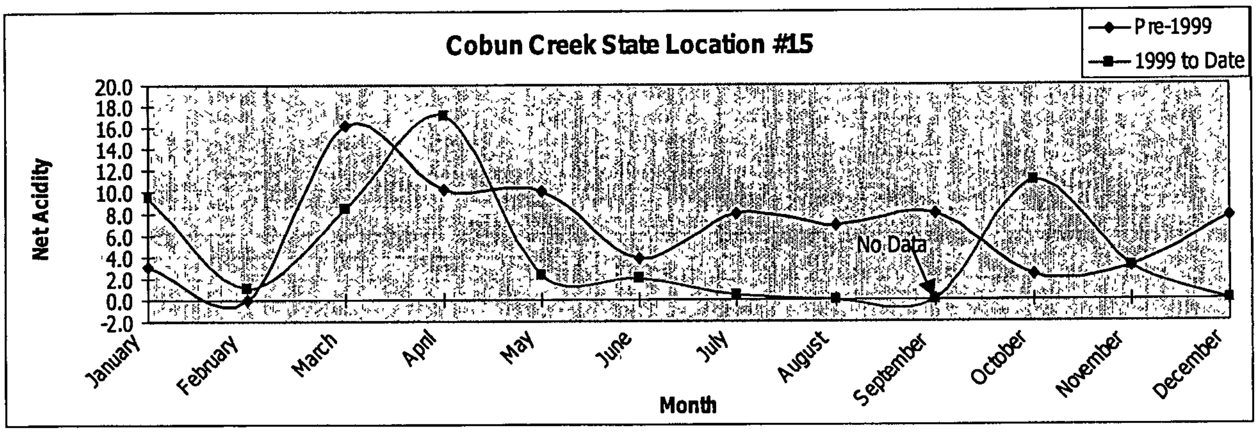



**Figure 7-13**

**Cobun Creek – Net Acidity Over Time**

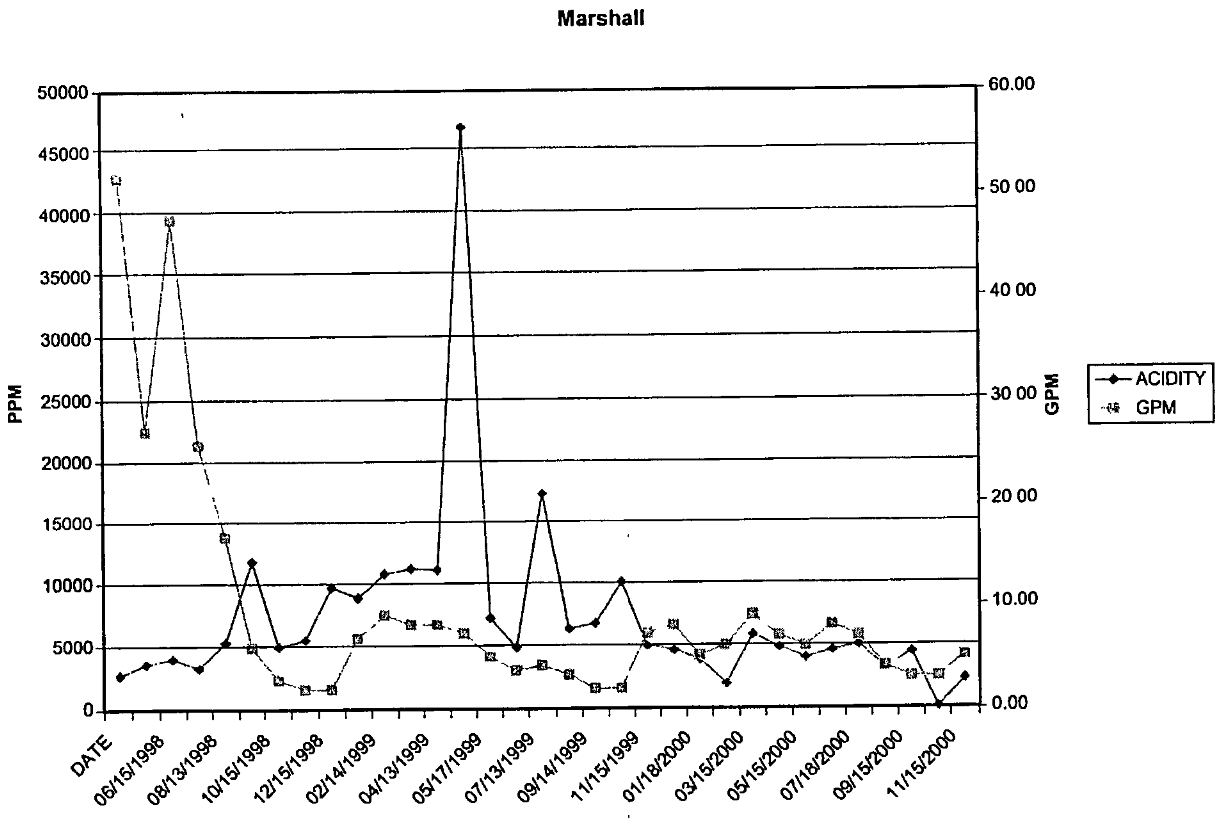



**Figure 7-14**

**Flow and Acidity of the Marshall House Discharge (from WVDEP)**
Note: Injection of grout was completed on November 4, 1998.

injection period may be related to very small flows, although the flow in October 1999 is the same as the flow in November 1999 when no peak in net acidity occurs. The central pool exhibits no significant change in net acidity (Figure 7-11). This is not unexpected since flows diverted to the central pool by the filling of the North Lobe were not anticipated to be more acidic. PM21 shows a significant increase in average net acidity in the post injection period (Figure 7-12). This is not related to injection of alkaline grout into the North Lobe but is due to stopping injection of sludge from the WVDEP treatment plant into the southwestern portion of the mine. Prior to 1999, the injected sludge contained enough residual alkalinity to buffer the AMD from PM21.

Figure 7-13 shows the average net acidity in Cobun Creek for the pre- and post injection period. Filling the North Lobe should have no impact on the net acidity of Cobun Creek. The fluctuation in net acidity in both the pre- and post- injection periods may be due to natural variations in flow and other factors effecting the small amounts of acidity.

Table 7-10 presents the average flows and acidities for the Marshall House, Seeps DEF and the Central Pool. PM21 is omitted because of the impacts of injection of sludge into the southwest portion of the mine which increased flows and decreased acidity in the period 1993-1999 and the decreased flow and increased acidity since 1998.

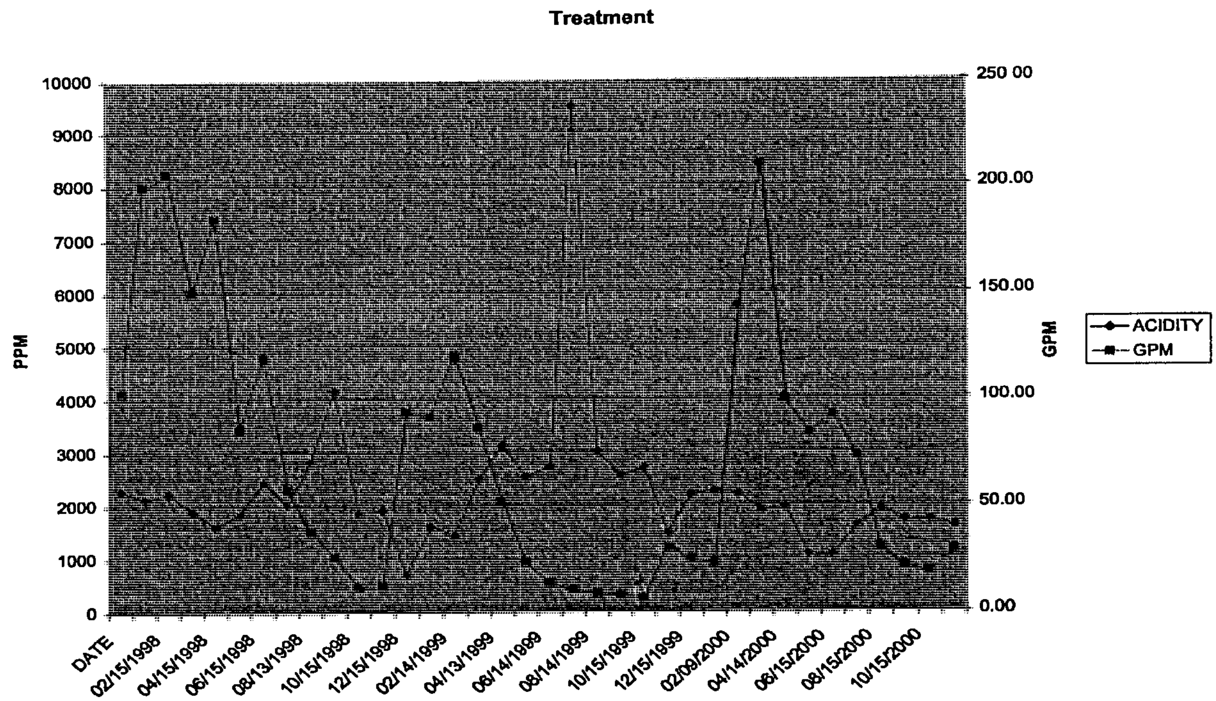



**Figure 7-15**

**Flow and Acidity – WVDEP Treatment Plant Inlet (from WVDEP)**
Note: Injection of grout was completed on November 4, 1998.

The average acidity for all three sources is greater and the average flows are less in the post injection period. Even with the increased acidity, the greatly decreased flows at the Marshall House and Seeps DEF result in, the average load of acid per day being less for the post injection period. However, the impact of the drought in 1999 needs to be considered.

Table 7-11 presents annual precipitation at Morgantown for 1993 through 2000.

**Table –7-10**
**Omega Mine Site: Pre- and Post-Injection Average Flows and Acidities**

| | 1993-1998 | | 1999 – October 2000 | |
|---|---|---|---|---|
| **Source** | **Avg. Flow (gpm)** | **Avg Acidity mg/l** | **Avg. Flow gpm** | **Avg. Acidity mg/l** |
| Marshall | 26.3 | 4.015 | 5.4 | 8.261 |
| DEF | 18.3 | 1.849 | 10.5 | 2.093 |
| Central | 12.8 | 2.381 | 10.5 | 2.792-* |

PM21- Not included due to cessation of injecting sludge into mine
*8 months January-August

**Table 7-11**

**Annual Precipitation – Morgantown, West Virginia**

| | Precipitation – Inches | |
|---|---|---|
| **Year** | **At Airport** | **At Lock and Dam** |
| 1993 | 36.48 | |
| 1994 | 51.78 | 53.30 |
| 1995 | 34.65 | 35.02 |
| 1996 | | 54.41 |
| 1997 | | 37.73 |
| 1998 | | 40.04 |
| 1999 | 38.85 | 36.09 |
| 2000 | | 42.10 |

To convert inches to millimeters multiply by 25.4

The normal annual precipitation for Morgantown is approximately 1,054mm (41.5 in). Table 7-11 indicates droughts in 1993 and 1995 comparable to 1999. The average precipitation in the pre-injection period using the airport data for 1993 and the Lock and Dam data for 1994-1998 yield an average of 1,088mm (42.83 in), almost 25mm (one inch) above normal for the six year period. Since the precipitation in 2000 is also above normal (about 15mm [0.6 inches]) the flows and acidities in 2000 may provide a better comparison with the pre-injection period.

Table 7-12 compares the pre-injection flow and acidity to flow in the period January-October 2000 as well as the average AMD load.

Table 7-12

**Omega Mine Site: Pre-Injection and 2000 Average Flows, Acidities, and AMD Loads**

| 1993-1998 | | | | Jan-Oct. 2000 | | |
|---|---|---|---|---|---|---|
| **Source** | **Avg. Flow (gpm)** | **Avg Acidity mg/l** | **Avg. AMD Load lbs/Day** | **Avg. Flow gpm** | **Avg. Acidity mg/l** | **Avg AMD Load Lbs/Day** |
| Marshall | 26.3 | 4,015 | 1,267 | 6.3 | 4,240 | 320 |
| DEF | 18.3 | 1,849 | 406 | 12.3 | 1,370 | 202 |
| Central | 12.8 | 2,381 | 365 | 13.1 | 2,095 | 329 |
| | | Total | 2,038 | | Total | 851 |

To convert pounds to kilograms multiply by 0.4536

The daily acid load in the year 2000 is less for all three sources, the Marshall House, Seeps DEF and Central Pool. The Marshall House daily acid load is 75 percent less in 2000 than in the pre-injection period. The average total daily load of acid from these three discharges in the pre-injection period was 924 kilograms per day (2,038 lbs/day) and 386 kilograms per day (851 lbs/day) in 2000. This reduction of approximately 544 kilograms per day (1,200 lbs/day) represents a decrease of 58 percent.

Figure 7-16 shows flow versus time and Figure 7-17 shows net acidity over time at the pipeline which delivers all mine discharges to the Omega treatment plant. The post-injection (1999-2000) flows are consistently less than pre-injection flows. The acidities in the pre- and post-injection periods are similar except for a large spike in May 1999.

To compare pre- and post-injection flows and acidities for the entire site, precipitation records were examined for periods of comparable conditions.

From January – July 1998, precipitation was 783mm (30.84 inches) and in the same period of 2000, the precipitation was 763 mm (30.02 inches). Since grouting did not begin until May 11, 1998 and with the lack of AMD buffering by the grout, it is believed the first seven months of 1998 can be considered pre-injection.

| Pipeline Before Treatment Combined State (DEP Location ID # 4) & Federal (DOE Loc. ID #25) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| 1993 | | 65 00 | 230.40 | 209 77 | 142.73 | 91 52 | 49.12 | 42 70 | 16 25 | 34 04 | 42.70 | 69 93 |
| 1994 | 230 77 | | | | 110 60 | 82 48 | 60 16 | 39 58 | 40.65 | 33 73 | 21.20 | 40 39 |
| 1995 | 53.41 | 49.55 | 77 37 | 42.46 | 100 50 | 109 30 | | 28.82 | 26 19 | 20 24 | 40.80 | |
| 1996 | 52 02 | 102 92 | 180.59 | 206 64 | 226.92 | 149.11 | 93 54 | 67.01 | 76 16 | 75.00 | 130.47 | 196 38 |
| 1997 | 86 99 | 84.37 | 207.71 | 113.34 | 165 06 | 94 53 | 44.49 | 33.60 | 25.23 | 24 64 | 79.11 | 65.86 |
| 1998 | 86.30 | 155 78 | 178 19 | 143 47 | 132.01 | 114.61 | 119 62 | 63.90 | 38.32 | 19 77 | 11.65 | 13.33 |
| Average | 101.90 | 91 52 | 174.85 | 143.14 | 146.30 | 106.92 | 73.39 | 45.93 | 37.13 | 34.57 | 54 32 | 77.18 |
| 1999 | 93 63 | 87.20 | 124.01 | 86 02 | 62.46 | 19.63 | 35 22 | 9 86 | 8 06 | 7 17 | 5 82 | 30 02 |
| 2000 | 24 01 | 8 99 | 143.00 | 208 99 | 99 01 | 83.01 | 92.02 | | | | | |
| Average | 58.82 | 48.09 | 143.00 | 147.51 | 80.74 | 51.32 | 63.62 | 9.86 | 8.06 | 7.17 | 5.82 | 30 02 |

| Pipeline Before Treatment Average Flows Pre- and Post-Grouting | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Month | January | February | March | April | May | June | July | August | September | October | November | December |
| Pre-1999 | 101.90 | 91.52 | 174.85 | 143 14 | 146.30 | 106.92 | 73.39 | 45.93 | 37.13 | 34.57 | 54.32 | 77 18 |
| 1999 to Date | 58.82 | 48.09 | 143 00 | 147.51 | 80.74 | 51.32 | 63.62 | 9 86 | 8.06 | 7.17 | 5.82 | 30 02 |

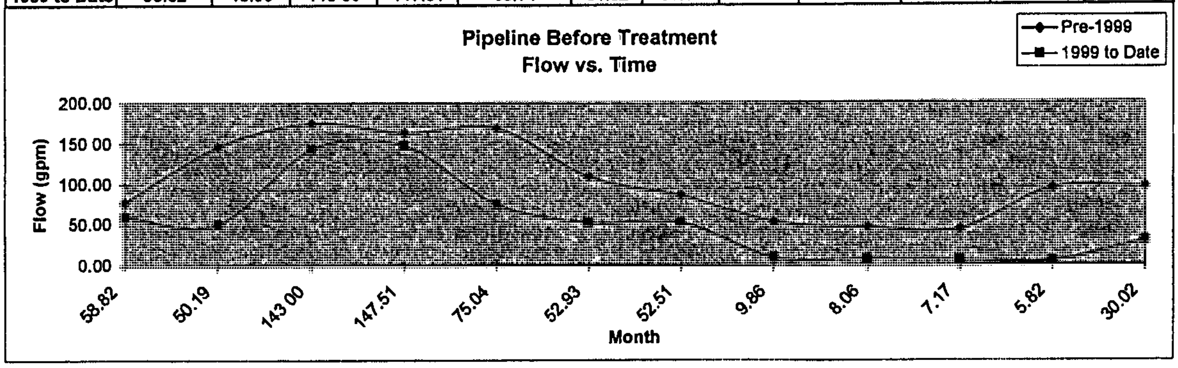



**Figure 7-16**

**Omega Mine Complex – Flows over Time at Treatment Plant Inlet**

| Pipeline Before Treatment<br>Combined State (DEP Location ID # 4) & Federal (DOE Loc. ID #25) | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Month** | **January** | **February** | **March** | **April** | **May** | **June** | **July** | **August** | **September** | **October** | **November** | **December** |
| **1993** | | 2712 00 | 2318.50 | 3077 00 | 3298 00 | 2378 00 | 2985 00 | 2348.00 | 2066.00 | 1793.00 | 1597 00 | 1900 00 |
| **1994** | 1562 00 | | | | 2643.00 | 2358 00 | 2041 00 | 1746.00 | 1609.00 | 2261.00 | 1887.00 | 1419.00 |
| **1995** | 1660.00 | 2087 00 | 1668 00 | 3432 00 | 1875.00 | 2046 00 | | 2167 00 | 1775.00 | 1668.00 | 766 00 | |
| **1996** | 1409.35 | 3006 00 | 2137.50 | 2591.33 | 1775 00 | 1795.33 | 1247 33 | 2262 67 | 2985.00 | 2011.67 | 1324.00 | 2428 50 |
| **1997** | 995.75 | 951.00 | 2470.00 | 2300.00 | 2417.00 | 1880.00 | 1805.00 | 1570.00 | 1557.00 | 1880.00 | 956.50 | 1230.00 |
| **1998** | 1192.50 | 1169.00 | 1904.00 | 1695.00 | 1555.00 | 1455.00 | 2430 00 | 1910.75 | 2681.00 | 2220.60 | 1830 00 | 1890.00 |
| **Average** | 1363 92 | 1985.00 | 2099.60 | 2619.07 | 2260.50 | 1985 39 | 2101.67 | 2000 74 | 2112.17 | 1972 38 | 1393.42 | 1773.50 |
| **1999** | 685.00 | 1605.00 | 1678.50 | 2349.75 | 2625.00 | 1533 75 | 2832.50 | 9480 00 | 2990.00 | 2540.00 | 2690 00 | 1460 00 |
| **2000** | 2180 00 | 2230 00 | 2200 00 | 1880 00 | 1930.00 | 1050.00 | 1050.00 | | | | | |
| **Average** | 1432.50 | 1917.50 | 1939.25 | 2114.88 | 2277 50 | 1291.88 | 1941.25 | 9480.00 | 2990 00 | 2540 00 | 2690.00 | 1460.00 |
| Pipeline Before Treatment<br>Pre- and Post-Grouting Average Net Acidity | | | | | | | | | | | | |
| **Month** | **January** | **February** | **March** | **April** | **May** | **June** | **July** | **August** | **September** | **October** | **November** | **December** |
| **Pre-1999** | 1363 92 | 1985.00 | 2099.60 | 2619.07 | 2260.50 | 1985.39 | 2101.67 | 2000.74 | 2112.17 | 1972.38 | 1393.42 | 1773.50 |
| **1999 to date** | 1432.50 | 1917.50 | 1939.25 | 2114.88 | 2277.50 | 1291 88 | 1941 25 | 9480.00 | 2990 00 | 2540 00 | 2690.00 | 1460.00 |

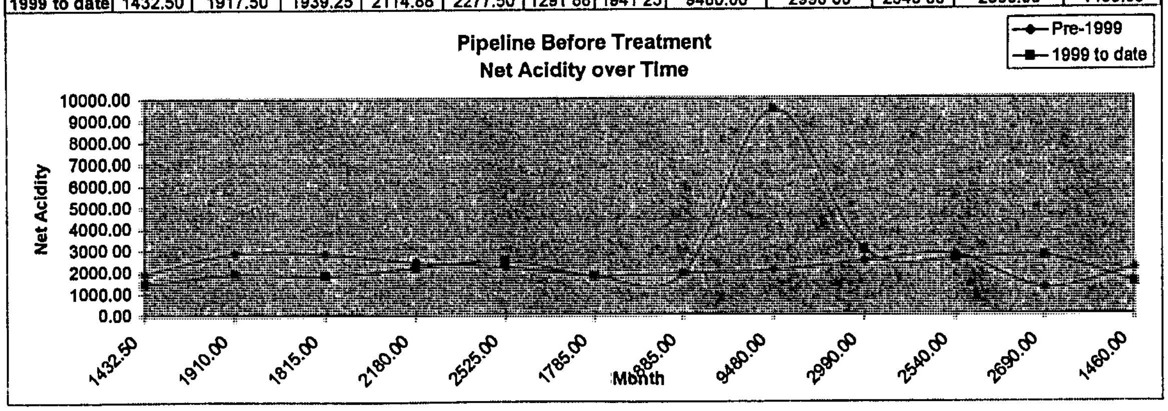



**Figure 7-17**

**Omega Mine Complex – Net Acidity over Time at Treatment Plant Inlet**

Table 7-13 shows the precipitation, average flows, average acidities and daily acid loads for water entering the treatment plant.

**Table 7-13**

**Jan-July 1998 and 2000 Precipitation, Average Flows**

**Acidities and AMD Loads for Water Entering Treatment Plant**

| | Precipitation | Avg. Flow | Avg. Acidity | Avg. AMD Load |
|---|---|---|---|---|
| Jan-July 1998 | 30.84 in | 133 gpm | 1629 mg/l | 2600 lbs/day |
| | 783 mm | 503 lpm | | 1179 kg/day |
| Jan-July 2000 | 30.02 in | 94 gpm | 1788 mg/l | 2016 lbs/day |
| | 763 mm | 355 lpm | | 914 kg/day |

The reduced flow in 2000 from 1998 for similar amounts of precipitation is likely due to some water being prevented from entering the filled North Lobe. The reduction in AMD load per day of 265 kg (584 pounds) represents an improvement of 22 percent.

# 8

# SUMMARY

## Introduction

The Omega Mine Injection Program had four goals:

- To fill the voids in the North Lobe of the Omega Mine to reduce contact of water and air with acid forming material with a secondary requirement that the grout have some alkaline leaching potential to help treat AMD.
- A grout mix which would set to sufficient strength to prevent mine subsidence.
- Use of a mixture of fly ash and FBC materials to demonstrate the synergistic attributes of the combined materials.
- A grout mix that would flow without separation and develop reasonable strength and dimensional stability.

These goals were met with the exception that the grout have some alkaline leaching to buffer the AMD. Evidence of meeting the project goals is summarized in the appropriate subsections presented below.

The conclusions and recommendations resulting from this study of a CCP grout injected into a large, abandoned deep mine to reduce acid mine drainage are conveniently considered under the following five general headings:

- Grout Mix Design
- Mine Grouting
- AMD Abatement
- Subsidence Control
- Future Work

## Grout Mix Design

An extensive laboratory testing program was conducted to evaluate both fly ash and FBC ash or mixtures of the two for injection into an abandoned deep mine to reduce AMD

The test program indicated that a blend of the two candidate materials provided an acceptable grout mix. The FBC ash had the potential to provide strength to the grout while the fly ash enhanced the fluidity of the grout. The addition of two percent cement provided dimensional stability to the hardened grout.

It was recommended that a grout blend of 49 percent FBC ash and 49 percent fly ash plus two percent cement with enough water added to produce a grout having a flow cone value of 60 seconds, be used. This mix demonstrated the synergistic attributes of combined fly ash and FBC ash, would flow long distances without separation and develop reasonable strength and dimensional stability. These characteristics provided confidence that the grout would fill 10.4 ha (26 acres) of the mine and encapsulate acid forming materials.

## Mine Grouting

The injection program at the Omega Mine is by far the largest project to date undertaken to reduce acid mine drainage. Almost 61,000 cm (80,000 cy) of CCP grout were injected into 10.4ha (26 acres) of the mine. A grout barrier was formed at the south side of the North Lobe to prevent mine water from flowing down the relatively steep dip (9 percent) into the North Lobe which is the lowest portion of the mine. Injection hole locations were determined from study of the mine plan. There was no set hole spacing. Due to the openness of the mine and steep dip of the mined seam, injection hole spacing in first mined areas was generally in excess of 30m (100 ft). In second mined areas, hole spacing was reduced to as little as 15m (50 ft) since site exploration (both borings and video camera) indicated that fallen roof material could impede movement of the grout. Due to the openness of some portions of the mine, grout moved up to 457m (1,500 ft) primarily because of the relatively steep dip of the mine. In previous work CCP grouts moved over 30m (100 ft) and the flowing grout moved slowly, like pancake batter (EPRI, 1996). From borehole video views of the grout, flow in the Omega Mine was fast and turbulent. The maximum grout take in one hole of 7,812cm (10,218 cy) is an indication of the ability of the grout to flow. The total quantity of the FBC-fly ash mix injected into 156 primary grout holes was 47,185cm (61,716 cy), an average of 302cm (395.6) cy per hole.

The total cost for drilling and injecting 60,500 cm (79,130 cy) of grout was $1,946,592 or $24.60 per cy.

The cost of using CCP grout for mine filling is competitive with typical mine grouting using a fly ash-cement grout, if suitable materials are available within a reasonable haul distance from the mine.

## Effectiveness of Filling

The volume of grout injected into the North Lobe, approximately 61,000cm (80,000 cy) is equivalent to 100 percent of the void volume in the first mined area and 75 percent of the mined void volume in the second mined area, where fallen roof material partially filled the mine void and would prevent grout from filling all voids.

The pressure grouting conducted in October 1998 near the end of the injection program indicates many portions of the mine had been previously filled by grout. Of the 46 holes that were pressure grouted, 18 had takes of only 0.7-1.5cm (1 or 2 cy). The total volume injected under pressure was 995cm (1301.8 cy) an average take of 21.6cm (28.3 cy) per hole. In contrast the average take of the FBC-fly ash grout in 156 primary injection holes was 302cm (395.6 cy) per hole.

Nine exploratory holes were drilled in the North Lobe late in the injection program using a rotary air rig. Eight encountered grout with no loss of air. This indicates the mine was filled at these locations. The other exploration hole encountered a 0.3m (1 ft) void and lost air, but air flow returned while drilling below the void. Subsequent pressure grouting of this hole resulted in a take of 92.5cm (121 cy).

Four core borings drilled approximately one year after grouting showed good roof contact by the CCP grout. Video camera observations in these holes confirmed good roof contact and showed good distribution of the injected grout. No voids were found.

## AMD Abatement

Reduction in acid mine drainage (AMD) usually infers some sort of water treatment to change the chemistry of the mine water and/or its discharge. Various treatment methodologies, such as percolation over and through limestone in air or through limestone under water in anoxic drains, dumping limestone fines near headwaters, and various kinds of other chemical dosing, all work to some extent but require replacing the limestone when it armors over or replenishing limestone or various chemicals as they are used in treatment. Work conducted in recent years in AMD reduction from mines and spoil piles has indicated that the best way to reduce AMD is to prevent the reaction from occurring rather than trying to treat the results (EPRI, 1996). [26]

Filling of the mine voids reduces or eliminates air and/or oxygenated water from reaching the AMD-forming materials thus preventing the generation of AMD. Using a material which will set and develop some strength further limits AMD production by decreasing the permeability of the material (grout) thus further limiting air and water movement. Complete filling of the mine workings results in limiting AMD formation by creating a barrier between the acid forming materials and air or water (EPRI, 1996).[26]

Testing of four grout core samples recovered from the mine a minimum of 9-10 months after injection showed a permeability range of $6.2 \times 10^{-7}$ to $8.9 \times 10^{-8}$ cm/sec. Thus the hardened grout is relatively impervious and with its dimensional stability should encapsulate acid forming materials and greatly reduce future formation of AMD.

The greatly reduced flows from the North Lobe following the grouting in 1998 could not be evaluated due to a drought in 1999. However, precipitation in 2000 was slightly above normal and the flows at the Marshall House and Seeps DEF have been reduced. Average flows in January-October 2000 at the Marshall House and Seeps DEF were 23.8 liters per minute (6.3 gpm) and 46.5 liters per minute (12.3 gpm), respectively. In the pre-injection period 1993-1998 the average flows were 99 liters per minute (26.3 gpm) and 69 liters per minute (18.3 gpm), respectively (Table 7-12).

It was anticipated that the flow from the central mine pool would increase, but the observed increase has been slight. In January-October 2000 the average flow was 49.5 liters per minute (13.1 gpm) compared to 48.4 liters per minute (12.8 gpm) in the pre-injection period of 1993-1998 (Table 7-12), an increase of only about 2 percent.

The quality of water flowing from the North Lobe has not changed significantly (Table 7-12). With injection of almost 61,000cm (80,000 cy) of grout, mostly highly alkaline, the lack of buffering of the AMD is disappointing. Apparently the ground water flows around the filled mine or encapsulated acid forming materials without being greatly affected by the chemistry of the hardened grout. Two other projects where CCP's were injected into mines producing AMD to improve water quality have not greatly changed the chemistry of the mine effluent (Lamminen, et al., 2000 and Rafalko and Petzrick, 2000).[39, 51]

The reactivation of AMD seepage into the Cobun Creek watershed in March, 1999 appears related to the mine filling on the basis of timing and water quality. This AMD is similar to that flowing from the North Lobe. However, the combined seepage flow is generally low and does not add greatly to total acid load. The source of the poor quality water from the Cobun Creek seeps and the North Lobe is uncertain. A possible source is precipitation moving downward to thin pyritic seams or zones in the mine roof where the water becomes acidic before reaching the grouted mine. However, the monitoring wells installed above the mine in 1999 generally indicate good water quality within a few feet of the mine roof.

In spite of essentially no change in mine water quality, the greatly reduced flows from the North Lobe (Marshall House) result in a reduction in daily acid load of 75 percent. Taking into account the increased flow and unchanged quality of the flow from the central pool the reduction in the daily acid load from the three largest sources of AMD from the Omega Mine is 58 percent.

The injection program met its goal of reducing the contact of water and air with acid forming materials. The secondary goal of buffering the AMD by alkaline leaching did not occur. However, the great amount of acid forming materials that are now encapsulated will not contribute to future AMD. It is anticipated that the materials now contributing to AMD formation will eventually be exhausted and water quality will improve as has been observed in the study of mine discharges from abandoned mines (Demchak, et al., 2000).[20]

## Subsidence Control

Injection of CCP grout with the procedures utilized results in virtually complete filling of the mine voids which provides excellent subsidence control. The fact that the hardened grout was found to have significant unconfined compressive strength coupled with a low permeability indicates it will provide good roof support while resisting passage of water thus limiting the potential for chemical degradation.

Although no strength criteria were specified since the priority was flowability to fill the mine, the 28-day strength of cubes formed during the injection program varied from 9,350-22000 kPa (1356-3192 psi). Seven cubes having an age of 496 to 652 days yielded strengths varying from 1,738 to 7,847 kPa (252 to 1138 psi) with an average strength of 4,744 kPa (688 psi). Six unconfined compressive strength tests were conducted on core samples recovered a minimum of 9-10 months after injection. The test results vary from 6,543 to 13,907 kPa (949 to 2017 psi) with an average for the six tests of 9,653 kPa (1400 psi).

The strength, ability to fill the mine and dimensional stability exhibited by the FBC-fly ash grout indicate it is an excellent material for subsidence control.

## Future Work

The injection program at the Omega Mine met its goals with exception of alkaline leaching to buffer AMD. However, continued poor quality AMD flow from the North Lobe and seeps above Cobun Creek raise questions on what might have been done differently or should be done on future AMD control projects.

Leaving the very lowest part of the mine ungrouted (near the Marshall House horizontal drains) may have prevented reactivation of the AMD seeps into the Cobun Creek watershed.

Starting grout injection at the lowest end of the mine may have provided better contact between grout and the mine roof and possibly moved grout up into fractures in the roof.

Since the project grout mix expands and cracks without the addition of 2% cement, a bench test would be of value to determine if a non-stabilized, alkaline grout in its expanded condition with much more surface contact area would buffer the AMD. If it would, such grout could be injected for control of AMD at sites where subsidence control is not a problem.

This project greatly benefited from an accurate mine map and substantial information on mine conditions and mine outflows. Yet when a significant flow of AMD occurred into the Cobun Creek watershed following a snow melt in March 1999, it became apparent that the hydrology of the mine and surrounding strata were not well understood and monitoring wells were installed. In future projects a detailed hydrogeologic study is recommended prior to developing any remediation plans.

The greatly reduced acid load will reduce treatment costs for the WVDEP. However, the savings will not be proportional to the reduction in acid load as only the cost of chemicals will be less. Labor costs will not change.

The WVDEP has an excellent database on mine flows and water quality. It is anticipated and hoped that this monitoring program will continue. Observation of future flows and water quality would add greatly to our understanding of how effective mine filling which encapsulates a large quantity of acid forming material can be in the abatement of AMD.

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