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

Annual Report
October 1994 - September 1995

Y.P. Chugh
D. Dutta
S. Esling
B. Paul
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DISCLAIMER

October 1995

Work Performed Under Contract No.: DE-FC21-93MC30252

For
U.S. Department of Energy
Office of Fossil Energy
Morgantown Energy Technology Center
Morgantown, West Virginia

By
Southern Illinois University at Carbondale
Carbondale, Illinois

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Morgantown, West Virginia 26507-0880

By
Southern Illinois University at Carbondale
Department of Mining Engineering
Carbondale, Illinois 62901

October 1995

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CHAPTER 1

INTRODUCTION AND SUMMARY

Introduction

On September 30, 1993, the U.S. Department of Energy-Morgantown Energy Technology Center (DOE-METC) and Southern Illinois University at Carbondale (SIUC) entered into a cooperative research agreement entitled "Management of Dry Flue Gas Desulfurization By-Products in Underground Mines" (DE-FC21-93MC30252). Under the agreement Southern Illinois University at Carbondale will develop and demonstrate several technologies for the placement of coal combustion residues (CCBs) in abandoned coal mines, and will assess the environmental impact of such underground CCB placement.

Dr. Yoginder P. Chugh, Director, Coal Combustion Residues Management Program of SIUC, is the Program Director and Principal Investigator, and Mr. Edwin M. Thomasson is Program Manager. Co-Principal Investigators are Dr. Deepak Dutta, Dr. Steven Esling, Dr. Bradley Paul, Dr. Hasan Sevim and Mr. Edwin Thomasson, all of SIUC. During the year Mr. Scott Renninger was appointed the Contracting Officer's Representative (COR) for the U.S. Department of Energy.

The overall program set forth in the cooperative agreement is for a period of four (4) years, beginning September 30, 1993. The overall program is divided into three phases, as follows:

Phase I ----- 30 months, Beginning September 30, 1993
Phase II ---- 6 months, Beginning April 1, 1996
Phase III --- 12 months, Beginning October 1, 1996

Under the terms of the cooperative agreement, Southern Illinois University is required to submit a Technical Progress Report every three months. However, the program has been in effect for the two full calendar years on September 30, 1995, and it was decided, with the concurrence of the COR, to submit an annual technical progress report, detailing the work and progress under the agreement for the second full year of operation. A similar annual technical progress report was submitted covering the first full year of operation, i.e. the period October 1, 1993 - September 30, 1994.

Accordingly, this report sets forth the technical progress under the agreement for the period October 1, 1994 -- September 30, 1995, of Phase I of the overall program.

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Summary and Highlights

At the close of the second year of operations the program was essentially on schedule both fiscally and technically. Indeed, the second year of operations concluded with most of Phase I tasks either completed or on-schedule for completion by the end of Phase I.

All necessary wells, including the two injection wells, two geotechnical wells, and two vent wells (which can be used as further injection wells if necessary) and groundwater monitoring wells have been drilled and completed. By using a borehole television camera, pictures of the underground mine area show that the wells are suitably located in void mine space. Thus underground placement of coal combustion by-products should proceed without serious problems.

Likewise, surface subsidence monuments have been installed and periodic observations are being made to provide background data on surface subsidence and the rate of subsidence over the placement area. A setback occurred when approximately half of the installed monuments were destroyed by truck traffic in the area; however, new monuments were installed and are in good order.

Virtually all laboratory tests on coal combustion by-products mixtures are complete, including TCLP, ASTM and modified SLP shake tests, and ASTM column leaching. Long-term column leaching tests, designed to continue for at least one year, are underway. All tests to date indicate that the individual coal combustion by-products, as well as the by-product mixtures, are non-hazardous in character and very unlikely to have any adverse affects on the environment or on groundwater at the underground placement site.

A full scale field demonstration of the collapsible intermodal container (CIC) technology for the handling and transportation of coal combustion by-products was conducted during the year. A final topical report on the technology and the field demonstration was completed and furnished to the Department of Energy.

Work continued on the study of material handling and transportation of coal combustion by-products by various means, and the economics of such handling and transportation. Various scenarios are being studied and evaluated including the use of pneumatic trucks, pressure differential rail cars, and collapsible intermodal containers (CIC's).

At the close of the year the Department of Energy granted an additional $78,000.00 to be used for the development and testing of additional technologies for the handling and transportation of coal combustion by-products. Cooperating in this undertaking will be the Illinois Central Railroad and Wilson Manufacturing Company. Preliminary plans call for the modification of existing steel intermodal containers so that they can be efficiently and effectively used in the handling and transportation of combustion by-products as well as modifying rail hopper cars for the same use. One or both technologies will be fully tested under plans now being developed.

Overall the program is progressing according to schedule, and no serious problems have developed. Good cooperation between the various program team members has been a hallmark of the past year. Fiscal management is sound, with all invoices and bills being processed in a timely manner. The fiscal tracking system has proved to be fully successful, providing program management and the Co-Principal investigators with timely data on all program accounts.

Objectives

The " Management of Dry Flue Gas Desulfurization By-Products in Underground Mines" program has the following basic objectives:

Objective 1. Based on an assessment of current technologies - develop and demonstrate a methodology for handling and transporting dry coal combustion residues (and mixtures of various residues) that is technically and economically feasible and that will alleviate the fugitive dust problem inherent in the handling and transportation of dry coal combustion and FGD residues.

Objective 2. Based on an assessment of current technologies - develop and demonstrate a methodology for disposing of coal combustion residues (and mixtures of various residues) in abandoned underground coal mine working from the surface through a borehole utilizing dry residues and a pneumatic placement technique. The goal of the methodology to be developed is to distribute the residues over an area of at least 300 feet from the injection borehole. The methodology developed must be both economically feasible and environmentally sound.

Objective 3. Based on an assessment of current technologies - develop and demonstrate a methodology for coal combustion residues (and mixtures of various residues) in abandoned underground coal mine workings from the surface through a borehole utilizing a mixture of residues and water in " paste" form so that the " paste" contains approximately 70% residue solids. The goal of the methodology to be developed is to distribute the residues over an area of at least 300 feet from the injection borehole. The methodology developed must be both economically feasible and environmentally sound.

Objective 4. Determine whether the placement of coal combustion residues in abandoned underground coal mine workings, using either a pneumatic placement technique or a " paste" hydraulic technique will control surface subsidence over the abandoned underground workings.

Objective 5. Assess environmental impacts (land, water, air) of underground management of FGD by-products.

The assessment of current technologies and development of technologies in objectives 1, 2, and 3 will be completed during Phase 1. The demonstration of the technology in objective 1 was completed during the year, and a final topical report prepared and furnished to the Department of Energy - METC. Demonstration of the technologies developed under objectives 2 and 3 will be done in Phase II and Phase III (April 1, 1996 - September 30, 1997). However, the foundation for the completion of Objectives 2 and 3 will be completed during Phase 1, and, indeed, are nearing completion at the time of this report. This foundation consists of determining the physical, chemical, engineering, and leachate characteristics of various coal combustion by-products and selected by-product mixtures; establishing baseline data on groundwater and other environmental characteristics; and developing economics analyses and economic models of the by-product placement methodologies.

Objectives 4 and 5 are dependent upon the completion of Objectives 2 and 3, and will require long-term measurements following the underground placement of the coal combustion by-products.

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CHAPTER 2

ENVIRONMENTAL CHARACTERIZATION

DR. BRADLEY PAUL
CO-PRINCIPAL INVESTIGATOR

Abstract

The overall purpose of the Environmental Characterization Studies is to determine the chemical and leachate characteristics of coal combustion by-products in order to make accurate predictions of the environmental effects that the placement of such by-products in underground coal mines can be made. This chapter presents a detailed report of the Environmental Characterization Studies conducted over the past year. The overall conclusion that is drawn from these studies is that the placement of coal combustion by-products in underground coal mines will not adversely affect groundwater of otherwise adversely affect the environment.

Introduction

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

Based on interaction between the environmental and physical properties characterization teams, two mixes have been selected for placement. A mix selected for pneumatic placement will consist of 80% FBC fly ash and 20% spent bed ash treated with 30% moisture during injection. A paste backfill mix consisting of 55% force oxidized scrubber sludge, 40% F-type fly ash, and 5% venturi scrubber sludge (lime waste) will be placed hydraulically.

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

1) TCLP tests on all mix components and proposed mixes.
2) ASTM column tests on mix components and mixes.
3) SLP tests on preliminary mixes.
4) SEM and EDX determinations of component mineralogy and composition.
5) CCE and paste pH determinations on all mixes and mix components.
6) Initial results of the rapid aging tests.

Material Characteristics Influencing Environmental Risk

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

1- The bulk composition of the material to be placed

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

2- The mineralogical structure of the material

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

3- Permeability of the material

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

4- Amenability to alkaline or acid side leaching conditions

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

Material Composition

To date only preliminary work has been done on bulk total composition. Samples of each of the major residues were subject to microwave digestion in a solution of nitric, hydrochloric, and hydrofluoric acid (aqua regia & HF). The solutions were neutralized with boric acid and analyzed by ICP. Samples were also analyzed by long capture time SEM.

Of course both methods have definite limitations. SEM techniques on these samples appear effective at measuring major and minor element composition, but even after 8 hours of capture time were not sensitive to trace elements. ICP analysis gives much better response with regard to trace element determination but requires soluablization of a solid sample. Most acid digestion procedures can produce biased answers due to the effect of refractory phases. ICP also has limited detection for some elements and high background equivalent noise levels for others.

Results from acid digestion and SEM have been supplemented with preliminary XRD analysis to identify mineral phases. The finding most clear from this work is that trace elements of environmental concern are present only in limited quantities.

Conventional PCC fly ash

Conventional PCC fly ash is to be used in the underground FGD disposal demonstration as part of the hydraulic mix. It's main functions are to provide pozzolan for setting strength of the mix and for preventing bleed-off water. As would be anticipated from previous work with fly ashes, X-Ray analysis indicates that most of the matrix is amorphous alumino-silicate glasses. The photo-micrograph shown in Figure 2.1 indicates that the fly ash is composed mostly of individual and agglomerated censospheres about 1 to 20 microns in size. Considerable surface area is exposed which may facilitate the early release of elements commonly found as surface coatings including boron and sodium. SEM analysis picked up considerable iron from the scope and both oxygen and carbon (carbonates are present in the incombustible matter in coal) where confirmed present in significant concentrations. The SEM semi-quantitative analysis was adjusted to give 6% iron and carbon and oxygen were excluded. The elemental percentages are reported below in Table 2.1. The results from a 3 minute microwave digestion are also reported.

At first glance the two methods for composition determination may appear to have little in common, however, when the weaknesses of each method are considered the results make considerable sense. SEM is a semi-quantitative analysis that is rather unreliable for concentrations under about 0.5% even with very long capture times. The results are fully consistent with the view that any element under 0.5% concentration will not be detected by the SEM. Indeed no element reported by SEM had under 0.5% concentration, and no element found by Digestion but not detected by SEM indicated over 0.14% concentration.

Microwave Digestion, on the other hand, is vulnerable to problems of incomplete dissolution of refractory complexes, and despite the use of hydrofluoric acid still had considerable undissolved material. Alumino-silicate complexes particularly are noted for resistance to dissolution. Aluminum, silicon, iron, manganese, and titanium are commonly found in alumino-silicate complexes and glasses and constitute 91% of the SEM composition of the PCC fly ash, but less than 16.5% of the acid digestion composition. When amorphous phases are subject to leaching it is common for elements to leach in far from stoichiometric amounts, which is quite consistent with the imbalance between iron, aluminum and silicon in the digestion composition compared to the SEM composition.

Considered as a whole the composition of the PCC fly ash contains few elements not typical of ordinary continental crust surface. The major elements that appear candidates for release into the environment are alkali metals, Ca, K, Mg, and Na. Because the surrounding groundwater is a 4% solids brine, none of these elements seems out of place. A greater amount of alumina dissolved than may have been the case with silicates but this seems unlikely to be an environmental issue.

Table 2.1
Bulk Composition of PCC Fly Ash

Element	SEM Analysis	Acid Digestion
Al	31.86%	9.52%
Ba	Not Det.	380 ppm
Be	Not Det.	44 ppm
Ca	6.37%	47.6%
Cd	Not Det.	122 ppm
Co	Not Det.	122 ppm
Cr	Not Det.	960 ppm
Cu	Not Det.	258 ppm
Fe	6% {adjusted to}	6% {adjusted to}
Hg	Not Det.	Not Det. (cold vapor)
K	Detected	8.8%
Mg	.69%	3.5%
Mn	Not Det.	.19%
Mo	Not Det.	288 ppm
Na	Not Det.	25.88%
Ni	Not Det.	580 ppm
Pb	Not Det.	Not Det. (Graphite Furnace)
S	1.09%	Not Analyzed
Sb	Not Det.	Not Det.
Si	46.86%	.65%
Tl	Not Det.	Not Det.
Ti	6.37%	Not Analyzed
V	Not Det.	.138%
Zn	.91%	.274%

[Note - Values reported in Table 2.1 are concentrations in solid phases, not leachate concentrations]

Some hazardous trace elements were found in the fly ash including chromium and cadmium, while others such as antimony, mercury, and lead were not. Where hazardous elements were found the issue of how much of the element is available for potential release becomes a question. It is clear first that no hazardous element is a major or minor component of the PCC fly ash. Certainly the ppm concentrations reported in Table 2.1 are high because of the exclusion of carbon and oxygen. The trace element composition of the refractory phases that could not be acid digested, even under the harshest conditions, cannot be known from the methods used to date. On the other hand, if a trace element cannot be mobilized in detectable concentrations by boiling aquariga and HF, it seems doubtful that it will be released in hazardous concentrations by cold mineral water. The concept of fixing dangerous elements in inert glass phases is common in the disposal of high level radio-active waste. Although the glass phases in the PCC fly ash are to be attacked in the hydraulic mix by alkali's from the venturi scrubber, as part of the pozzolanic reactions used to produce setting, these pozzolanic cement binders are themselves noted for being highly stable. It therefore seems reasonable to treat the elements released by the acid digestion as the maximum amount of an element ever available for release into the environment over a short enough time span to be a bonafied environmental concern.

In estimating maximum trace element quantities available for leaching it is noted first that the PCC fly ash is approximately 91% alumino-silicates and 9% other components. If one assumes that all the other components dissolved in the acid digestion then 9% of the total composition produced about 84% of the leached trace elements and only 11% of the PCC fly ash ever dissolved. This would be consistent with visual observations. Making slight allowances for the missing carbon and oxygen the ppm concentration for trace elements considered potentially mobile is about 1/10th the value reported in Table 2.1 and the estimated maximum trace elements for release into the environment are given below in Table 2.2. Since 10,000 tons of hydraulic mix are to be placed and 40% of the mix is PCC fly ash, 4,000 tons of material are assumed in the analysis.

Table 2.2
Estimated Maximum Trace Elements
Potentially Available from DOE Demonstration

Element	Quantity
Ba	304 lbs
Cd	98 lbs
Co	98 lbs
Cr	768 lbs
Cu	206 lbs
Mo	238 lbs
Ni	464 lbs

Of course a table of the type provided above must be accompanied by several caveats. Clearly it is impractical to digest a sample large enough to be representative in the sense of assaying by ASTM procedures even with good laboratory splitting. Similarly trace elements in fly ash come ultimately from trace elements in the coal which are known to have greater variability than the major and minor elemental compositions. The estimates, however, serve as at least order of magnitude indicators of how much of a trace element may be placed in the ground in an extractable form. The table might be further extended to estimate the maximum amount of water that could be contaminated above some regulatory limit and this amount compared to the amount on groundwater in a region of some size. This could indicate the maximum percentage of the water that could be impacted if all the extractable trace element was simultaneously released in an instant. The table might also be used in conjunction with groundwater models to limit the maximum amount of a contaminant that can be found in a leachate plume emanating from the proposed test fill.

Several trace elements seem conspicuous by their absence. Arsenic and selenium values are not reported in any of the tables above. Of course SEM would not detect these elements. Concentrations of Ar and Se found in the acid digestion samples were low enough relative to background equivalent concentrations for ICP that quantitative results cannot be confidently reported. Similarly boron values have been excluded. The microwave digestion procedure involves neutralization of HF by boric acid prior to introduction into the ICP (un-neutralized HF would attack the ICP components). The result is very high levels of introduced boron in the digested samples. Method blanks were used but examination of the data to date suggests that most of the boron is introduced and that memory effects are producing significant noise in reported values. Thus, reporting of boron data at this time seems premature. Finally, Hg and Pb were not detected even though more powerful graphite furnace and cold vapor techniques were used instead of ICP. The low boiling point for Hg and the lack of a capture mechanism in the vacuum desiccator leaves doubt about the capture of Hg. This is also a potential problem for Se. The absence of lead probably relates to the digestion technique used.

Future Plans.

To improve the quality of results for the final topical report it is planned to use three digestion techniques for the material. In the first technique long open beaker boiling times in aqua-regia and HF will be used to digest the sample. It has been reported by some labs that more complete dissolution is obtained by this technique than by microwave digestion. The EDL system purchased for this project will allow precise graphite furnace quantification of arsenic. Also the HF is boiled off directly without the need for boric acid neutralization which should enable more reliable data for the boron ultimately available in the sample. Since boron from the fly ash is one of the few potential environmental issues the effort seems worth while for the boron data alone.

The samples will be fired in a tube furnace to volatilize Se which will then be condensed out into solution. Again the EDL system in conjunction with a graphite furnace should allow low detection limits and reduce the noise that prevented quantification up to this point. The direct fire technique also captures F which is not available in the current data. Direct volatilization of the low boiling point elements is probably a more reliable way to quantify concentrations. Similarly Hg will be direct fired in a tube furnace at 600°C. The vapor will then be preconcentrated with an amalgam attachment ahead of a cold vapor cell. The technique represents the state of the art for trace mercury detection.

Finally semi-quantitative XRF will be run to try to quantify non-amorphous phases. Neutron activation is also to be run to provide a cross-check and alternate means of detection for trace elements that might not be mobilized by digestion procedures.

Force Oxidized Scrubber Sludge

The force oxidized scrubber sludge shown in Figure 2.2 is composed of about 98% pure gypsum, though the internal crystal structure differs from natural gypsum. The crystals range in size from 25 to 200 microns in length and 3.5 to 30 microns in thickness. SEM analysis is being done at this time. Microwave digestion results are reported in Table 2.3 below. ICP analysis did not cover sulfur and oxygen directly, however percentages are reported using the assumption that calcium was all in the form of gypsum. There are few unreacted carbonates or incompletely oxidized sulfites in this wallboard quality high grade material. Dissolution in the boiling acid bath seemed almost complete so large disparities between SEM and digestion results are not to be expected.

Two overall observations stand out in terms of the leachate that might be produced by this material. First, the FGD sludge is remarkably low in trace elements of environmental concern. Estimated quantities of trace elements that might be introduced by the 5,500 tons of FGD sludge to be placed as part of this demonstration are shown below in Table 2.4. Second, it is noted that all of the major elements in this material are in a soluble form and one may expect calcium, sulfate, and sodium to be readily released into the environment. Calcium and sodium pose little risk to a brine solution. Sulfate may or may not require a variance for the site although there is little conceivable reason why a variance could not be granted.

Table 2.3
Composition of Gypsum FGD

Element	SEM (Work in Prog)	Acid Digestion
Al		.33%
Ba		62 ppm
Be		7 ppm
Ca		28.73%
Cd		7 ppm
Co		4.7 ppm
Cr		12 ppm
Cu		19 ppm
Fe		24 ppm
Hg		Not Det
K		475 ppm
Mg		Not Det
Mn		62 ppm
Mo		Not Det
Na		1.29%
Ni		Not Det
Pb		Not Det
S		22.98%
Sb		Not Det
Si		.43%
Tl		Not Det
V		14 ppm
Zn		19 ppm

Table 2.4
Maximum Trace Element Quantities Introduced by
Peabody #10 Mine Demonstration

Element	Quantity
Ba	682 lbs
Be	77 lbs
Cd	77 lbs
Co	52 lbs
Cr	132 lbs
Cu	209 lbs
V	154 lbs
Zn	209 lbs

Work remaining to be done on the FGD material is the same as that planned for the fly ash discussed above.

Industrial Venturi Scrubber Sludge

The venturi scrubber sludge was added to the proposed hydraulic mix after the microwave digestions were complete so results currently available are limited to SEM scans. Acid digestion by the three procedure battery specified above for fly ash is still planned for this material. The industrial venturi scrubber uses suspended lime particles from a lime kiln to scrub sulfur from the stack gasses. Secondary calcium carbonate is also formed as CO2 is scrubbed. Most of the sulfite particles are small and long term exposure to water in the pond may have hydrated much of the CaO. The SEM analysis shown below in Table 2.5 detected but did not include the secondary carbonate.

Table 2.5
Composition of Venturi Scrubber Sludge

Element	SEM	Acid Digestion
Al	5.03%	
Ca	79.78%	
Fe	5.35%	
S	6.46%	
Si	3.38%	

Little can be said about the trace element composition of the venturi scrubber sludge since SEM could not be expected to pick-up trace elements. The iron, aluminum, and silica suggest some inert impurities in the material. The ratio of calcium to sulfur leaves little doubt that considerable calcium is in the form of carbonate and calcium oxide, rather than sulfite.

FBC Fly Ash

FBC fly ash shown in Figure 2.3 has a distinctly different structure and relative absence of cenospheres so common in PCC fly ash. At the lower firing temperature of an FBC unit, small particles about 1 to 4 microns in size fuse together to make agglomerates up to 80 microns in size. The finer size and angular nature of the particles in FBC ash will make this material dustier in handling, but may also allow it to be carried more easily during pneumatic injection. One feature that was noted during SEM work was that the fine cube-like particles appear to be calcium and magnesium sulfates and that most of the sulfate is contained in the finer particles. The appearance of magnesium sulfates would suggest that a dolomitic limestone was used as the sorbant in the boiler. The fact that the sulfate is mostly contained in small high surface area particles of soluble mineral phases suggests that this material may have a high initial release of sulfate. The extent of any environmental problems that result may depend in part on how much of the early release sulfate becomes tied-up in ettringite, which also forms early in the set-up chemistry for FBC fly ash.

SEM analysis is still in progress on this material. Response to microwave digestion was intermediate between the almost complete dissolution observed for the scrubber sludge and the limited dissolution observed for the PCC fly ash. The percentages reported below in Table 2.6 do not reflect carbon and oxygen from unreacted carbonate, or the oxygen in the calcium oxide or other oxide phases and thus trace element percentages are almost certainly exaggerated.

The composition shown in Table 2.6 suggests that leachates from this material will be dominated by alkali metals which will do little to degrade the 4% solids brine surrounding the mine site. At this point no attempt is made to predict maximum leachable quantities of trace metals to be introduced with the pneumatic mix. It is clear from the low percentages of aluminum and silica that incomplete dissolution has impacted the results above and the SEM work being done should improve the situation by quantifying the refractory phases. The expected XRF analysis and wet cement chemical analysis on the ash should furnish data on the missing oxygen, carbonate, and sulfate compositions. All of this will significantly reduce the concentrations above.

FBC Spent Bed

The FBC spent bed composition as digested appears quite similar to the fly ash although the particle size is much larger and in the 200 to 300 micron range. Only the microwave digestion results are available at this time and are reported in Table 2.7 below. Again estimates of the maximum trace element quantities introduced into the environment are withheld pending a more complete data set.

Table 2.6
Composition of FBC Fly Ash

Element	SEM (Work in Prog)	Acid Digestion
Al		5.8%
Ba		210 ppm
Be		19 ppm
Ca		70%
Cd		19 ppm
Co		45 ppm
Cr		89 ppm
Cu		172 ppm
Fe		8.38%
Hg		Not Det.
K		2.1%
Mg		6.44%
Mn		.17%
Mo		13 ppm
Na		5.22%
Ni		140 ppm
Pb		Not Det.
Si		.84%
Sb		Not Det.
Tl		Not Det.
V		159 ppm
Zn		.39%

Table 2.7
FBC Spent Bed Composition

Element	SEM (Work in Prog)	Acid Digestion
Al		5.67%
Ba		195 ppm
Be		22 ppm
Ca		76.6%
Cd		22 ppm
Co		53 ppm
Cr		34 ppm
Cu		75 ppm
Fe		4.91%
Hg		Not Det
K		.83%
Mg		5.95%
Mn		.22%
Mo		30 ppm
Na		3.54%
Ni		75 ppm
Pb		Not Det.
Sb		Not Det.
Si		1.22%
Tl		Not Det.
V		135 ppm
Zn		.23%

Hydraulic Conductivity

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

Pressure and leachate volumes taken in both the ASTM column and Rapid Age test procedures in this program allow estimates of hydraulic conductivity to be made by the constant head method. In assessing hydraulic conductivity it should be understood that FGD materials are often soluble or composed of high temperature phases that may not be in equilibrium with a low temperature environment below the water table. The tendency for pores to open or close and minerals to reform means that initial hydraulic conductivity may not correspond well to long term hydraulic conductivity. ASTM columns were run only 16 days and thus unless reactions proceed very fast the indications given by this data may be representative of only initial hydraulic conductivity or short trends. All of the FGD materials in this program are fine grained and when tightly packed give fairly low hydraulic conductivities. Estimated initial hydraulic conductivities are given below in Table 2.8.

Table 2.8
Initial Hydraulic Conductivities of FGD Materials
and Additive Fly Ashes

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

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

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

In estimating hydraulic conductivity of the pneumatic and hydraulic mixes to be placed in the field the question of how to estimate compaction as well as reactions between mix components needs to be taken into account. It was decided that in Rapid Age and ASTM column tests of the mixes no compactive effort would be assumed for the mixes, giving the worst case scenario for hydraulic conductivity. The cement chemistry for FBC fly ash and spent bed both rely on sulfo-pozzolanic reactions and formation of expansive minerals such as entringite. Not surprisingly the ASTM column on the pneumatic mix gave roughly the same behavior as the individual materials starting in the 10^-5 cm/sec range and falling an order of magnitude to 10^-6 cm/sec. Test results are also available for 3 replicate Rapid Age column experiments run with the pneumatic mix and the brine solution discussed later in the report.

The results for all 4 tests are similar and are shown graphically in Figure 2.4.

As would be anticipated when materials are simply poured into place, there is considerable non-uniformity in compaction and initial hydraulic conductivity in replicate experiments ranged from 1.24 X 10^-4 cm/sec to 1.28 X 10^-5 cm/sec. There are also some indications that the water flow history may influence hydraulic conductivity. High water flows may be able to remove and dissolve enough mineral to keep pours open longer against expansive and swelling reactions. One Rapid Age test sealed itself after just 7 days. As leachate volumes fell, the material became more resistant to future leachate flows. The two other columns continued for up to 2 months as of the last data used for this report. The fall in permeability to the low 10^-6 cm/sec range is clear, and although the data is not shown here, attempts are being made to keep these columns operating by setting the pressure all the way to the top of the design limit. In previous work with FBC fly ash it has been found that columns will try to seal themselves and that once sealed the column cannot be revived. The higher the flow of water, the better the column seems to resist sealing. The Rapid Age procedure involves several pore volumes of water every day instead of 1 pore volume as in the ASTM column procedure. Hydraulic conductivity seems to fall less rapidly in the case of the Rapid Age test. If the Rapid Age Tests seal themselves the apparatus will be adapted to allow pressures of up to 60 psi to be used. The current apparatus for both Rapid Age and ASTM column tests will have difficulty making measurements once hydraulic conductivity gets into the upper 10^-7 cm/sec range.

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

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

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

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

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

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

Risk of Acid Side Leaching - Material Considerations

Fly ashes in particular are know to contain elements of environmental concern, including Cr, Cd, As, Se, and Pb. Most of these elements are likely to leach in higher concentrations in an acid environment than an alkaline environment, and state regulators are much more restrictive in permitting placement of coal combustion residues when the residues produce acidic leachates. Paste pH measurements are normally used to indicate whether residues are alkaline or acidic. It should be noted that while most fly ashes and scrubber sludges have a net alkaline composition, that some fly ashes will produce acid paste pHs. Because of the 40%+ calcium oxide content of the FBC ashes, there is little surprise that the fly and bed ashes produced paste pHs of 12.25 and 12.36 respectively, or that the proposed pneumatic mix gave a paste pH of 12.26. The venturi scrubber uses particles from a lime kiln to scrub the stack gasses and likewise has a high calcium oxide content that produces a paste pH of 12.29. The force oxidized scrubber sludge is over 98% pure gypsum and has a characteristic paste pH of 8.28.

Only the PCC F-type fly ash produced a curious result. From previously long-term open column leaching tests it was known that the pH of leachates from this ash were in the 7.5 to 8.5 range. The alkali in this ash, however, seemed slow to mobilize. In one experiment the paste was mixed and the pH meter was then allowed to sit in the mix. After 5 to 10 minutes, the pH appeared to be in the 3.5 to 3.7 range; although by letting it sit a little longer, the residues seemed to show a steady increase in the almost stable pH. Since it was already known that this fly ash had one of the highest CCE values of any of the fly ashes collected from southern and central Illinois power stations, the low paste pH seemed especially out of place. Paste pH was then determined using the same Mettler Auto-Titrator used for CCE determinations. The same buffers and pH standards were used, only the auto-titrator maintains agitated conditions during pH determination. As in the more quiescent beaker test, the pH started low and rose. However, the rising trend in the pH meter that was easily missed under static conditions was unmistakable under agitated conditions. The paste pH finally stabilized at 11.78. The experiment was repeated four times for confirmation.

The possibility that fly ash alkali release may be sluggish compared to other materials of similar composition may indicate large scale experiment involving FBC fly ash. A pond containing over 1 million gallons of very acid mine water was treated with the same FBC fly ash proposed for the pneumatic mixes at the Peabody #10. The FBC fly ash has a CCE value of 71 as determined by the agricultural lime method. In both the field setting and bench top laboratory experiments it took twice as much FBC fly ash to neutralize the water as was calculated by the CCE and acidity of the water. Both experiments were done under fairly quiescent water conditions.

The apparent fixation of some of the carbonate values in fly ash may be one reason why some F-type fly ashes from the midwest appear to be acidic while having positive calcium carbonate equivalent values. When attempts are made to generalize and commercialize the underground disposal concepts being demonstrated in this study it may be important to note that paste pH values may be a function of time or rate of change used to define the end-point pH and the agitation conditions under which the measurement was taken. The PCC fly ash to be used in this study has been tested in open and ASTM columns and found to produce alkaline leachates. In SLP shake tests the leachate was alkaline, and in the TCLP the pH was raised above the buffer level but not to neutrality. Thus all materials in this study are prone to alkaline side leaching. The fact that careless paste pH measurement of the PCC fly ash could mislead a study or a permit application into emphasis on acid side leaching should be noted as a warning.

Risk of Acid Side Leaching - Environmental Considerations

Leachate and Permitting Considerations Based on TCLP

TCLP test results are presented in Tables 2.9 and 2.10. The final column reports the theoretical detection limit for the analytical technique being used. Elements not detected are reported as less than the detection limit rather than zero. All concentration values are reported in parts per million. Part per million units do not apply to pH, calcium carbonate equivalent (reported as a percentage of neutralizing power of pure calcium carbonate), or conductivity (reported in micro-mhos).

Permitting decisions for this material will most likely be made relative to groundwater standards. The closer water bearing units at the Peabody #10 site are located about 12 feet above the coal. These units are of marginal permeability and contain brines. Surface groundwater resources apparently are not in communication with the deep brines since there is no detectable contamination of the near surface water resources. No data on the trace element content of the deep brines is available, but the salt content of the brines is generally around 2 to 4%. The water resources in the most apparent risk are not of class I quality, so comparisons in this analysis will be based on State of Illinois Class II groundwater standards.

There are few issues involved with the quality of the TCLP leachate relative to class II groundwater standards. The violations discussed below were noted.

1- The ADM FBC fly ash and spent bed and the Pcc F-fly ash and the lime waste all produce end point pH values in excess of the 9 limit often used for mine sites. It is known from long term column testing done for other research projects that the high pH values do not persist and in time drop below the 9 limit. It is also interesting to note that none of the tests exceeded the 12.5 RCRA limit. The wet scrubber sludge had an endpoint pH of 5.29 which is below the mine discharge limit of 6. It should be remembered that the TCLP is buffered at pH 4.9 and the calcium carbonate equivalent value and the alkalinity value leave no doubt that this is an alkaline material. The endpoint pH for the hydraulic mix is inside the 6 to 9 limit. The mixture of spent bed and FBC fly ash is of course not within the 6 to 9 limit.

2- The total dissolved solids exceed the class II groundwater limit for all mix components and all mixtures. Since the brines in the region generally run about 2 to 4% dissolved solids and all the TCLP tests came out with a fraction of a percent dissolved solids, it is unlikely that groundwater quality will be deteriorated on this parameter.

3- Only the industrial boiler scrubber sludge did not exceed the class II groundwater limit for sulfate in the TCLP. All other components and mixes exceeded the sulfate limit. Of course such a result is almost inherent in any SO2 control product when the class II limit is only 400 ppm. The anions for the deep brines in the region have not all been determined, but with the dissolved solids already as high as 2 to 4%, mostly with salt like cations, it is likely that sulfate is already above class II standards and any exceedance of the standard will already be for natural causes. Sulfate values in the leachate are high enough that it is conceivable that the exact composition of the deep brines may have to be tested. Since the monitoring wells are now being installed to provide baseline data, the sulfate values should probably be available by next report.

4- The class II standard for selenium is 50 ppb. Although selenium was not detected in this round of tests, the detection limit is 75 ppb. It is none-the-less apparent that selenium should not be a significant problem.

5- Arsenic concentrations were above the class II groundwater limit for both F-fly ash and the industrial scrubber sludge (lime waste). The mixtures proposed for actual placement were all within class II limits so no problem is expected.

6- The F-fly ash is significantly above the class II limit for boron, and the hydraulic mixture thus also violates the class II limit. The basis for boron limits is plant toxicity. Concern about poisoning sensitive plant species with salt water in a low grade aquifer 400 feet deep seem unwarranted, and if a variance request were necessary, it could be granted without threat to the environment. Of course sea water already contains high levels of boron and it has been found in other work done at SIUC that many limestones in the region are high in boron. It is quite likely that deep brines that have been in extended contact with boron rich rocks will already violate class II limits for natural causes.

In the setting, where it is proposed to place the FGD byproduct mixes, there seems little reason to believe that any of the proposed mixes could pose a threat to the environment. Although there are some parameters that could be raised as a concern in the permitting process it seems unlikely that any issue will arise that cannot be dealt with by simply establishing the existing background water quality.

Leachate and Permitting Considerations Based on ASTM Columns

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

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

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

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

4- The ASTM column test produces leachate samples over time. One of the faults of shake tests is that they contact fresh material with solution, measure element concentrations, and assume that the concentrations seen are representative of the leaching of the bulk material. This is not the case with FGD byproducts and associated combustion residues. Many of the most noted toxic elements are volatilized at low temperature and are fumed out over the surface of particles exposed to the combustion exhaust gasses. Shake tests on fresh material leach unrepresentative surfaces and once the surface trace elements are leached away they will never be available in the bulk material in such concentrations again. Figure 2.1 represents the boron content of several consecutive ASTM column leach samples taken after 1, 2, 4, and 16 days. The pattern seen is highly representative of the leaching of volatile toxics from the surfaces of combustion residues.

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

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

a- FBC Fly Ash, Ag, Be, Cd, Co, Cr, Cu, Hg, Mg, Ni, Pb, V.

b- FBC Spent Bed, Ag, Be, Co, Cr, Cu, Hg, Mg, Ni, Pb, Sb, V, Zn.

c- Force-Oxidized FGD, Ag, As, Be, Cd, Co, Cr, Cu, Ni, Pb.

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

a- FBC Fly Ash, Al, B, Ba, Fe, Mn, Mo, Si, V, Zn.

b- FBC Spent Bed, Al, Ba, Fe, K, Mn, Mo, Na, Si.

c- Force Oxidized FGD, Al, Fe, K, Mg, Mn, Mo, Si, V, Zn.

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

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

a- FBC Fly Ash--- As, Sb

b- FBC Spent Bed--- As, B, Cd

c- Force-Oxidized FGD --- B (see figure 1) --- Sb-, Se

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

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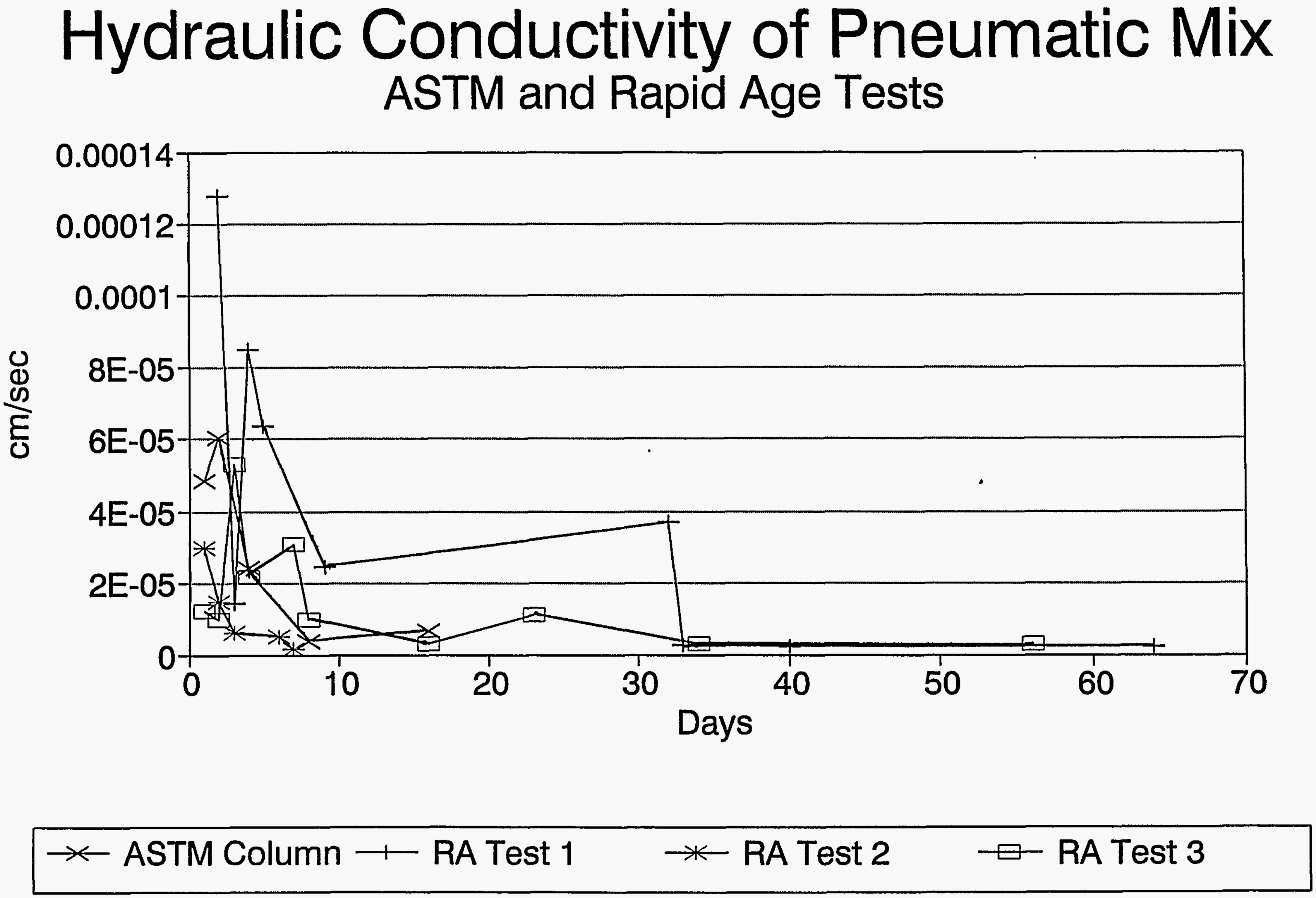
Figure 2.1


Hydraulic Conductivity of Pneumatic Mix
ASTM and Rapid Age Tests
cm/sec
0.00014
0.00012
0.0001
8E-05
6E-05
4E-05
2E-05
0
0
10
20
30
40
50
60
70
Days
ASTM Column
RA Test 1
RA Test 2
RA Test 3

CHAPTER 3

MIX DEVELOPMENT AND GEOTECHNICAL CHARACTERIZATION

DR. D. DUTTA
DR. X. YUAN
CO-PRINCIPAL INVESTIGATORS

Abstract

Geotechnical characterization of mixes of coal combustion by-products to be placed in underground coal mines by pneumatic and hydraulic placement techniques continued throughout the year. This chapter reports on the work performed, particularly in determining mixture strength and stiffness (modulus of elasticity). The use of small additions of lime and lime waste to improve strength characteristics and stability of the mixtures in water was investigated with success, and is reported in this chapter.

Specific Objectives

The geotechnical characterization of mixes for pneumatic and hydraulic placements is intended for the determination of their short- and long-term strengths, elastic moduli, stress-strain characteristics, swelling and slump characteristics, linear expansions, heat of reactions, mass loss at extreme temperatures, and density. Besides, hydraulic placement of coal combustion residues (CCR) and flue gas desulfurization (FGD) by-products in abandoned mine workings may negatively impact the stability of mine workings and surface subsidence since Illinois coal seams are generally associated with thick (2-4 ft) and weak (300-1000 psi) floor strata. Short-term subsidence due to wet backfilling can cause damage to surface structures and impact land use patterns. In addition, the stability of underground bulkhead may be negatively impacted due to active pressures imposed by hydraulic or pneumatic placement of by-products. The immediate floor strata associated with No. 6 coal seam (Herrin Seam) at Peabody No 10 demonstration mine are known to be weak (Chugh et al., 1989). Hence, objectives of the geotechnical assessment also include: 1. analysis of the stability of abandoned mine workings prior to and after disposal of combustion by-products; 2. estimation of the surface movements and their characteristics due to wet disposal of by-products; 3. assessment of the stability of isolation structures such as bulkhead to withstand pressures due to the disposal of FGD by-products; 4. monitoring of long term surface and sub-surface movements prior to and after backfilling of coal combustion by-products. The results of the demonstration studies at Peabody No. 10 mine will be generalized for other areas within the Illinois Coal Basin.

Technical Progress

In the last year (October 1, 1994 to September 30, 1995), geotechnical characterizations of the pneumatic and hydraulic mixes were continued to determine their strength and stiffness (modulus of elasticity) properties. Preliminary strength results of hydraulic mixes consisting of scrubber sludge and PCC fly ash were not satisfactory (Annual Report, 1994). To improve the strength characteristics and the stability of mixes in water, lime and lime waste (a high alkaline sludge from the limestone processing plants) were considered as new raw materials for the hydraulic mix design. Results of strength and modulus using lime as a binder were reported in an earlier quarterly report (October 1-December 31, 1994). Mix characteristics using the lime waste are also reported in earlier quarterly reports (January 1-March 31 and April 1-June 30, 1995). A separate section in this report summarizes different geotechnical characteristics of hydraulic mixes using the lime waste as a binder.

The composition of the pneumatic mixes were finalized based on the current production ratio of the raw materials (FBC fly ash and spent bed bottom ash) and the strength characteristics. Finite element modeling was continued to study the effects of the strength of backfill materials and partial backfilling.

Selection of two injection panels (one for pneumatic injection and the other for hydraulic injection) in Peabody #10 mine was done after lengthy deliberations with the program director, program manager, principal investigators, and mine personnel. Drilling of two injection boreholes and two geotechnical instrumentation boreholes over the two panels (one for the hydraulic injection panel and the other for the pneumatic injection panel) were completed during the reporting period. Also, two additional boreholes termed as "vent holes" were drilled over the two injection panels. These two boreholes will act as "vent holes" for normal injection through the injection holes. These "vent holes" can be converted into injection holes in the case current injection holes are not usable for the injection of backfill materials.

Surface subsidence monuments were installed over the two panels for monitoring the surface movements prior to the underground placements. Figure 3.1 shows different boreholes drilled over the pneumatic and hydraulic injection panels (Figure 3.1 does not show any water monitoring wells).

Finite Element Modeling

The causes of subsidence on the surface in the Illinois coal basin are the weakening of floor strata resulting in floor heaves and pillar punching into soft floors resulting in pillar and roof failures. The finite element modeling was conducted to determine the amounts of floor heave, roof sag, and pillar punching over a period of time with and without back filling. If the backfilling can reduce floor heaves, roof sags, and pillar punching, then it can reduce the subsidence on the surface. Also, reductions in floor heaves, roof sag, and pillar punching due to partial backfilling and backfilling with different material stiffness were investigated using finite element modeling.

Figure 3.2 shows the mesh of the two-dimensional, plane strain finite element model used for the study. The mesh is a vertical section with parallel entries and the crosscuts are parallel to the plane of the mesh. This implies that the entries are tunnel like openings and are infinitely long. The pillars are also treated as infinitely long in the perpendicular direction. This type of two-dimensional analyses ignore the effects of the crosscuts which result in pillar stresses that are less in analyses than in actuality. To circumvent this problem, Pariseau and Sorenson (1979) have suggested to increase the actual unit weight of the overburden strata by the factor $(1 + W_e/W_p)$ where W_e and W_p are the entry and pillar widths, respectively. This technique leads to pillar stresses from two-dimensional analyses that are within a few percent of three-dimensional results (Pariseau and Sorenson, 1979). In the current model, the three-dimensional effect is simulated by increasing the unit weight of overburden strata by the above mentioned factor.

The model is formulated as a large displacement, small strain problem and the geometric non-linearity is incorporated into the model. All the strata in the model are assumed to be linear elastic except the immediate floor. The immediate floor is assumed to have time dependent behaviors. An empirical creep equation of the form $\varepsilon = a\sigma^b t^c$ (where ε is the creep strain, σ is the

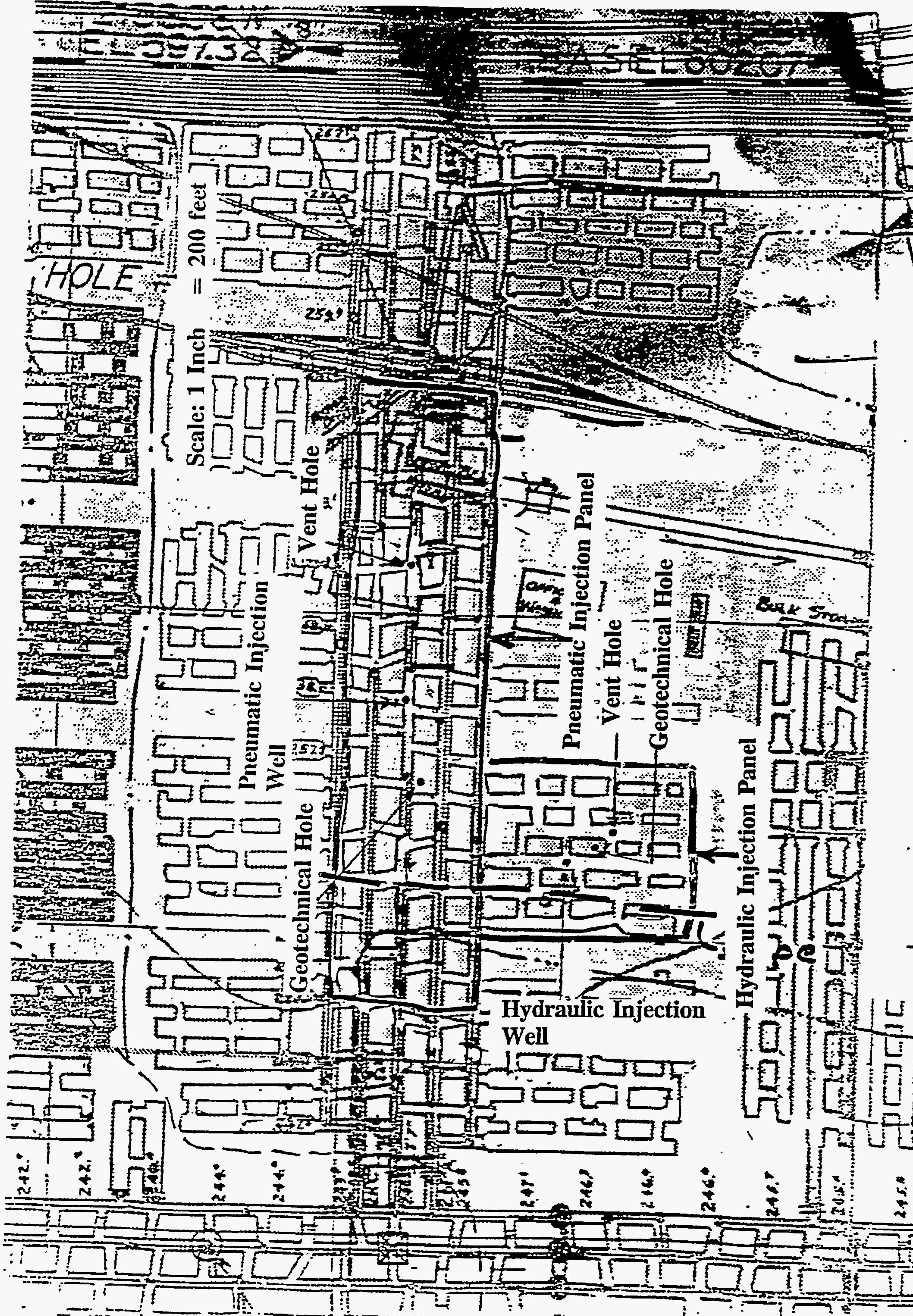


Figure 3.1 - Different boreholes over pneumatic and hydraulic injection panels.

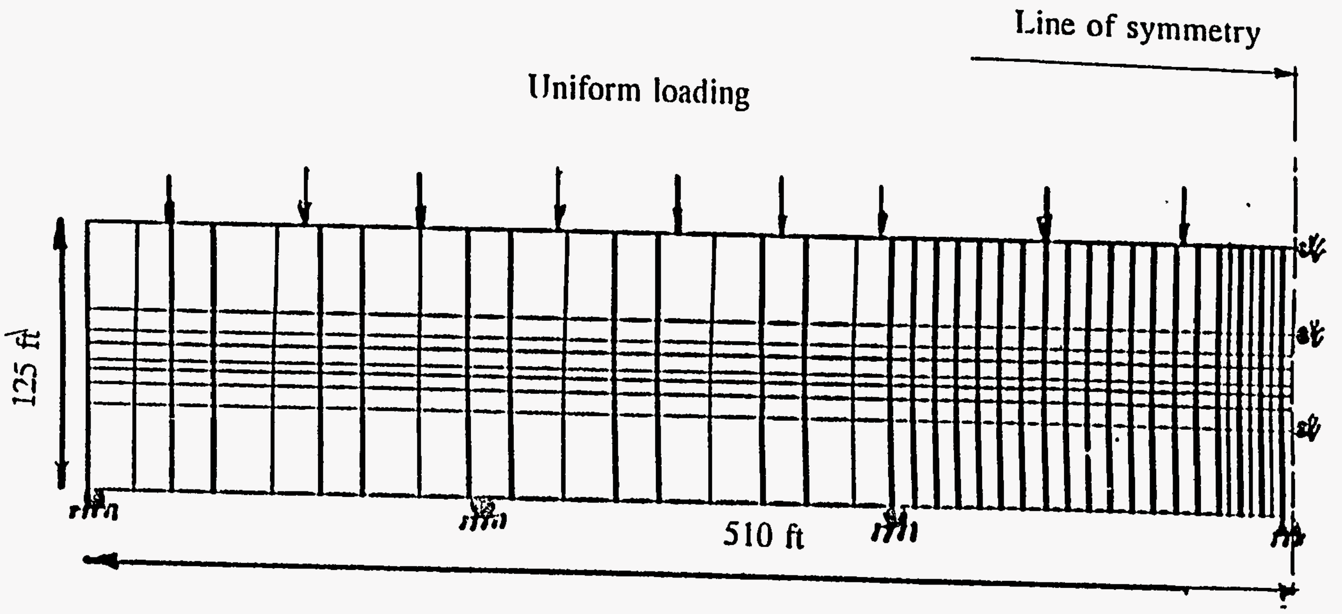


Figure 3.2 - Finite element mesh used in the model

stress, t is the time in days, and a, b, c are constants) is used to simulate the time dependent behavior of the floor strata. Chugh et. al (1994) have done extensive testing of time dependent floor strata behaviors in the Illinois coal basin and developed a visco-elastic model for the time dependent floor behavior. Data from those test results were used to determine the creep constants a, b, and c. These constants are: $a = 2.44x10^{-4}$, $b = 1.1$, and $c = 0.82$.

A typical geologic log as shown in Figure 3.3 is used for modeling. The extraction height of the coal seam is seven ft and the immediate floor is 10 ft thick clay stone (weak floor). The immediate roof is four feet thick gray shale overlain by a eight ft thick limestone stratum. In the model, 68 ft of overburden strata are considered and the rest of the overburden, up to the surface, is modeled by using vertical overburden loads. A floor thickness of 50 ft beneath the coal seam is used for modeling.

Though the model simulates two complete barrier pillars of 80 ft length and two half barrier pillars (40 ft length) at two extreme ends, only half of the mesh is used because of the symmetry (see Figure 2). A five entry panel configuration with 20 ft wide entries and 40 ft wide pillars is used in the model. The length of the actual mesh is 510 ft with one half barrier pillar (at the extreme left) and one full barrier pillar. The model is loaded on the top by uniform vertical loads to simulate 282 ft of overburden which corresponds to 350 ft of overburden thickness (282 ft of loading and 68 ft of actual overburden materials). The bottom of the model is constrained in the Y direction and allowed to move in the X direction. While the right hand side of the model is constrained in the X direction and allowed to move in the Y direction, a uniform displacement is prescribed in the left side of the model to simulate a horizontal stress of 1000 psi in limestones. The region of interest is one half entry and one half pillar at the extreme right (see Figure 3.2) where the element size is 2 ft by 1 ft.

Table 3.1 shows material properties used for different strata and backfill materials for modeling purposes. The backfill material is assumed to have linear properties and four different values of Young's modulus are used (also shown as percentages of coal's Young's modulus in Table 3.1).

The modeling procedures involve two sets of steps--one without any backfill and one with a backfill material. The first set of steps involve:

1. Apply pre-mining stresses to the model without any opening.
2. Excavate all the openings simultaneously within 30 days to simulate formation of entries. Redistribution of stresses occur and vertical displacements in the roof and floor are determined due to the creation of openings.
3. Run the model to simulate nine years at an increment of 30 days. Stresses and strains are updated at every increment. Time dependent strains induce time dependent floor, pillar and roof movements.

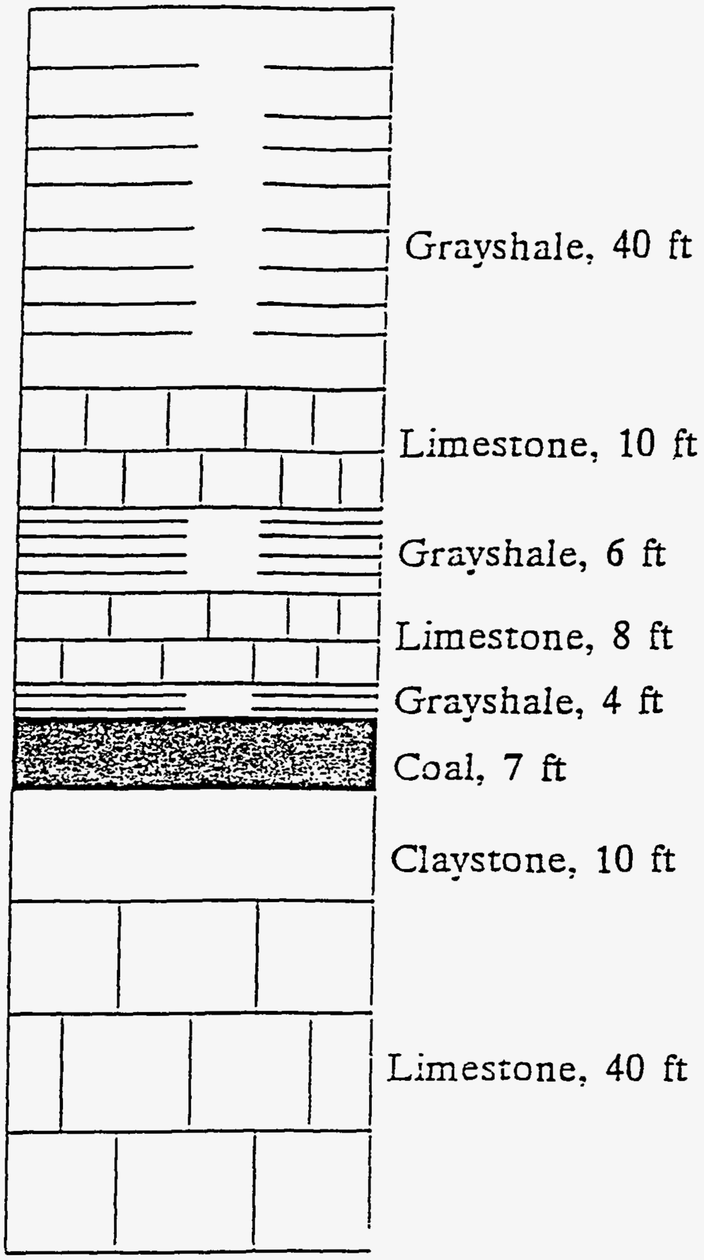


Figure 3.3 - Geologic log used for finite element modeling

Table 3.1 Material properties used for finite element modeling

Materials	Young's Modulus, psi	Poisson's Ratio	Unit Weight, pcf
Limestone	700,000	0.15	170
Gray shale	250,000	0.25	160
Coal	150,000	0.30	75
Claystone	25,000	0.35	160
Backfill materials	6,250 (4%)* 12,500 (8%)* 18,750 (12%)* 25,000 (16%)*	0.25	70

Note: * indicates percentage of coal's Young's modulus of Elasticity

In the second set of steps, the excavations are created as before but the openings are backfilled after one year. The backfill material is born stress-free in the already deformed openings. The model is run for nine years and stresses and strains are updated at an increment of 30 days. Floor heaves, roof sags, and pillar punching are determined from vertical displacements.

Results and Discussions

Figures 3.4 and 3.5 show the maximum floor heave and roof sag at the entry centers for different E (Young's Modulus) values of the backfill material. Also shown is the case when partial backfilling is done (one foot of gap between the roof and the backfill material). The case of partial backfilling considered here is the worst case scenario where complete backfilling cannot be achieved at the injection borehole. In practice, however, complete backfilling is achieved in the region around the injection borehole and voids remain at the periphery of the panel.

Figures 3.6 and 3.7 show the percentage reduction in floor heaves and roof sag due to backfilling. The maximum floor heave after nine years without backfilling is 8.3 inches. If backfilling is done after one year when the maximum floor heave of 5.2 inches has already occurred, the floor heave increases by an amount 0.2 inches for a backfill material whose Young's modulus is 16% that of the coal. Figure 3.6 shows that this amounts to 88% reduction in floor heaves. For an extremely weak backfill material (whose Young's modulus is 4% that of the coal), the reduction in floor heave is 60% due to backfilling. In the worst case scenario when there is one foot gap between the backfill material and the roof, the reduction in floor heave is 48%. A similar reduction is obtained for roof sag as shown in Figure 3.7. Figure 3.8 shows the amount of pillar punching into the floor in the case of no backfilling, backfilling with different E values, and partial backfilling. The figure shows that the amount of pillar punching can be significantly reduced by backfilling.

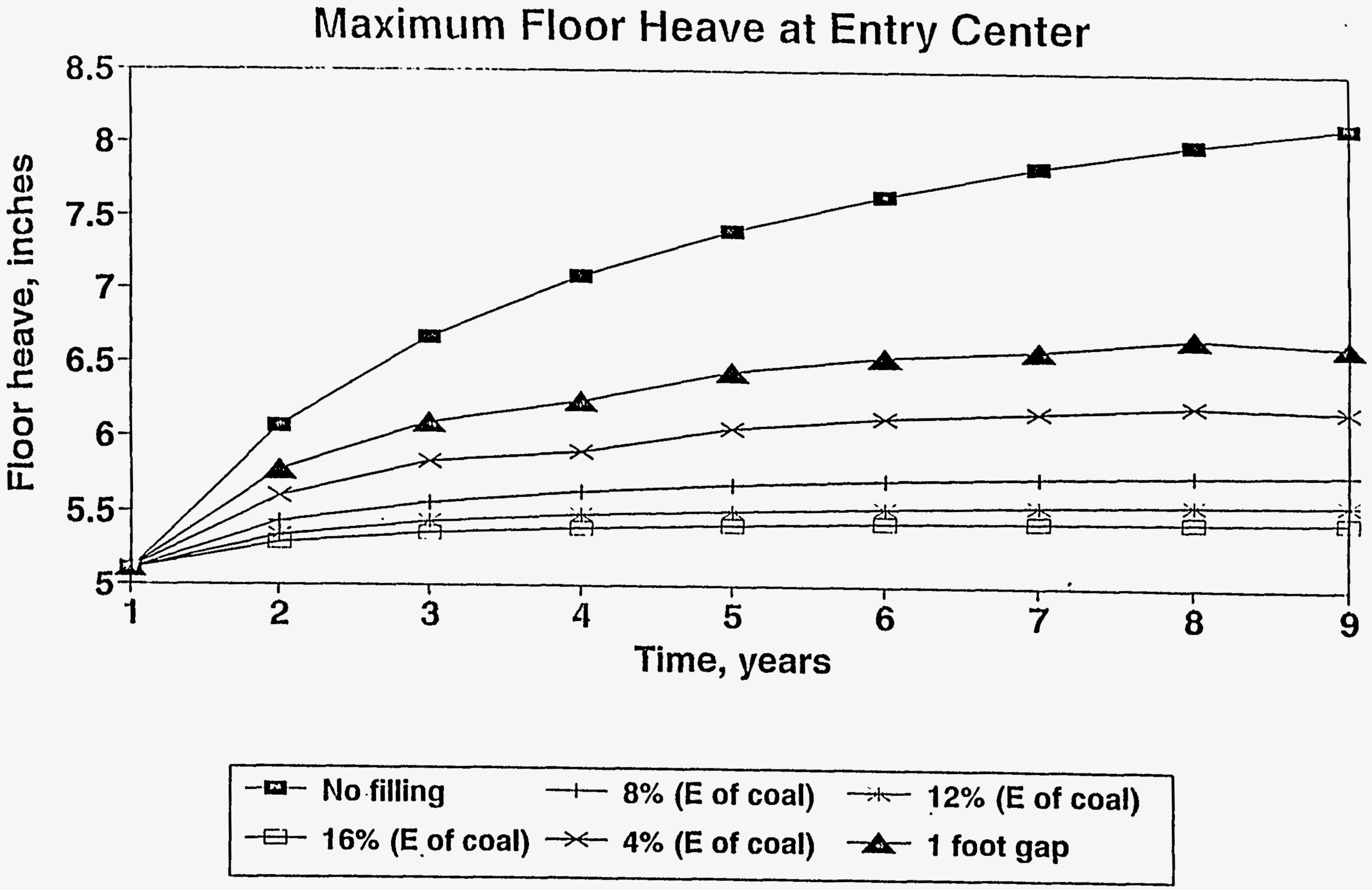


Figure 3.4 - Maximum floor heave at the
entry center

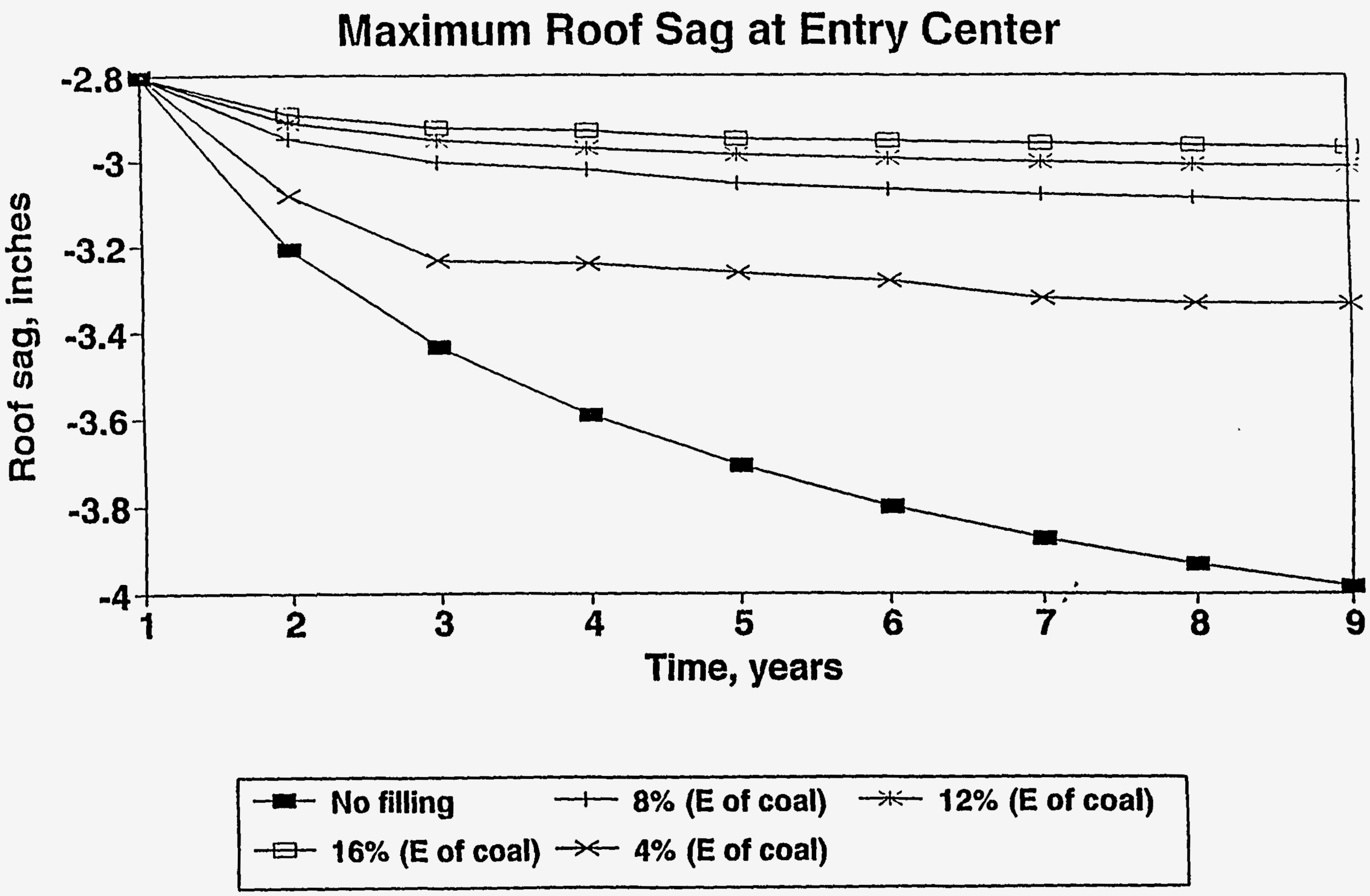


Figure 3.5 - Maximum roof sag at the entry center

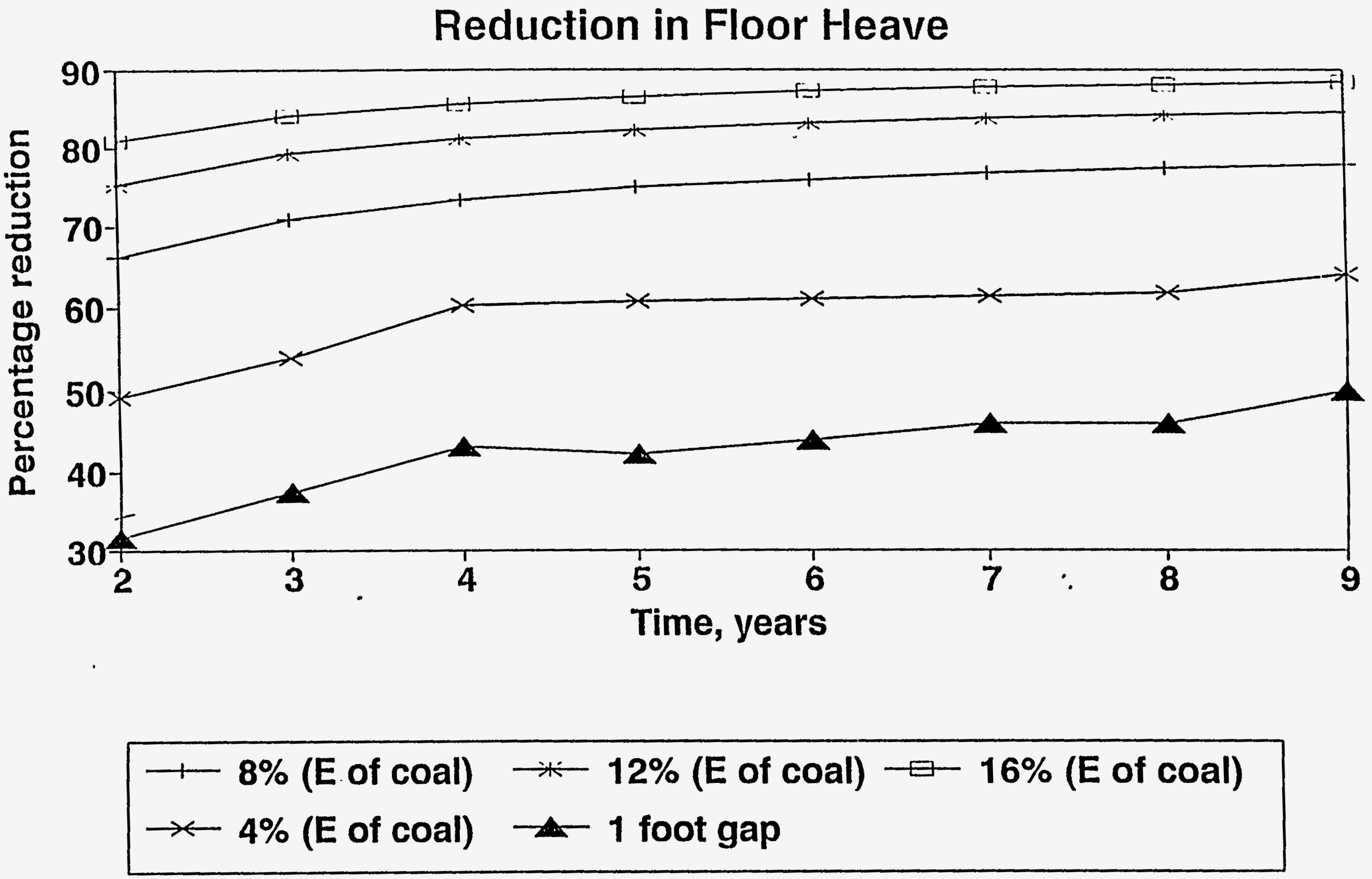


Figure 3.6 - Percentage reduction in floor heave due to backfilling

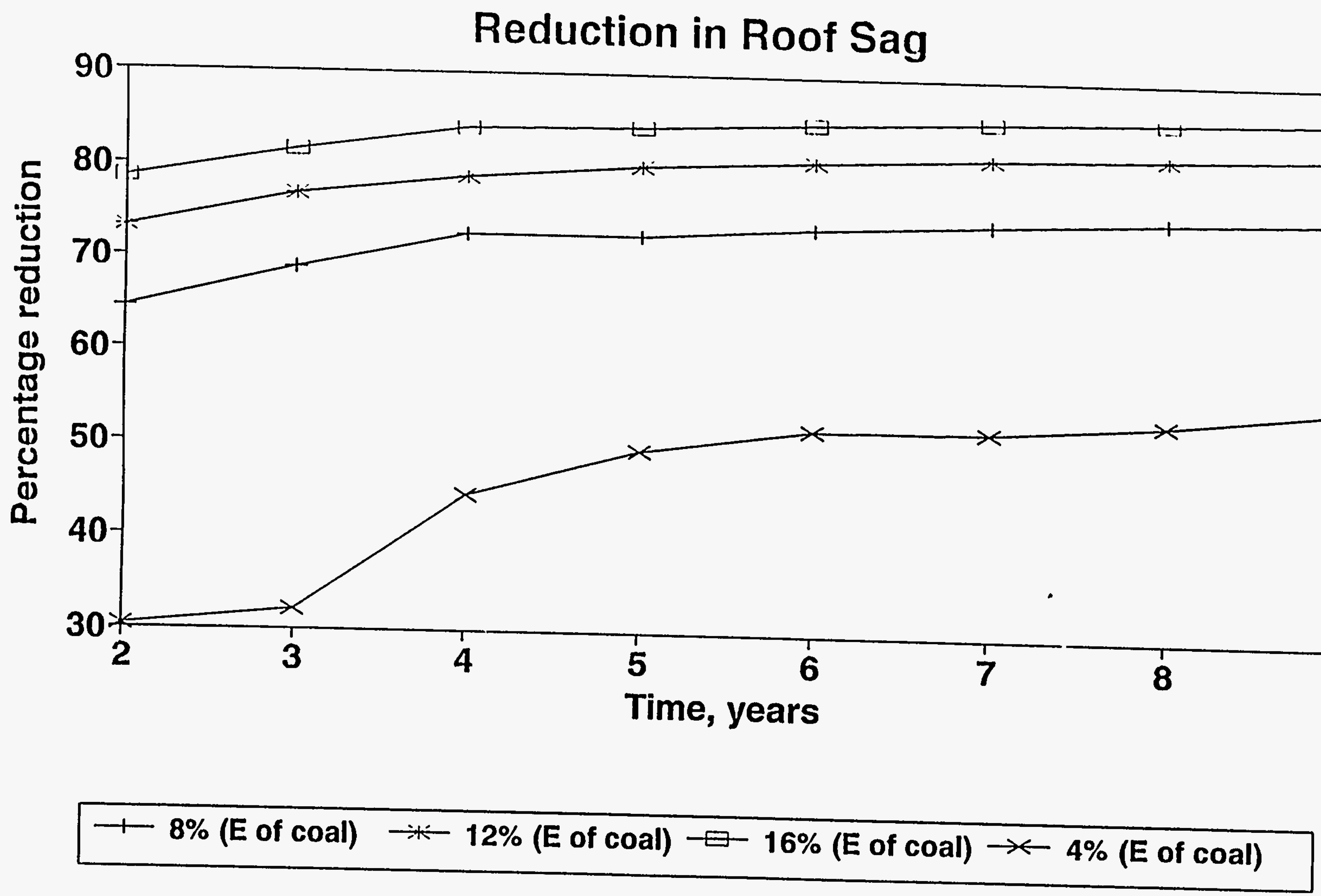


Figure 3.7 - Percentage reduction in roof sag due to backfilling

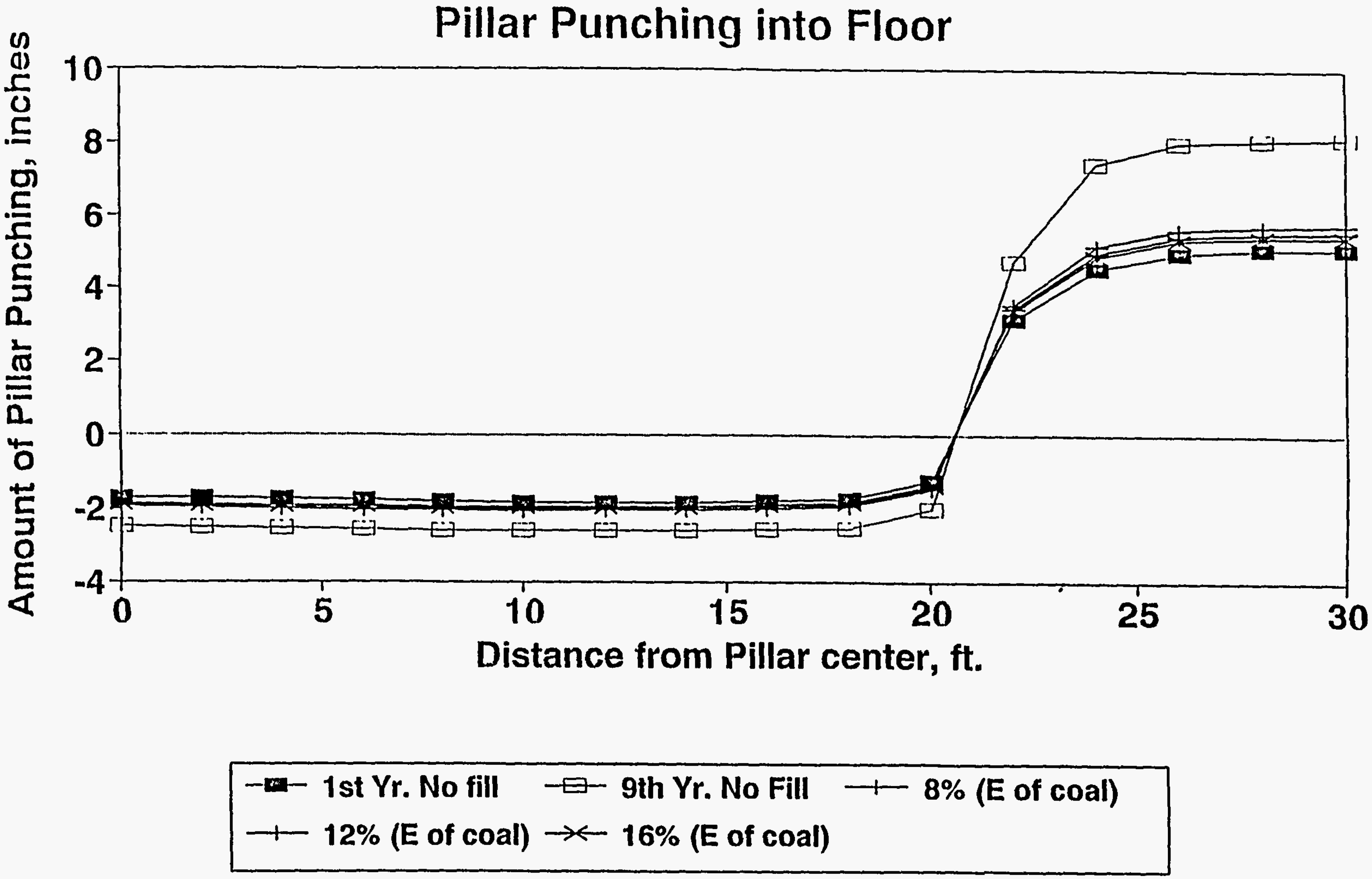


Figure 3.8 - Amount of pillar punching into the floor

Figure 3.9 shows the maximum compressive stresses on the backfill materials for different E values of the back fill material. The figure shows that the maximum compressive stress on the backfill material is between 60 to 170 psi. It implies that not much stress redistribution occurs due to backfilling with weak backfill materials. The reduction of floor heaves, pillar punching, and roof sags can be attributed to the confinement provided by the backfill material. The failure of the backfill material when placed underground is negligible because the stresses on it is very low The analyses show that a weak backfill material can effectively reduce floor heaves, roof sags, and the amount of pillar punching into the floor, which, in turn, can reduce surface subsidence.

Characterization of Pneumatic Mixes

Out of 16 initial trial mixes conducted during the period October 1, 1993-September 30, 1994 (see Annual Report, October 1-1993, September 30, 1994), four mixes of different proportions were selected for further geotechnical characterizations. The mixes were 80-20 and 70-30 (FBC fly ash and spent bed) with 25% and 30% moisture. Initial tests on trial mixes show the average demolded densities as 71 and 73 pcf for 80-20 and 70-30 mixes, respectively, with 25% nominal moisture. The same mixes with 30% nominal moisture have average densities of 71 and 75 pcf, respectively. The 28-day compressive strengths of 80-20 and 70-30 mixes with 25% nominal moisture are 11 and 3 psi, respectively. This indicates an extremely low strength which is attributed to low actual moisture contents of the samples (see Annual Report, 1994, page 38). The same mixes with 30% nominal moisture have average strengths of 92 and 34 psi, respectively. Though the increase in average demolded density of the mixes due to the increased water content is trivial, the strength increase is highly significant (Annual Report, 1994). Further geotechnical characterizations of mixes were conducted at 25 psi compacted pressure.

The pneumatic mix is finalized to have either 80% FBC fly ash and 20% spent bed, or 70% FBC fly ash and 30% spent bed. Considering the current production of FBC fly ash and spent bed, a 80-20 (fly ash-spent bed) mix is more desirable. When water is added to the mix of FBC fly ash and spent bed, the heat of reaction raises the temperature to 99.5 degree centigrade. At this high temperature, the water in the mix vaporizes. It has been reported earlier that a nominal moisture content of 40% is reduced to an actual moisture content of approximately 18% by the time of sample preparation (see Annual Report, 1994, page 38). Similarly, a nominal moisture content of 30% is reduced to an actual moisture content of 10%-12%.

Two tests were performed to study the drop of temperature and moisture with time. Figure 3.10 shows the drop of temperature and moisture with time when the mixing is done in a bowl mixer. A 30% nominal moisture content in the mix drops to 18% after five minutes of mixing (shown at time 0 in Figure 3.10) and eventually drops to 11% after 30 minutes. The temperature drops to 62 degrees centigrade from 99.5 degrees centigrade in the first 30 minutes. It takes two hours for the temperature to drop to the room temperature.

In the field, a drum type mixer (like a concrete mixer) will be used to mix FBC fly ash, spent bed, and water for pneumatic injection. The second test to study the drop of temperature and moisture was done by simulating a drum type mixer. In the drum type mixer, the material falls under its own weight and more surface area comes in contact with the air. This accelerates the temperature drop. Figure 3.11 shows the temperature and moisture drop when simulating a

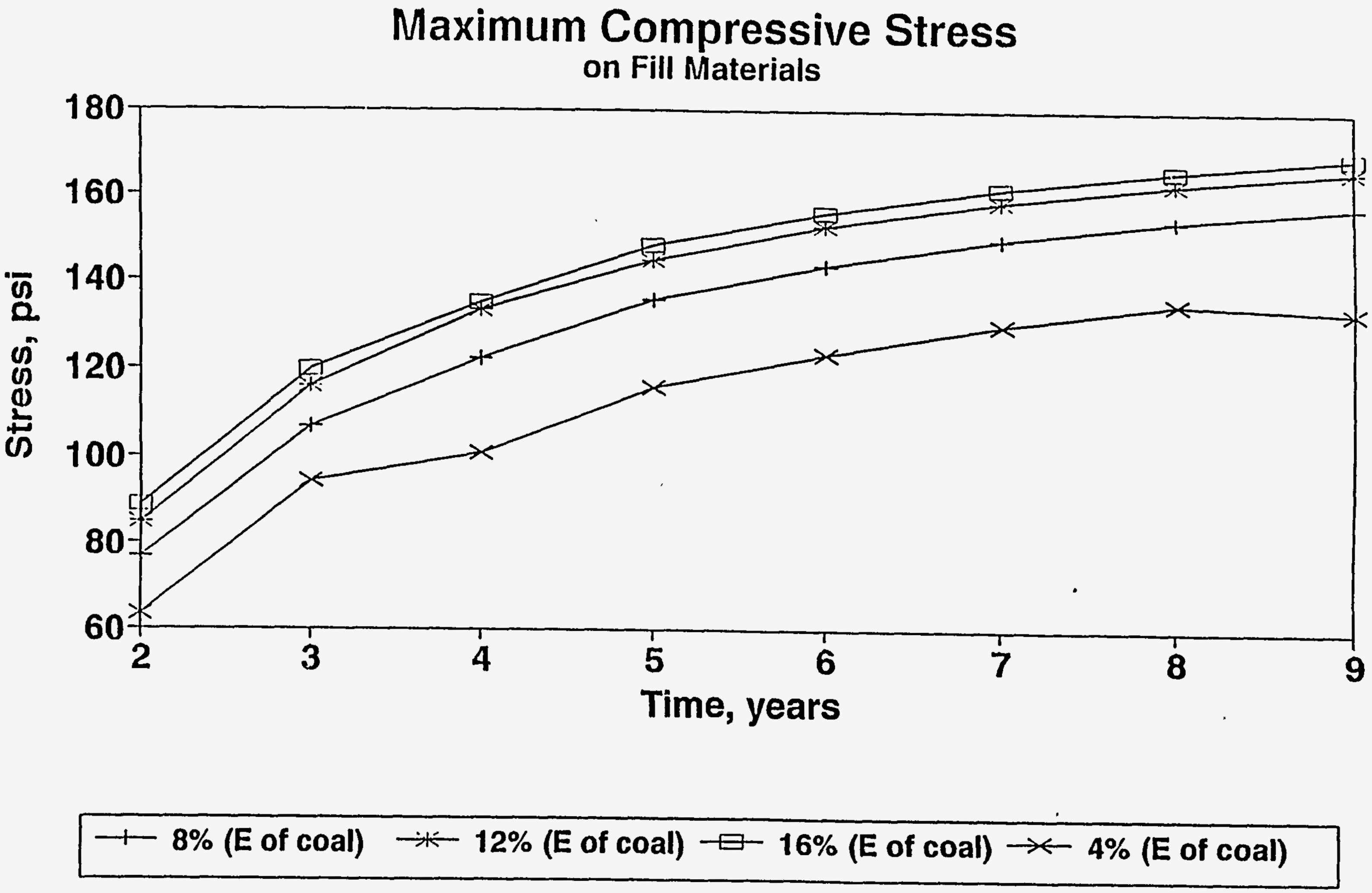


Figure 3.9 - Maximum compressive stress on the back-fill material

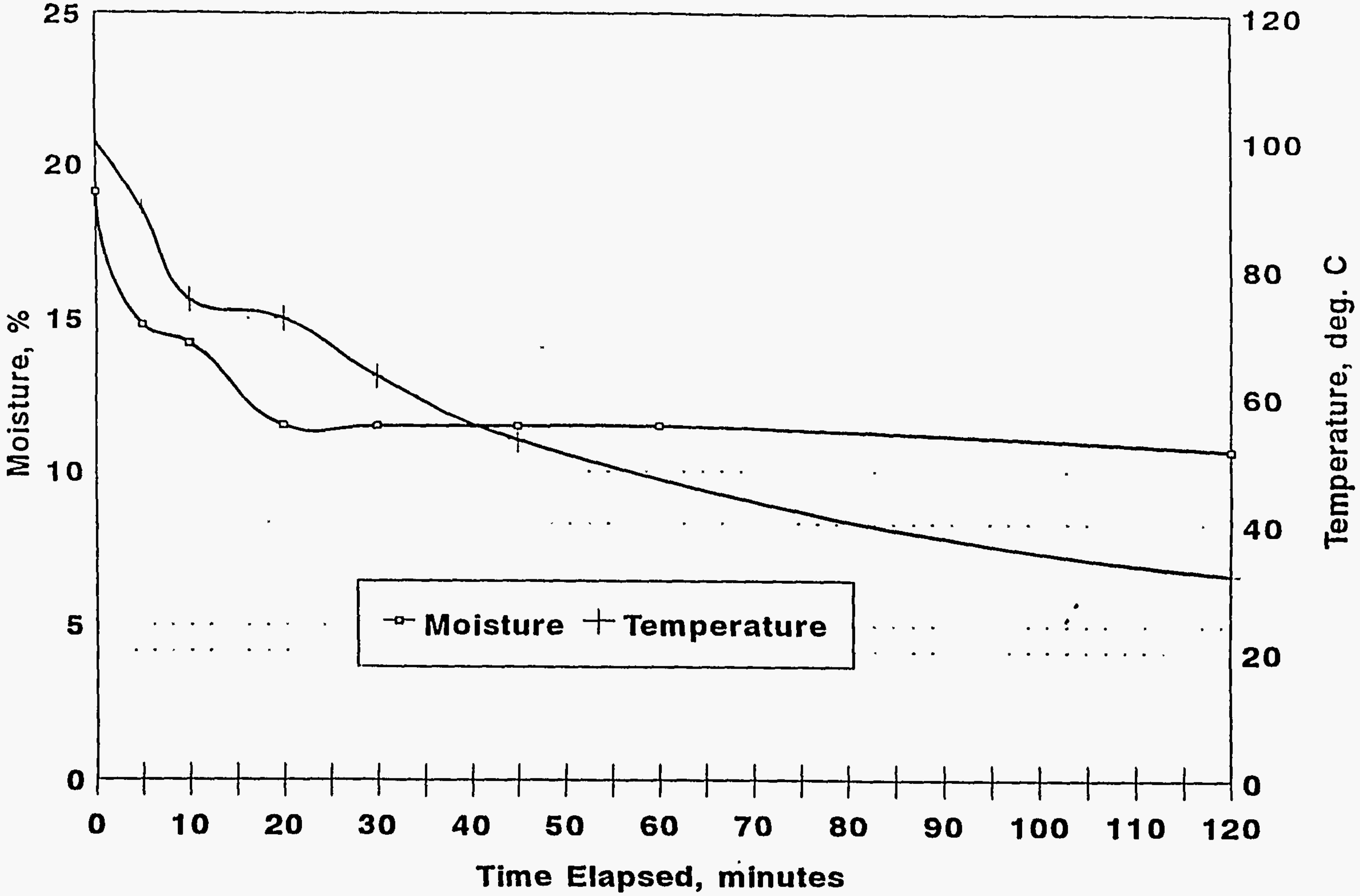


Figure 3.10 - Temperature vs. elapsed time when mixing in a bowl mixer

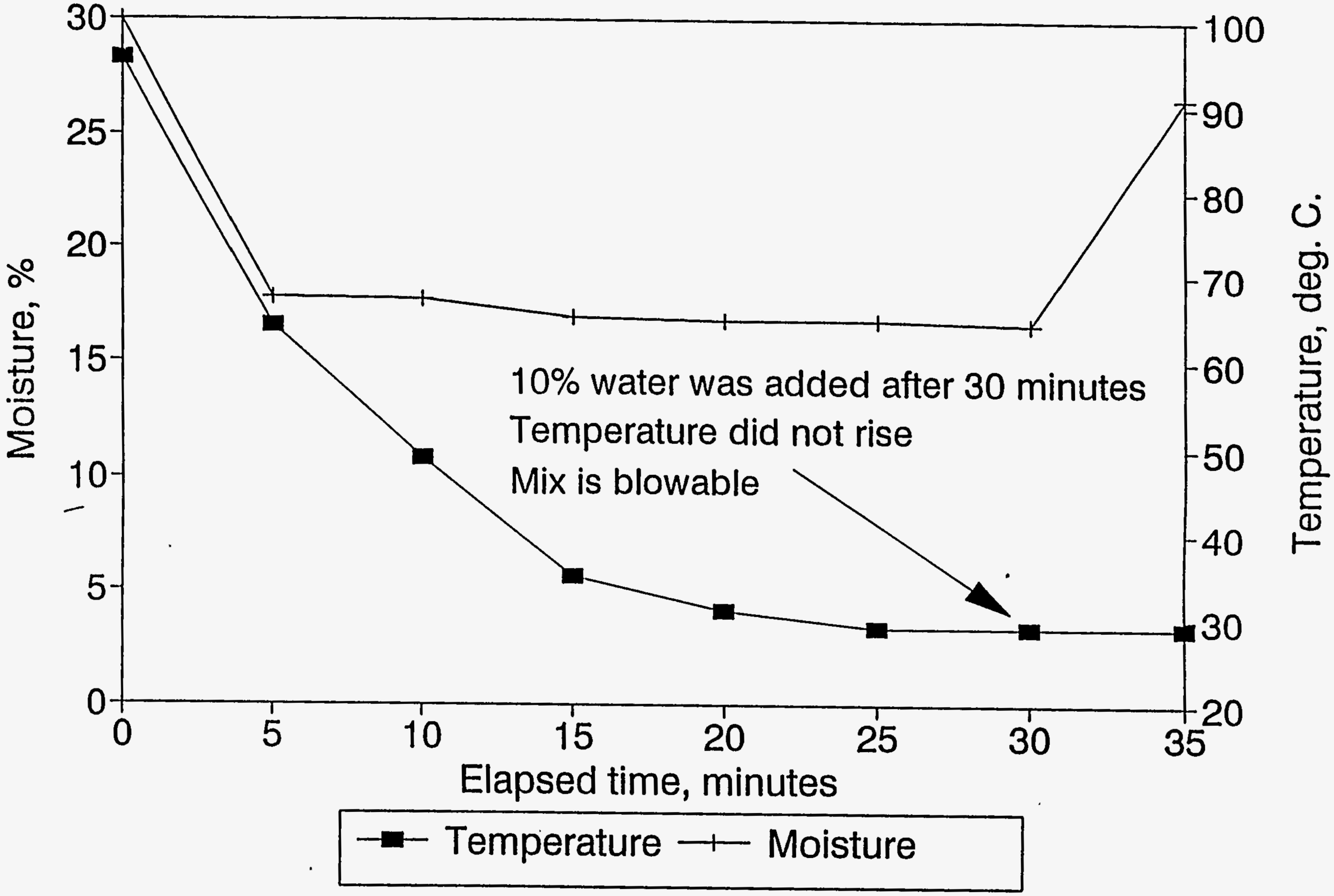


Figure 3.11 - Temperature vs. elapsed time when simulating a drum type mixer

drum type mixer. As before, a 30% nominal moisture content drops to 18% after five minutes. At the same time, the temperature drops to 62 degrees centigrade. Because of the low temperature, the rate of moisture evaporation in the mix is reduced significantly. In 30 minutes, the temperature drops to the room temperature and the actual moisture content of the mix is 17%. A 10% moisture is added to the mix after 30 minutes. This does not increase the temperature further. The actual moisture content of the mix at 27% is blowable. This test suggests that 30% moisture should be added to the mix (FBC fly ash and spent bed) and mixed in a drum type mixer for five minutes. An extra 10% water should be added after five minutes. This will ensure an actual moisture content of 27% to 28% in the mix and the mix is blowable at this moisture content.

Uniaxial compressive strength and Young's modulus of pneumatic mixes

Nine three-inch diameter cylindrical samples with 80% FBC fly ash, 20% spent bed and 28% actual moisture content were tested to determine their 7-day, 28-day and 90-day uniaxial compressive strengths and Young's moduli. Typical stress strain curves for seven-day and 90-day samples are shown in Figures 3.12 and 3.13, respectively. The seven-day compressive strengths of three samples were 28, 39, and 54 psi with an average of 40 psi. The average Young's modulus of the three samples was 3850 psi. The 28-day average compressive strength of the samples was 72 psi and the average Young's modulus was 5195 psi. The 90-day average compressive strength of the samples was 148 psi and the average Young's modulus was 8735 psi. The average modulus of two-week water-saturated samples is greater than 8,000 psi.

The design of pneumatic mix calls for a mixture which can be blown into the underground. This necessitates low percentage of moisture and consequently low strength and modulus values. (The strength can be increased considerably by adding additional moisture or making the mix a paste or by increasing the compaction pressure).

Finite element modeling confirms that a very high strength material is not needed to effectively control the surface subsidence which is mainly caused by floor heave and pillar punching into the floor. The stiffness of the material will govern the amount of reduction in the floor heave and consequently the surface subsidence. All the above test results are for unconfined compression condition. In the underground confinement condition, the strength and stiffness values will increase due to the confinement.

Injection Wells and Underground Conditions

Observations by borehole camera in the pneumatic injection hole indicate dry borehole walls. The hydraulic injection hole indicates seepage of water from the sandstone unit 15 to 30 ft above the roof-line. The heights of the openings in both panels are approximately 6.75 ft. The pillars in the pneumatic panel do not show any sign of spalling and the old timber posts are in good shape. The immediate roof of the pneumatic panel is limestone. The pillars in the hydraulic panel show signs of spalling and the old timber posts are buckled, indicating load on them. The immediate roof of the hydraulic panel is 2.5 ft thick shale. A six-inch separation exists between the shale and the overlying limestone. Openings in both panels are in good conditions.

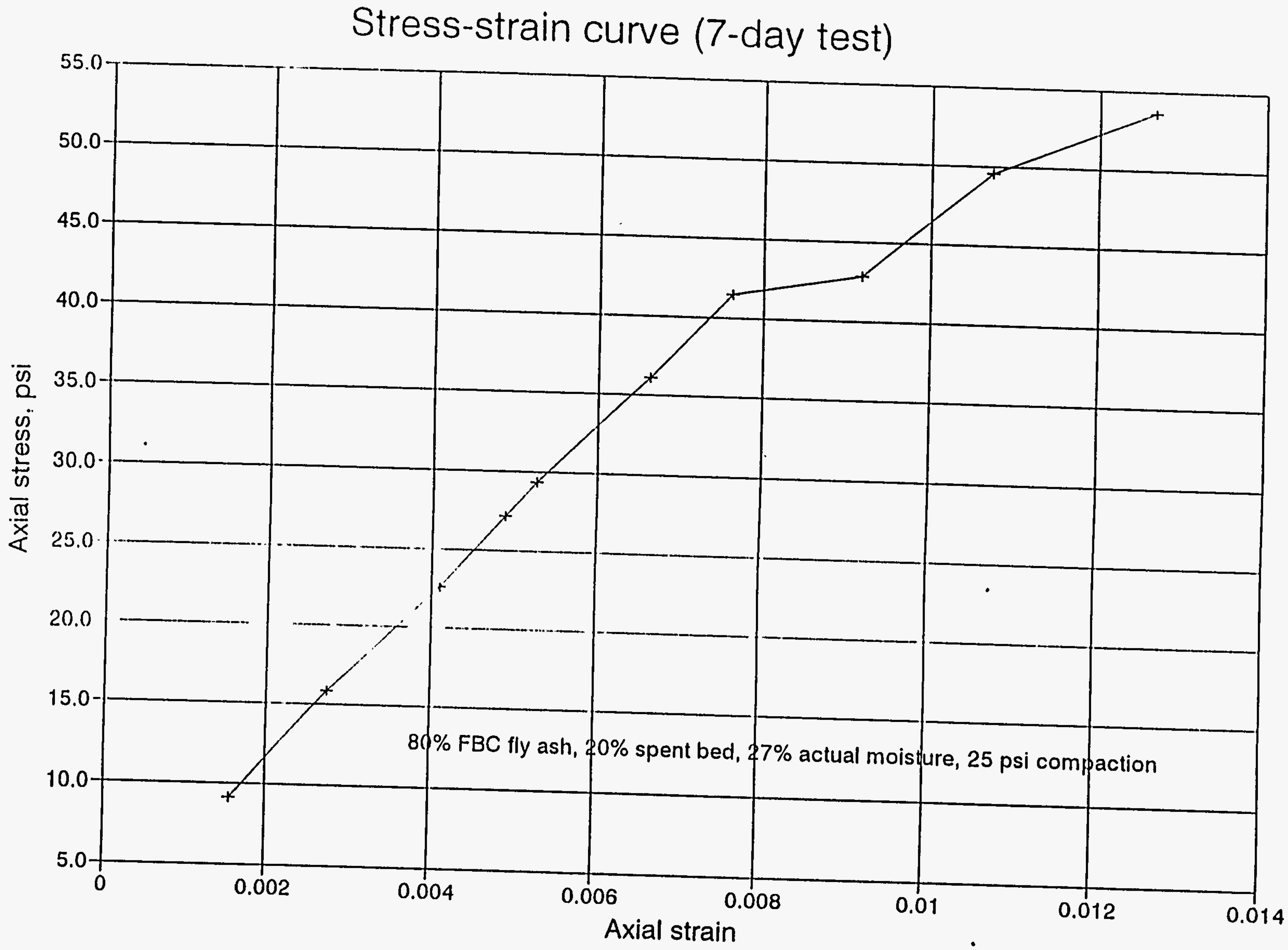


Figure 3.12 - Typical stress-strain curve for 7-day test

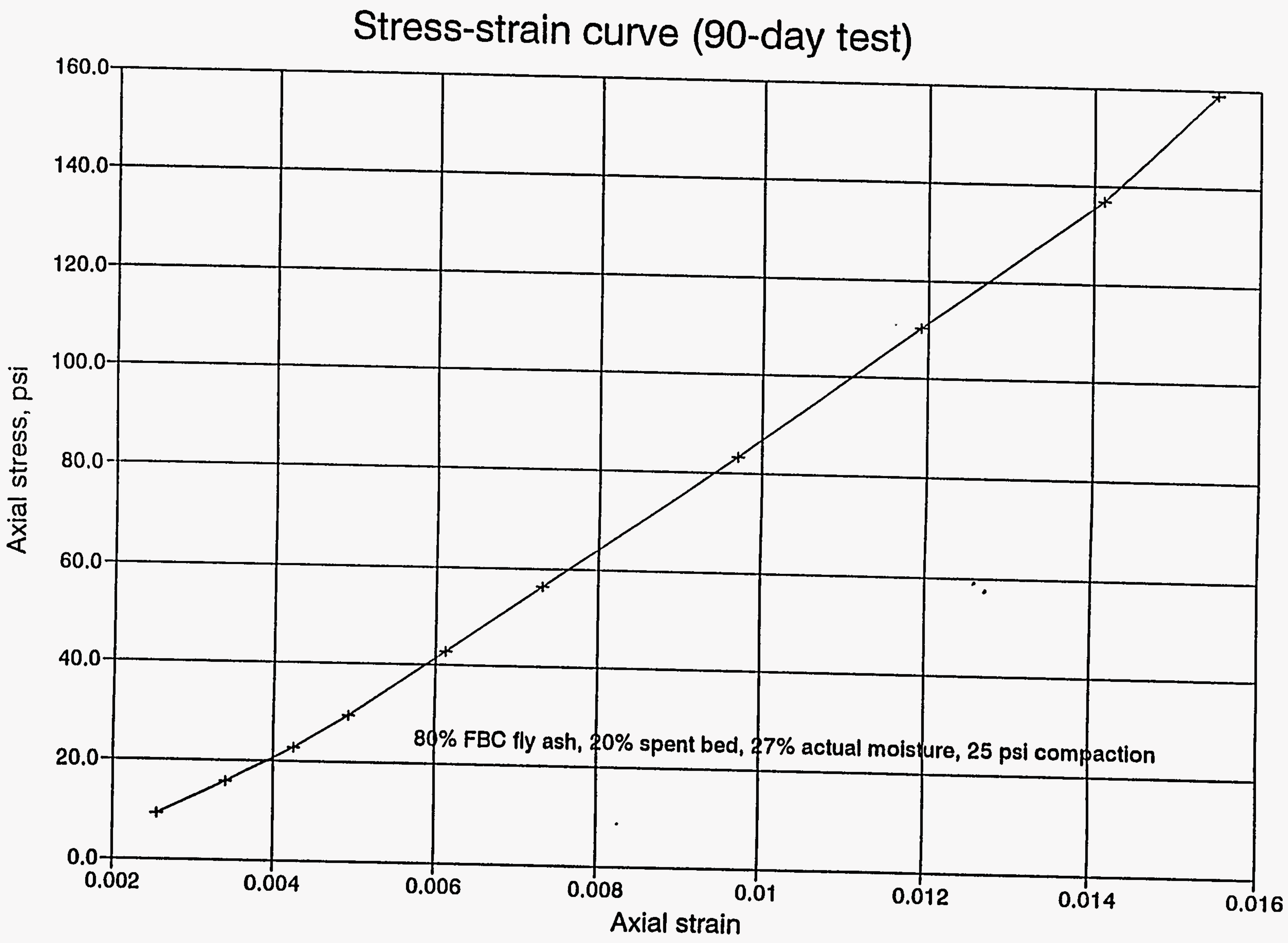


Figure 3.13 - Typical stress-strain curve for 90-day test

Oxygen in the air coming through the boreholes from the panels was 3%, indicating high content of methane. Methane in the underground air cannot be measured because of the low oxygen content in the air. No carbon monoxide in the underground air is detected. A complete analysis of underground air quality is planned for the future.

Installation of Subsidence Monuments and Subsurface Instrumentation

Sixty subsidence monuments have been installed over the pneumatic and hydraulic injections panels. Figure 3.14 shows the grid employed to install the subsidence monuments. The grid consists of subsidence-monument-lines in the transverse and longitudinal directions over the panels Also, four subsidence-monument-lines at angles to the longitudinal directions of the two panels were installed. This grid would provide enough surface movement data for developing contours of vertical surface movements over the two panels prior to and after the backfilling. The surface monuments were spaced at 25 ft intervals in the longitudinal and transverse directions. The subsidence-monument-lines extend at least 100 ft beyond the panel edges.

A subsidence monitoring point consists of a seven ft long bolt that is 7/8-inch in diameter and has a frost free design. The monitoring point is installed by augering a hole (four inches diameter) down to a depth of three ft, inserting the seven ft long roof-bolt (a rebar) through a three ft of foam insulated, two inches diameter PVC pipe, and then hammering the bolt between 3.0 and 3 5 ft into the ground. As the frost heave zone in this area is reported as 18 inches deep from the surface, the subsidence monuments are free from the effects of ground freezing.

Two geotechnical holes were drilled (for locations, see Figure 3.1) for installing settlement sensors in the boreholes above the coal seam (up to 50 ft above the roof-line). The settlement probe system consists of the probe with the cable reel, an integral readout device, and the corrugated plastic casing with settlement sensors. Fifty feet of corrugated plastic casing with sensors (metal rings) at every ten ft were installed at the bottom of each geotechnical borehole of six-inch diameter. Each corrugated casing is attached to the bottom of a four-inch diameter PVC pipe-column installed all along the borehole. The PVC pipe column is built by coupling 10 ft sections of PVC pipes. The annulus between the borehole wall and the PVC pipe-column is filled with silica sand.

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

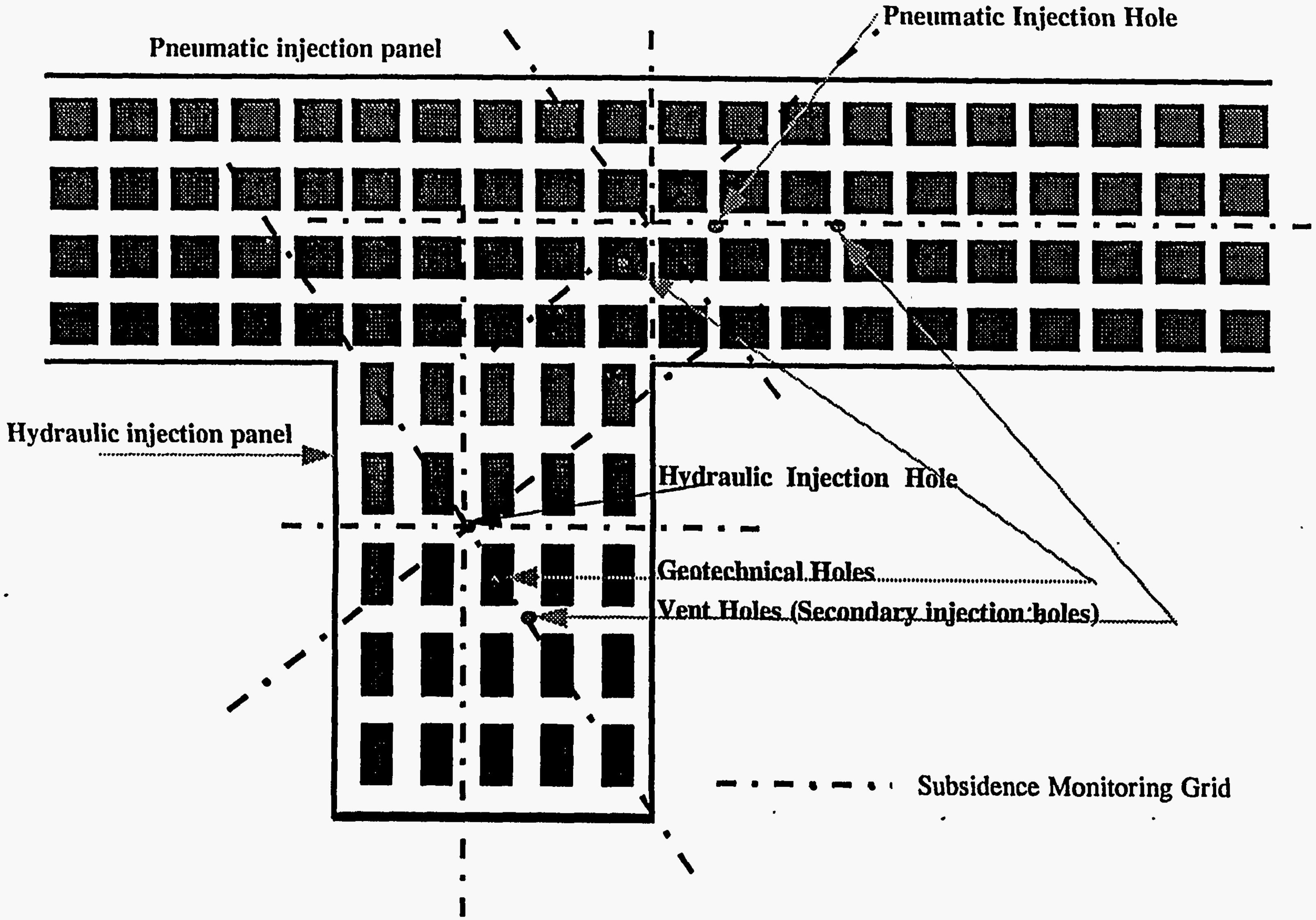


Figure 3.14 - Layout of subsidence monuments

Mechanical Characteristics Of Mixes

Study of the short term strength

For the study of the short term strength, eight samples consisting of 40% fly ash, 55% scrubber sludge, and 5% lime waste were made with 29% water. They were divided into three groups according to the different curing time, as shown in Table 3.2

Table 3.2 Samples and their curing time

No. of sample	Group	Days in the mold	Days in air chamber	Days in steam chamber	Days in oven at 40 Centigrade
A	1	2	9	0	0
B	1	2	9	0	0
C	1	2	9	0	0
11	2	2	1	2	1
33	2	2	1	2	1
A1	3	2	3	2	1
A2	3	2	3	2	1
A3	3	2	3	2	1

Table 3.3 Water contents of sample group 2 and group 3 at different curing stages

No. of sample	Group	At molding	Demold	After air chamber curing	After steam chamber curing	After keeping in oven	After test
11	2	29%	24.38%	22.75%	26.94%	16.63%	16.63%
33	2	29%	23.92%	21.38%	25.85%	14.38%	14.38%
A1	3	29%	21.11%	19.98%	24.54%	13.75%	13.75%
A2	3	29%	21.00%	20.68%	24.37%	14.36%	14.36%
A3	3	29%	21.18%	20.35%	23.96%	11.71%	11.71%

The water contents of group 2 and group 3 samples were measured during different curing stages (Table 3.3). Mechanical tests for determining the ultimate uniaxial compressive strength and Young's modulus of the samples were conducted. The samples were tested in a frame which is a specially designed apparatus for the propose of determining the ultimate uniaxial strength either a cylindrical or a cubic sample. The test procedure was made in the light of standard test method for compressive strength of cylindrical concrete specimens described in ASTM The test results are given in Figure 3.15 through 3.17

Figure 3.15

Stress—Strain relationship curve of sample group 1
(40% F.A. , 55% S.S., 5% L. W. , mixed with 29% water)

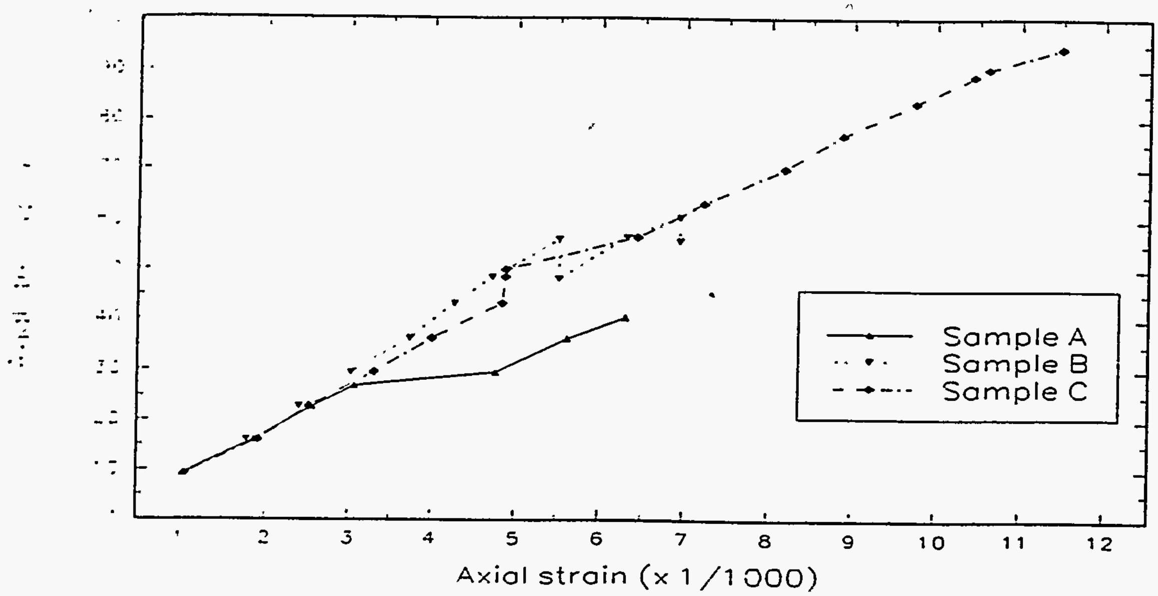


Figure 3.16

Stress—strain relatioship curve of sample group 2
(40% F.A., 55% S.S., 5% L.W., mixed with 29% water)

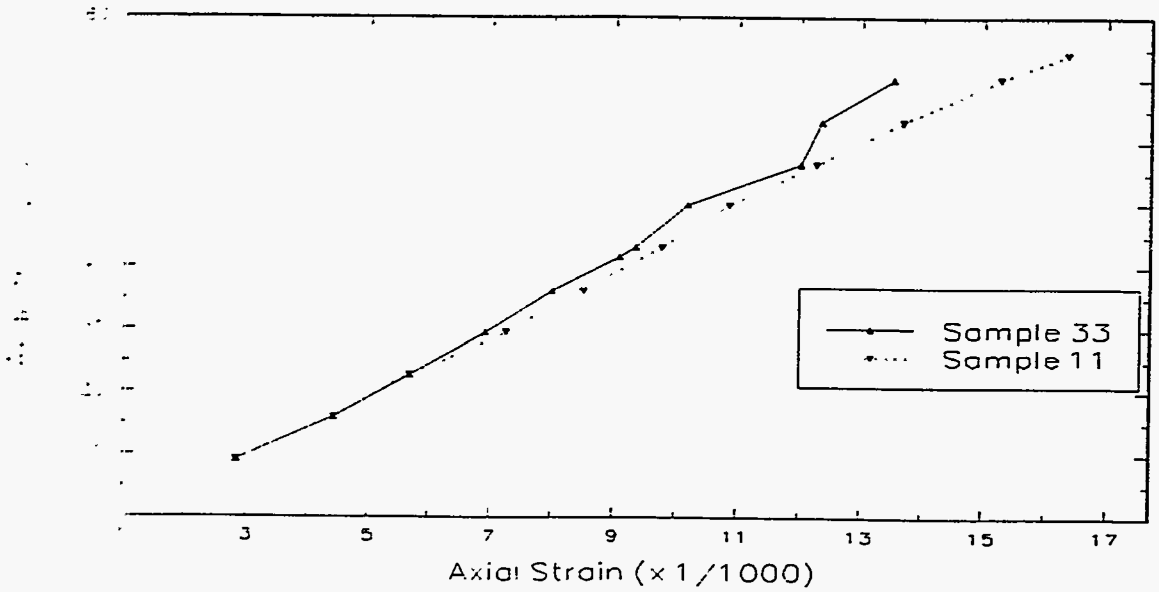
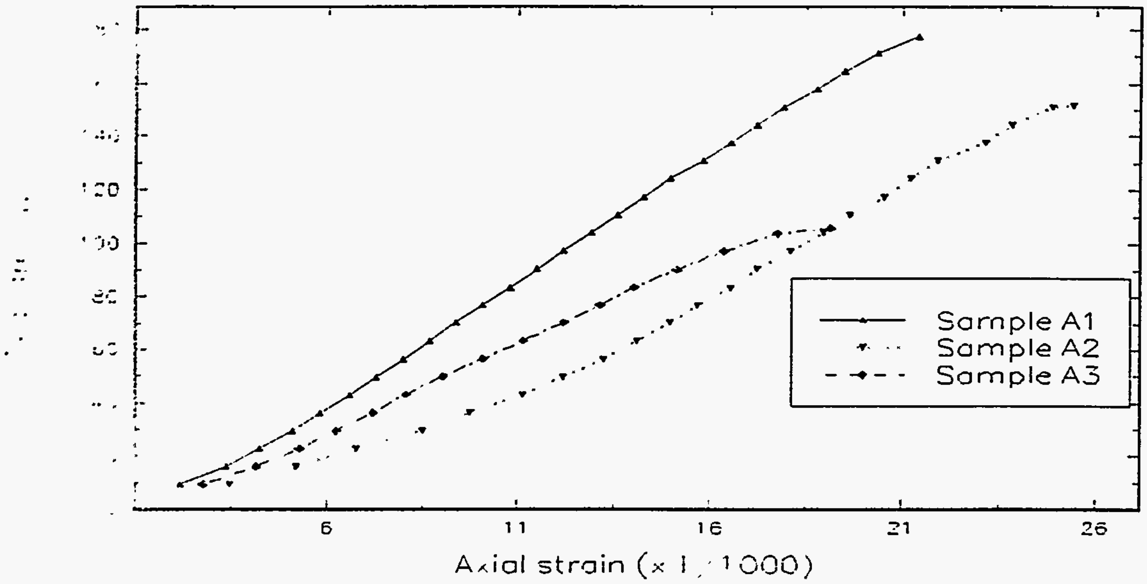


Figure 3.17

Stress—strain relationship curve of sample group 3
(40% F.A., 55% S.S, 5% L.W., mixed with 29% water)



The mean ultimate compressive strength is 63 psi for group 1, 72 psi for group 2, and 145 psi for group 3. The mean Young's modulus is 8500 psi for group 1, 4250 psi for group 2, and 5100 psi for group 3. It is obvious that the samples which were put into the steam chamber and oven had higher ultimate compressive strength, but lower Young's modulus. The reason for the reduction of Young's modulus is that the samples cured in the steam chamber (at a temperature of about 80 degrees centigrade for two days) followed by oven curing (at a temperature of 40 degrees centigrade for one day) developed small cracks, and thus had a much larger deformation and a more obvious hardening phenomenon than those samples which were not cured the steam chamber / oven.

Study on the long-term strength of mixes

To determine the long-term strength characteristics of hydraulic mixes, two groups of samples were tested. Group 1 consisted of five cylindrical samples of 40% fly ash, 55% scrubber sludge, 5% lime waste, and 31% water. After being molded, the samples were put into an aerial curing chamber for 25 days. After curing, the samples were tested. The average axial compressive strength is 158 psi, and the average Young's modulus is about 9900 psi. Figure 3.18 shows the test result.

Group 2 includes 4 cylindrical samples of 44% fly ash, 55% scrubber sludge, 5% lime waste. and 29% water. The curing time for this group of samples was 35 days. The average axial compressive strength is about 184 psi, and the corresponding Young's modulus is about 10100 psi. Figure 3.19 shows the test results of this group. Compared with the shorter curing time strength test results mentioned above, the average strength of the samples which were cured for a

Figure 3.18

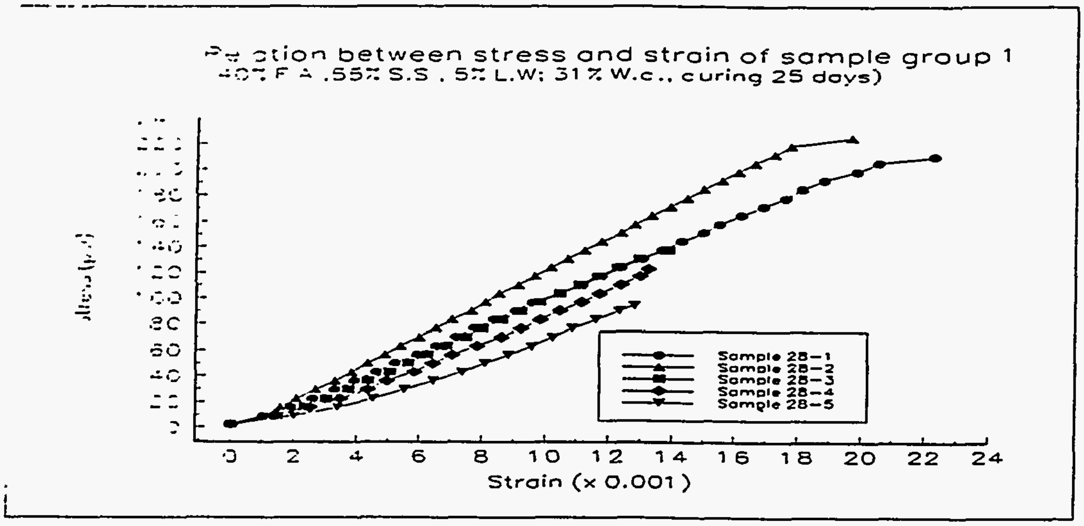
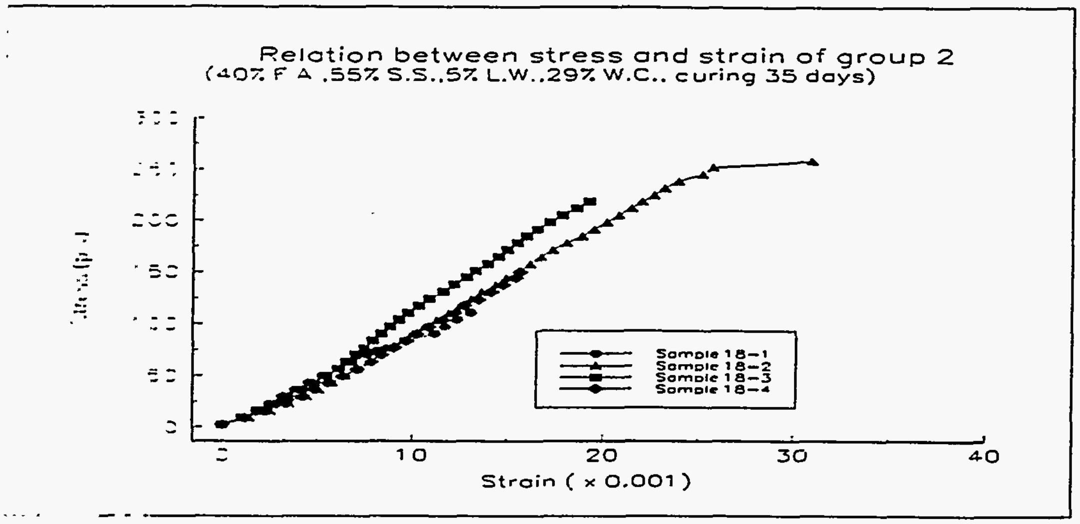


Figure 3.19



longer time period was greater than that of the samples which were cured for a shorter time period. Figures 3.20 and 3.21 give these comparison results.

Figure 3.20 Relation of strength and curing time

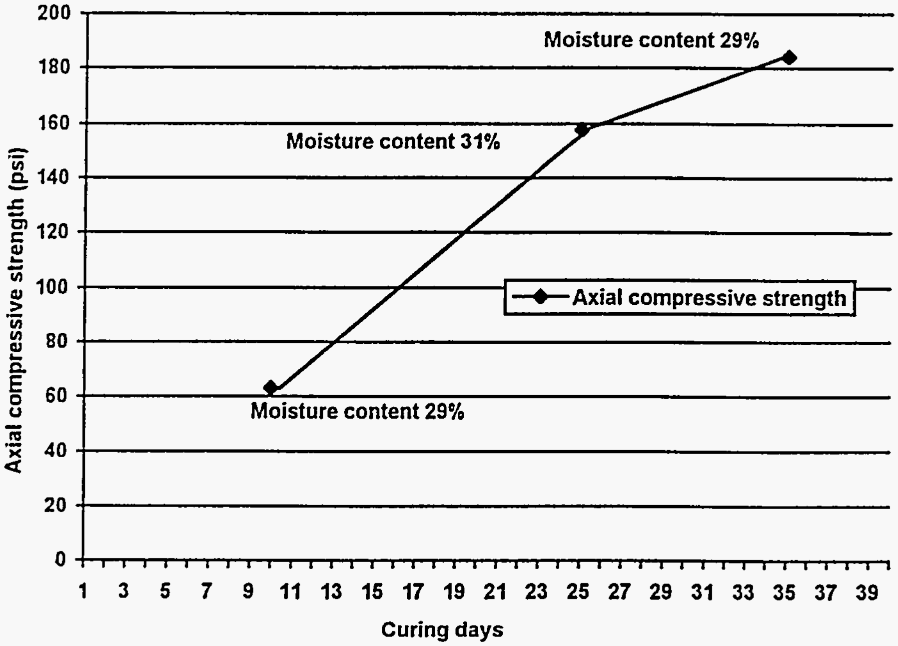


Figure 3.21 Relation of Young's modulus and curing time

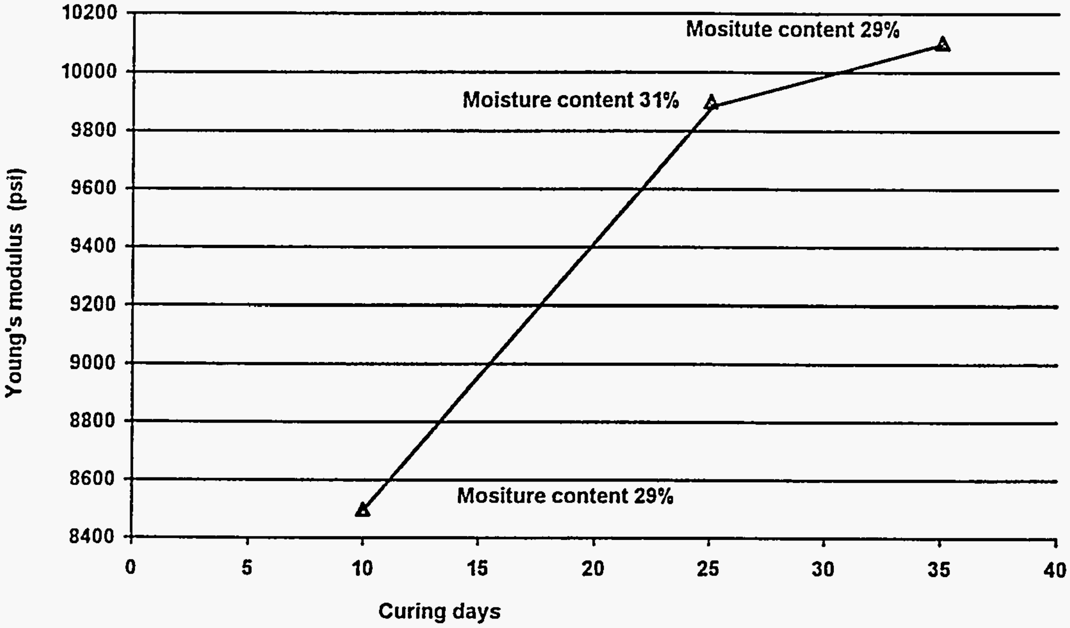


Figure 3.22

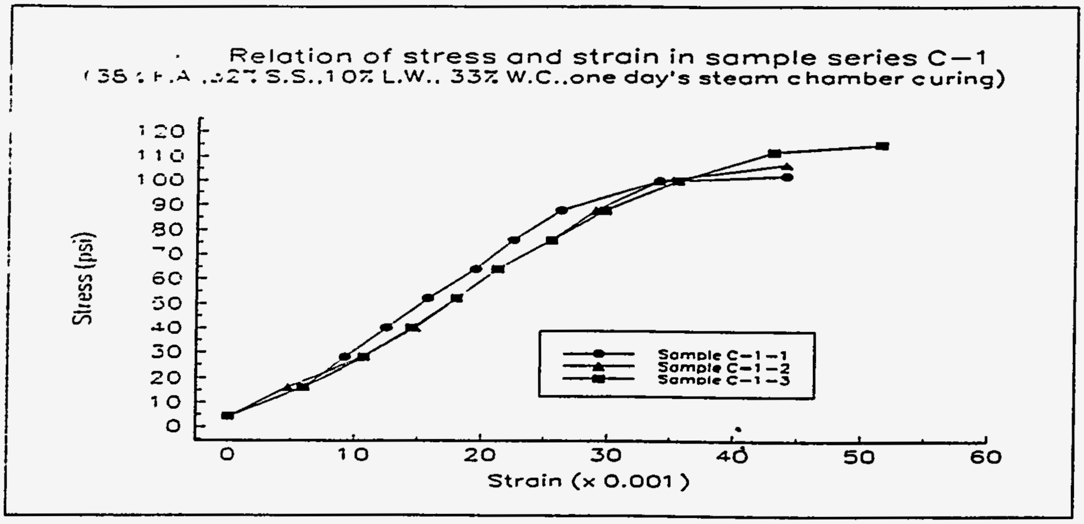
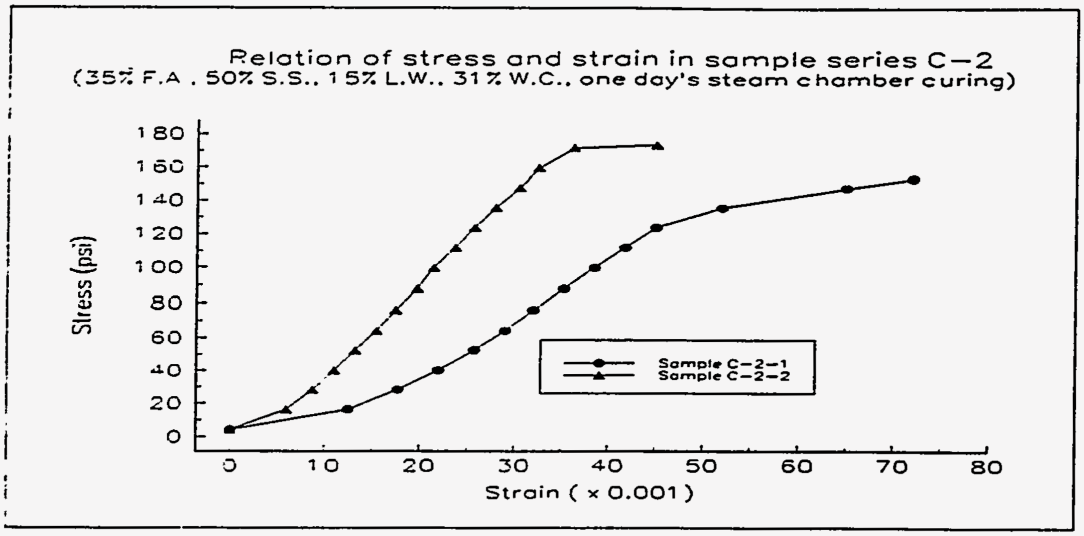


Figure 3.23



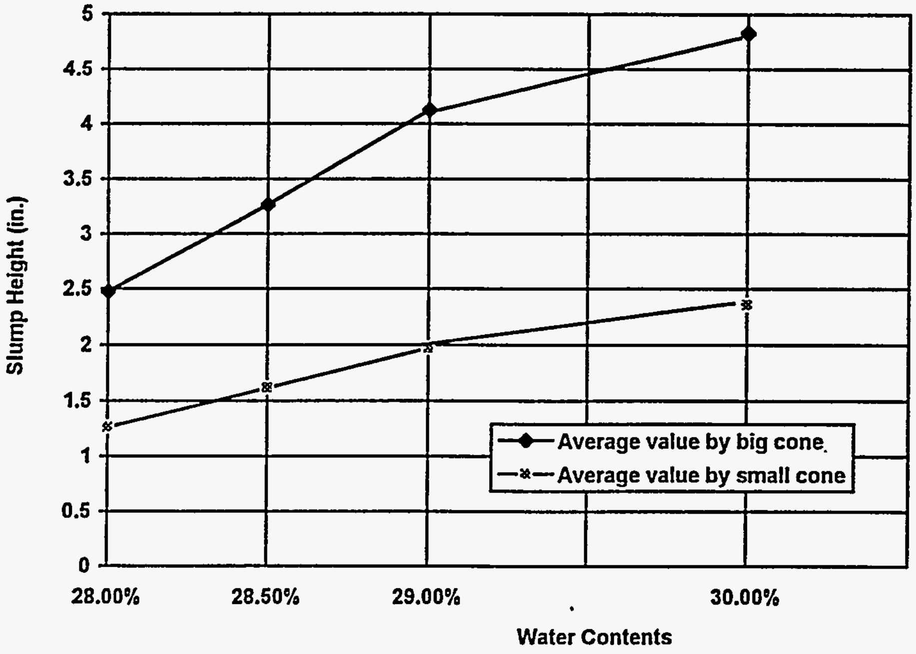
The mixtures tested include the following combinations:

(1). 40% fly ash, 55% scrubber sludge, 5% lime waste, 28% water;
(2). 40% fly ash, 55% scrubber sludge, 5% lime waste, 28.5% water;
(3). 40% fly ash, 55% scrubber sludge, 5% lime waste, 29% water;
(3). 40% fly ash, 55% scrubber sludge, 5% lime waste, 30% water.

Slump height

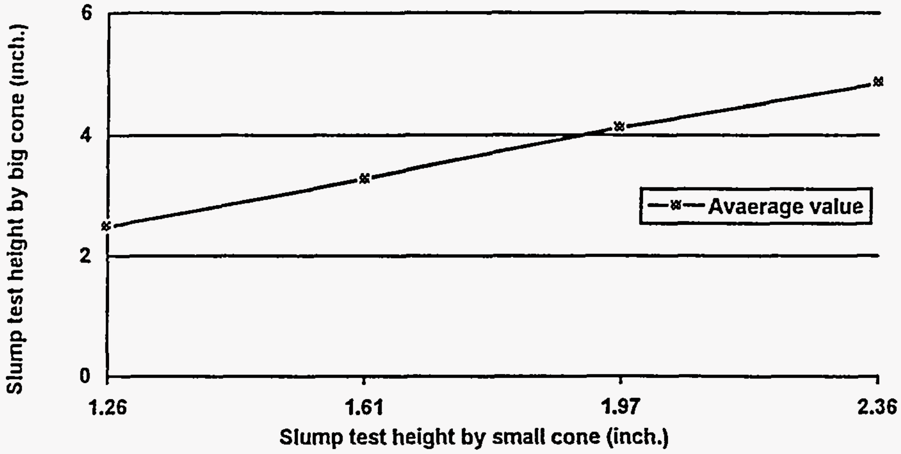
The slump tests were conducted in light of the test procedures mentioned in ASTM. The slump heights are shown in Figure 3.25.

Figure 3.25 Curve of Slump test results



The relationship between the slump heights obtained by big cone and those by the small cone is plotted in Figure 3.26, which indicates that the slump height of the small cone is approximately half of that of the big cone.

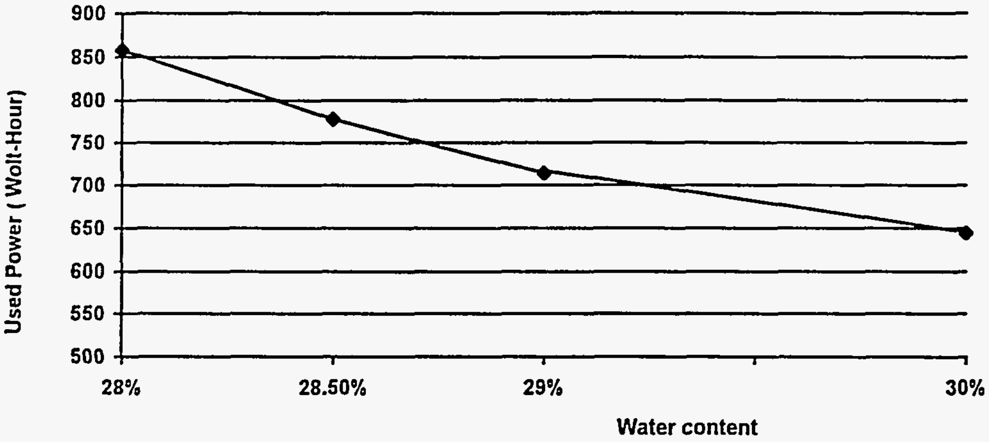
Figure 3.26 Relationship between the slump test results by big cone and small cone



Power used in the mixing of mixes

In each slump test, the power used in the mixing of materials was measured. Figure 3.27 gives the results.

Figure 3.27 Relationship of used power with moisture content



It can be seen that the power required for mixing the materials decreases with the increase in moisture content.

Conclusions

Based on the studies mentioned above, the following brief conclusions can be made:

A. Curing time can increase the strength of mixes. The longer the curing time, the stronger the mixes. However, the rate of increase of strength decreases with the increase of curing time.

B. Lime waste has an obvious influence on the strength of the mixes.

C. Moisture has a noticeable influence on the slump height of the mixes.

D. The correlation between the slump height of ASTM cone and the slump height of small cone is very good. Therefore, the small cone has a potential and should be used in the slump test because it is more convenient than the ASTM cone.

E. The higher the moisture of the mixes, the smaller the power required for mixing the mixes.

References

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2. ANSI/ASTM c39-72, Standard test method for compressive strength of cylindrical concrete specimens, pp 25-27;

3. ASTM Designation c 143-74, Standard test method for slump of Portland cement concrete, pp 97-98.

4. Chugh, Y. P., W. M. Patel, and R. Caudle (1994) Simplified three-dimensional roof and pillar interaction analysis model, *Final Report Submitted to the Generic Mineral Technology Center (Mine System Design and Ground Control)*, Mining and Mineral Resources Research Institute, Virginia Polytechnic Institute and State University, Blacksburg, Virginia 24601, June, pp. 179.

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CHAPTER 4

MATERIALS HANDLING AND SYSTEM ECONOMICS

DR. H. SEVIM
CO-PRINCIPAL INVESTIGATOR

Abstract

Safe and economic transportation of coal combustion by-products from power plants to mine sites is an important segment of this overall program. A number of transportation alternatives have been examined. In this chapter transportation alternatives are applied to hypothetical cases pertaining to central and southern Illinois. The operating scenarios are described and a comparative economic analysis is conducted. Each alternative is evaluated for varying distances and tonnages to reveal its favorable operating range.

Objectives

The objectives of the materials handling research are: 1) To identify the systems that are technically, economically, and environmentally feasible in handling and transporting the coal combustion residues from the power plant to the injection site, 2) To demonstrate the operation of one or two of the identified systems.

The objectives of the system economics research are: 1) To conduct economic analyses of the selected materials handling and underground residue placement systems, and 2) To develop a generalized "Economic Evaluation" model that can be used in evaluating various types of materials handling and placement systems for different distances and tonnages.

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

Summary Of Activities From October, 1994 To September 30, 1995

In the past, several handling and transportation systems have been evaluated, among these three were found to be technically feasible and environmentally acceptable. These systems were:

1. Pneumatic trucks (PT)
2. Pressure differential rail cars (PD-car)
3. Collapsible intermodal containers (CIC)

Detailed engineering and economic analyses of these systems have been conducted during the last 12 months. Operating scenarios have been revised after visiting existing operations, meeting with several equipment manufacturers, and deliberating with research partners. The collected information have been used in "EXCEL" spreadsheets to execute engineering and cost computations of the systems. These spreadsheets have been revised a few times to make them user friendly and to reflect the four modules of handling and transportation mentioned above. A document describing the line items in these spreadsheets was also completed. In this document, the source of data entry, the assumptions behind the computations, and the step-by-step computations are described so that the reader can verify and understand the logic of the spreadsheets.

The economic evaluation program that was developed earlier has been modified to reflect leasing as well as capitalization options. The algorithm of this program is based on the principle of "Net Cash Cost" analysis which is also known as "After-Tax Cost" analysis. This program provides the net present cost of the system, annual equivalent cost, after-tax cost-per-ton, and before-tax price-per-ton to be charged to make the minimum required rate of return on investment.

The three systems mentioned above were economically evaluated using the developed spreadsheets and the after-tax-cost analysis program. Revised system operation scenarios and the outcomes of the economic analyses were presented at the 12th International Pittsburgh Coal Conference as a technical paper entitled "Economics of Coal Combustion Residue Transportation". A copy of this paper is included in this report as Appendix A.

Development of an Integrated Software

Three software were used in the engineering and economic evaluations of the handling /transportation systems: the sketches of the systems were drawn using AUTOCAD, engineering and economic computations were done on EXCEL spreadsheets, and economic analyses were conducted using a software written in FORTRAN language. The evaluations were performed by switching from one computing environment to another and preparing input files using a text editor, which made the system difficult to use and raised the need for an integrated software package.

Delphi, a rapid application development (RAD) tool, was used in constructing the needed integrated software. A WINDOWS application has been developed putting all the components of the system together so that the user can perform various tasks "under one roof." As a result, the user can input parameter values in better organized dialog boxes, and see the results of cost and engineering calculations on the spot without the need to know the complicated underlying structure. Helpful plots of system schematics and item explanations are shown on the screen as item-sensitive features. The procedural steps involved in the system are accomplished using a context-sensitive menu system.

In this WINDOWS application, a template has been developed for each of the three environmentally sound systems. When the user selects one of these systems, the input data are presented to him in various dialog boxes. The source of a particular data entry can be retrieved by clicking on that entry. If the entry is not satisfactory, the user can overwrite it by typing in his own number. In other words, default values are provided for every single entry in the template of the selected alternative. For convenience to the user, the input data and computations are organized in 10 categories as shown in Table 4.1. The data given in this table corresponds to a hypothetical PD-car transportation system delivering 200,000 ton of residue per year to a mine 200 miles away from a power plant.

Table 4.1 - Calculation of transportation cost for PD-car system

DISTANCE (MILES)...................... = 200
PRODUCTION (TONS/YEAR) .... = 200000
MINIMUM REQUIRED RATE OF RETURN (%) .. = 12
EFFECTIVE TAX RATE (%) .... = 40

Year	0	1	2	3	4	5
Revenue.........	0	0	0	0	0	0
-Oper. Cost.....	0	-1652	-1652	-1652	-1652	-1652
-Depreciation...	0	-650	-1106	-776	-547	-403
Taxable Income..	0	-2302	-2758	-2428	-2199	-2055
-Tax............	0	921	1103	971	880	822
Net Income......	0	-1381	-1655	-1457	-1319	-1233
+Depreciation...	0	650	1106	776	547	403
-Capital Cost...	-4404	0	0	0	0	0
Net Cash Flow...	-4404	-731	-549	-681	-772	-830

Year	6	7	8	9	10
Revenue.........	0	0	0	0	0
-Oper. Cost.....	-1652	-1652	-1652	-1652	-1652
-Depreciation...	-382	-361	-180	0	0
Taxable Income..	-2034	-2013	-1832	-1652	-1652
-Tax............	814	805	733	661	661
Net Income......	-1220	-1208	-1099	-991	-991
+Depreciation...	382	361	180	0	0
-Capital Cost...	0	0	0	0	0
Net Cash Flow...	-838	-847	-919	-991	-991

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

Salvage Value..................................	1920
Book Value.....................................	0
Capital Gain (or Loss).........................	1920
Tax Liability..................................	-768
After-Tax Capital Gain (or Loss)...............	1152
Book Value.....................................	0
After-Tax Cash Flow Due to Cap. Gain (or Loss)..	1152

AFTER-TAX NET PRESENT VALUE ($1,000) ....= -8426
AFTER-TAX ANNUAL EQUIVALENT COST ($1,000) ...= -1491
AFTER-TAX COST PER TON ($) ....= -7.46
BEFORE-TAX PRICE TO BE CHARGED ($/TON) ......= 12.43

A technical paper summarizing the features and capabilities of the integrated software was presented at the 1995 International Ash Utilization Symposium held in Lexington Kentucky on October 23-25, 1995. A copy of the paper is included in this report as Appendix B.

Development of a Simulation Program

During the course of conducting evaluation of the systems it was realized that the systems components are serially connected and that a decision on the size of a component (or capacity), or, the number of units within a component, will effect the size of other components, or the number of units within the other components. The relationships among the components are not as simple as they appear. The sketch of a PD-car system, as shown in Figure 4.1 will be used to illustrate these points.

As seen in Figure 4.1, one set of PD-cars is being filled from the fly ash silo at the power plant while the other set is being emptied at the mine site. When the filling is completed, a local train will deliver the filled cars to the mine site and another local train will pick up the empty cars from the mine silo and deliver them to the plant. An appropriate number of trucks will shuttle between the silo and the injection point to deliver the residue to be placed in the old underground workings. The number of PD cars in each set is a function of the annual residue production, local train schedule, and the rate of unloading into the silo at the mine site. The number of trucks between the silo and injection site is a function of the distance, capacity of trucks, loading and unloading rates, and the injection rate. The capacity of the silo is a function of the filling and emptying rates. For a well functioning and economic system, all the components must be sized properly. An effective way of designing systems with interrelated components, such as the system shown in Figure 1, is to build a computer simulation model of the system and simulate different scenarios using this model.

During the last quarter, simulation models have been developed for the three selected systems using the General Purpose System Simulation (GPSS/H) language. Among the three models, PD-car simulation model has been extensively tested and debugged. The other two models are currently being tested.

Application of the Simulation Model and the Integrated Software

The PD-car simulation model has been run for an hypothetical case where 200,000 tons of fly ash had to be transported annually to a mine that is 200 miles away from the plant. The life of the project is assumed to be 10 years. The objective is to be able to transport and inject the entire 200,000 tons with the least number of PD cars, least number of trucks, least number of personnel, and a small silo. The advantage of having a simulation model is that once the model is validated several scenarios can be simulated in a short period of time.

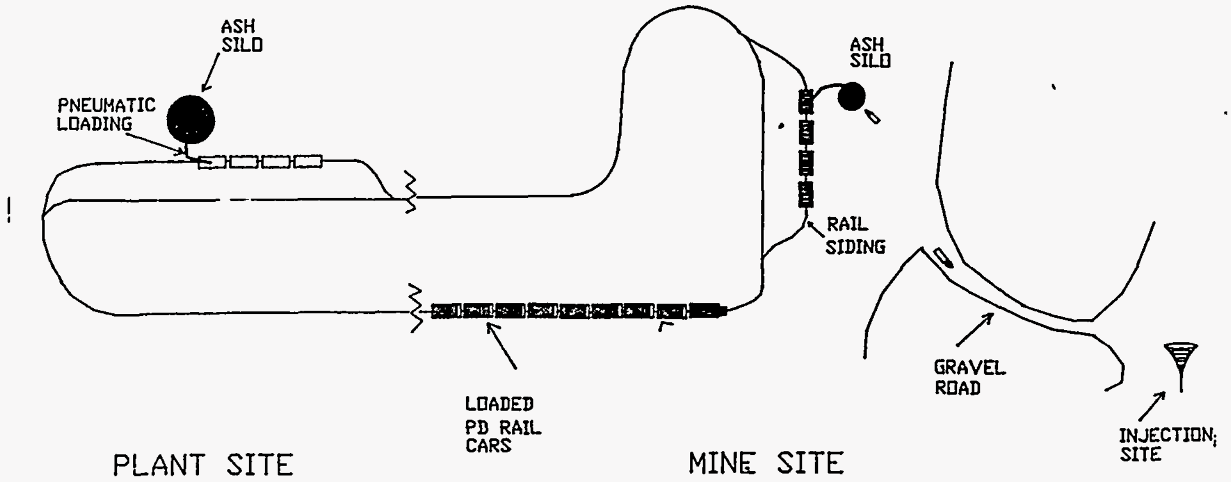

Figure 4.1 - Dry FGD residues transportation by PD cars

After trying a number of scenarios, it is found that the best operating scenario would be to have two sets of 20 PD cars, 3 trucks, and a silo of 400 tons. This scenario requires one (1) shift of operation per day, six days per week of injection, two train trips per week, and approximately 90 tons per hour of injection and 96 tons per hour of pumping rate from the PD cars into the silo.

The simulation results are then entered in the spreadsheet of the integrated software where the engineering and economic calculations are performed. The entered, as well as the calculated, values are exhibited in Table 4.1. The outcomes of the economic evaluation are shown in Table 4.1 As seen in this table, the net cash flows of years 1 to 10 are negative since only costs are involved in the analysis, and the objective is to find the price to be charged per ton of material delivered in order to make the minimum required rate of return. The net cash flow of year 0 simply reflects the capital cost invested in the project. The salvage value of the project after 10 years of operation is estimated to be $1,920,000. Capital cost items in this project are assumed to be depreciated within 5 to 7 years. Hence, at the end of the project life all capital cost items would be totally depreciated and any salvage value would be subjected to capital gain tax, giving an after tax cash flow of $1,152,000. At 12% minimum required rate of return, the annual equivalent cost of the project is determined to be $1,491,000 indicating an after-tax cost per ton of $7.46. At 40% effective tax rate, the $7.46 after-tax cost corresponds to a before-tax price of $12.43. In other words, the company who undertakes this project must charge $12.43 per ton in order to obtain a 12% rate of return on its investment.

There are a number of critical variables such as railroad rate, capital cost items, and wages and salaries that can affect the project economics. Sensitivity analyses on these variables are essential to reveal the project's merits. With the aid of the integrated software, these sensitivity analyses can be conducted in a short period of time. Figure 4.2 shows such a sensitivity analysis which was conducted on the railroad charge for the above hypothetical case. The original railroad charge is $0.0226 /ton/mile when the calculated price to be charged is $12.47 /ton. As shown in this figure, a 20% reduction in the railroad rate would bring this price down to $11.27. Similarly, a 40% reduction would bring it to $10.27, providing a significant reduction in the price to be charged.

CHAPTER 5

UNDERGROUND PLACEMENT

DR. Y. P. CHUGH
CO-PRINCIPAL INVESTIGATOR

Abstract

The basic objective of the DOE-SIUC Cooperative Research Program is to demonstrate the technologies for placing coal combustion by-products in underground coal mines. The actual demonstration of these technologies will take place in Phase III of the program (October 1, 1996 - September 30, 1997). This chapter describes the work to date to prepare for these demonstrations.

Introduction

As set forth in Chapter 1 of this report, the basic objectives of the DOE-SIUC Cooperative Research Program is to develop and demonstrate two technologies for the placement (injection) of coal combustion by-products or mixtures of such by-products in the void spaces of abandoned underground coal mine workings. These two technologies are (1) pneumatic injection of basically dry combustion by-products, and (2) hydraulic injection of high solid content by-product "paste" (i.e., a by-product mixture of approximately 70 percent by-products and 30 percent water). The actual demonstration of these two technologies will take place during Phase III of this program, i.e., during the period October 1, 1996 - September 30, 1997. However, much preliminary work must be accomplished before the actual demonstration can take place.

Accomplishments to Date

Elsewhere in this report the laboratory testing of the coal combustion by-products to be used in this program is described in detail. The selection of suitable by-product mixtures for both pneumatic and hydraulic placement has been finalized. The pneumatic mixture will consist of about 80 percent fly ash and 20 percent spent bed by-products, with a moisture content of 30 percent. The hydraulic mixture will consist of about 40 percent fly ash, 55 percent scrubber sludge, and about 5 percent lime or lime waste, with sufficient water to make a "paste" consisting of approximately 70 percent solids. Some testing to determine more of the specific properties of these coal combustion by-product mixtures is continuing.

A fundamental reason for placing coal combustion by-products in abandoned underground coal mines is to alleviate surface subsidence. In order to determine the effectiveness of placing the by-products underground, the rate of surface subsidence (absent underground placement) needs to be known. To obtain this baseline data, a series of subsidence monuments have been installed around the placement area at the Peabody Number 10 mine. Measurements at these monuments are being taken monthly to establish the baseline subsidence data prior to underground placement.

Also, the injection wells, vent wells, and a geotechnical well have been drilled in the placement area, and are being maintained in a ready state for the actual underground placement demonstration. Drilling the wells at this time was an economic decision, as it was essential that groundwater monitoring wells be drilled to establish baseline groundwater data. By drilling the wells necessary for underground placement at the same time as drilling the monitoring wells, a single drilling contract accomplished all drilling. The maintenance of the injection wells until the time of the demonstration will be a minor task.

The Cooperative Research Program also calls for a full-scale surface demonstration of the two placement technologies, to be accomplished during Phase II of the program (April 1 - September 30, 1996). During the past few months work has began to develop the specifications for both the pneumatic and hydraulic placement equipment. Wherever possible it is intended to rent or lease, rather than purchase, the necessary equipment. Whether the acquisition of the needed equipment will take place during Phase I or Phase II of the program will largely be determined by budgetary considerations. Also, transportation and storage of the equipment will be considered.

Some preliminary work has begun on assembling the necessary data for obtaining the essential permits for the final underground placement demonstrations. This work will continue, and all necessary permits will be obtained prior to the demonstrations. A close working relationship with the permitting agencies, both State and Federal, has been maintained throughout the program.

In the remaining portion of Phase I of the program, subsidence measurements will continue to be taken to fully develop the baseline subsidence database, as will groundwater measurements to develop the baseline data. Also, the search for suitable equipment for both pneumatic and hydraulic placement will continue, with the objective of obtaining the most suitable equipment at the lease overall cost. Assembling data for obtaining necessary permits will be accelerated, in order that the permits may be obtained in a timely fashion.

Overall, the underground placement tasks are technically on schedule, and fiscally within projected budgets. It is believed that the actual underground placement demonstrations will be conducted early in Phase III of the program.

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CHAPTER 6

ENVIRONMENTAL ASSESSMENT AND GEOTECHNICAL STABILITY

DR. STEVE ESLING
CO-PRINCIPAL INVESTIGATOR

Abstract

During the first year of the project, the uncertain status and availability of the original mine site delayed progress toward the completion of some study tasks. Work during the second year, however, has put the project back on schedule. Twelve monitoring wells were installed at the mine site; five for the collection of water quality samples and seven for monitoring hydraulic head. Work also continued on refining the conceptual model of site geologic and hydrologic conditions. A detailed geotechnical description of a continuous core collected at the site and geophysical logs from each of the boreholes drilled for monitoring well installation have improved the model.

Work this past year supports earlier study findings; that the mine site has a favorable geologic and hydrologic setting for the disposal of coal combustion residues. Bounding strata have a low intrinsic permeability and at present the target panels are dry. Air shafts and fracture zones may transmit underground water from shallow permeable units to the mine, but current groundwater discharge cannot flood the target panels. Packer tests suggest that the thick sequence of Pennsylvanian Strata over the mine have a bulk hydraulic conductivity less than 1 x 10-8 cm/s.

Introduction

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

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

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

Goals and Specific Objectives

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

1. Develop a conceptual model for the stratigraphy in the vicinity of the target panels, specifically locating any units which may function as aquifers;
2. Determine the flow system in the vicinity of the mine panels and the hydrologic properties of the bounding strata, including hydraulic conductivity.
3. Assess the impact, if any, of injecting coal combustion residues into the underground mine panels on groundwater quality;
4. Assess the environmental impact, if any, of temporary surface impoundments or spills, and;
5. Assess the impact, if any, of injecting the residues into underground mine panels on the extraction of economic resources.

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

Literature Review

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

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

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

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

Results and Discussion

During the first year of the project, the uncertain status and availability of the original mine site delayed progress toward the completion of some study tasks. By the end of the second year of the project, however, the Environmental Assessment studies were back on schedule. Work competed during the second year included:

1 Continued refinement of the conceptual model of geologic and hydrologic conditions at the mine;

2. Detailed description of packer tests and data analysis methods;

3. Development of the programs for controlling the automated data loggers on the site;

4. Collection, description, and interpretation of a continuous core collected from top of bedrock to the unit below the mined coal;

5 Completion of packer tests in the boreholes drilled for the installation of monitoring wells; and

6. Installation of twelve monitoring wells, five designed for groundwater chemistry and seven designed to provide data on water levels.

This section presents the results completed in each of the above topics.

Conceptual Model

Work continued on developing a conceptual model of site geology and hydrology. Timothy McDonald, the graduate assistant working on this project, spent two days at Peabody Coal Company offices working with the computerized well log data base of the Peabody #10 mine. Structure contour and isopach maps were prepared for the major units, including the top of the Herrin (No. 6) Coal, top of bedrock, and thickness of the Anvil Rock Sandstone. The computer system at Peabody has a slow response time and further work on figures summarizing geology at the mine will continue at Southern Illinois University at Carbondale (SIUC) during the third year of the project. The Department has a package (Terra Sciences) for analyzing stratigraphic relationships that should provide a quicker turnaround. Summary figures prepared thus far indicate errors in the well log files. Work continues on correcting these problems so that figures summarizing the conceptual model can be prepared.

The conceptual model is also based on an analysis of well logs on file with the Illinois State Geological Survey as well as a review of the literature on the geology and hydrology of the area around the mine. Individuals working in the area of Environmental Assessment visited the mine prior to its closure in August, 1994 to examine surface workings and underground conditions in one of the target areas. One advantage a project like this has over others is the existing data base. Before a site is selected for solid waste disposal, its geology as well as its surface and groundwater hydrology must be evaluated. The process of extracting coal from surface or underground mines generates a significant data base on these site characteristics. In effect, data that would have to be generated before a landfill operation is permitted already exists for underground mine disposal sites.

The ultimate goal of developing a conceptual model is to locate any artificial or natural pathways capable of transmitting contaminants originating from the coal combustion residues to potable groundwater supplies. Possible pathways include permeable units, fractures, fault zones, and artificial pathways such as improperly sealed boreholes. The developing conceptual model was used to design the groundwater monitoring program and to determine potential impacts from surface activities such as temporary storage of the residues. The conceptual model was refined as additional hydrologic and geotechnical data became available during monitoring well installation and it will continue to change to fit data collected during on-site monitoring. Ultimately, the conceptual model will include the following components:

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

SIUC and the ISGS personnel are working together to develop the conceptual model of the site. The following discussion describes the state of this model at the end of the second year of study.

Site Description

The study area is the Peabody Number 10 Mine, with large underground works located in portions of the South Fork and Cotton Hill Townships, T 13 N, R 4 W and T 14 N, R 4 W in Christian and Sangamon Counties, Illinois. Pawnee is the nearest town, situated about 2 miles west of the mine. The town of Kincaid is located about four miles east. Springfield is about 20 miles west of the mine. Portions of Sections 10 and 11, T 13 N, R 4 W in Christian County, herein referred to as the study area, contain extensive surface works, including disposal areas, as indicated on the Pawnee 7.5 Minute Quadrangle. The underground mine panels targeted for disposal are located in these sections.

Geology

The surficial deposits of the study area average about 40 feet thick and consist of about 10 feet of loess over unconsolidated glacial diamicton with minor lenses of sand and gravel (Glasford Formation). The unconsolidated surficial deposits are underlain by Pennsylvanian Age shale, limestone, clay, coal and minor sandstone units of the Bond, Modesto, and Carbondale Formations (Willman and others, 1975; Bergstrom and others, 1976). The stratigraphy of the Pennsylvanian bedrock is complex. Most Pennsylvanian lithologies are not continuous over the study area and the sandstones and gray shales exhibit extreme changes in thickness laterally.

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

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

Hydrology

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

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

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

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

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

Packer Testing and Analysis Methods

ISGS personnel completed a literature search on the topic of packer testing and prepared a draft report on conducting and analyzing single and double packer tests (Appendix G). Methods discussed include conventional methods described in the Bureau of Reclamation's Ground Water Manual (1985) to less common methods such as Dagan (1978). The draft document was used during the packer testing conducted during the month of August, 1995.

Data Acquisition Software

The Environmental Assessment Team prepared the software for data acquisition in the field, and in particular the programs for taking vibrating wire transducer readings in the monitoring wells and processing these readings into hydraulic head. Four programs are included in Appendix H to this report entitled Unit 1, Unit 2, Loot, and Pillage. The function of each of these programs is summarized below:

Unit 1:

This program controls a CR10 data logger. The code instructs the CR10 to take pressure and temperature readings from four vibrating wire transducers at one hour intervals. Four readings of temperature and pressure from each transducer are taken in succession every hour and averaged prior to storage. One of the vibrating wire transducers will monitor atmospheric pressure at the site. The other three will be in groundwater monitoring wells.

Unit 2

This program also controls a CR10 data logger and readings are taken as described above for Unit 1. Three of the four vibrating wire transducers will be installed in groundwater monitoring wells. The fourth transducer will monitor head in the panel targeted for hydraulic injection.

Loot

This program takes the raw file in printer format and splits the different instruments into separate files within a folder labeled by the retrieval date. The program converts day and hour data from the data logger into a decimal day.

Pillage

This program converts raw data on temperature and pressure collected from the vibrating wire transducers into pressure in centimeters of water. Data are also corrected relative to atmospheric pressures. Pillage only reads files that have first been processed with the program Loot. The program converts the data for the vibrating wire transducers into a two column format (with one column holding time and the other pressure) that can be read by a graphics program.

The programs described above have had some debugging, but detailed analysis requires installation of the CR10 units and transducers in the field.

Core Description

Researchers monitored coring at the Peabody 10 Mine June 21 through June 23. One continuous core was collected to a depth of 355.55 feet, boxed, and returned to the ISGS for detailed description. The final geotechnical core log was created using Geotechnical Graphics System Software to combine all the field logging notes into a presentable and legible core log. Appendix I with this report contains the geotechnical log of the core prepared by the ISGS which notes such features as rock lithology, Rock Quality Designation (RQD), total fracture frequency, and description of individual rock joints. The core was stored in the ISGS core storage facility. Geophysical logs were taken of the borehole before it was sealed.

The core log supports earlier findings that disposal at this site poses little risk to the environment. Shale dominates the section above the mine. Rare sandstone units appear to have low hydraulic conductivity.

Packer Tests

ISGS investigators with the Environmental Assessment portion of the project and SIUC designed and fabricated a hydraulic injection system for the packer tests. This injection system performed well in the field and was capable of maintaining constant pressures up to 85 psi above the top of the injection pump. Appendix J contains tables summarizing the test data.

Apparently, the bulk hydraulic conductivity of the Pennsylvanian Strata overlying the mine is less than 1 x 10 -8 cm/s. In some boreholes, values of hydraulic conductivity were below the intrinsic accuracy of the injection system.

The flow from the packer was read with the automated data logger, as well as manually. The CR10 pulse counting program (Unit 2 Pulse Counting) is included in Appendix H with other data logger control and processing programs.

Monitoring Well Installation

The original test plan submitted to the U.S. Department of Energy (USDOE) was modified with regard to the location and installation of groundwater monitoring wells. In an earlier report, we suggested that a monitoring well screened in the coal would not yield samples. Although this might be true, the collapse of strata above the Anvil Rock sandstone, the unit nearest the coal with sufficient permeability to yield samples, prevented installation of a screen in the Anvil Rock in most boreholes. We decided to take a proactive course of action; installing screens in the more permeable lithologies in the section, up to a maximum of three screens per borehole. For consistency, a screen was set within the coal in all boreholes which pierced the coal.

In addition, the number of monitoring wells for sampling was reduced from six, as proposed. to five. The hydraulic and pneumatic injection sites are located near each other, and can share one of the monitoring wells. Most of the boreholes drilled contain a nest of wells (up to three) screened at different horizons in order to assess vertical hydraulic gradients. Nested wells also departs from the original plan, but will provide data on vertical components of flow and the opportunity to test more than one interval in a borehole with single well test methods.

The wells were installed in a pattern that assures at least one well will be upgradient and one well downgradient of groundwater flow (Figure 6.1).

Well installation generally followed accepted guidelines (USEPA, 1986). Some modifications to the original well design as described in the test plan were made because of technical difficulties associated with installing monitoring wells deep into low-permeable strata. Boreholes were drilled by water rotary or mud rotary methods. All monitoring well casings were installed in a six inch (minimum) diameter hole. A six inch ID steel casing was driven through the surficial unconsolidated sediments to the top of the bedrock to prevent collapse. The casings for wells designed to yield water quality samples were constructed of 2 inch ID threaded schedule 80 PVC pipe. Those installed to monitor hydraulic head alone were constructed of 1.25 inch ID schedule 40 PVC pipe. PVC is considered an acceptable well casing material when aggressive organic leachate mixtures will not be contacted (Barcelona and others, 1985). No solvent cement was used. instead pipe junctions were sealed with an O ring to ensure against leakage. The monitoring wells were screened over an interval of 5 feet. Wells designed to provided samples on groundwater quality were screened in coal as close to the mine panel as possible. Those wells designed to provide head data alone were generally screened in the more permeable lithologies in the geologic section. Drilling continued to the unit immediately below the coal in boreholes over a barrier and to a depth within 10 to 20 feet of the top of the coal for those boreholes drilled over the mine.

The annulus around the screen was filled with quartz sand (Ottawa quartz type). The screen slots (0.01 inch) were selected so that they retained 90% of the sand pack materials. The sand pack extends to a minimum of 1 ft above the top of the screen to minimize interference from the grout Grout was pumped through a tremie pipe in stages. Because bentonite stores water, and delays the recovery of natural hydraulic heads in low permeable materials, the annulus above the sand pack was filled with Portland cement (Type I) to just below the frost line. Concrete fills the remaining annulus to a point above grade to minimize surface water interference. An external locking PVC casing was set in the concrete to protect the well casings at the surface. The original test plan called for steel locking casings, but those acquired for this project would not fit in the six inch ID steel surface casing. Appendix K contains the summary notes on well installation.

Summary

The Environmental Assessment portion of the project is now back on schedule. Completed tasks or those with significant progress toward completion, as described in the test plan for the project include: development of a conceptual model of site hydrology and geology; monitoring well installation; geotechnical logging; and measurement of hydrologic properties. The Environmental Assessment test plan was modified. Additional wells were installed to provide vertical components of flow at the site.

The proposed study area has a favorable geologic and hydrologic setting for the disposal of coal combustion residues. The target panels avoid complex geologic features in the region. Bounding strata have a low intrinsic permeability and at present the target panels are dry. Natural and artificial pathways such as abandoned boreholes, air shafts, faults, and fracture zones may transmit groundwater to the mine, but no significant discharge was found within the target panels. The mine is located at between 325 and 375 feet below the land surface. The nearest potable groundwater supplies occur at depths less than 250 feet. Most individual users obtain water from the shallow sand and gravel deposits within the unconsolidated surficial materials near the surface.

Plans

During the next six months, (October 1 - March 31, 1996) participants in the Environmental Assessment portion of the project will install groundwater sampling equipment and instrumentation for monitoring hydraulic head, conduct long term slug tests on the monitoring wells, collect background water quality samples, and prepare preliminary generic flow models. In addition, we will continue.to refine the conceptual model of site geology and hydrology.

Suitable contaminant transport models in the public domain, developed by the principal investigators, and available commercially will be compared and evaluated by running a series of hypothetical scenarios through each algorithm. Advantages and limitations of each modeling program will be determined by this process. The groundwater flow model will also go through a series of tests in order to determine a suitable grid spacing. It is important to recognize that calibration of the flow and transport models requires head and groundwater geochemistry; data that will be unavailable or at best sparse before the end of March, 1996. Preliminary work, however, should show advantages and limitations of the different programs.

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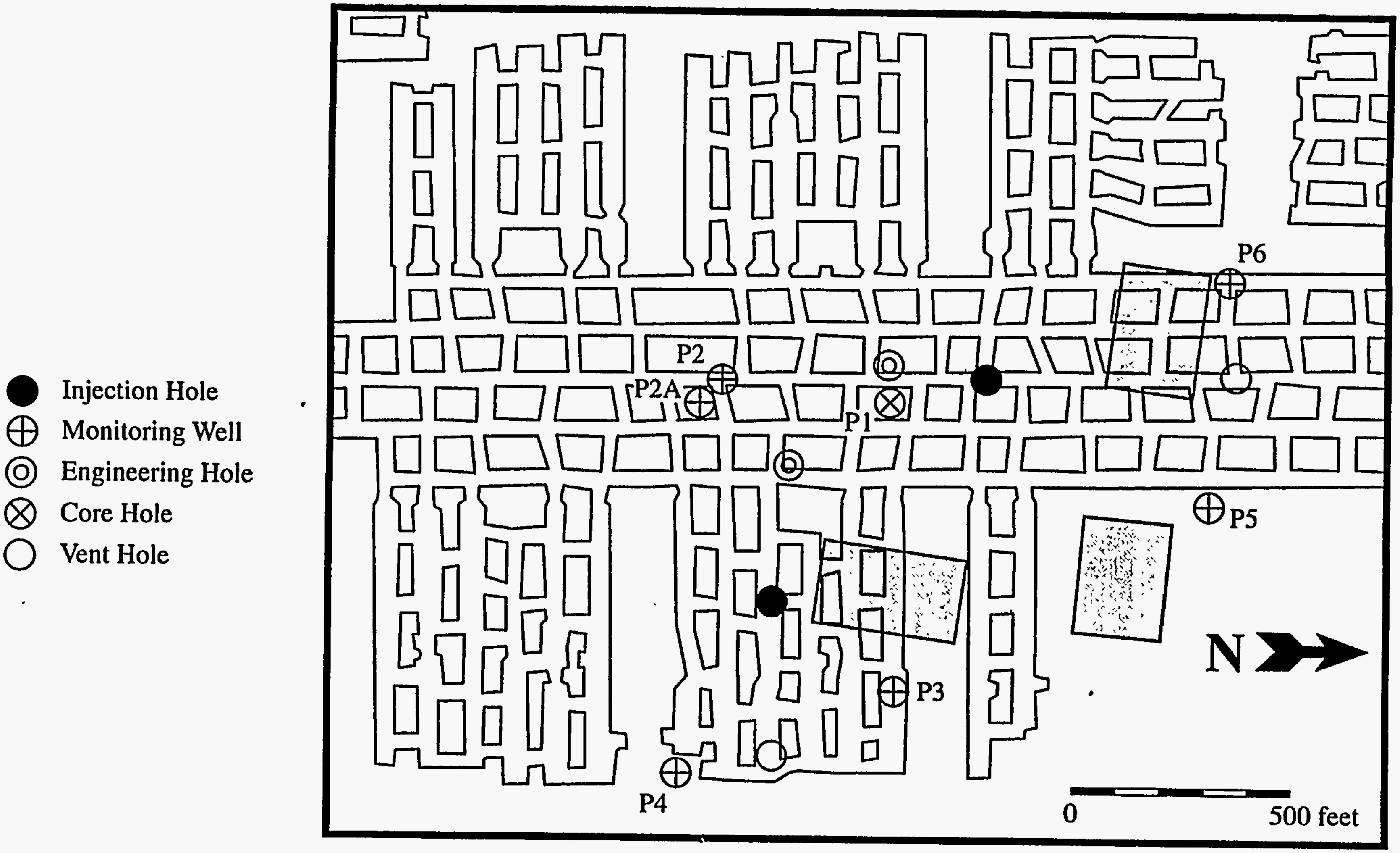

Figure 6.1 - Map showing location of monitoring wells. All locations approximate.

CHAPTER 7

FIELD DEMONSTRATION

MR. EDWIN THOMASSON
CO-PRINCIPAL INVESTIGATOR

Abstract

The Department of Energy - Southern Illinois University Cooperative Research Agreement specifies that the Collapsible Intermodal Container (CIC) technology partially developed by SEEC, Inc. be further developed and field tested during Phase I of the overall program. This chapter describes the development and testing of the CIC technology. The full-scale test showed that the CIC can be filled with fly ash successfully and can be transported over long distances without degregation of its contents. However, emptying the CIC proved unsuccessful, and indicated that additional engineering and development work is needed to fully develop the CIC technology

Introduction

A specific part of the DOE - SIUC Cooperative Research Agreement was to develop and demonstrate the Collapsible Intermodal Containers (CIC) under development by SEEC, Inc. of Mendota Heights, Minnesota. The first year of the program was developed to the further development and testing of the CIC.

SEEC had a single CIC manufactured according to their specific design. As designed, the CIC is a portable and collapsible storage bin that allows bulk commodities to be moved from place to place and between different types of vehicles, by handling the container , not the contents. The CIC is specifically configured to ride in empty coal rail cars. As developed, the CIC is about ten feet in height, about nine feet in diameter, and is constructed of an ultra-violet resistant rubber compound bonded to a fabric comprised on nylon interwoven with kevlar. Each CIC will hold about twenty tons of coal combustion by-products.

A consulting engineering firm conducted a computed-aided finite element analysis of the CIC, involving the simulation of stresses occurring in four specific cases, using a base load of twenty tons. These specific cases were:

1. Stresses on filled CIC when lifted
2. Effects on CIC during transport
3. Effects on a triple hopper rail car during transport
4. Effects of repeated impact.

The analyses indicated that in no cases would transporting filled CICs in a rail hopper coal car compromise the integrity of either the CIC or the rail car, and all elements of the CICs and the rail car were within the limits established by the Association of American Railroads.

Also, the first CIC was subjected to an overload test, where it was filled with sand to a weight exceeding its rated capacity. In actuality, the CIC was filled to a weight of twenty five tons (when the rated capacity was twenty tons), and allowed to "hang" from a heavy crane for some time. As with the finite element analysis, the overload test indicated that the integrity of the CIC was not compromised when subject to a load that exceeded the rated capacity by about five tons.

The overall concept of the CIC system is that coal combustion by-products generated at a coal-burning electric power plant could be loaded into CIC's then "back-hauled" to the coal mine by the coal unit train for further utilization or disposal. Empty, collapsed CIC's could be taken to the power plant either by truck or on one or two cars hooked to the unit coal train. The "back-haul" concept of transporting coal combustion by-products from the power plant to the coal mine should result in significant transportation cost savings.

On November 17, 1994, a field test of the CIC concept was held. The Baldwin Power Station of the Illinois Power Company was selected as the site for the demonstration.

Three CIC's (all that then existed) were filled with fly ash at the Baldwin Station. Each CIC was fitted into a steel framework to hold it upright, and placed on a flat-bed truck. The truck was positioned under the fly ash silo, and the CIC filled by gravity flow from the silo through a standard ash dispensing spout that can move up and down. The filling spout is designed to fill standard PD ash trucks. Each CIC was fitted with a removable interface to provide a dust-free connection between the silo ash spout and the CIC.

Filling the CIC's proceeded without difficulty. As the silo loading ramp was fitted with a scale, the CIC's were filled by weight. Filling rate was approximately two tons per minute; thus fill a CIC to its twenty-ton capacity took about ten minutes.

After being filled, the CIC's were transported by the flat-bed truck to a rail siding, where a standard three-bay hopper rail coal car awaited. The CIC's were lifted from the steel loading framework and placed in the hopper car. Loading was without undue difficulty, even though a boom crane not specifically designed for such use was used. In fact, the loading of the final (third) filled CIC took only about five minutes.

Following being loaded into the hopper car, the car was sent on a cross-country journey to Norflok. Virginia. In Norfolk the CICs were taken to a pier that handles standard intermodal containers (rigid 8' x 8' x 20' containers). An overhead gantry crane and an articulating gantry crane (which performs ship-to-shore container transfers) were used to move one CIC out of, and back into, the rail car. No problems were encountered in these activities. After the Norfolk tests, the three CIC's were returned, still fully loaded, to the Baldwin Power Station.

Again, at the Baldwin Station, the CIC's were removed from the rail car and placed on a flat-bed truck. The truck transported the CIC's to the Peabody Coal Company coal cleaning plant, a distance of about five miles from the Baldwin station.

The final test was the emptying of the CIC's at the Peabody plant, and here significant difficulties were encountered. Although the CIC's had totally protected the fly ash load from weather (the fly ash was almost exactly as it had been when loaded, i.e., dry), attempting to empty the CIC's by inverting them by using a specially-constructed inverting frame failed to work properly. apparently the fly ash had compacted during the 2000 mile trip, and would not flow by gravity alone even when inverted fully. Compressed air, which might have been injected into the filled CIC to partially "fluidize" the fly ash was not available at the disposal sit, and emptying the CIC's took hours, instead of minutes. Only two of the three CIC's were emptied by the inverting method.

The third CIC was emptied about one month later, using commercial vacuuming equipment. This method of emptying proved successful, as the commercial vacuum system had no problems in removing the fly ash efficiently and in a timely manner from the CIC.

Several conclusions can be drawn from the full-scale field test of the SEEC CIC technology. First, CIC's can be filled with a reasonable degree of efficiency at a standard power plant fly ash silo. In a full-scale commercial operation the silo loading ramp might need to be configured to better handle the CIC's, and, if possible, provide direct access to a rail hopper car with CIC's in place, thus avoiding the step of transferring fully loaded CIC's from a flat-bed truck to the rail car.

Second, rail transportation was not a problem for the fully-loaded CIC's. On a rail trip of about 2000 miles, neither the rail car nor the CIC's encountered any significant problems. It should be noted that accelerometers installed on both the CIC's and the rail cars showed longitudinal accelerations of almost 28 g's. However, even with this stress, neither the CIC's nor the rail car showed any damage or other indications of distress. Further, the fly ash load of the CIC's was fully protected from the weather.

However, the field demonstration did show that further engineering and design work need to be done on the CIC's, particularly to develop a more efficient and effective emptying system. SEEC, Inc. is continuing the development of the CIC's and the overall concept of bulk commodity handling and transportation. However, with the conclusion of the field demonstration herein described, SEEC's involvement in the DOE-SIUC Cooperative Research Agreement was concluded.

A final topical report on the SEEC, Inc. development and field testing of the CIC's was completed in July, 1995. Prepared jointly by Dr. Jeffrey L. Carpenter of SEEC, Inc. and Mr. Edwin Thomasson, Program Manager, DOE-SIUC Cooperative Research Program, the final topical report has been furnished to the Department of Energy-Morgantown Energy Technology Center.

CHAPTER 8

FUTURE PLANS

For the remaining period of Phase I of the DOE-SIUC Cooperative Research Program emphasis will be placed on developing the pneumatic and hydraulic placement systems, determining specifications for the equipment needed for the systems, and contacting suppliers to determine sources of equipment. Wherever possible, equipment will be acquired on a rental or lease basis, rather than purchase, to minimize costs. Also to be continued with emphasis is the acquisition of the data necessary to apply for and acquire the permits from the involved regulatory agencies essential for the conduct of both the surface and underground demonstration.

Baseline subsidence data and hydrologic monitoring will continue, so that a sound data base can be developed. Maintenance of the wells that have been drilled at the placement site will continue.

A final topical report on coal combustion by-products characterization will be prepared and furnished to the Department of Energy. Likewise, a final Technical Progress Report for Phase I of the program will be prepared. Finally, close contact will be maintained with all participating parties, and with the Department of Energy officials and representatives.

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APPENDIX A

Economics of Coal Combustion Residue Transportation

ECONOMICS OF COAL COMBUSTION RESIDUE TRANSPORTATION

H. SEVIM AND S. GWAMAKA

Department of Mining Engineering
Southern Illinois University
Carbondale, IL 62901, USA

ABSTRACT

A group of researchers at the Southern Illinois University is engaged in a research project whereby technical, environmental, and economic feasibility of coal combustion residue disposal into old underground coal mines is being investigated. Safe and economic transportation of residues from power plants to mine sites is an important segment of this project. A number of transportation alternatives have been examined, and among these, pneumatic trucks, pressure differential rail cars, and collapsible intermodal containers have been found to be promising. In this paper, all three alternatives are applied to hypothetical cases pertaining to central and southern Illinois. The operating scenarios are described and a comparative economic analysis is conducted using "After-Tax Cost" method. Each alternative is evaluated for varying distances and tonnages to reveal its favorable operating range.

INTRODUCTION AND BACKGROUND

The disposal of coal combustion residues in an environmentally sound and economically acceptable manner is a growing concern in the coal and utility industries in the US. In 1990, approximately 100 million tons of combustion residues were produced on a dry basis. The residue production is expected to increase significantly after 1995 due to the Clean Air Act Amendment of 1990. To meet the requirements of the Act, numerous power plants will have to adopt advanced combustion and flue gas desulfurization (FGD) technologies which will significantly increase the amount of by-products. The current most widely used residue disposal method is surface disposal to landfills or ponds near the power plants. Surface disposal has a number of environmental concerns such as surface and subsurface acidic discharge, dust, decreased land value and land use. Also, surface disposal is becoming costlier due to increasing residue production, stringent environmental regulations, and diminishing land availability.

Underground disposal of residues may alleviate the problems associated with surface disposal. Furthermore, since the FGD by-products typically have high calcium hydroxide content and cementitious properties, if properly disposed in the old workings of the underground coal mines they may mitigate subsidence and acid mine drainage.

Under a contract from the US Department of Energy, Southern Illinois University at Carbondale (SIUC) is investigating the technical, environmental, and economic feasibility of residue disposal into the old workings of the coal mines. The project is focused on Illinois and it is progressing successfully in 5 coordinated areas since September 1993. These areas are: 1) Residue Characterization. 2) Materials Handling, 3) Underground Residue Placement, 4) Field Demonstrations. and 5) Economic Evaluation.

In this paper. the progress in materials handling and systems economics areas is reported. The objective of materials handling research is to identify the systems that are technically, economically, and environmentally feasible in handling and transporting the coal combustion residues from the power plant to the injection site. To achieve this objective a number of alternatives have been evaluated, and few more are still under investigation. So far, the research has been focused on dry residue transportation. Among the alternatives evaluated, three were found to be promising; pneumatic trucks, pressure differential rail cars, and collapsible intermodal containers (CIC^{TM}).

Pneumatic Trucks (PT)

Pneumatic trucks, also referred to as bulk tank trucks, are widely used in transporting low density dry flowable powder and granules as well as high density materials such as cement, limestone and fly ash. A pneumatic truck is composed of three main components: (i) tractor, (ii) tank trailer. and (iii) blower. The most common method of loading the material is gravity feeding from a silo by a collapsible spout which engages to the gate on top of the tank. The tank is air tight when the lids of the gates are closed. During offloading, the material flows through the piping below the tank due to pressure difference created by the blower. The PT transportation scenario for this project is schematically shown in Figure 1. The trucks are loaded from the fly ash bin of the plant and they deliver the material directly to the injection point at the mine site. There, the pressure necessary for offloading the fly ash into the injection hopper is supplied by the blower mounted on the truck. These trucks are approximately 25 tons in capacity and can offload in about 25 minutes.

Pressure Differential Rail Cars (PD cars)

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

The PD car transportation scenario is schematically shown in Figure 2. As seen in this figure, there are three sets of rail cars; the first set of cars is at the plant and is being filled from the fly ash bins: the second set of cars, which has already been filled and attached to the empty coal unit train. is traveling to the mine; and the third set is at the mine site and is being offloaded into pneumatic trucks at a pace synchronized with the injection system capacity.

When the train arrives at the mine, the PD cars are demurred and parked at the junction of the siding. Then. the unit train pulls under the coal silo for coal loading. After coal loading is completed. the empty PD cars which have been waiting at the siding are attached at the back of the loaded unit train. The train then leaves for the plant. Meanwhile, the loaded PD cars which

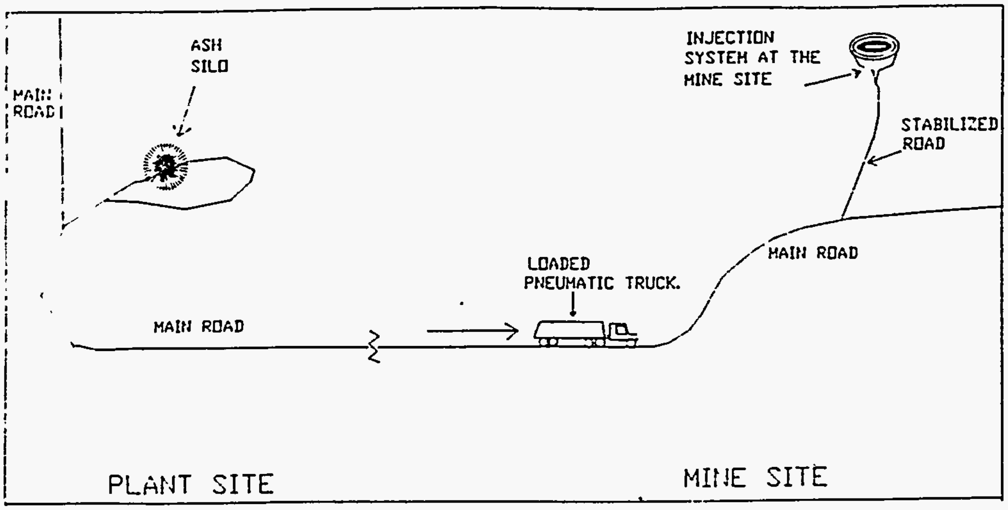

Figure 1. Residue transportation by pneumatic trucks

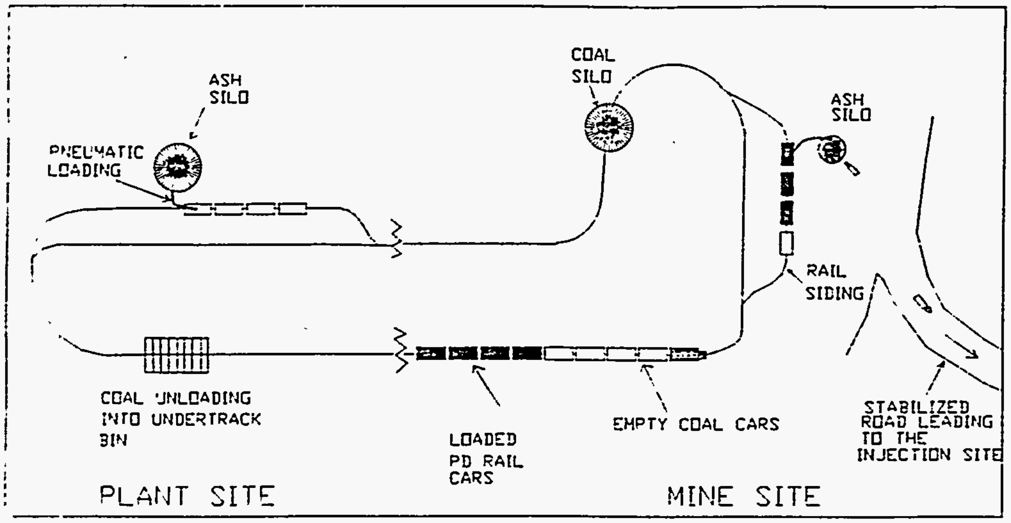

Figure 2. Residue transportation by pressure differential rail cars

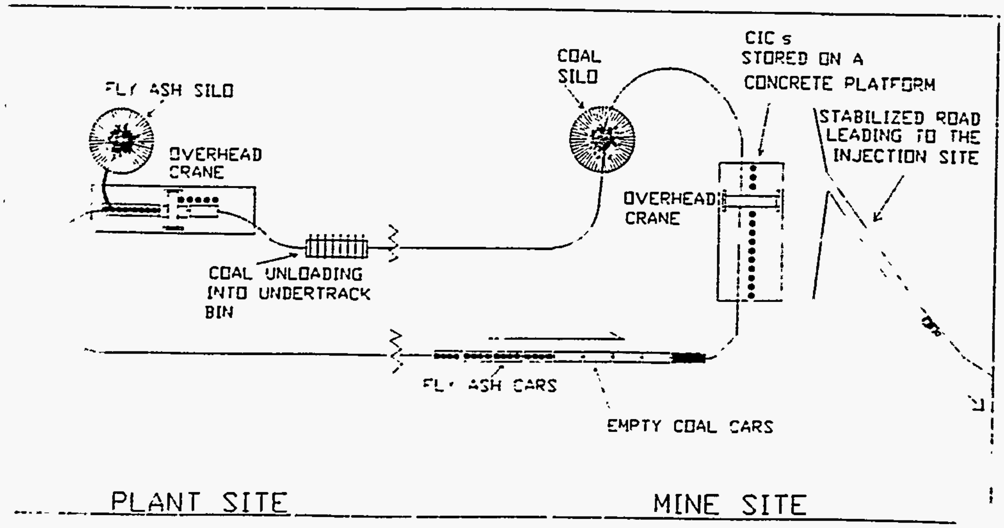

Figure 3. Residue transportation by collapsible intermodal containers

have been left at the junction are pulled into the rail siding and their content is transferred into a silo by the use of a blower. The pneumatic trucks fill their tanks from this silo and then deliver the material to the injection hopper.

When the train arrives at the plant, the empty PD cars are demurred and parked at the junction of the siding. The unit train pulls over the undertrack bin and dumps the coal. After coal dumping is completed, the PD cars which have already been filled and waiting for the train are attached to the back of the empty coal train. Subsequently, the train leaves for the mine. Meanwhile, the empty PD cars are pulled into the rail siding and the fly ash loading process restarts.

The above described operating scenario is valid only when the tonnages of coal and fly ash transported justify a dedicated unit train between the plant and the mine. Otherwise, two sets of PD cars will suffice. The first set will be at the plant site, and when it is filled, a local train will deliver them to the mine. Meanwhile, the empty PDs will be brought back from the mine by the coal train. The exchange of empty cars with the full ones will be scheduled in such a way that there will be no excess accumulation of residue at the mine site.

Collapsible Intermodal Containers (CIC^{TM})

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

The CIC transportation scenario is shown schematically in Figure 3. The coal train arrives at the plant and offloads coal into an undertrack bin. Next, the CICs which have already been filled with fly ash and staged along the rail are lifted, one at a time, by an overhead crane and placed into the bays of the empty coal cars. Two specially designed lifting brackets mounted on both sides of the CIC facilitate lifting and placement by the crane into the bay of the car. Four CICs occupy a car. each taking one of the four bays of a typical coal car. The overhead crane is on rubber tires and travels along the rail track looking inside the cars. When all the CICs are loaded, the train leaves for the mine.

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

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

ECONOMIC EVALUATION

The alternatives were applied to the conditions of central and southern Illinois. Engineering analyses were conducted for each alternative to determine the values of the system parameters and the operating and capital costs. The systems then were evaluated using the "Net Cash Cost (NCC)" method which is also known as the "After-Tax Cost" method. For each year of the project life, the NCC is calculated using the following equation:

$$NCC = (\text{Operating Cost} + \text{Depreciation})\,(1 - \text{tax rate}) - \text{Depreciation} + \text{Capital Cost} \qquad (1)$$

The alternatives were evaluated at three levels of transportation distance; 30, 100, and 200 miles, and at three levels of production rates; 100,000, 200,000, and 300,000 tons per year, to determine the range in which they are favorable. It is noted that in central and southern Illinois, most of the underground mines where residues can be disposed are not more than 200 miles away from the power plants, and the annual residue production does not exceed 300,000 tons. It is assumed that the mine sells 3 tons of coal to the plant for every ton of residue it receives from that plant.

The projects were evaluated over five years, at the end of which all capital cost items were liquidated at fair market prices. A minimum required rate of return of 12 %, an effective tax rate of 40 %, and MACRS depreciation method were applied to all projects. All cost estimates were made in terms of 1995 dollars, and the rate of cost increases is assumed to be the same as the inflation rate so that the evaluations can be done in constant dollars. Some of the important cost data and assumptions specific to each alternative is given below.

Pneumatic Trucks: The depreciation and useful lives of pneumatic trucks are 5 and 10 years, respectively. The capital cost of a unit is approximately $120,000, and the salvage value of a 5-year old unit is assumed to be 50% of its original cost. The number of units in the fleet is determined in such a way that the fleet availability is at least 80% at all mines.
PD Cars: The depreciation and useful lives of PD cars are 7 and 25 years, respectively. The capital cost of a PD car is approximately $80,000, and the salvage value of a 5-year old car is assumed to be 80% of its original cost. This system requires 1/4 mile rail siding both at the plant and the mine site. The rail company will charge the "multiple-car" rate which will be 4 to 5 times higher than the "backhaul" rate since this system will not fit in "backhaul" classification. A car will only qualify for backhaul rate if it is utilized to transport residue when it goes back to the origin.
CICs: The depreciation and the useful lives of CICs are 5 and 20 years, respectively. The capital cost of a CIC is $15,000, and the salvage value of a 5-year old CIC is assumed to be 75% of its original cost. This system requires a 50-foot wide by 1/4 mile long concrete pad along the rail track at both the plant and the mine sites to accommodate the overhead cranes. At the rate of 300,000 tons per year, this system will be charged with a "backhaul" rate by the railroad. However, for lesser tonnages it will be treated as a "multiple-car" since a dedicated unit train can not be justified.

Table 1. Calculation of Transportation Cost for Pneumatic Trucks
(30 Miles - 100,000 Tons Case - central and southern Illinois)

Year	0	1	2	3	4	5
Revenue........	0.	0.	0.	0.	0.	0.
-Operating Cost.	0.	-432.	-432.	-432.	-432.	-432.
-Depreciation...	0.	-72.	-115.	-69.	-41.	-41.
Taxable Income..	0.	-504.	-547.	-501.	-473.	-473.
-Tax Savings....	0.	202.	219.	200.	189.	189.
Net Income......	0.	-302.	-328.	-301.	-284.	-284.
+Depreciation...	0.	72.	115.	69.	41.	41.
-Capital Cost...	-360.	0.	0.	0.	0.	0.
Net Cash Flow...	-360.	-230.	-213.	-232.	-243.	-243.

AFTER-TAX CAPITAL GAIN ($1000) .... = +116.
NET PRESENT VALUE ($1000).................... = -1126.
AFTER-TAX ANNUAL EQUIVALENT COST ($1000) ... = -312.
AFTER-TAX COST PER TON ($) .... = -3.12
BEFORE-TAX PRICE ($) .... = 5.21

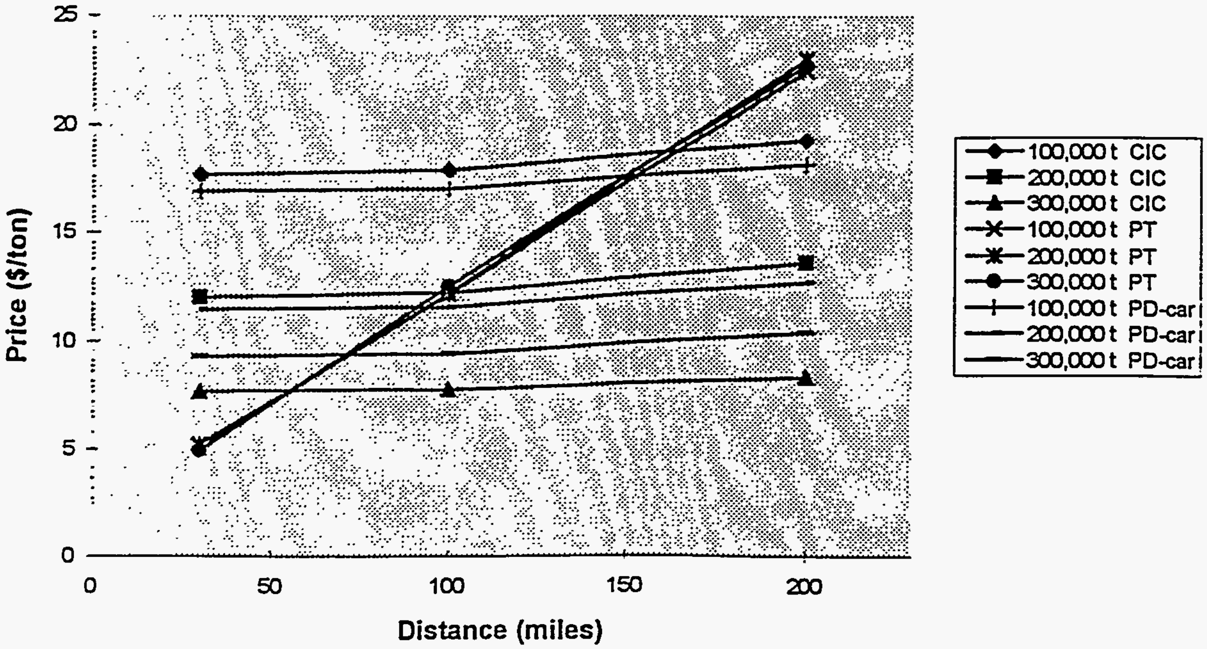

Figure 4. Prices for transporting residues by PTs, CICs, and PD cars

INTERPRETATION OF THE RESULTS

For each particular alternative, a price-per-ton value is calculated for each distance-tonnage combination. Table 1 illustrates the calculation of the price-per-ton when 100,000 tons of ash is transported for 30 miles by PTs. As seen, $360,000 were invested at time 0 to purchase 3 trucks to do the job. The negative cash flows in the next five years indicate that these are costs in each year. The salvage value of these trucks at the end of five years is $180,000, bringing an after-tax capital gain of $116,000. The annual equivalent cost is calculated to be $312,000 at 12% discount rate. The price to be charged per ton of material transported is $5.21. This value is obtained by dividing the after-tax cost of $3.12/ton by (1- tax rate). In other words, if $5.21 is plugged back into the economic model, the NPV of the project would be exactly 0, indicating a return of 12% on investment. The outputs of the 27 runs are presented in Figure 4. The following observations can be made from Figure 1:

1. The price to be charged in PT alternative is not sensitive to tonnage, but very sensitive to distance. That is, the longer the distance the higher the price.
2. The prices to be charged in PD-car and CIC alternatives are not sensitive to distance, but very sensitive to tonnage. That is, the more the tonnage the lower the price.
3. For 100,000-ton annual production, PT alternative provides prices that are lower than those of PD-car and CIC alternatives up to a distance of approximately 150 miles. For shorter distances, the difference in prices between the PT and the other two alternatives is very significant. The prices of PD-car and CIC alternatives show similar trend, but on the average, the price of PD-car runs $0.9 less than the CIC.
4. For 200,000-ton annual production, PT alternative provides prices lower than those of the PD-car and CIC alternatives up to a distance of approximately 90 miles. Again, the difference is very significant at shorter distances. The prices of PD-car and CIC show again similar trend, but on the average, the price of PD-car runs $0.7 less than the CIC.
5. For 300,000-ton annual production, PT alternative is lower priced than PD-car and CICs up to 60 miles in distance. Otherwise, the CIC price is on the average $1.80 lower than the PD-car, reversing the price differential seen between the PD-car and CICs at lower tonnages. This is due to the CICs being charged at a rail backhaul rate at 300,000 tons per year.

CONCLUSIONS

The results above are based on cost estimates and system assumptions for central and southern Illinois. Importantly, this model can be applied elsewhere by substituting the actual costs and system factors for other plant sites and geographic regions. As seen in item 5 above, price sensitivity is such that a backhaul rate at any tonnage can reverse the economics in favor of CICs, especially for long distances. In a final system decision, one must also consider intangibles. For example, in moving large tonnages, the impact of increased truck traffic and road deterioration may favor CICs. It is also noted that the transportation option between a power plant and a mine may be narrowed by existing facilities at these sites. For instance, availability of rail sidings may favor PD cars over other alternatives.

ACKNOWLEDGMENT

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APPENDIX B

An Interactive Software for the Evaluation of Residue Transportation and Haldling Alternatives

AN INTERACTIVE SOFTWARE FOR THE EVALUATION OF RESIDUE TRANSPORTATION AND HANDLING ALTERNATIVES

H. Sevim, D. Lei, and S. Gwamaka

Department of Mining Engineering
Southern Illinois University
Carbondale, IL 62901

ABSTRACT

The disposal of coal combustion residues in abandoned sections of underground coal mines is considered as a promising alternative to surface disposal in landfills and ponds near the power plants. For underground disposal to be competitive, safe and economic transportation of residues from the plants to the mines is essential. An interactive software has been developed to evaluate different transportation and handling alternatives for a given plant-mine scenario in order to find the best alternative. The software is a Windows application that has been developed using Delphi, a rapid application development (RAD) tool. The capabilities of the software is illustrated by a hypothetical case.

INTRODUCTION

The technical, environmental, and economic feasibility of coal combustion residue disposal into old underground coal mines in Illinois is being investigated at the Southern Illinois University. The project has been carried out in cooperation with the U.S.D.O.E.-METC since September of 1993, and it is progressing in 5 coordinated areas: 1) Residue Characterization, 2) Materials Handling, 3) Underground Residue Placement, 4) Field Demonstration, and 5) Economic Evaluation.

The objective of the Materials Handling research is to identify the systems that are technically, economically, and environmentally feasible in handling and transporting the residues from the power plant to the injection site. There are a number of existing, as well as futuristic, handling and transportation alternatives that can serve a given plant-mine scenario. The engineering and economic evaluations to find the most suitable alternative can be very demanding. Also, each alternative has several critical variables, for which the designer would like to conduct sensitivity analyses to reveal their impact on the overall project economics. It is for these reasons that a generalized "Evaluation" model has been developed and keyed into a software package.

SOFTWARE

As will be seen in the "EXAMPLE" section, the evaluation of any transportation/handling system requires a wealth of input data, an operating policy, scheduling, engineering and cost computations, and finally an economic evaluation. Assuming that at least two or three systems are technically feasible for a given plant-mine scenario, the evaluations of these systems require ample time of data collection, familiarity with spreadsheet environment to be able to manipulate the collected data, familiarity with engineering and economic computations, and familiarity with computer programming. Furthermore, the large number of critical variables in the systems warrant sensitivity analyses. Such a task requires repetitive computations.

The developed software efficiently overcomes the above cited difficulties and problems. Through the materials handling research, a few transportation and handling alternatives were identified as environmentally sound. Data on system design, operating schedules, unit operating and capital costs were collected for these alternatives. Operating policies were developed and sketches drawn using Computer Aided Design (CAD) software. Spreadsheets were developed using Microsoft EXCEL for engineering and cost computations. An economic evaluation model based on "After-Tax Cost" method was also developed and coded in FORTRAN.

Initially, the evaluations were performed by switching from one computing environment to another and preparing input files using a text editor, which made the system difficult to use and raised the need for an integrated software. Delphi, a rapid application development (RAD) tool was used for this purpose. A Windows application has been developed integrating all components of the system in such a way that the user can accomplish various tasks "under one roof." As a result, the user can input parameter values in better organized dialog boxes, and see the results of cost and engineering calculations right on the spot without the need to know the complicated underlying structure. Helpful plots of system schematics and item explanations are shown on the screen as item-sensitive features. The procedural steps involved in the system are accomplished using a context-sensitive menu system.

In this application, a template has been developed for each environmentally sound alternative. When the user selects one of these alternatives, the input data are presented in various dialog boxes. The source of a particular data entry can be retrieved by clicking on that entry. If the entry is not satisfactory, the user can overwrite it by typing in his own number. In other words, default values are provided for every single entry in the template of the selected alternative.

To date, templates have been developed for three alternatives: 1) Pneumatic Trucks, 2) Pressure Differential Rail Cars, and 3) Collapsible Intermodal Containers (CIC^{TM}). With a certain degree of familiarity with the software, other alternatives can be easily generated from the three templates built in the software. For instance, any type of truck transportation can be simulated by the use of Pneumatic Trucks template. Similarly, any rail car transportation can be simulated by the use of Pressure Differential Rail Cars template.

It is noted that the template is necessary for the first time use of the transportation alternative. When the user confirms the entries and makes the desired changes in the default values, he/she can save the generated file by using the "File" menu item. Next time, he can open this file through the "File" menu item and edit it. Printing the outputs is also done through this menu item.

EXAMPLE

The alternative selected for demonstration of the software is the transportation of residues by pressure differential (PD) rail cars. As seen in Figure 1, by selecting the "Alternatives" menu item from the menu bar, the user calls the sub-menu which shows the three alternatives that were built in the software. When the user clicks on the PD rail car alternative, the software provides the sketch of the system. The description of the system operation appears on the box next to the sketch.

After becoming familiar with the system, the user can click on the "Parameters" menu item and activate the spreadsheet-like dialog box for entering the "General Parameters" values. The data under General Parameters are shown in Table 1. Next on the menu bar is the "Unit Costs". By activating this item, the user is supplied with equipment capital costs and various unit operating costs. Any question on the source of these unit costs can be answered by clicking on that unit cost, which will retrieve the information from the data bank and present it in a box. Should the user have better information on that unit cost, he can overwrite the default value by typing on that line. The unit operating and capital cost items for the PD-car alternative are also shown in Table 1. "Scheduling" is the next item on the menu bar. Under this category, the scheduling and engineering computations are performed for both the plant and mine sites. Figure 2 shows the spreadsheet-like dialog box for the plant site together with an information box giving the information on the "length of rail siding." The specific items under each site are shown in Table 1 under subtitles "POWER PLANT" and "MINE SITE."

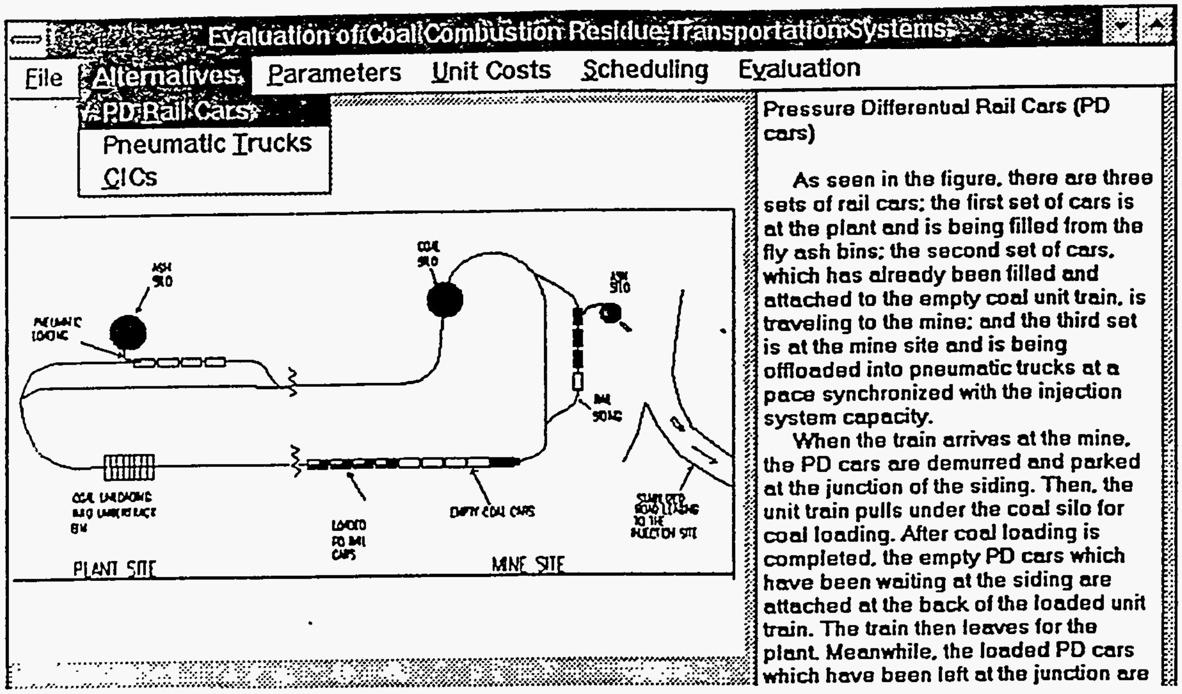
The last menu item is "Evaluation". When this item is clicked, three sub-items appear on the menu: 1) Cost Calculations, 2) Financial Data, 3) Start Evaluation. By invoking the Cost Calculations item. all operating and capital cost calculations are performed for both the plant and mine sites. In Table 1, these calculations are listed under PLANT CAPITAL COSTS, PLANT OPERATING COSTS. MINE CAPITAL COSTS, and MINE OPERATING COSTS.

Under "Financial Data", the user is expected to enter the cost of each capital investment item used in the system. its depreciation life, and the year it was invested. The effective tax rate and the required rate of return are also entered under this item. After the item is completed, the user can invoke the "Start Evaluation" item. This will run the "After-Tax Cost" model and will produce an output file. This file will contain the Net Cash Flows for each year of the project life, the After-Tax Net Present Value. After-Tax Cost and Before-Tax Price to be charged to the customer. For this hypothetical case, the project life was 5 years, effective tax 40%, and minimum required rate of return 12%. It is found that a company undertaking the transportation of 200,000 tons of residue per year from a power plant located 200 miles from the mine, should charge $12.24 per ton in order to secure 12% return on its investment.

CONCLUSIONS

An interactive software has been developed to evaluate different alternatives of transporting coal combustion residues from the power plants to underground coal mines. A few environmentally sound alternatives have been built in the software as templates. The user can conveniently use these templates to evaluate a given mine-plant scenario. With moderate knowledge of the software structure. the user can also generate other transportation and handling alternatives that

Figure 1. First window for the PD car transportation alternative

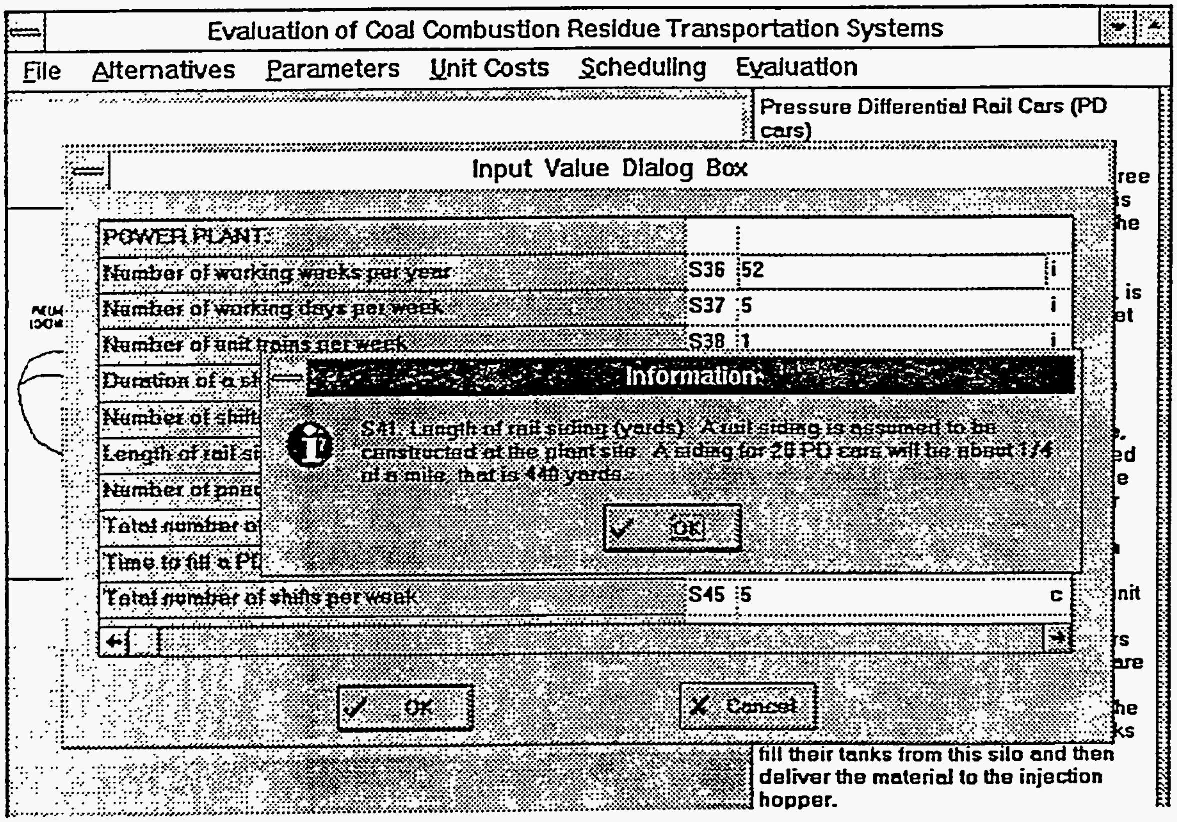


Pressure Differential Rail Cars (PD cars)

As seen in the figure, there are three sets of rail cars: the first set of cars is at the plant and is being filled from the fly ash bins; the second set of cars, which has already been filled and attached to the empty coal unit train, is traveling to the mine; and the third set is at the mine site and is being offloaded into pneumatic trucks at a pace synchronized with the injection system capacity.

When the train arrives at the mine, the PD cars are demurred and parked at the junction of the siding. Then, the unit train pulls under the coal silo for coal loading. After coal loading is completed, the empty PD cars which have been waiting at the siding are attached at the back of the loaded unit train. The train then leaves for the plant. Meanwhile, the loaded PD cars which have been left at the junction are

Figure 2. Window for power plant scheduling



Pressure Differential Rail Cars (PD cars)

POWER PLANT:			
Number of working weeks per year	S36	52	i
Number of working days per week	S37	5	i
Number of unit trains per week	S38	1	i
Duration of a s			
Number of shif			
Length of rail s			
Number of pneu			
Total number o			
Time to fill a PD			
Total number of shifts per week	S45	5	c

S41. Length of rail siding (yards). A rail siding is assumed to be constructed at the plant site. A siding for 20 PD cars will be about 1/4 of a mile, that is 440 yards.

fill their tanks from this silo and then deliver the material to the injection hopper.

Table 1. Input parameters and cost computations for the PD rail car transportation

GENERAL PARAMETERS	
Distance between the power plant and the mine (miles)	200
Annual fly ash production (tons)	200000
Project life.	5
Pneumatic truck capacity (tons)	24
Rail car capacity (tons)	100
CAPITAL COST ITEMS:	
Cost of a pneumatic truck	120000
Cost of a PD car	80000
Cost of rail siding ($/yard)	570
Ash silo capital cost (250 tons capacity)	225000
OPERATING COST ITEMS:	
Fuel consumption rate (gal/hour)	2.5
Maintenance cost rate (% of capital cost - in decimals)	0.02
Overhead cost rate (% of operating cost - in decimals)	0.1
Tire cost per truck mile ($/truck/mile)	0.06
Hourly charge of contracted labor at the plant site	35
Hourly wage for truck drivers and silo operator at the mine	27
Fuel cost ($/gallon)	1.1
Railroad charge ($/ton/mile)	0.0226
Insurance cost (% of capital cost - in decimals)	0.01
Cost of road construction ($/sq. yd)	5
PD car maintenance cost (when they are leased) ($/mile /car)	0.03
Leasing cost of a PD car ($/month/unit)	900
Leasing cost of a pneumatic truck ($/month/unit)	0
POWER PLANT:	
Number of working weeks per year	52
Number of working days per week	5
Number of unit trains per week	2
Duration of a shift (hours)	8
Number of shifts per day	1
Length of rail siding	440
Number of pneumatic rail cars required per trip	20
Total number of PD cars required for the operation	40
Time to fill a PD car (hours)	0.5
Total number of shifts per week	5
Number of hours per year a contracted labour will work	1040
PLANT CAPITAL COST:	
Rail siding	250800
Pressure differential rail cars	0
Total capital investment	250800

PLANT OPERATING COST:	
Rail road charge	904000
PD leasing cost	432000
PD car maintenance cost (when they are leased)	49920
Insurance	2508
Maintenance	5016
Contracted labor at the plant	36400
Overhead cost	142984
Total operating cost	1572828
MINE SITE:	
Initial distance between rail siding and injection site (miles)	1
Area of new road to be constructed (sq. yd)	28000
Number of working weeks per year	52
Number of working days per week	5
Pneumatic truck cycle time (minutes)	60
Duration of a shift (hours)	8
Number of shifts per day	1
Length of rail siding (yards)	440
Total number of truck trips required per shift per truck	8
Total number of trips required per shift	32
Number of trucks required per shift	4
Number of ash silo operators at the mine per shift	1
Total number of truck drivers required per shift	4
MINE CAPITAL COSTS:	
Rail siding ($570/yard)	250800
Ash silo (250 tons capacity)	225000
Pneumatic trucks	480000
Total capital investment	955800
MINE OPERATING COSTS:	
Leasing cost of pneumatic trucks	0
Road construction cost	140000
Insurance	9558
Operators wage	280800
Tire cost	998
Fuel cost	22880
Maintenance	19116
Overhead cost	47335
Total operating cost	520687
TOTAL CAPITAL INVESTMENT	1206600
TOTAL OPERATING COST	2093515
Operating cost per ton	10.47

he/she may desire to evaluate. The most important benefit of the software lies in its ability to provide a wealth of information on the selected transportation alternative, to evaluate a number of alternatives in a short period of time, and to facilitate sensitivity analysis on important parameters.

ACKNOWLEDGMENT

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APPENDIX C

Report on Flow Measurements in Pneumatic Stowing of Dry FGD Wastes

Abstract

The minimum pressure required to break a stationary plug of fly ash with a pulse of air was measured in an experimental apparatus. Plugs of various lengths were prepared in 6.5 inch ID clear acrylic tubing. The plugs were subjected to bursts of pressurized air and the results were observed. Particles in the upper portion of the plug were displaced in a repeatable pattern. The downstream side of the plugs were severely fractured. A linear relationship was found to exist between plug length and break pressure. A pressure of approximately 0.5 psi/ft was required to break the plugs. At pressures ten to twenty times higher than the minimum pressure, the entire upper portion of the plug was sheared off. The remaining bottom portion of the plug formed into a wavy pattern.

Current work is being carried out to acquire pressure drop and mass flow rate data in a two inch conveying line. This should help in design of a conveying system into the mine. Throw distance calculations were carried out to determine how far the fly ash would travel in the mine. It was found that if the particles could be kept from agglomerating, the fly ash could carry up to 80 meters. That information coupled with the pressure drop and mass flow rate data should give some insight into actual conveyance in the mine itself.

Finally, friction factors are being measured so that the experimental data concerning plug breakage can be compared with a theoretical model in the hopes that plug breakage could be predicted in the mine after plugging has occurred.

Introduction & Background

Pneumatic stowing is a well established practice for preventing mine subsidence. Subsidence occurs when the structure of the mine deteriorates and the mine roof collapses. Filling the mine cavity with support material can prevent this process from occurring. Fly ash serves as an excellent support material. because it forms a cement when mixed with water. Furthermore, by using the fly ash as backfilling material, the cost of ash disposal is reduced. Finally, fly ash is alkaline substance which will reduce mine acid runoff.

In a typical operation, the backfilling process begins from the end of a mine shaft [1]. The transport line is then withdrawn as the mine is filled in. Because entry to an abandoned mine is too dangerous. a new technique for backfilling sealed mines is being developed.

Boreholes will be drilled to allow the transport of fly ash, and exhaust of transport and displaced air. This technique offers challenges not found with conventional stowing methods. To reduce drilling costs it is necessary to determine the maximum conveying distance. The material must be conveyed in broad channels where the cross-sectional area will limit the conveying velocity which can be achieved. At these low velocities, saltation and plug formation are serious concerns.

In an effort to prevent saltation, it is necessary to determine the minimum conveying velocity in the dilute phase. Dilute phase pneumatic transport is a well understood concept. It has been researched thoroughly in the past and can be implemented if all the proper equipment is available. The necessary equipment for dilute phase transport includes a bin, feeder, air mover, pipeline and collection devices. Most of the challenges brought about by the fly ash would come in the collection devices such as baghouse filters.

The baghouse filters that would normally be employed with very fine particles, such as fly ash, create large pressure drops and can only filter small amounts of air. Because of this, moving large volumes of air would require many baghouse filters which can become fairly expensive. In any event, dilute phase transport can only go so far. At that point, plugs will develop and the breaking of them will become necessary.

The breaking of stationary plugs has not been explored in great detail. Tusuji, Morikawa and Honda [2] found that for course particles, the critical pressure required to break a plug is proportional to plug length. These investigators determined the critical pressure by gradually increasing the air flow rate through a plug and measuring the pressure drop when the plug was broken. Furthermore, the critical pressure decreased with an increase of pipe diameter.

The breaking pressure will undoubtedly be dependent upon the properties of the plug material. Cohesiveness, particle size distribution, shape and air retention are some of the properties which determine the breaking pressure. [3] In addition to the properties mentioned above, the structural integrity of each plug must be considered.

Experimental Apparatus & Procedure

Dilute Phase Transport Experiments

Figure 1 is a diagram showing the experimental setup of the two inch line for dilute phase pneumatic conveying. A six inch setup similar to the two inch system was attempted, but it was found that dust collection was a large problem due to the large amounts of air necessary to convey fly ash in a six inch line. It was decided that bigger dust collection equipment would not be purchased due to the time limitations and large cost for the proper equipment.

Figure 1. Diagram of two-inch system for acquisition of pressure drop and flowrate data.

The two inch air line supply is compressed house air which is damped out by a surge tank which can support flowrates of up to 75 scfm of air. This is more than enough air to sustain pneumatic flow. The solid particles will be dropped into the line by means of a gate valve coupled with orifice plates so that the mass flow rate will be controlled. The mass flow rate of the solids will be determined by means of disappearance from the hopper. As the particles proceed down the line, they cause a drop in pressure drop. The pressure drop per unit length will be measure via Omega pressure transducers and data acquisition system connected to an IBM AT. This data should allow for scale up of flow to the actual conveying used to bring the fly ash into the mine.

Plug Breakage Experiments

Figure 2 shows the configuration of the apparatus used to test plug breakage. Compressed air is fed into the air reservoir. The reservoir is a five foot section of four inch carbon steel pipe. The pressure is measured by a gauge near the air inlet. The reservoir is connected to the test section by a fast acting, two inch, pneumatically actuated ball valve. Opening this valve allows the pressurized air to enter the test section.

Figure 2. Experimental setup of plug breaking apparatus.

The test section consists of a twelve foot long section of 6.5 inch ID clear acrylic tubing. The tubing is flanged at both ends. The flanges permit connection of the test section to the pneumatic valve and the collection bin.

To form the plugs, the test section is removed and rotated vertically. The lower end is sealed with a cap and plug retainer. The retainer prevents the plug from filling the entire end of the tube. It also serves to produce a uniform void volume between the plug and the valve. Fly ash is poured into the tube while the test section is seated on a scale. When the desired weight of ash has been added to the tube, the plug is allowed to settle. To enhance the settling, the tube is tapped repeatedly with a dead-blow hammer. The settling process continues until the bulk density reaches an approximate value of 64.4 lb/ft^3. This bulk density is approximately equal to what would be achieved if the material would be allowed to settle naturally for one hour.

When the fly ash has settled, the test section is attached to the apparatus and the air reservoir is pressurized. The ends of the plug deform leaving the length of plug contacting the top of the tube somewhat shorter than the length of the bottom of the tube. This minimum length is then recorded as the plug length. When the reservoir pressure has been recorded, the pressure is released into the test section. Before the air reaches the plug it expands through a short length of connecting pipe and a void volume before the plug. The effect of the pressure wave on the plug is observed and recorded. The pressure exerted on the face of the plug is calculated from the expansion volume.

Results & Discussion

Throw Distance Calculations

The theoretical distance of FBC fly ash was calculated using momentum and energy balances. It makes the following assumptions:

- Two streams: one of air alone, and one with solids and air which mix perfectly at the injection point
- Solids stream:
- 10 pounds of solids/pound of air
- Velocity = 30 m/s
- Air stream:
- Near supersonic velocity; velocity = 320 m/s
- Volumetric flowrate of air = 2000 standard ft^3/min

After the streams mix and expand, the air in the mine tunnel is .1 m/s. By calculating the settling velocity of various sized particles, the amount of time the particles stay airborne can be found and the throw distance calculated. Figure 3 shows a graph of the theoretical throw distance vs. the particle size. The minimum point is the transition between the Stokes regime and the intermediate regime in the settling velocity.

Figure 3. Throw Distance vs. Particle Diameter for FBC Fly Ash

The larger particles go fairly far because of the larger amount of momentum that they possess. The smaller particles go far as well, due to very small settling velocities. Therefore, either the particles should be agglomerated into very large particles, or should be kept from agglomerating at all to keep their settling velocities small.

Plug Breakage Studies

The minimum break pressure of fly ash plugs was experimentally determined for plugs of 2, 4, and 8 feet in length. The results of these experiments are given in figure 4. A linear relationship appears to exist between break pressure and plug length. The critical break pressure is 0.5 psi/ft of plug.

Figure 4. Pressure Difference vs. Plug Length for Stationary

FBC Fly Ash Plugs (Inside Diameter = 6.5 in.)

Figure 5 shows the typical fracture pattern at the minimum break pressure. The top view indicates a parabolic indentation on the upstream end of the plug. This shape was seen in both broken and unbroken plugs. Beyond the parabola, a roughly circular hole forms which is typically about two inches in diameter. This hole is the starting point of a path which meanders through the remainder of the plug. Fissures frequently form along the meandering path. These fissures may extend deep into the plug. A fissure is evident in the side view of the broken plug. On plugs which did not break, the air seemed to be held in the parabolic area.

Figure 5. Top and side vies of a plug broken at its critical pressure from left to right.

Several tests were also performed at much higher pressure. Table 1 indicates the pressures and plug lengths corresponding to these experiments. Figure show the break pattern to be significantly different at higher pressures. In case one, the plus was short and fragile. When it was hit with the blast of air, the plug was completely demolished. In cases 2-5, only the bottom half of the plug, in a wavy formation, remained after pressure was applied. The plugs were moved about six inches before breakage in cases 3 and 4.

Table 1. High pressure plug breakages.

Figure 6. Side view of a plug broken at a high pressure from left to right.

The fly ash has excellent air retention properties. To insure that the plugs were similarly compact, the bulk density of each plug was calculated. The mean bulk density for each plug length was calculated and is reported in Table 2.

Table 2. Mean bulk densities of plugs of various lengths.

The greatest challenges in this study was the creation of uniform plugs. The fly ash is highly aeratable. Its ability to retain air made it difficult to form plugs which did not develop air pockets as they settled. Striking the tube with a dead-blow hammer helped to remove the air pockets somewhat, but when that failed, the plugs were unusable.

The plug forming problem was further complicated by the necessity of rotating the tube from a vertical to a horizontal position. During rotation, the ends of the plug deforms somewhat. Because of this, it was extremely difficult to predict the plug length before rotation. Sometimes, the rotation of the tube caused the formation of cracks in the plug. When cracks were found, the plug was considered unusable.

This method of building plugs suffers from another problem. Since the plug is allowed to settle in a vertical orientation, the compaction is not uniform across the length of the plug. This situation does not ideally represent a plug which might form during the backfilling process. In an effort to better represent real plug formation, attempts were made to extrude plugs from a blow tank into horizontal pipes. Two difficulties prevented the use of this method. First, it was difficult to release the air from the fly ash. Thus, the plugs would be only minimally compacted and settling would produce gaps between the pipe and plug. Secondly, the only pressure vessel available had a two inch diameter outlet. The entire cross-section of six inch pipe could not be filled after the expansion. The extrusion method was abandoned because of these difficulties.

To show that the pressure applied to the plug was the minimum, pressures less than that were attempted and shown to hold for a longer period of time. At pressures slightly less than the critical pressure, the plug would initially hold, but after approximately one minute, the continued pressure would finally seep through the plug and break in apart. It was not considered to be a break due to a pressure blast unless the plug broke in less than five seconds.

Previous work in plug breaking does not lend itself to direct comparison to this investigation. In the earlier investigations, an equilibrium developed between the air permeating through the plug and the pressure drop. [1] When the air flow rate reached a critical value, the plug would deteriorate. In this study, the plug is subjected to a pulse of pressurized air. A shock wave distorts the front side of the plug forming the parabolic pattern depicted in Figure 5. If the pressure is above the critical pressure, the air continues to break through in a meandering pattern following the path of least resistance. Because of these differences, the data from the present study cannot be reliably compared to previous investigations.

Conclusions

The throw distance calculations suggest that to minimize throw distance, the particle agglomeration must be kept to a minimum and the volumetric flowrate of air should be high as possible. The velocity of the air going into the mine should not have much of an effect due to the large expansion area. Therefore, compressed air may not be the best solution if blowers can produce high standard volumes of air.

The experiments have shown that fly ash plugs can be broken with pulses of pressurized air. The minimum pressure required to break the plugs is proportional to the plug length. At the minimum pressure only particles in the upper portion of the plug were noticeably displaced. The air surged through the upper layers of the plug leaving a parabolic pattern in its meandering path. At pressures significantly higher than the minimum break pressure, a different flow pattern is observed. With pressures ten times the minimum pressure, the top portion of the plug is sheared off. The remaining material is distributed across the bottom of the pipe in a wave like pattern.

Current Work

Dilute Phase Conveying

As indicated in the experimental section of this report, fly ash is being conveyed in a two inch line. The pressure losses over a fifteen foot section are measured with pressure transducers and a data acquisition system. Experiments are being conducted to determine pressure drops for various solids and air flow rates. This data can be used to predict pressure losses for a much larger system.

Plug Breaking

A model is being developed to predict the minimum break pressure of fly ash plugs. This model will be compared to results of the minimum break pressure experiments to assess its validity. If this model proves to be an effective method of predicting the minimum break pressure in pipe it will be modified to correct physical differences of the mine. To test the model it was necessary to obtain the value of internal angle of friction and angle of friction between ash and the wall material. The results of these tests have been included in the appendices.

APPENDIX D

An Investigation into the Breakage Characteristics of Fly Ash Plugs During Pneumatic Backfilling of Abandoned Coal Mines

A Paper Published At
International Freight Pipeline

An Investigation into the Breakage Characteristics of Fly Ash Plugs during Pneumatic Backfilling of Abandoned Coal Mines

Gregory A. Jama, William H. Link, and George E. Klinzing
Department of Chemical and Petroleum Engineering
University of Pittsburgh
Pittsburgh, PA 15261

Abstract

The minimum pressure required to break a stationary plug of fly ash with a pulse of air was measured in an experimental apparatus. Plugs of various lengths were prepared in 6.5 inch ID clear acrylic tubing. The plugs were subjected to bursts of pressurized air and the results were observed. Particles in the upper portion of the plug were displaced in a repeatable pattern. The downstream side of the plugs were severely fractured. A linear relationship was found to exist between plug length and break pressure. A pressure of approximately 0.5 psi/ft was required to break the plugs. At pressures ten to twenty times higher than the minimum pressure, the entire upper portion of the plug was sheared off. The remaining bottom portion of the plug formed into a wavy pattern.

Introduction

Pneumatic stowing is a well established practice for preventing mine subsidence. Subsidence occurs when the structure of the mine deteriorates and the mine roof collapses. Filling the mine cavity with support material can prevent this process from occurring. Fly ash serves as an excellent support material, because it forms a cement when mixed with water. Furthermore, by using the fly ash as backfilling material, the cost of ash disposal is reduced.

In a typical operation, the backfilling process begins from the end of a mine shaft [1]. The transport line is then withdrawn as the mine is filled in. Because entry to an abandoned mine is too dangerous, a new technique for backfilling sealed mines is being developed.

Bore holes will be drilled to allow the transport of fly ash, and exhaust of transport and displaced air. This technique offers challenges not found with conventional stowing methods. To reduce drilling costs it is necessary to determine the maximum conveying distance. The material must be conveyed in broad channels where the cross-sectional area will limit the conveying velocity which can be achieved. At these low velocities, saltation and plug formation are a serious concern.

The breaking of stationary plugs has not been explored in great detail. Tsuji, Morikawa and Honda [2] found that for course particles, the critical pressure required to break a plug is proportional to plug length. These investigators determined the critical pressure by gradually increasing the air flow rate through a plug and measuring the pressure drop when the plug was broken. Furthermore, the critical pressure decreased with an increase of pipe diameter.

The breaking pressure will undoubtedly be dependent upon the properties of the plug material. Cohesiveness, particle size distribution, shape and air retention are some of the properties which determine the breaking pressure. [3] In addition to the properties mentioned above, the structural integrity of each plug must be considered.

Current work concerns the breaking of fly ash plugs in seven inch clear acrylic tubing. The pressure required to break the plugs is measured and the flow patterns are observed. This investigation will provide a basis for determining the feasibility of a new pneumatic stowing technique.

Experimental

Figure 1 shows the configuration of the apparatus used to test plug breakage. Compressed air is fed into the air reservoir. The reservoir is a five foot section of four inch carbon steel pipe. The pressure is measured by a gauge near the air inlet. The reservoir is connected to the test section by a fast acting, two inch, pneumatically actuated ball valve. Opening this valve allows the pressurized air to enter the test section.

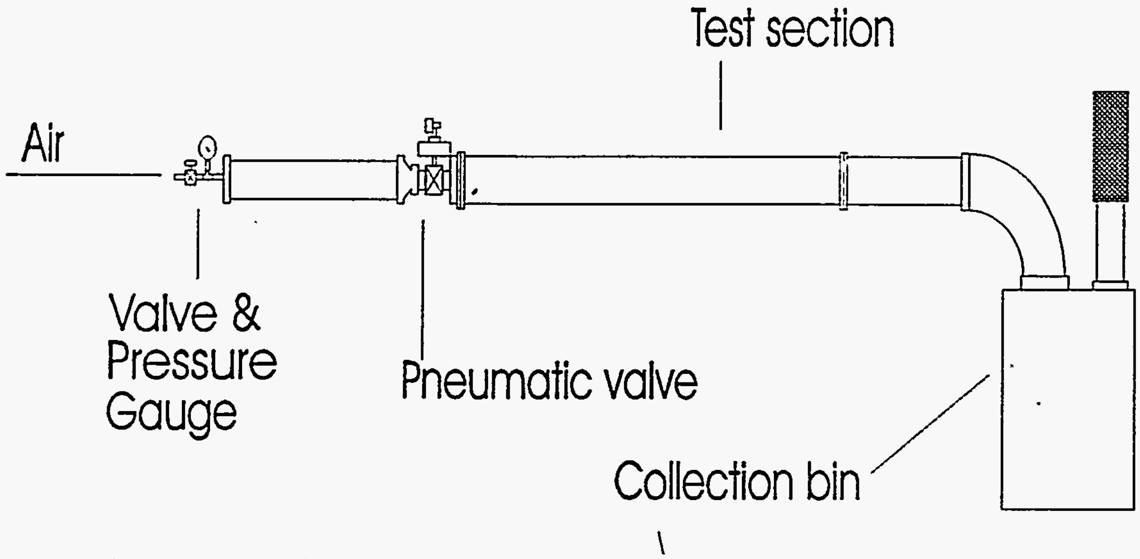


Figure 1. Experimental setup

The test section consists of a twelve foot long section of 6.5 inch ID clear acrylic tubing. The tubing is flanged at both ends. The flanges permit connection of the test section to the pneumatic valve and the collection bin.

To form the plugs, the test section is removed and rotated vertically. The lower end is sealed with a cap and plug retainer. The retainer prevents the plug from filling the entire end of the tube. It also serves to produce a uniform void volume between the plug and the valve. Fly ash is poured into the tube while the test section is seated on a scale. When the desired weight of ash has been added to the tube, the plug is allowed to settle. To enhance the settling, the tube is tapped repeatedly with a dead-blow hammer. The settling process continues until the bulk density reaches an approximate value of 64.4 lb/ft^3. This bulk density is approximately equal to what would be achieved if the material would be allowed to settle naturally for one hour.

When the fly ash has settled, the test section is attached to the apparatus and the air reservoir is pressurized. The ends of the plug deform leaving the length of plug contacting the top of the tube somewhat shorter than the length of the bottom of the tube. This minimum length is then recorded as the plug length. When the reservoir pressure has been recorded, the pressure is released into the test section. Before the air reaches the plug it expands through a short length of connecting pipe and a void volume before the plug. The effect of the pressure wave on the plug is observed and recorded. The pressure exerted on the face of the plug is calculated from the expansion volume.

Results

The minimum break pressure of fly ash plugs was experimentally determined for plugs of 2, 4, and 8 feet in length. The results of these experiments are given in figure 2. A linear relationship appears to exist between break pressure and plug length. The critical break pressure is 0.5 psi/ft of plug.

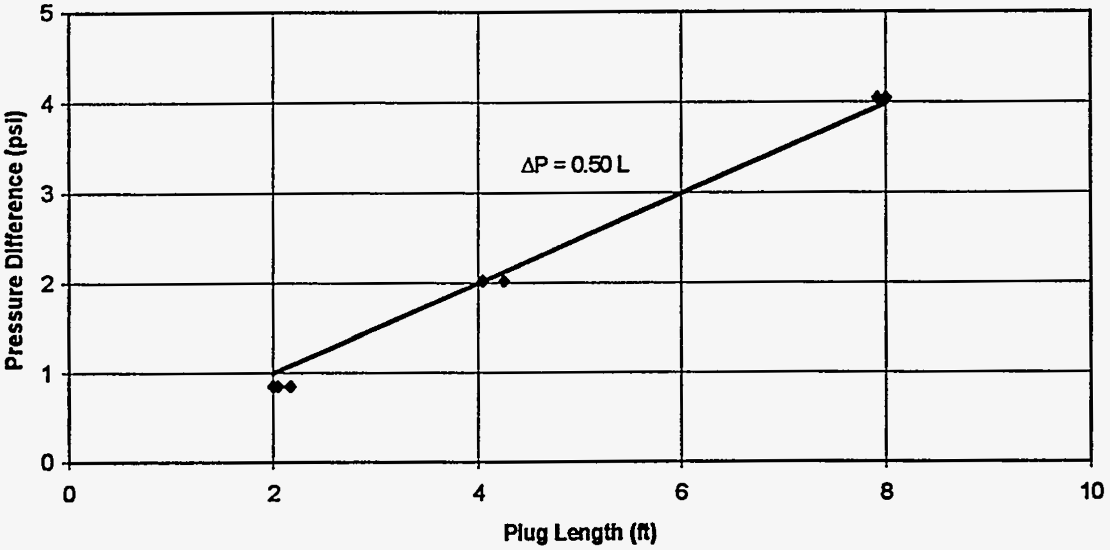
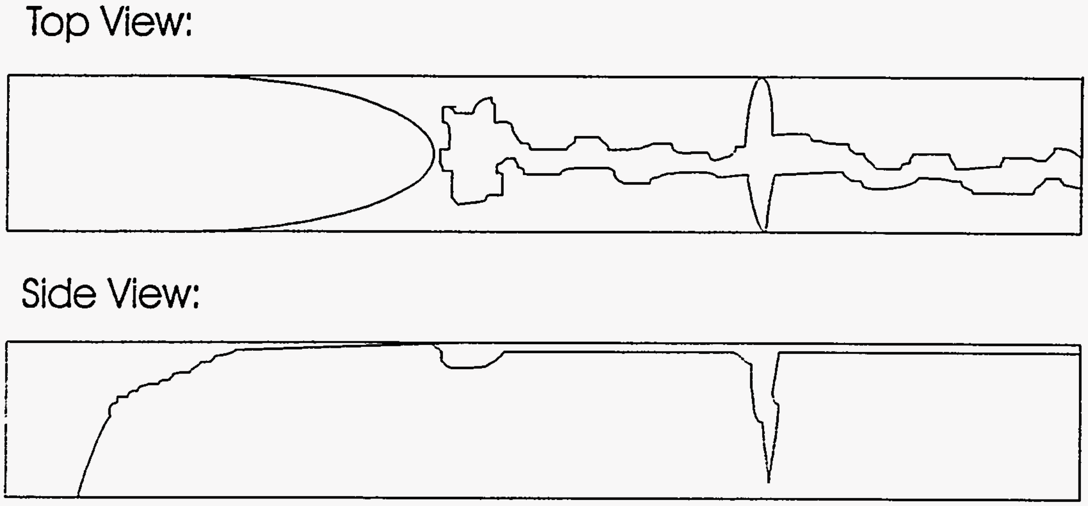


Figure 2. Pressure Difference vs. Plug Length for Stationary FBC Fly Ash Plugs (Inside Diameter = 6.5 in)

Figure 3 shows the typical fracture pattern at the minimum break pressure. The top view indicates a parabolic indentation on the upstream end of the plug. This shape was seen in both broken and unbroken plugs. Beyond the parabola, a roughly circular hole forms which is typically about two inches in diameter. This hole is the starting point of a path which meanders through the remainder of the plug. Fissures frequently form along the meandering path. These fissures may extend deep into the plug. A fissure is evident in the side view of the broken plug. On plugs which did not break, the air seemed to be held in the parabolic area.


Top View:

Side View:

Figure 3. Top and side views of a plug broken at its critical pressure from left to right.

Several tests were also performed at much higher pressures. Table 1 indicates the pressures and plug lengths corresponding to these experiments. Figure 4 shows the break

Case	*Plug Length (ft)*	*Pressure Difference (psi)*
1	1.33	38.9
2	3.17	23.3
3	3.25	38.9
4	3.58	31.1
5	5	38.9

Table 1. High pressure plug breakages

Figure 4. Side view of a plug broken at a high pressure from left to right.

pattern to be significantly different at higher pressures. In case one, the plug was short and fragile. When it was hit with the blast of air, the plug was completely demolished. In cases 2-5, only the bottom half of the plug, in a wavy formation, remained after pressure was applied. The plugs were moved about six inches before breakage in cases 3 and 4.

The fly ash has excellent air retention properties. To insure that the plugs were similarly compact, the bulk density of each plug was calculated. The mean bulk density for each plug length was calculated and is reported in Table 2.

Approximate Plug Length (ft)	*Mean Bulk Density (lb/ft^3)*
2	64.1
4	64.4
8	64.6

Table 2. Mean bulk densities of plugs of various lengths.

Discussion

The greatest challenge in this study was the creation of uniform plugs. The fly ash is highly aeratable. Its ability to retain air made it difficult to form plugs which did not develop air pockets as they settled. Striking the tube with a dead-blow hammer helped to remove the air pockets somewhat, but when that failed, the plugs were unusable.

The plug forming problem was further complicated by the necessity of rotating the tube from a vertical to a horizontal position. During rotation, the ends of the plug deforms somewhat. Because of this, it was extremely difficult to predict the plug length before rotation. Sometimes, the rotation of the tube caused the formation of cracks in the plug. When cracks were found, the plug was considered unusable.

This method of building plugs suffers from another problem. Since the plug is allowed to settle in a vertical orientation, the compaction is not uniform across the length of the plug. This situation does not ideally represent a plug which might form during the backfilling process. In an effort to better represent real plug formation, attempts were made to extrude plugs from a blow tank into horizontal pipes. Two difficulties prevented the use of this method. First, it was difficult to release the air from the fly ash. Thus, the plugs would be only minimally compacted and settling would produce gaps between the pipe and plug. Secondly, the only pressure vessel available had a two inch diameter outlet.

The step from two inches to six inches disrupted the homogeneity of the extrusion. The entire cross-section of six inch pipe could not be filled after the expansion. The extrusion method was abandoned because of these difficulties.

To show that the pressure applied to the plug was the minimum, pressures less than that were attempted and shown to hold for a longer period of time. At pressures slightly less than the critical pressure, the plug would initially hold, but after approximately one minute, the continued pressure would finally seep through the plug and break in apart. In was not considered to be a break due to a pressure blast unless the plug broke in less than five seconds.

Previous work in plug breaking does not lend itself to direct comparison to this investigation. In the earlier investigations, an equilibrium developed between the air permeating through the plug and the pressure drop. [1] When the air flow rate reached a critical value, the plug would deteriorate. In this study, the plug is subjected to a pulse of pressurized air. A shock wave distorts the front side of the plug forming the parabolic pattern depicted in Figure 4. If the pressure is above the critical pressure, the air continues to break through in a meandering pattern following the path of least resistance. Because of these differences, the data from the present study can not be reliably compared to previous investigations.

Conclusions

The experiments have shown that fly ash plugs can be broken with pulses of pressurized air. The minimum pressure required to break the plugs is proportional to the plug length. At the minimum pressure only particles in the upper portion of the plug were noticeably displaced. The air surged through the upper layers of the plug leaving a parabolic pattern in its wake. The air penetrated the remainder of the plug while creating fissures along its meandering path. At pressures significantly higher than the minimum break pressure, a different flow pattern is observed. With pressures ten times the minimum pressure, the top portion of the plug is sheared off. The remaining material is distributed across the bottom of the pipe in a wave like pattern.

References

1. Peter, G. Measurements Made On Pneumatic Stowing Machines for Determining the Relationship Between Stowing Efficiency, Length of Pipe Quantity of Air and Pressure. *Transl. 3780, National Coal Board Transl. A. 261, from Glukhauf year 88*, v. 33/34, (1952), 807-819.

2. Tsuji, Y., Morikawa, Y., and Honda, H. A Study On Blowing Off a Stationary Plug of Coarse Particles in a Horizontal Pipe. *Journal of Powder & Bulk Solids Technology.* 3 (1979) 4, 30-35.

3. Dhodapkar, S.V., Plasynski, S.I., and Klinzing, G.E. *Plug Flow Movement of Solids. Powder Technology.* 81 (1994) 3-7.

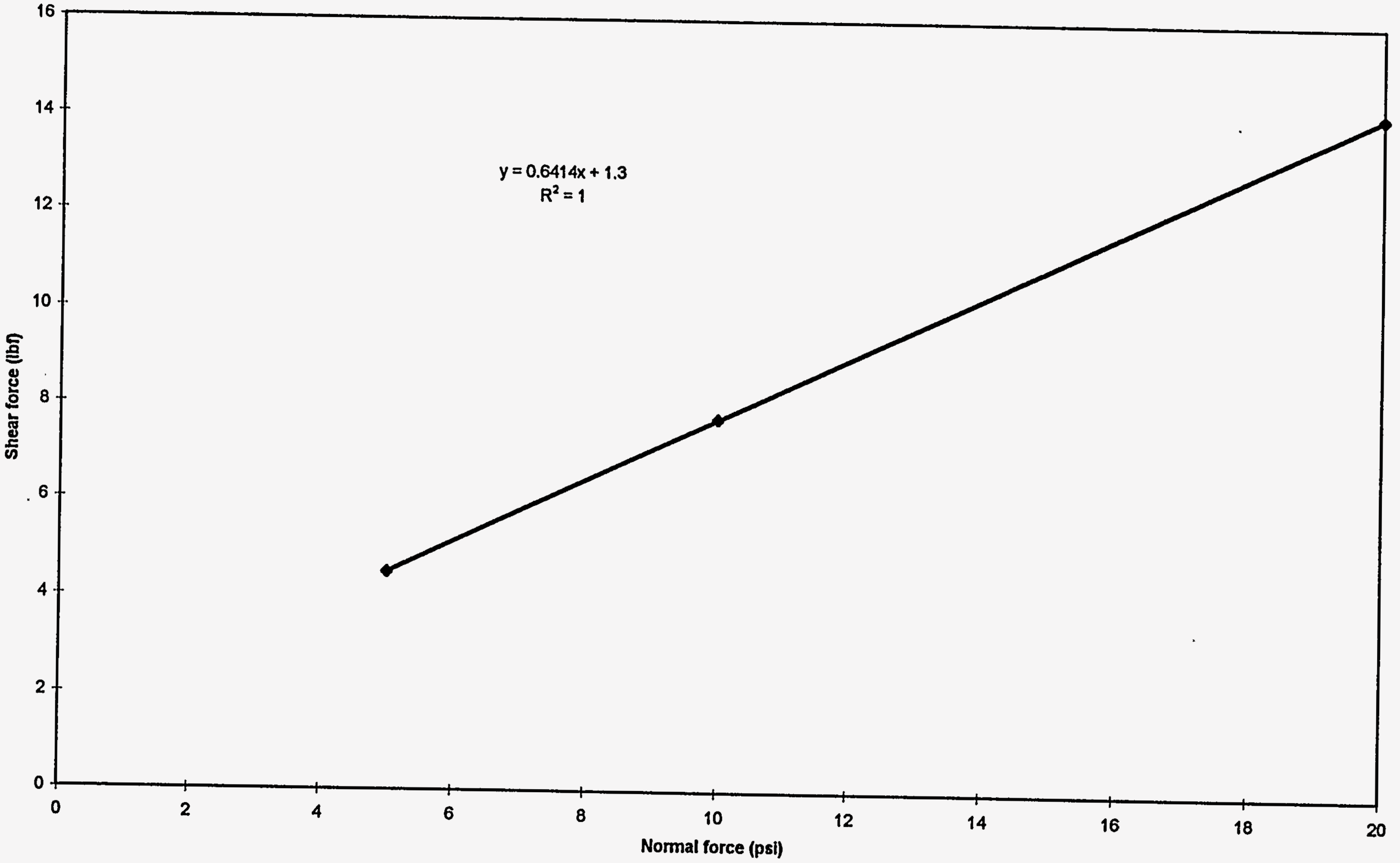
MATLAB code

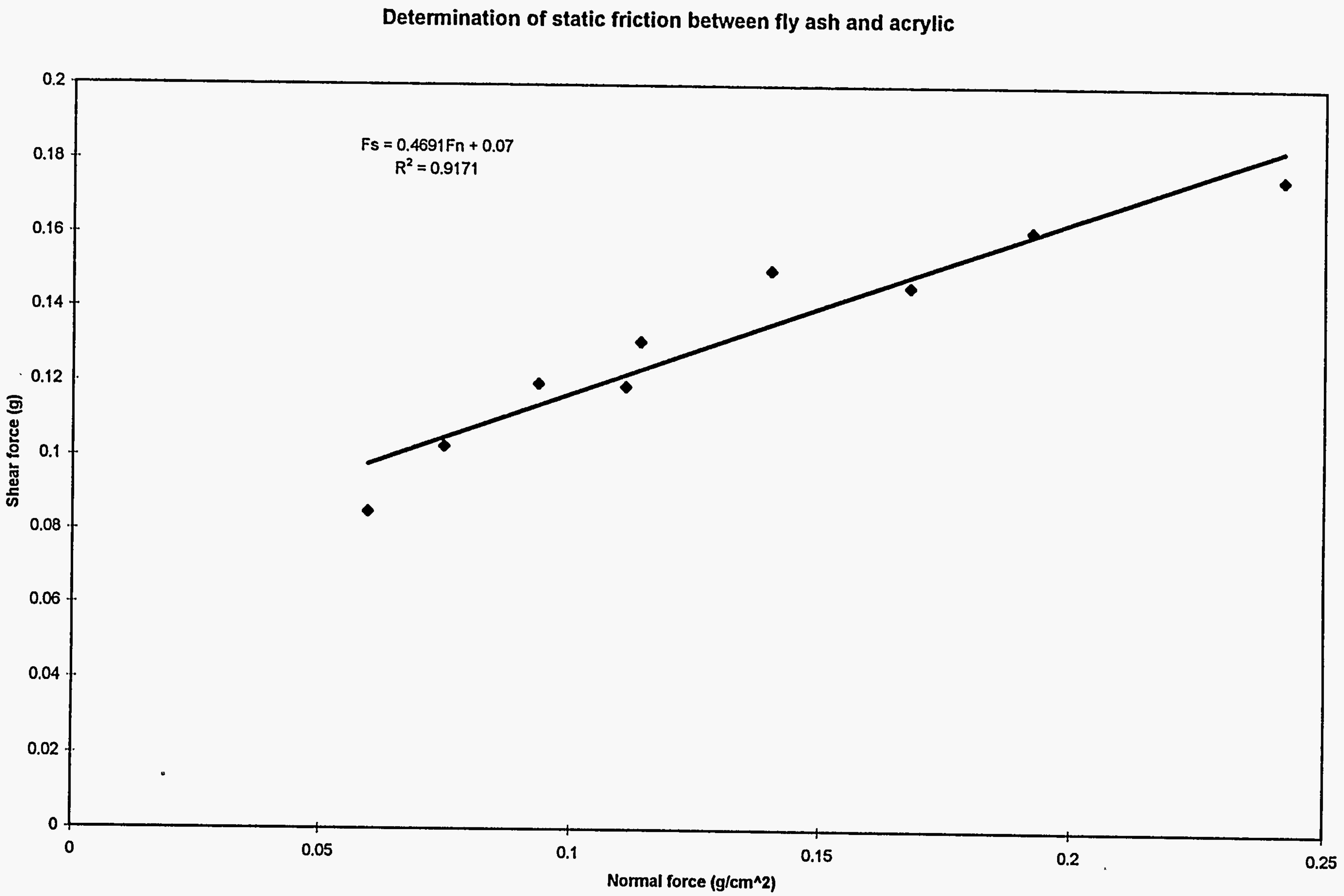
```
%superson.m
%Ash at loading of 10 moving at 30 m/s plus a gas stream of air
%at speed of sound

load = 10;      %loading = pounds of solids/pounds of air
rhoa = 3.6;     %density of air coming out of the nozzle = 3 atm
Vn1 = 30;       %velocity out of nozzle with ash in it
Vn2 = 320;      %velocity of just air
Vdot2 = .944; %volumetric flowrate = 2000 scfm
mua = 1.85e-5;          %viscosity of air
x = 100*1e-6; %diameter of a particle
rhos = 2300;    %density of particle

An1 = 5*.025^2*3.1415/4;    %area of 5 nozzles
Vdot1 = An1*Vn1;            %volumetric flowrate
Mdota1 = rhoa*Vdot1;        %mass flow rate of air out of nozzle 1
Mdots1 = Mdota1*load;       %mass flow rate of solids out of nozzle 1
Mdot1 = Mdota1 + Mdots1;  %mass flow rate out of nozzle 1
An2 = Vdot2/Vn2;    %area of nozzle in carrying only air
Mdot2 = rhoa*Vdot2;         %mass flow rate out of nozzle 2
Mair = Mdot2 + Mdota1;               %mass flow rate of air total
Msol = Mdots1;                       %mass flow rate of solids total
Vsol = Vn1;
%Basis of one second
Kesolinit = .5*Msol*Vsol^2;          %KE of solids initially
Keairinit = .5*Mdot2*Vn2^2 + .5*Mdota1*Vn1^2;
Keinit = Kesolinit + Keairinit;
Vt = (Keinit/.5/(Mair + Msol))^.5;   %initial velocity
Kesolaftermix = .5*Msol*Vt^2;                 %KE of solids after two stream mix
d = 0;
Vcarry = Vt*(An1 + An2)/(2*6);                %air velocity in tunnel
V = Vt - Vcarry;
Mp = rhos*(3.1415*x.^3/2);           %mass of a particle
numparticles = Msol./Mp;
Ke = Kesolaftermix./numparticles;    %KE of one particle
increment = .1;
t = 0;
velterm                                       %calls velterm program

%This next section iterates through a process until the particle
```

```
%reaches the ground and finds how far the particle has traveled.

while vterm*t < 2,
  V = Vcarry + (Ke/.5/Mp)^.5;
  Rep = x*V*rhoa/mua;
  if Rep > .2
    Cd = 18.5./Rep.^.6;
  else
    Cd = 24/Rep;
  end;
  Fdrag = Cd.*(3.1415.*x.^2/3)*rhoa*V^2/2;
  Ke = Ke - Fdrag*increment;
  d = d + increment
  t = d/V;
end;
d
```

```
%velterm.m
%This program calculates the terminal velocity of a particle.
%It is called by superson.m

Cdratio = 1;
c =.9*Vcarry;                  %solids velocity
D = 2.76;                      %pipe diameter
lambdaz = 1000;                %very rough guesstimate
g = 9.81;

v = x.^2*(rhos - 1)*9.81/18/mua;
Re = 1*x.*v/mua;
if Re > 2
  v = .153*9.81^.71*x.^1.14*(rhos - 1)^.71/1^.29/mua^.43;
end;


vterm = ((Cdratio)*((Vcarry - c)/v)^2*2*D*g/(lambdaz*c*Vcarry))^.5;
```

Determination of internal friction factor for fly ash





Determination of static friction between fly ash and acrylic

APPENDIX E

Letter Progress Report

Eric Powell & Associates, Inc

ERIC POWELL & ASSOCIATES, INC.

By Fax; 618-453-7455

Post Office Box 20406,
Knoxville, TN 37940-0406
Phone; 423-577-0073

October 20, 1995

Mr. Edwin M. Thomasson,
Department of Mining Engineering,
Southern Illinois University at Carbondale,
Carbondale, Illinois 62901-6603

Reference; Progress Report, Dry Flue Gas Desulfurization By-Products Backfill.

Dear Ed;

Progress on the pneumatic backfill aspects of the project has consisted of the following:

1. Visit to the University of Pittsburgh to review and observe the fly ash plug breaking demonstration being supervised by Dr. George Klinzing. Suggested to Gregory Jama and Bill Link (equipment operators) that further tests should be undertaken with the fly ash plug kept under a pressure of about 50 psig prior to the higher pressure air blast being applied. This could not be done during the time of my visit as additional lengths of pipe were required and were not immediately available.

2. During a visit to the University of Kentucky on March 23, 1995, for a discussion on the sealing of highwall addits with Dr. Jack Groppo, using dry FGD, I received the latest articles on pneumatic stowing published in the German mining magazine, Gluckauf. These have been reviewed for any new practices in backfilling underground mines (suggested by P.C.). However all the information published relates to active underground mines.

3. Work is proceeding on the design of the ASME coded pressure vessels including the feed discharge control. The drawings and calculations will be available at the appropriate time for bids from ASME coded shops. Work is in hand to locate positive displacement blowers and air compressors that may be available to rent for the short period the backfill tests are to run.

Yours sincerely,

Eric

Eric Powell

Appendix F

Equipment Specifications
Paste Injection of FGD Wastes

October 20, 1995

EQUIPMENT SPECIFICATIONS

PASTE INJECTION OF FGD WASTES

1. Scrubber Sludge Stockpile

A stockpile area next to the injection plant is needed to store up to 825 tons of filter cake hauled from Springfield. It is assumed that the haul trucks will be end dump. A concrete pad is most desirable for the stockpile, but if not avialable, the ground surface should be prepared to remove all trash and rocks larger than 1/2 inch diameter. Trucks would dump at the base of the stockpile and the front end loader would be used to heap the stockpile if necessary. The size of the stockpile is about 600 cubic yards. The foot print of the stockpile will be about 70 feet by 70 feet.

2. Front End Loader

A rubber-tired front end loader of the Cat 1 yard class is required to handle filter cake, both to work the stockpile and to feed the filter cake to the hopper. This machine should be procured on a rental basis.

3. Belt Feeder with Hopper

This hopper with conveyor and drive should be procured on a rental basis from the sand and gravel or construction industry in the local area. Ideally, the belt width should be 24 inches and the hopper should be 3-5 cubic yards capacity. Skirting would be used to control the loading of scrubber sludge on the belt. The belt speed may have to be adjusted to achieve the correct feed rate of 41 tons per hour. Belt speed should be 50 to 100 feet per minute. The discharge of the belt would be directly into the agitated tank. Often a screen is part of these units, but a screen is not needed. The discharge height should be 15 feet. The drive should be electric 480 volt, 3 phase. Horsepower is 3-5 hp.

4. Scrubber Sludge Mix Tank

A steel tank approximately 12 feet diameter by 12 feet high will be used for preparing and storing slurry. Possibly, a disused fuel tank can be procured for this purpose. Steel wall thickness should be 3/16 inches. A 3-inch flanged outlet should be installed about 1 foot off the bottom for connection to the circulation pump. The tank would be open on top. A support bridge will be placed on the top of the tank for support of the agitator.

5. Scrubber Sludge Agitator

A CIP type agitator should be procured for mixing scrubber sludge slurry. Tank size and slurry specific gravity should be specified to the manufacturer. Cost will be about $6,000. Slurry specific gravity is 1.53. Manufacturers are:

John Scheuer
APPCOR
2045 West Ninth Ave.
Denver, CO 80204
Tel: 303 534-1433
Fax: 303 534-1456

Mike Kelzer
Warman Intl, Inc.
P.O. Box 7610
Madison, WI
Tel: 608 221-2261
Fax: 608 221-5810

The manufacturer can provide details of the bridge support structure mounted on top of the tank.

6. Centrifugal Pump

A rubber-lined centrifugal pump would be adequate for circulating the scrubber sludge. The circulation rate is 100 USgpm in a 2-inch line. The head on the pump would be about 50 feet. It is preferred that the pump not require gland water for sealing. A Warman 2/1.5 BAH rubber lined pump fits the application. See Warman address above. Another suitable manufacturer is:

Galigher or BGA Intl., also Envirotech or A-S-H
440 West 8th South
P.O. Box 209
Salt Lake City, UT 84110
Tel: (801) 359-8731
Fax: (801) 530-7533

7. Slurry Pipeline

The velocity of the slurry in the circulation loop should be high to avoid pipe scaling. With a 2-inch pipeline, the velocity will be 10 feet/sec at a flow rate of 100 USgpm. Therefore, 2-inch steel schedule 40 pipe has been chosen. This pipe with victaulic style 77 couplings should be purchased from a local supplier during plant setup when all dimensions are known. Long radius elbows should be used.

8. Valves for Scrubber Slurry

An inexpensive 3-inch shutoff valve should be installed between the tank and the centrifugal pump. Of course, the pump should be installed as close to the tank flange as possible.

A remotely controlled 2-inch valve is needed for controlling the flow of slurry into the mixer. Contact:

Red Valve Co.
P. O. Box 548
Carnegie, PA 15106
Tel: (412) 279-0044
Fax: (412) 279-7878

9. Fly Ash Screw Feeder

Contact a Springfield cement company for sources of rental equipment. The fly ash feed rate is 20 tons/hr. Because the mix won't be critical [we are disposing waste], the screw feeder can be calibrated by speed adjustment with a sheave or other such simple method. Two feeders are needed, one for each silo.

10. Fly Ash Silo

Contact a Springfield cement company for sources of rental equipment. The silos should be tip-up portables. The silos should have bag houses for dust control when trucks are loading the silos pneumatically.

11. Paste Mixer

The mixer should be a twin shaft paddle mixer, radial arm pan mixer or equivalent positive action type mixer. A rotating drum mixer is not suitable. Check the concrete industry in the Chicago area for a used mixer. The mixer should be about 1 cubic yard capacity. Rapid makes a suitable pan mixer, but a used one or a rental unit may be hard to find:

Shaft Machines - Dynatec
Paul Sulman
Toronto Area
(905) 886 6957
(905) 888 1388
(905) 886 8891 Fax

After some potential mixers are discovered, please contact MSD to discuss.

12. Paste Pump

The paste pump selected is a Schwing trailer mounted concrete pump with "Rock" valve, model BPA 750 RLD. It has a maximum output of 39 yd^3/hr. Call the Schwing plant at White Bear, Minnesota for the location of the nearest dealer for rental of the unit.

Roger Phares, Regional Manager
Schwing America
5900 Centerville Road
White Bear, MN.55110
(612) 429-0999

13. Instrumentation

Milltronics manufacture's economical level indicators [The Probe] to be used for the slurry tank and pump hopper:

Milltronics
709 Stadium Drive
Arlington, TX 76011
(817) 277-3543
(817) 277-3894 Fax

Milltronics may also make a density meter to be used on the circulating loop. Another manufacturer is:

Kay-Ray/Sensall, Inc.
Mt. Prostect, IL
(708) 803-5100

The power meter for the mixer drive can be procured from:

Load Controls, Inc.
Technology Park
10 Picker Rd
Sturbridge, MA 01566
(508) 347-2606

Pressure transducers are difficult for paste. Please contact MSD for details.

A PLC is needed for controlling the process. The instrument department at SIU can probably best give advice on which equipment should be selected.

A control shed to house the PLC and monitors should be procured.

APPENDIX G

Packer Test and Data Analysis Procedures

DRAFT

Date of Last Revision: July, 1995

Packer Tests: Critical Literature Review

Miguel Restrepo and Edward Mehnert
Illinois State Geological Survey

1. INTRODUCTION

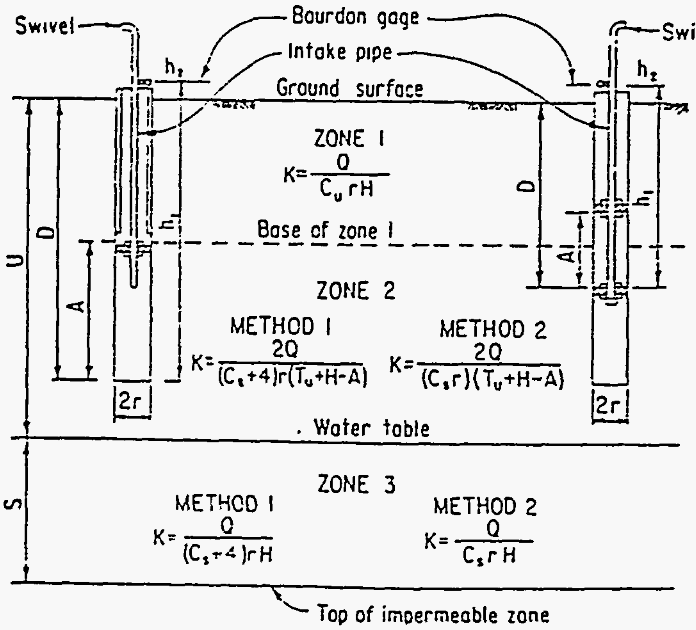
1.1 Objectives

Packer tests are used to estimate the hydraulic conductivity of a geologic formation. The purpose of this work is to provide and present a set of procedures to obtain and mathematical computations used to analyze data from a packer test. Examples dealing with single and double packer tests are analyzed using the equations and procedures presented in the report. Finally, some conclusions and observations about packer testing and analysis are presented.

1.2 Conventions and variables used in this report

The conventions and variables used in this report are consistent with those used in the Ground Water Manual [USBR, 1985]. The following variables were used (most are shown on figure 1):

A = length of test section [L].
a = surface area of test section [L^2].
C_s = conductivity coefficient for semispherical flow in saturated materials through partially penetrating cylindrical test wells [dimensionless].
C_u = conductivity coefficient for unsaturated material with partially penetrating cylindrical test wells [dimensionless].
D = distance from ground surface to bottom of test section [L].
H = $h_1 + h_2 - L$ = effective head [L].
h_1 (above water table) = distance between Bourdon gage and bottom of hole for single packer or distance between gage and upper surface of lower packer for double packer [L].
h_1 (below water table) = distance between gage and water table [L].
h_2 = applied pressure at gage [L].
K = coefficient of permeability [LT^{-1}].
L = head loss in pipe due to friction [L]; ignore head loss for Q<4 gallons per second in a 1.25" pipe; use length of pipe between gage and top of test section for computations.
Q = steady flow into the well [L^3T^{-1}].
r = radius of test hole [L].
S = thickness of saturated material [L].
U = thickness of unsaturated material [L].
T_u = U - D + H = distance from water surface in well to water table [L].
X = 100 H / T_u = percent of unsaturated stratum [dimensionless].



K = coefficient of permeability, feet per second under a unit gradient
Q = steady flow into well, ft^3/s
$H = h_1 + h_2 - L$ = effective head, ft
✓ h_1 (above water table) = distance between Bourdon gage and bottom of hole for method 1 or distance between gage and upper surface of lower packer for method 2, ft
✓ h_1 (below water table) = distance between gage and water table, ft
h_2 = applied pressure at gage, 1 lb/in^2 = 2.307 ft of water
L = head loss in pipe due to friction, ft; ignore head loss for $Q<4$ gal/min in $1\frac{1}{4}$-inch pipe; use length of pipe between gage and top of test section for computations
$X = \frac{H}{T_u}(100)$ = percent of unsaturated stratum
A = length of test section, ft
r = radius of test hole, ft
C_u = conductivity coefficient for unsaturated materials with partially penetrating cylindrical test wells
C_s = conductivity coefficient for semi-spherical flow in saturated materials through partially penetrating cylindrical test wells
U = thickness of unsaturated material, ft
S = thickness of saturated material, ft
$T_u = U - D + H$ = distance from water surface in well to water table, ft
D = distance from ground surface to bottom of test section, ft
✓ a = surface area of test section, ft^2; area of wall plus area of bottom for method 1; area of wall for method 2
Limitations:
✓ $Q/a \leq 0.10$, $S \geq 5A$, $A \geq 10r$, thickness of each packer must be $\geq 10r$ in method 2

Figure 1. System configuration and variables used (from USBR, [1985])

2. DESCRIPTION OF PACKER TESTS

2.1 General background on packer tests

Packer tests are pressure tests in which a one or two packers are used to isolate a part of a borehole into rock. The borehole does not need to be vertical, it can be angled or even horizontal (Ground Water Manual [USBR, 1985]). Water is injected at a known pressure and, at steady state conditions, the water discharge is measured, and with that information the value of the conductivity can be estimated. According to Braester and Thunvik [1985], packer tests are used for in-situ determination of the hydraulic conductivity of a geological formation, especially in formations with low values of hydraulic conductivity such as hard rock. These methods were originally developed for the exploration of prospective sites for conventional structures like dams and tunnels; it has been used recently mainly for the investigation of sites for radioactive waste repositories and underground hydrocarbon storage reservoirs

2.2 Single packer tests

Single packer tests are used when determining the hydraulic conductivity of a consolidated rock formation that would require the installation of casing during drilling. However, the hole should be neither cased nor cemented at the moment the packer test, otherwise the results would be wrong. In general the procedure to follow is:

- The hole is drilled to the maximum depth so that there is no need to immediately case it.

- The drilling tools are then removed.

- The hole should be washed and bailed or blown out thoroughly in order to minimize "borehole skin effects". A detailed discussion on these effects is presented in section 2.5 of this report.

- The packer is seated above the bottom of the drilled hole so that the test section is included completely in the "just-drilled" portion of the hole. Care must be taken with the packer so that "equipment compliance effects" are minimized as much as possible. These effects are presented in detail in section 2.4 of this report.

- Water is then pumped until steady state is reached (i.e., three or more consecutive readings of pressure and discharge taken at five minute intervals are approximately constant). It is important to get steady state conditions since compliance effects are minimized this way. Pickens et al. [1987] noted that the most important compliance effect is observed at shut in.

- Pressure and discharge readings are recorded at 5 minute intervals.

- The packer is removed. The test can be repeated in other formations as the hole is drilled.

2.3 Double packer tests

Double packer tests are used when determining the hydraulic conductivity of a consolidated rock formation that is stable and does not require either casing or cementing during drilling. In general, the procedure to follow is:

- The hole is drilled to the desired depth.

- The drilling tools are then removed.

- The hole should be cleaned and bailed or blown out thoroughly in order to minimize "borehole skin effects". For a detailed discussion of these effects, see section 2.5 of this report.

- Two packers are installed at an arbitrary distance along the water injection pipe or rod to isolate the test section. Since double packer tests are made sequentially until the length of the hole is tested. the packers must be placed as close as possible to the bottom of the hole at the beginning. They should be lifted a distance equal to the length of the test section (A). Care must be taken with the packer so that "equipment compliance effects" are minimized as much as possible (see section 2.5).

- Water is pumped until steady state is reached (i.e., changes in consecutive readings of pressure and discharge are approximately constant). It is important to get steady state conditions since compliance effects are minimized this way.

- Pressure and discharge readings are recorded at 5 minute intervals.

- The test is repeated until the entire length of the hole is tested.

This test can be run using multiple pressures. The procedure consists of applying pressure in three or more approximately equal steps. For example, suppose that the maximum allowed differential pressure is 50 psi. then the test would be run at 10, 20, 30, 40, and 50 psi Care must be taken to maintain each pressure step for approximately 20 minutes, and readings should be made at 5-minute intervals. Doing so, the pressure can be raised to the next step until the maximum allowed differential pressure is attained: the process is then reversed. with the pressure being maintained for 5 minutes at approximately the same middle and the lowest pressure steps. In the example shown before, the process for the whole multiple pressure test would be: 10. 20, 30, 40. 50. 40, 30, 20, and 10 psi, approximately. A graph of intake against pressure for all the steps in a multiple pressure test is handy for assessing hydraulic conditions of the tested geological formation.

2.4 Pressure loss in test pipe

Drill rods are the most common intake pipes used in packer tests. NX and NW rods can be used with good results as long as the intake of the test section does not exceed 12 to 15 gallons per minute and the length of the test section is less than 50 feet. It is recommended that a pipe of 1.25 inches of diameter or larger be used in the test. In figures 10-1 through 10-4, plots of head losses per 10-foot section are shown for several water discharges. with different diameter of drill rod and a 1.25 inches diameter rod (the graphs were compiled from experimental data).

Another source of pressure losses comes from the pumping equipment. The most common type of pump for pumping the water is the mud pump. Usually such devices are of the multiple cylinder type with a uniform fluctuation in pressure. It is very difficult, if not impossible, to account for this effect in the calculation of the permeability values. In order to avoid this problem, the use of centrifugal pumps having sufficient capacity to develop back pressure is recommended.

2.5 Interference of packer tests

Pickens et al. [1987] analyzed the effect of borehole pressure history, thermally induced borehole pressure response. borehole and formation skin effects, and equipment compliance effects. Each of these effects will be presented in some detail.

Borehole pressure history was found to affect.significantly by orders of magnitude the value of the hydraulic conductivity. According to Pickens et al [1987], in order to minimize the effect of borehole pressure history, the starting pressure of the test should be substantially different from the formation pressure and the annulus pressure, which is the down hole absolute pressure corresponding to conditions with the borehole fluid level at the wellhead.

Thermally induced borehole pressure response, or the pressure changes resulting from thermal expansion of the borehole fluid, can have an enormous impact on the measurement of the borehole pressure response: a change of temperature in the order of 1 to 2 degrees Celsius, was found to cause a change in pressure in the order of 10 to 100 meters, but these are theoretical results (Pickens et al. [1987]). In real measurements, this effect is moderated by pressure dissipation in the aquifer. This effect is then more critical when measuring a low permeability formation because the pressure dissipation in the aquifer depends on the permeability. Thus, for a high permeability aquifer, the effect of thermally induced borehole pressure response is less critical than in a low permeability aquifer.

Borehole and formation skin effects are perhaps the most important to be considered when performing packer tests. Skin effects are basically the alteration of the permeability of the formation at the well face due, mainly, to the drilling process. In general, skin effects have a great impact (orders of magnitude) on the value of the permeability. This conclusion is common to the work of both Pickens et al. [1987], and Braester and Thunvik [1984]. According to the work of Pickens et al. [1987] (which seems to be more complete on this issue), the value of the permeability can increase due to fracturing and/or erosion, or can decrease due to mineral reactions, mud cake formation, granite flour accumulation, etc. The most common skin effect is drilling mud invasion. The extent of this effect (i.e., the penetration of the mud cake) depends on several factors, like properties of the drilling fluid, porosity of the formation, the ratio of the formation pore size to the size of particulate in the mud, permeability of the formation, permeability of the mud cake, detention time of the mud filling the hole, pressure differential between mud-filled borehole and the formation, etc. However, although the skin effect is known to greatly affect permeability measurements, it is very difficult to quantify the value in real conditions. The determination of the thickness, permeability of a skin, and even its presence in a testing well is nowadays very difficult if not impossible.

3. INTERPRETATION OF PACKER TEST MEASUREMENTS

3.1 Single packer tests

Single packer tests can be analyzed using the Groundwater Manual methodology. This type of packer test is not used often, therefore there are not many methods available in the literature.

3.1.1 Groundwater Manual method

This is the most widely used methodology for the analysis of single packer tests. The estimation of the hydraulic conductivity involves the direct evaluation of an equation involving a simple set of parameters.

3.1.1.1 Assumptions

For the Groundwater Manual method, the following assumptions are made for single packer tests:

- Homogeneous and isotropic formation.
- Confined or unconfined aquifer.
- Measurements and values in English units.

3.1.1.2 Data requirements

For the single packer test, the following data are required to obtain permeability K (remember that English units should be used):

- Radius r of the borehole
- Length of the test section A. For this case, it is the distance between the packer and the bottom of the borehole.
- Depth h_1 from pressure gage to the bottom of the borehole. When using a pressure sensor, use the pressure recorded in the test section prior to pumping as the h_1 value.
- Applied pressure h_2 at the gage, or, in the case of a sensor being used, use the pressure recorded during pumping.
- Steady flow Q into the well. Flow is considered steady when at least three readings at 5-minute intervals are essentially equal.
- Nominal diameter and length of intake pipe between the gage and the upper packer.
- Thickness U of the unsaturated material, or material above the water table.
- Thickness S of the saturated material above a relatively impermeable bed.
- Distance D from the bottom of the test section and the ground surface.

- Starting test time and time for each of the measurements.

- For streambeds and lakebeds below water, the effective head H is computed as the difference between the elevation of the free water surface in the pipe and the elevation of the gage plus h_2.

- In the case of a pressure sensor, the effective head should be computed as the difference in pressure before water is pumped into the test section and the pressure readings made during the test.

- The surface area of the test section, a, is the area of the sidewalls of the wet borehole plus the area of the bottom.

- The method is applicable as long as: the value of Q/a is less than or equal to 0.10 (English units) and S/A greater than or equal to 5; also, A/r should be greater than or equal to 10, all measured in English units.

3.1.1.3 Computational Procedure

- Get the head loss coefficient (figure 10-1, 10-2, 10-3 or 10-4 of the Groundwater Manual [USBR, 1985]) = K_l

- Compute $L = K_l$ x (distance from bourdon gage to bottom of pipe)

- Compute $H = h_1 + h_2 - L$

- Get $T_u = U - D + H$

- If $D > U$ then the permeability can be computed using zone 3 formulae. Otherwise, proceed with the following calculations:

- Compute $X = H / T_u$

- Compute theoretical X' from:

$w = T_u / A$

$X' = 100 - 600 \{1/(wX') [2-100/(wX')] \}^{0.5}$

Note: this is a recursive equation that requires the use of a numerical technique such as the fixed point algorithm.

- If $X > X'$ then the test is in zone 1 otherwise is in zone 2.

- Once it has been determined in which zone is the test being analyzed, the appropriate formulae should be used:

Zone 1: $K = Q / (rHC_u)$

Zone 2: $K = Q / [(C_s+4)r(T_u+H-A)]$

Zone 3: $K = Q / [(C_s+4)rH]$

Where: $C_s = [A/r]2\pi/\underline{\ln}[A/r]$

$C_u = rH/[2\pi(2AH-A^2)][\sinh^{-1}(A/r)-(A/r)]$

3.2 Double packer tests

The analysis of double packer tests can be made using the Groundwater Manual method. However, since double packer tests are more widely used than single packer tests, alternative methods of analysis are available. In particular, Hvorslev method, Moye method and Dagan method are presented in this report. The method by Dagan [1978] has been mentioned in the literature as the method with a sound theoretical background, and the one that predicts a more accurate value for the hydraulic conductivity (Braester and Thunvik [1984]).

3.2.1 Groundwater Manual method

Double packer test interpretation was developed using the same theoretical assumptions of the single packer test case. The formulae employed for the double packer test data interpretation are very similar to the formulae employed for single packer tests data interpretation.

3.2.1.1 Assumptions

The Groundwater Manual method assumes the following characteristics for the double packer tests:

- Homogeneous and isotropic formation.
- Confined or unconfined aquifers.
- Measurements in English units.

3.2.1.2 Data requirements

For the double packer test, the following data are required in order to compute the hydraulic conductivity K (remember to use English units):

- Radius r of the borehole
- Length of the test section A. In this case, it is the distance between the packers.
- Depth h_1 from pressure gage to the bottom of the borehole. When using a pressure sensor, use the pressure recorded in test section prior to pumping as the h_1 value.
- Applied pressure h_2 at the gage, or, in the case of a sensor being used, use the pressure recorded during pumping.
- Steady flow Q into the well. Flow is considered steady when at least three readings at 5-minute intervals are essentially equal.

- Nominal diameter in inches and length of intake pipe in feet between the gage and the upper packer.

- Thickness U of the unsaturated material above water table.

- Thickness S of the saturated material above a relatively impermeable bed.

- Distance D from the bottom of the test section and the ground surface.

- Starting test time and time for each of the measurements.

- For streambeds and lakebeds below water, the effective head H is computed as the difference between the elevation of the free water surface in the pipe and the elevation of the gage plus h_2.

- In the case of a pressure sensor, the effective head should be computed as the difference in pressure before water is pumped into the test section and the pressure readings made during the test.

- The surface area of the test section, a, is the area of the borehole wall only.

- In order to apply the method, the value of Q/a must be less than or equal to 0.10 (English units) and S/A must be greater than or equal to 5. Also,the value of A/r must be greater than or equal to 10, all measured in English units. In addition, the length of each packer in this case should be greater than or equal to 10r.

3.2.1.3 Computational procedure

- Get the head loss coefficient, K_l, from Figures 10-1, 10-2, 10-3, and 10-4 from the Groundwater Manual [USBR, 1985].

- Compute $L = K_l$ x (Distance from Bourdon Gage to bottom of pipe)

- Compute $H = h_1 + h_2 - L$

- If $D > U$, then the test is in zone 3 and so the conductivity K can be computed from the formula presented below, without going through the computation of T_u, X and X'.

- Get $T_u = U - D + H$

- Compute $X = H / T_u$

- Compute theoretical X' from:

$$w = 1/(T_u/A)$$

$$X' = 100 - 600 \{1/(wX') [2-100/(wX')] \}^{1/2}$$

Note: it is a recursive equation that requires the use of a numerical technique such as the fixed point algorithm.

- If X > X' then the test is in zone 1 otherwise is in zone 2.

- Once it has been determined in which zone is the test being analyzed, the appropriate formulae should be used:

Zone 1: $K = 2Q / [(rC_s) (T_u+H-A)]$

Zone 2 and Zone 3: $K = Q / [C_s rH]$

Where: $C_s = [A/r]2\pi/\underline{\ln}[A/r]$

3.2.2 Dagan method

Braester and Thunvik [1984] showed that the method presented by Dagan [1978] estimated hydraulic conductivity more accurately than the methods proposed by Hvorslev [1951] and Moye [1967]. The method is not as straightforward as the others presented in the Ground Water Manual, and requires the solution of a system of linear equations with a dense matrix of dimension 20 (Dagan [1978]). However, it can be solved easily using a spreadsheet.

3.2.2.1 Assumptions

The following assumptions are incorporated into the Dagan [1978] method:

- Unconfined aquifers.

- Isotropic formations.

- Water is injected into a double packer system set-up.

- Measurements are taken at steady state conditions.

3.2.2.2 Data requirements

The method proposed by Dagan [1978] needs the following input data:

- The well radius, r.

- The length of the test section, A.

Note: This method is accurate if and only if A > 50r. If this condition is not satisfied, the results may not be accurate (Dagan, 1978).

- The water discharge, Q.

- The effective head, H.

- Length of the test section (distance between packers), A.

- Water table depth, U.

- Distance from ground surface to lower packer, D.

- Note that in this method the units are not restricted, so either English or SI units can be used in a consistent way.

3.2.2.3 Computational procedure

The model proposed by Dagan (Dagan [1978] and Braester and Thunvik [1984]) consists of distributing the water discharge Q induced into the well into a finite number of discharges q_i so that:

$$[M]^T\{u\} = \{\Delta P\}$$

$$Q = \Delta L \Sigma q_i \quad i=1,\ldots,N$$

$$u_i = q_i / K$$

$$\Delta P = H + I$$

Where: $I = 0$ if $U < D$
$I = U - D + A/2$ otherwise.

So the method basically requires the assembling of a system of simultaneous equations to get the values for u_i, solve the system, and then compute K from:

$$K = Q / [\Delta L \Sigma u_i]$$

$$\Delta L' = A/N$$

The definition of the elements of the matrix M is as follows:

$$M_{ij} = 1/4\pi \, \underline{\ln} \, \{[(|a_{ij}| + a_{ij} + r^2/2|a_{ij}|)/(|a_{ij}-\Delta L| + a_{ij} - \Delta L + r^2/2|a_{ij}-\Delta L|)]$$
$$[(|c_{ij}-\Delta L| + c_{ij} - \Delta L + r^2/2|c_{ij}-\Delta L|)/(|c_{ij}| + c_{ij} + r^2/2|c_{ij}|)]\}$$

where:

$$a_{ij} = (j - i + 0.5)\Delta L$$

$$c_{ij} = (j + i - 0.5)\Delta L - 2D$$

$N = 20$ always (this is to avoid numerical instability problems, according to the recommendations of Dagan [1978])

3.2.3 Other methods: Hvorslev and Moye equations

Hvorslev [1951] and Moye [1967] methods are presented in good detail by Braester and Thunvik [1984]. The two methods are very straightforward, and consist mainly of a simple equation, including a natural logarithm.

According to Braester and Thunvik [1984], Hvorslev and Moye methods tend to overestimate hydraulic conductivity.

3.2.3.1 Assumptions

The methods proposed by Hvorslev [1951] and Moye [1967] are studied in the work by Braester and Thunvik [1984]. They consist of a simple equation that computes the hydraulic permeability, K, based on the discharge, Q, the induced pressure, ΔP, and the geometric characteristics of the aquifer. Both methods assume that:

- The tested region is located at such depth that the water table is not affected by the injection of small volumes of water for a relatively short period of time.

- According to the previous assumption, the phreatic level is assumed to be a fixed boundary.

- The aquifer is unconfined, homogeneous and isotropic.

3.2.3.2 Data requirements

The data required for Hvorslev and Moye methods are:

- The radius of the borehole, r.

- Water discharge, Q.

- Effective head, H

- Length of the test section, A.

- Any consistent units are valid.

3.2.3.3 Computational procedure

The computational procedure for Hvorslev and Moye methods is very simple; basically, given the information mentioned in the previous section, the hydraulic conductivity K is computed with the following equations:

- Hvorslev method:

$$K = Q \underline{\ln} [A/2r + \{ 1 + (A/2r)\}]/(2\pi AH)$$

- Moye method:

$$K = Q\{ 1 + \underline{\ln}(A/2r)\}/(2\pi AH)$$

3.3 Application examples

Two application examples are presented, one relating a single packer test, and the other for a double packer test. The example of the single packer test is the same as one of the examples in the Ground Water Manual [USBR, 1985]. The double packer test example is one of the cases from the Harza Engineering report [1984]

3.3.1 Single packer

The example that will be presented in the following subsections was taken from the Ground Water Manual [USBR, 1985].

3.3.1.1 Data Provided

For this example, assume the following data was given:

$U = 75$ ft, $D = 25$ ft, $A = 10$ ft, $h_1 = 32$ ft, $h_2 = 57.8$ ft,
$Q = 0.045$ ft^3/s (cfs), $L = 1.7$ ft, $r = 0.5$ ft.

3.3.1.2 Solutions

- Applying the Groundwater Manual Method, we get:

$$K = 1.5x10^{-5} \text{ ft/s}$$

Now, as a way to compare the result obtained for the only methodology available specific for single packer tests, we obtained the conductivity using the methods for double packer tests:

- Moye method gives: $K = 2.7x10^{-5}$ ft/s

- Hvorslev method gives: $K = 2.5x10^{-5}$ ft/s

3.3.1.3 Comments on the Results

Looking at the results we can see that the highest estimate for the conductivity is the Ground Water Manual value. Hvorslev and Moye formulae give a similar result, which is lower than the Ground Water Manual value. Single packer tests are not very common in practice; double packer tests, on the other hand, are of much frequent occurrence.

3.3.2 Double packer

The example that will be presented in the following subsections was taken from the Harza Engineering report [1987]

3.3.2.1 Data Provided

For this example, assume the following data was given:

$U = 95.2$ ft, $D = 490.0$ ft, $A = 21.0$ ft, $h_1 = 35.0$ ft, $h_2 = 42.0$ ft,
$Q = 0.000446$ ft^3/s (cfs), $L = 0.0$ ft, $r = 0.125$ ft.

3.3.2.2 Solutions

- Applying the Groundwater Manual Method, we get:

$K = 2.96x10^{-6}$ ft/s

- Moye method gives: $K = 3.14x10^{-6}$ ft/s

- Hvorslev method gives: $K = 2.97x10^{-6}$ ft/s

The application of Dagan method is quite complicated and was thus performed using a spreadsheet. The spreadsheet was configured so that with two simple macros it is possible to compute the solution fast and efficiently, changing just a few values in the cells of the spreadsheet file. The result obtained in this case was:

- Dagan method gives: $K = 2.76x10^{-6}$ ft/s

3.3.1.3 Comments on the results

Looking at the results we can see that the lowest estimate for the conductivity is the Ground Water Manual value. Hvorslev and Moye formulae give a similar result, which is higher than the Ground Water Manual value. The value for Dagan method is the lowest of all the others, but very close to the values provided by Hvorslev and the Ground Water Manual. This result is not surprising, however. The fact that Hvorslev, Moye and the Ground Water Manual methods tend to overestimate the value for the hydraulic conductivity has been noted by Braester and Thunvik (1984). The requirement of $A/r > 50$ is satisfied. Therefore, the results are accurate enough using Dagan method.

Appendix H

Data Acquisition Software

Program Unit 1

Vibrating Wire Transducers for the atmospheric pressure and three monitoring wells;

Allocate 17 for input memory

Labels:
- 2: vibrating wire transducer (vwt) readings
  - 1: atmosphere
  - 2: Well 2
  - 3: Well 1
  - 4: Well 3

Short term measurement from atmospheric pressure vibrating wire transducer.

```
*1
120
Take vwt measurement
Set counter for loop at 0
P30
        0
        0
        25  (counter 1 is stored in location 25)
Loop to take four measurements from the vwt
P87
        00
        4
Increment counter
P32
        25
Take temperature measurement from atmospheric vwt
P4
        1
        15
        1 (in channel 1H)        HOOK-UP
        1 (excitation channel 1E)        HOOK-UP
        1
        2500
        1 (memory location 1)
        0.001
        0
Take pressure measurements from atmospheric vwt
P28
        1
        2 (in channel 1L)        HOOK-UP
        1 (excitation channel 1E)        HOOK-UP
        26
        41
        500
        500
        5 (memory location 5)
        0.5 (actual multiplier should be 1, double values in memory, done for resolution purposes)
        0
```

Average temperature and vwt measurements

```
P89
        25
        1
        4
        10
P80
        3
        1 (input location for results of the averaging)
P71
        1 (average 1 measurement and put it in locations 1)
        1
P80
        3
        5 (input location for results of the averaging)
P71
        1 (average 1 measurements and put it in location 5)
        5
P95 End loop
P86 Set output flag low
        20
```

At this point memory locations 1 and 5 contain the average temperature and pressure for vwt 1, respectively.

Check to see if pressure value changes within 0.002 with a loop
Subtract old and new value. Old value is in location 9.

```
P35
        5
        9
        13 (subtract and put the result in 13)
```

Take absolute value of the result which is in location 13

```
P43
        13
        13
```

Check Tolerance

```
P89
        13
        3
        .002
        30
P86 Set output flag high
        10
```

Output this value along with a label, the temperature, and the time.
Set label

```
P80
        1
        2 (vwt measurement array label)
P30
        1
        0
        26 (location 26 has the value 1)
P70
        1
        2 (label for vwt)
```

```
P70
        1
        1
P70
        1
        5
P77
        110
P31 Move new value to old value location
        5
        9
P95 End if
Set the output flag low
P86
        20
End of the program for atmospheric vwt

Long term measurement from vibrating wire transducers.
Locations 1, 5, 9, 13 are used
*2
3600
Take vwt measurements
Set counter for loop at 0
P30
        0
        0
        27  (counter 3 is stored in location 27)
Loop to take four measurements from each vwt
P87
        0
        4
Increment counter
P32
        27
Take Temperature Measurement from 3 well vwts
P4
        3
        15
        3 (in channel 2H, 2L, 3H)       HOOK-UP
        1 (excitation channel 1E)       HOOK-UP
        1
        2500
        5 (memory locations 2 through 4)
        0.001
        0
```

```
Take pressure measurements from atmospheric and well vwts
P28
        3
        6 (in channel 3L, 4H, 4L)       HOOK-UP
        1 (excitation channel 1E)       HOOK-UP
        26
        41
        500
        500
        6 (memory locations 6 through 8)
        0.5 (actual multiplier should be 1, double values in memory, done for resolution purposes)
        0
Average temperature and vwt measurements
P89
        27
        1
        4
        10
P80
        3
        2 (input location for results of the averaging)
P71
        2 (average temperature measurements and put them in locations 2, 3, 4)
        2
P80
        3
        6 (input location for results of the averaging)
P71
        2 (average pressure measurements and put them in locations 6, 7, 8)
        6
P95 End loop
P86 Set output flag low
        20
```

At this point memory locations 2, 3, 4 contain the average temperature for vwt 2 through 4 and memory locations 6, 7, 8 contain the average vwt measurements for vwt 2 through 4.

```
Check to see if pressure values change within 0.002 with a loop for vwt 2 through 4
Set counter to 1
P30
        1
        0
        28
Start loop
P87
        0
        3
Increment counter
P32
        28 (counter now has the value 2)
Subtract old and new values.  Old values are in locations 10,11,12.
P35
        6--
        10--
        14--(subtract locations and put the result in 14,15,16)
```

```
Take absolute value of the results which are in locations 14, 15, 16
P43
        14--
        14--
Check Tolerance
P89
        14--
        3
        .002
        30
P86 Set output flag high
        10
Output this value along with a label, the temperature, and the time.
P80
        1
        2 (vwt measurement array label)
P70
        1
        28 (label for vwt)
P70
        1
        2--
P70
        1
        6--
P77
        110
P31 Move new value to old value location
        6--
        10--
P95 End if
Set the output flag low
P86
        20
P95 End loop
End of the Program for vwt
P10
        17 (battery voltage)
```

Program Unit 2

Vibrating Wire Transducers for four monitoring wells

Allocate 27 for input memory

Labels:
    2: vibrating wire transducer (vwt) readings (5-8)

Available memory: 29,902 spaces

```
Long term measurement (vwts in wells)
*1
3600 seconds
Set counter for loop at 0
P30
        0
        0
        25  (counter 1 is stored in location 25)
Loop to take four measurements from each vwt
P87
        0
        4
Increment counter
P32
        25
Take Temperature Measurement from vwts
P4
        4
        15
        1 (in channel 1H, 1L, 2H, 2L)        HOOK-UP
        1 (excitation channel 1E)            HOOK-UP
        1
        2500
        1 (memory locations 1,2,3,4)
        0.001
        0
Take pressure measurements from vwts
P28
        4
        5 (in channel 3H, 3L, 4H, 4L)        HOOK-UP
        1 (excitation channel 1E)            HOOK-UP
        26
        41
        500
        500
        5 (memory locations 5,6,7,8)
        0.5
        0
Average temperature and vwt measurements
P89
        25
        1
        4
```

```
        10
P80
        3
        1 (input location for results of the averaging)
P71
        8 (average 8 measurements and put them in locations 1-8)
        1
P86
        20
P95 End loop
```

At this point memory locations 1 through 4 contain the average temperature for vwt 5 through 8 and memory locations 5 through 8 contain the average vwt measurements for vwt 5 through 8. Compare the new values to the old values

Check to see if values change within 0.002 with a loop for all vwts
Set counter to 0

```
P30
        0
        0
        26 (location of the second counter)
P87
        0
        4
```

Increment counter

```
P32
        26
```

Subtract old and new values. Old values are in locations 9,10,11,12.

```
P35
        5--
        9--
        13--(subtract locations 5 and 9 and put the result in 13)
```

Take absolute value of the results which are in locations 13,14.15,16.

```
P43
        13--
        13--
```

Check Tolerance

```
P89
        13--
        3
        .002
        30
P86
        10
```

Output value along with a label, the temperature, and the time.

```
P80
        1
        2 (lysimeter vwt measurement array label)
P70
        1
        26
P70
        1
        1-- (temperature)
```

```
P70
        1
        5-- (vwt measurement)
P77
        110
P31 Move command
        5--
        9--
P95 End if
Set the output flag low
P86
        20
P95 (end loop)
End of the Program for vwt
Take battery voltage
P10
        27
```

Program Unit 2

Pulse Counting Program

Labels:
    2: vibrating wire transducer (vwt) readings (5-8)

Available memory: 29,902 spaces

```
*1
3600 seconds
Take pulse reading
P3
        1 (repetitions)
        1 (channel number)           HOOK-UP
        2 (switch closure, or 1 low level AC, or if we get many pulses, 4 for sixteen bit counter)
        1 (input location)
        1 (multiplier)
        0 (offset)
Output value
Set the output flag low
P86
        10 (set flag to high)
P80
        0
        511 (array label)
P70
        1
        1
P77 (we may not want the time saved)
        0111
Set the output flag low
P86
        20
Take battery voltage
P10
        2
```

```
Program Loot;

{This program takes the raw file dumped from the CR10 and splits the different
 instruments into separate files within a folder labeled by the retrieval
 date.  The program also converts day and hour data from the data logger
 into a decimal day.}


uses sane;

var
   results, time_equation, vwt1_equation, vwt_equation,
   subtraction, spacer, string_year: string;

   temporary: decstr;

   infile1, outfile1, outfile2, outfile3, outfile4, outfile5, outfile6,
   outfile7, outfile8, outfile9, outfile10, outfile11, outfile12:text;

   value, temperature, pressure, day, hour, level, correction, correction1,
   correction2, correction3, realid: extended;

   id, id2, n, i, year: longint;

   tab, digit: char;

   ying, integral, comma, bogus, wampus, legit: boolean;


procedure assign_vwt;

begin
    case n of
        1:   temperature:=value;
        2:   pressure:=value;
        3:   begin
            day:=value;

               case id2 of
               1, 2, 3, 4:  correction:=correction1;
               5, 6, 7, 8:  correction:=correction2;
               end;

            end;
        4:   begin
            hour:=trunc(value/100)+(value/100-trunc(value/100))/0.6;
            day:=day+hour/24+correction;
            end;
    end;
end;


procedure check_legit;
```

```
begin

comma:=false;
legit:=false;
if digit='0' then legit:=true;
if digit='1' then legit:=true;
if digit='2' then legit:=true;
if digit='3' then legit:=true;
if digit='4' then legit:=true;
if digit='5' then legit:=true;
if digit='6' then legit:=true;
if digit='7' then legit:=true;
if digit='8' then legit:=true;
if digit='9' then legit:=true;
if digit='.' then legit:=true;
if digit='.' then legit:=true;
if digit='.' then comma:=true;
if not legit then n:=6;

end:


procedure build_number;

begin

      bogus:=false;
      temporary:='';
      while not (bogus) do begin
        read(infile1, digit);
        check_legit;
        if legit then begin
          if comma or (seekeoln(infile1)) then bogus:=true;
          if not comma then temporary:=temporary+digit;
        end
        else begin
          temporary:='0';
          n:=6:
        end:
      end:

      value:=str2num(temporary);

end:


procedure read_vwt;

begin
while ((n<4) and not (seekeoln(infile1))) do begin
  build_number;
  n:=n+1:
  if n<6 then assign_vwt:
end;
```

```
 if not(seekeoln(infile1)) then readln(infile1);

 end;

 procedure write_vwt;

 begin

     case id2 of
         1: writeln(outfile1, temperature:6:3, tab, pressure:6:3,
         tab, day:6:3);
         2: writeln(outfile2, temperature:6:3, tab, pressure:6:3,
         tab, day:6:3);
         3: writeln(outfile3, temperature:6:3, tab, pressure:6:3,
         tab, day:6:3);
         4: writeln(outfile4, temperature:6:3, tab, pressure:6:3,
         tab, day:6:3);
         5: writeln(outfile5, temperature:6:3, tab, pressure:6:3,
         tab, day:6:3);
         6: writeln(outfile6, temperature:6:3, tab, pressure:6:3,
         tab, day:6:3);
         7: writeln(outfile7, temperature:6:3, tab, pressure:6:3,
         tab, day:6:3);
         8: writeln(outfile8, temperature:6:3, tab, pressure:6:3,
         tab, day:6:3);

     end;
end;

procedure check_id2;

var

temp:string;

begin
temp:='';
n:=0;
read (infile1, digit);
check_legit;
if legit then begin
  temp:=digit;
  read (infile1, digit);
  check_legit;
  if legit then begin
    if comma then begin
      realid:=str2num(temp);
      id2:=num2integer(realid);
      read_vwt;
      if n=4 then write_vwt;
    end
    else begin
      temp:=temp+digit;
```

```
      read (infile1. digit);
      check_legit:
      if comma then begin
        realid:=str2num(temp):
        id2:=num2integer(realid);
        read_vwt;
        if n=4 then write_vwt;
      end
      else begin
        readln(infile1);
      end:
    end:
  end
  else begin
    readln(infile1);
  end:
end
else begin
  readln(infile1);
end;

end;

procedure continue_processing;

begin

wampus := true:
if id = 2 then check_id2;
if (id=2) then wampus:= false;
if wampus then readln(infile1);

end:


procedure enter_data:

var
   filename1, filename2: string;

begin

writeln ('Beginning split process.');
writeln:
write('Enter the name of the file folder that will receive the output files:  ');
readln(filename1);
writeln:
write ('Input file name:  ');
readln (results);
reset (infile1, results);
writeln:

write ('Enter the time correction for Computer 1:  '):
readln(correction1);
```

```
write ('Enter the time correction for Computer 2:  ');
readln(correction2);

filename1:='main:doe:processed data:'+filename1+':';

rewrite (outfile1, filename1+'vwt1');
rewrite (outfile2, filename1+'vwt2');
rewrite (outfile3, filename1+'vwt3');
rewrite (outfile4, filename1+'vwt4');
rewrite (outfile5, filename1+'vwt5');
rewrite (outfile6, filename1+'vwt6');
rewrite (outfile7, filename1+'vwt7');
rewrite (outfile8, filename1+'vwt8');

end;





begin {main program}

tab:= char(9);

enter_data;

writeln;
writeln('Program is running, please wait.');
writeln;

legit:=false;
comma:=false;

while not (eof(infile1)) do
begin
    read (infile1, digit);
    check_legit;
    if legit then begin
      realid:=str2num(digit);
      id:=num2integer(realid);
    end
    else begin
      readln(infile1);
    end;

    if legit then read (infile1, digit);
    check_legit;
    if legit then begin
      if comma then continue_processing;
      if not comma then readln(infile1);
    end;
end;
```

```
close (infile1);
close (outfile1);
close (outfile2);
close (outfile3);
close (outfile4);
close (outfile5);
close (outfile6);
close (outfile7);
close (outfile8);

writeln;

writeln('Program is complete.');
writeln;
writeln ('Press return to leave the program.');

readln;


end.
```

```
Program Pillage;

{This program adjusts vwt data to atmospheric pressures.
It only reads files that have first been processed with the program entitled 'Loot'.
The program converts the data for the vibrating wire transducers into a
two column format that can be read by a graphics program.}


uses sane;

var
   results, filename1, filename2, vwt1_equation,
   vwt_equation, subtraction, time_equation: string;

   infile1, infile2, outfile1, outfile2:text;

   value, temperature, pressure, day, hour, level, difference,
   temperature1, pressure1, hour1, day1,
   temporary, water_level, water_level1,
   time, A1, B1, C1, yang: real;

   intday1, intday, i, code, total_vwt, ying, year: longint;

   tab: char;

   not_file_end, first_write, first_call, already_written: boolean;

procedure write_dummy_temp;

begin

A1:=temperature;
B1:=pressure;

case i of
   2:  water_level:=((-1.54*2*B1+23.48)-0.0063*((-104.78+378.11
       *A1-611.59*A1*A1+544.27*A1*A1*A1-240.91
       *A1*A1*A1*A1+43.089*A1*A1*A1*A1*A1)-19));
   3:  water_level:=((-0.91*2*B1+12.996)-0.0081*((-104.78+378.11
       *A1-611.59*A1*A1+544.27*A1*A1*A1-240.91
       *A1*A1*A1*A1+43.089*A1*A1*A1*A1*A1)-19));
   4:  water_level:=((-1.61*2*B1+27.1446)+0.0055*((-104.78+378.11
       *A1-611.59*A1*A1+544.27*A1*A1*A1-240.91
       *A1*A1*A1*A1+43.089*A1*A1*A1*A1*A1)-19));
   5:  water_level:=((-1.27*2*B1+19.477)-0.0124*((-104.78+378.11
       *A1-611.59*A1*A1+544.27*A1*A1*A1-240.91
       *A1*A1*A1*A1+43.089*A1*A1*A1*A1*A1)-19));
   6:  water_level:=((-1.33*2*B1+21.78)-0.0021*((-104.78+378.11*A1-611.59
       *A1*A1+544.27*A1*A1*A1-240.91*A1*A1*A1*A1
       +43.089*A1*A1*A1*A1*A1)-19));
   7:  water_level:=((-0.91*2*B1+11.208)-0.0071*((-104.78+378.11
       *A1-611.59*A1*A1+544.27*A1*A1*A1-
       240.91*A1*A1*A1*A1+43.089*A1*A1*A1
       *A1*A1)-19));
```

```
   8:  water_level:=((-1.32*2*B1+20.0772)-0.0037*((-104.78+378.11
       *A1-611.59*A1*A1+544.27*A1*A1*A1-
       240.91*A1*A1*A1*A1+43.089*A1*A1*A1*A1
       *A1)-20));
end;

water_level1:=temporary;

case i of
   2,3,4,5,6,7,8:  water_level:=(water_level-water_level1)*70.3077;
end;

time:=day;

writeln(outfile1, water_level:6:3, tab, time:6:3);

already_written:=true;

end;


procedure write_dummy;

begin

A1:=temperature;
B1:=pressure;

case i of
   2:  water_level:=((-1.54*2*B1+23.48)-0.0063*((-104.78+378.11
       *A1-611.59*A1*A1+544.27*A1*A1*A1-240.91
       *A1*A1*A1*A1+43.089*A1*A1*A1*A1*A1)-19));
   3:  water_level:=((-0.91*2*B1+12.996)-0.0081*((-104.78+378.11
       *A1-611.59*A1*A1+544.27*A1*A1*A1-240.91
       *A1*A1*A1*A1+43.089*A1*A1*A1*A1*A1)-19));
   4:  water_level:=((-1.61*2*B1+27.1446)+0.0055*((-104.78+378.11
       *A1-611.59*A1*A1+544.27*A1*A1*A1-240.91
       *A1*A1*A1*A1+43.089*A1*A1*A1*A1*A1)-19));
   5:  water_level:=((-1.27*2*B1+19.477)-0.0124*((-104.78+378.11
       *A1-611.59*A1*A1+544.27*A1*A1*A1-240.91
       *A1*A1*A1*A1+43.089*A1*A1*A1*A1*A1)-19));
   6:  water_level:=((-1.33*2*B1+21.78)-0.0021*((-104.78+378.11*A1-611.59
       *A1*A1+544.27*A1*A1*A1-240.91*A1*A1*A1*A1
       +43.089*A1*A1*A1*A1*A1)-19));
   7:  water_level:=((-0.91*2*B1+11.208)-0.0071*((-104.78+378.11
       *A1-611.59*A1*A1+544.27*A1*A1*A1-
       240.91*A1*A1*A1*A1+43.089*A1*A1*A1
       *A1*A1)-19));
   8:  water_level:=((-1.32*2*B1+20.0772)-0.0037*((-104.78+378.11
       *A1-611.59*A1*A1+544.27*A1*A1*A1-
       240.91*A1*A1*A1*A1+43.089*A1*A1*A1*A1
       *A1)-20));
end;
```

```
water_level1:=pressure1;

case i of
   2,3,4,5,6,7,8:  water_level:=(water_level-water_level1)*70.3077;
end;

time:=day;

writeln(outfile1, water_level:6:3, tab, time:6:3);

already_written:=true;

end;

procedure switch_files;

begin

reset (infile1, filename1+'dummy');

case i of
   1: rewrite (outfile1, filename1+'vwt1');
   2: rewrite (outfile1, filename1+'vwt2');
   3: rewrite (outfile1, filename1+'vwt3');
   4: rewrite (outfile1, filename1+'vwt4');
   5: rewrite (outfile1, filename1+'vwt5');
   6: rewrite (outfile1, filename1+'vwt6');
   7: rewrite (outfile1, filename1+'vwt7');
   8: rewrite (outfile1, filename1+'vwt8');
end;

while not(eof(infile1)) do
begin

   readln(infile1, water_level, time);

   writeln(outfile1, water_level:6:3, tab, time:6:3);

end;

close (infile1);
close (outfile1);

end;

procedure compare_day;

begin

difference:= day1-day;
if difference > 0.0208 then code:=1;
if difference < -0.0208 then code:=2;
if (difference <= 0.0208) and (difference >= -0.0208) then code:=3;
end;
```

```
procedure enter_data;

begin

writeln ('Beginning correction process.');
writeln;

write('Enter in the name of the folder containing data to be processed:   ');
readln(filename1);
writeln;

filename1:=('main:doe:processed data:'+filename1+':');

end;



procedure set_file;

begin

reset (infile1, filename1+'vwt1');

case i of
   2:  reset (infile2, filename1+'vwt2');
   3:  reset (infile2, filename1+'vwt3');
   4:  reset (infile2, filename1+'vwt4');
   5:  reset (infile2, filename1+'vwt5');
   6:  reset (infile2, filename1+'vwt6');
   7:  reset (infile2, filename1+'vwt7');
   8:  reset (infile2, filename1+'vwt8');
end;

rewrite (outfile1, filename1+'dummy');

end;



procedure process_atmospheric_pressure;

begin

reset (infile1, filename1+'vwt1');
rewrite (outfile1, filename1+'dummy');
while not(eof(infile1)) do begin

readln(infile1, temperature, pressure, day);

B1:=pressure;
A1:=temperature;
```

```
water_level1:=((-0.98*2*B1+12.348)-0.0025*((-104.78+378.11*A1-611.59*A1
*A1+544.27*A1*A1*A1-240.91*A1*A1*A1*A1+
43.089*A1*A1*A1*A1*A1)-19));

writeln(outfile1, water_level1:6:3, tab, day:6:3);

end;

close (infile1);
close (outfile1);

i:=1;
switch_files;

end;



{main body of program}

begin

tab:= char(9);

enter_data;

total_vwt:=8;

writeln;
writeln('Program is running, please wait.');
writeln;

process_atmospheric_pressure;

for i:=2 to total_vwt do

begin

first_call:=true;

set_file;

if not(eof(infile2)) then readln(infile2, temperature, pressure, day);
readln(infile1, pressure1, day1);

not_file_end:=true;
already_written:=false;

while not_file_end do

begin

if eof(infile1) then not_file_end:=false;
```

```
    compare_day;

    case code of
      1:  if not already_written then write_dummy_temp;
      2:  begin
          temporary:=pressure1;
          end;
      3:  begin
          if not already_written then write_dummy;
          temporary:=pressure1;
          end;

    end;
    case code of
      1:  begin
          if not eof(infile2) then
          begin
          readln(infile2, temperature, pressure, day);
          already_written:=false;
          end;
          if eof(infile2) then readln(infile1, pressure1, day1);
          end;
      2:  readln(infile1, pressure1, day1);
      3:  begin
          if not eof(infile2) then
              begin
              readln(infile2, temperature, pressure, day);
              already_written:=false;
              end;
          if eof(infile2) then readln(infile1, pressure1, day1);
          end;
    end;


end;

    close (infile2);
    close (infile1);
    close (outfile1);

    switch_files;


end;

close (infile1);
close (infile2);
close (outfile1);

erase (filename1+'dummy');

writeln;
```

```
writeln('Program is complete.');
writeln;
writeln ('Press return to leave the program.');

readln;


end.
```

APPENDIX I

Core Description

GEOLOGICAL BORING LOG: Peabody 10

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PROJECT: Management of DGD By-Products in Underground Mines

SURF ELEV: 402. FT

LOCATION: SE1/4, NE1/4, SW1/4, Sec10, T13N, R4W, Christian

TOTAL DEPTH: 355.55 FT

LOGGED BY: N. Kawamura

METHOD: HX-wireline core

DATE DRILLED: 6/21/95 to 6/23/95

Depth (feet)	Description	Lithology	Drilling Data	Core Recov (%) 25 50 75	RQD (%) 25 50 75	Fractures per foot 1 2 3 4	Joints Log	Joints Description
0	GLACIAL TILL and soil		Hollow-stem auger					
5								
10								
15								
20								
25								
30								
35								
40								

GEOLOGICAL BORING LOG: Peabody 10

PROJECT: Management of DGD By-Products in Underground Mines

SURF ELEV: 402. FT

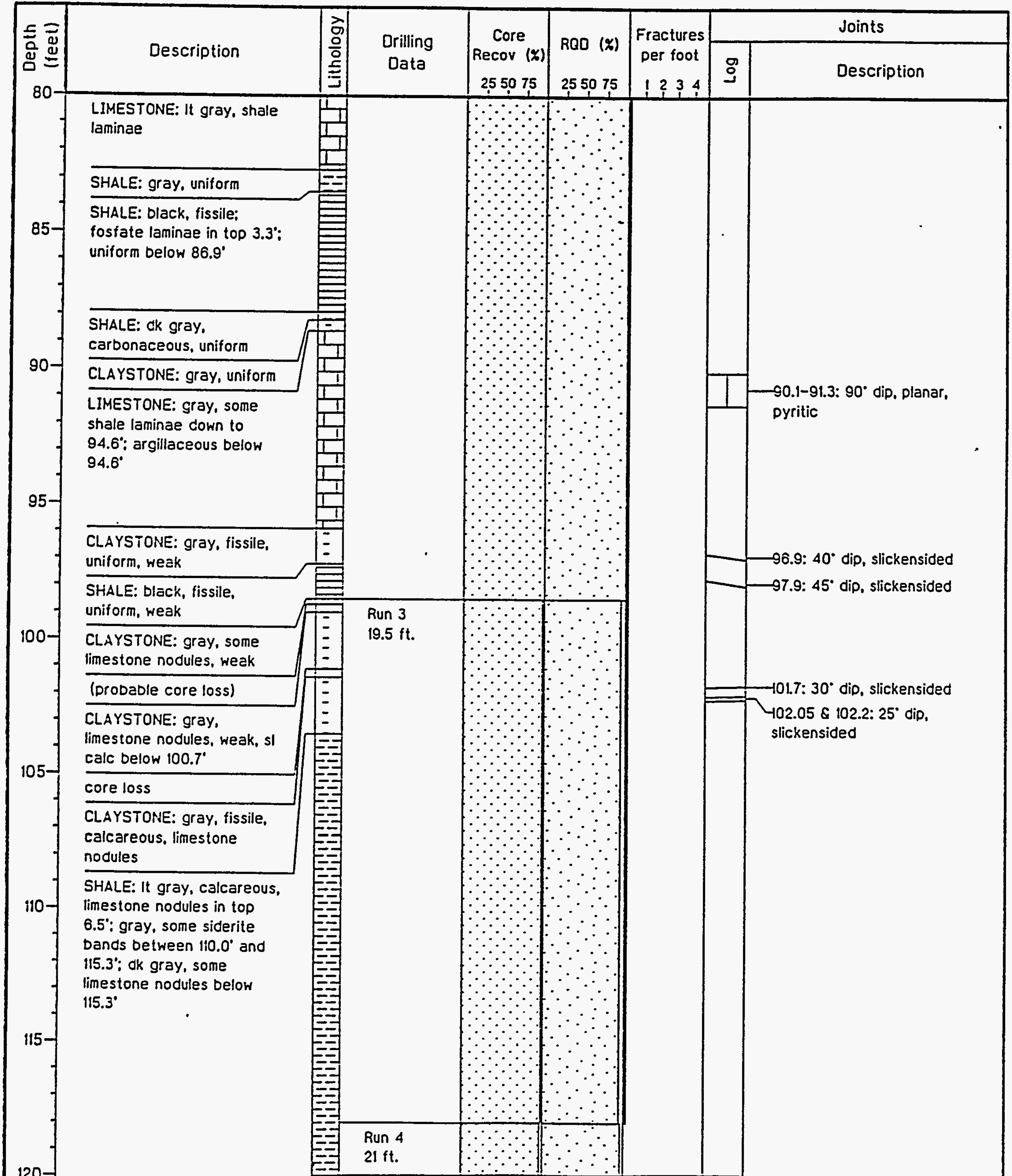
LOCATION: SE1/4, NE1/4, SW1/4, Sec10, T13N, R4W, Christian

TOTAL DEPTH: 355.55 FT

LOGGED BY: N. Kawamura

METHOD: HX-wireline core

DATE DRILLED: 6/21/95 to 6/23/95



LIMESTONE: lt gray, shale laminae

SHALE: gray, uniform

SHALE: black, fissile; fosfate laminae in top 3.3'; uniform below 86.9'

SHALE: dk gray, carbonaceous, uniform

CLAYSTONE: gray, uniform

LIMESTONE: gray, some shale laminae down to 94.6'; argillaceous below 94.6'

CLAYSTONE: gray, fissile, uniform, weak

SHALE: black, fissile, uniform, weak

CLAYSTONE: gray, some limestone nodules, weak

(probable core loss)

CLAYSTONE: gray, limestone nodules, weak, sl calc below 100.7'

core loss

CLAYSTONE: gray, fissile, calcareous, limestone nodules

SHALE: lt gray, calcareous, limestone nodules in top 6.5'; gray, some siderite bands between 110.0' and 115.3'; dk gray, some limestone nodules below 115.3'

90.1-91.3: 90° dip, planar, pyritic

96.9: 40° dip, slickensided

97.9: 45° dip, slickensided

101.7: 30° dip, slickensided

102.05 & 102.2: 25° dip, slickensided

GEOLOGICAL BORING LOG: Peabody 10

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PROJECT: Management of DGD By-Products in Underground Mines

SURF ELEV: 402. FT

LOCATION: SE1/4, NE1/4, SW1/4, Sec10, T13N, R4W, Christian

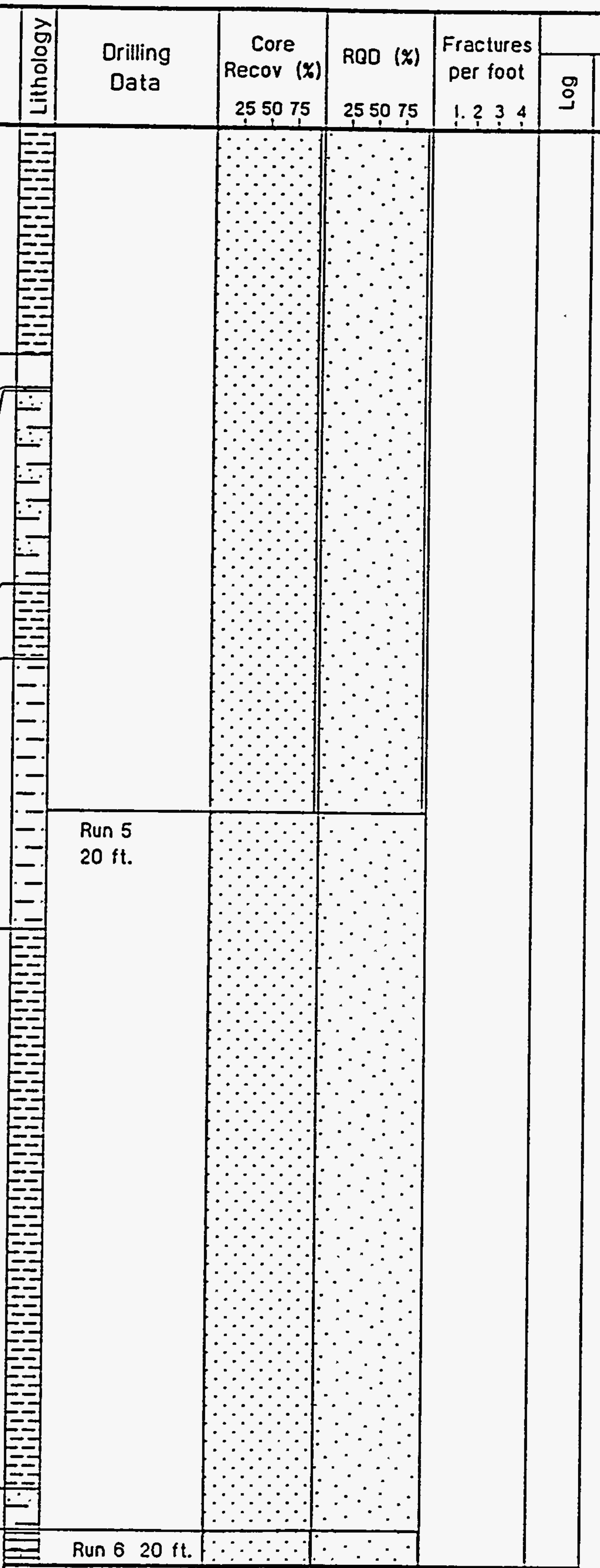
TOTAL DEPTH: 355.55 FT

LOGGED BY: N. Kawamura

METHOD: HX-wireline core

DATE DRILLED: 6/21/95 to 6/23/95

Depth (feet)	Description
120	SHALE: dk gray, some limestone nodules
	(probable core loss)
	LIMESTONE: lt gray, weak, fossiliferous, argillaceous
130	SHALE AND SILTSTONE interlaminated: dk gray shale laminae and lt gray siltstone laminae, calcareous, micaceous
	SHALE: gray, sl micaceous, siltstone laminae
135	SILTSTONE: lt to med gray, micaceous, some wavy shale and sandstone laminae, calcareous in top 1.75'
	SHALE. gray, micaceous, locally very weak, some siltstone laminae
	SHALE AND SILTSTONE interlaminated: lt to med gray, micaceous
160	SHALE: dk gray, (over)



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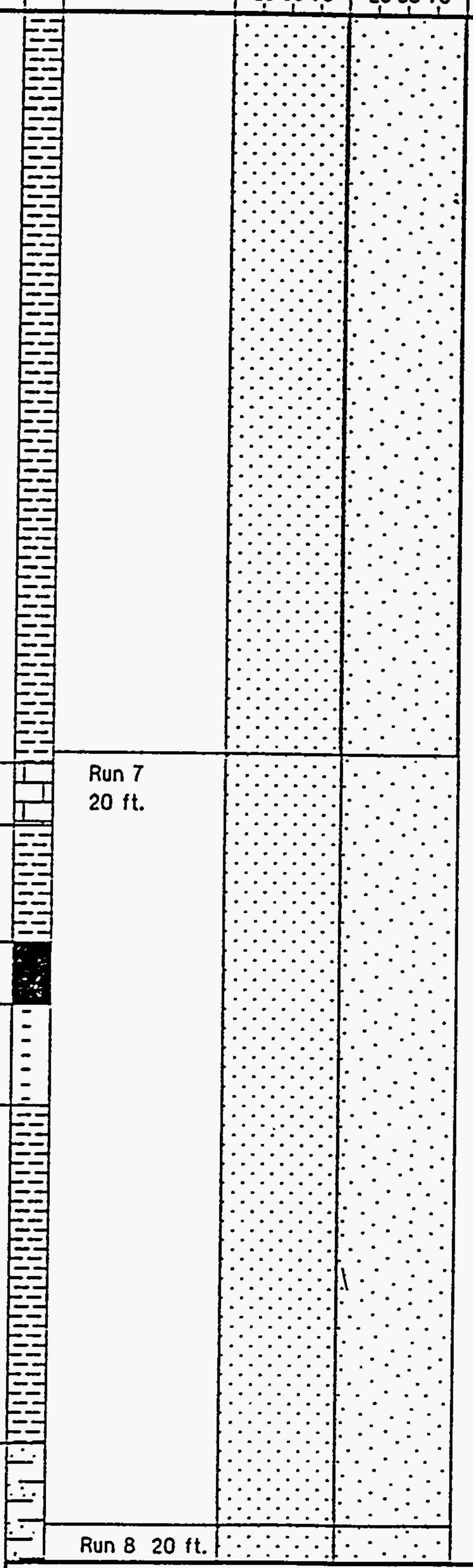
GEOLOGICAL BORING LOG: Peabody 10	*Page 5 of 9*
PROJECT: Management of DGD By-Products in Underground Mines	SURF ELEV: 402. FT
LOCATION: SE1/4, NE1/4, SW1/4, Sec10, T13N, R4W, Christian	TOTAL DEPTH: 355.55 FT
LOGGED BY: N. Kawamura METHOD: HX-wireline core	DATE DRILLED: 6/21/95 to 6/23/95



Depth (feet)	Description	Lithology	Drilling Data	Core Recov (%) 25 50 75	RQD (%) 25 50 75	Fractures per foot 1 2 3 4	Joints	
							Log	Description
160	fissile, carbonaceous, uniform down to 179'; silty between 165.1' and 166.1'; dk gray, fossiliferous, limestone nodules below 179'; (probable core loss at 159.1'-159.3', and 159.6'-159.8')							
165								
170								
175								
180	LIMESTONE: lt gray; weak, argillaceous, fossiliferous		Run 7 20 ft.					
	SHALE: dk gray, carbonaceous, uniform							
185	COAL: black, pyritic, fissile – CHAPEL (No.8) COAL							
	CLAYSTONE: gray, fissile, very weak							
	SHALE: gray, micaceous, siltstone laminae							
190								
195								
	SHALE AND SILTSTONE interlaminated: gray, micaceous							
200			Run 8 20 ft.					

GEOLOGICAL BORING LOG: Peabody 10
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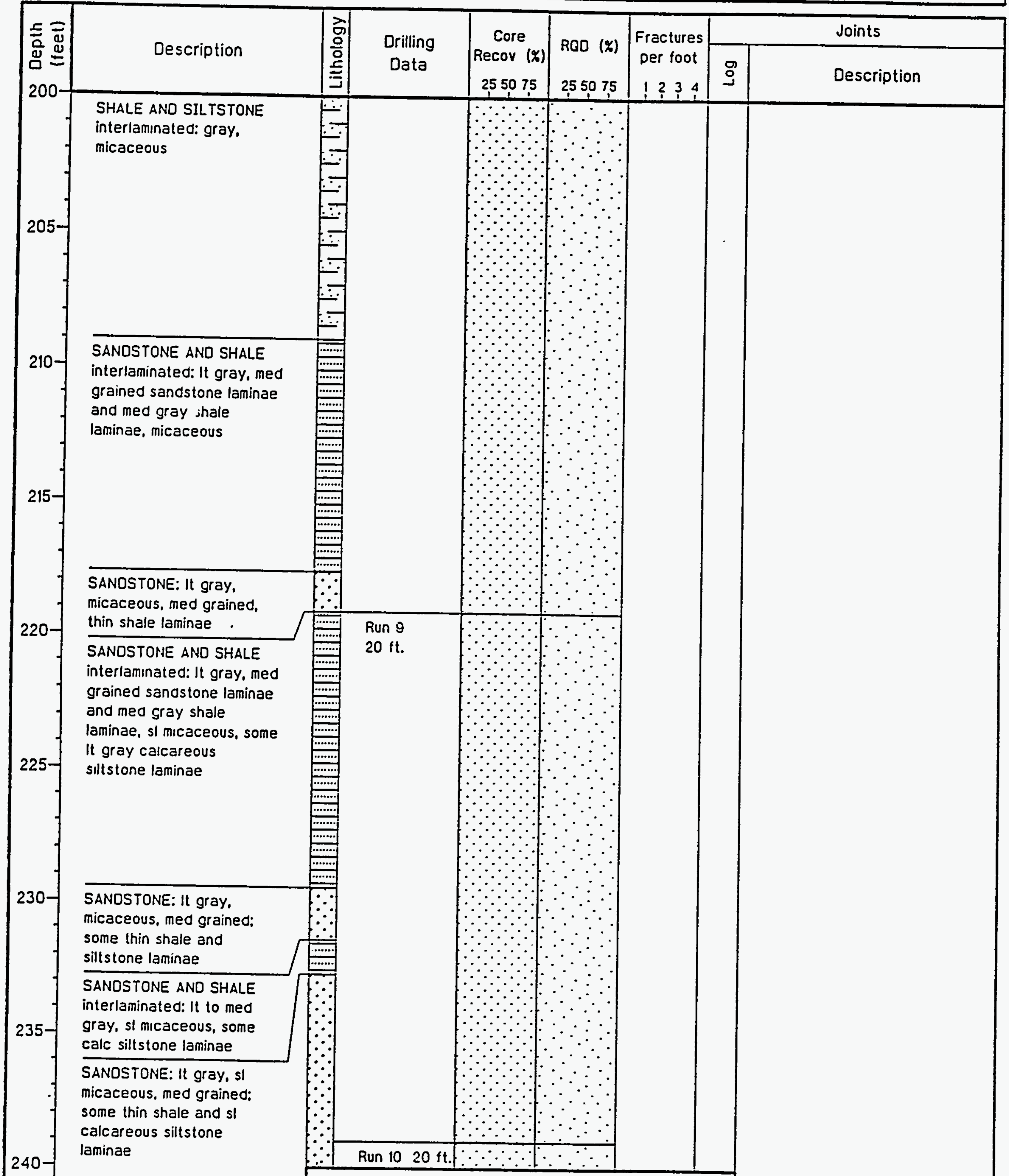
PROJECT: Management of DGD By-Products in Underground Mines
SURF ELEV: 402. FT

LOCATION: SE1/4, NE1/4, SW1/4, Sec10, T13N, R4W, Christian
TOTAL DEPTH: 355.55 FT

LOGGED BY: N. Kawamura
METHOD: HX-wireline core
DATE DRILLED: 6/21/95 to 6/23/95



SHALE AND SILTSTONE interlaminated: gray, micaceous

SANDSTONE AND SHALE interlaminated: lt gray, med grained sandstone laminae and med gray shale laminae, micaceous

SANDSTONE: lt gray, micaceous, med grained, thin shale laminae

SANDSTONE AND SHALE interlaminated: lt gray, med grained sandstone laminae and med gray shale laminae, sl micaceous, some lt gray calcareous siltstone laminae

SANDSTONE: lt gray, micaceous, med grained; some thin shale and siltstone laminae

SANDSTONE AND SHALE interlaminated: lt to med gray, sl micaceous, some calc siltstone laminae

SANDSTONE: lt gray, sl micaceous, med grained; some thin shale and sl calcareous siltstone laminae

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GEOLOGICAL BORING LOG: Peabody 10

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PROJECT: Management of DGD By-Products in Underground Mines

SURF ELEV: 402. FT

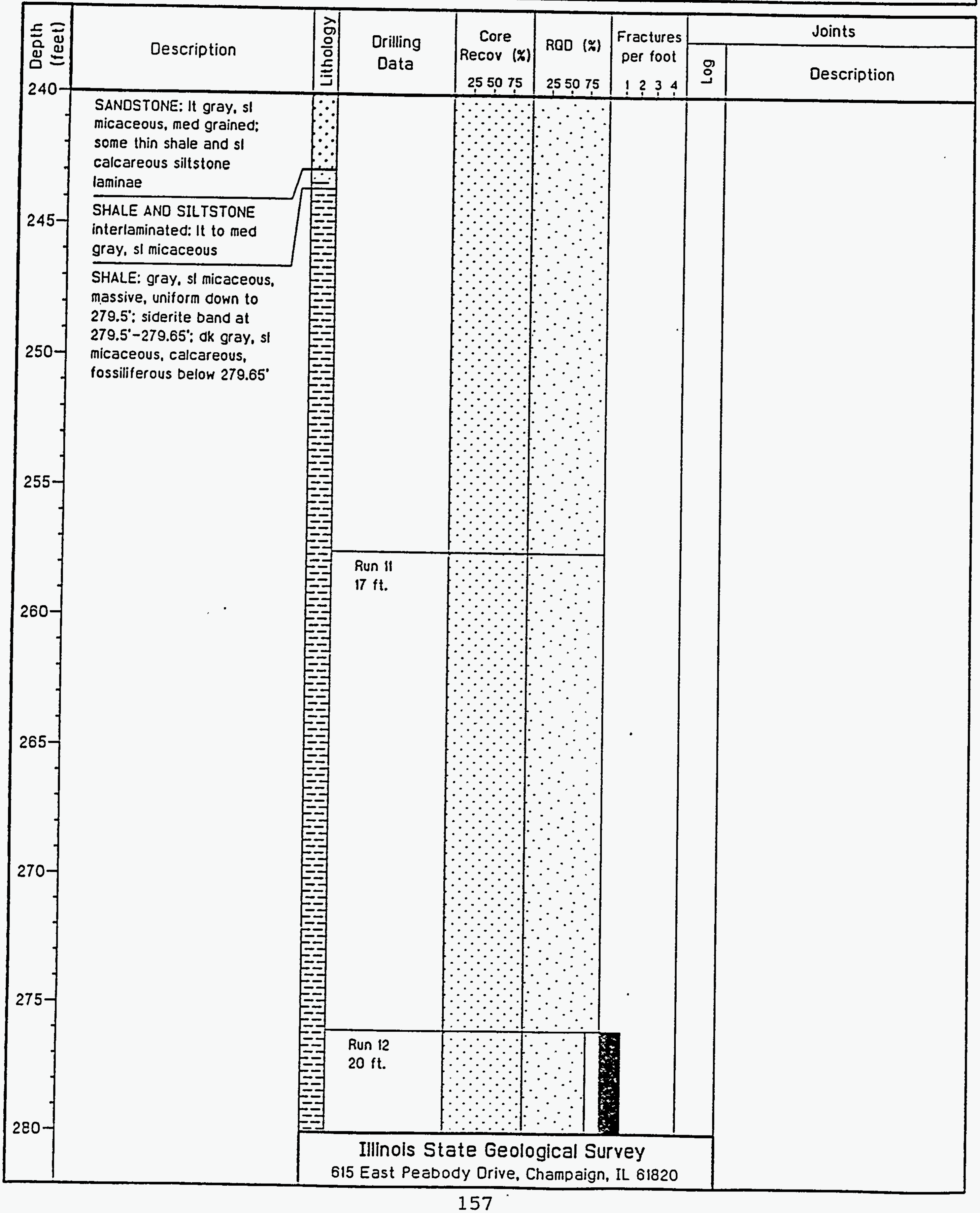
LOCATION: SE1/4, NE1/4, SW1/4, Sec10, T13N, R4W, Christian

TOTAL DEPTH: 355.55 FT

LOGGED BY: N. Kawamura

METHOD: HX-wireline core

DATE DRILLED: 6/21/95 to 6/23/95



SANDSTONE: lt gray, sl micaceous, med grained; some thin shale and sl calcareous siltstone laminae

SHALE AND SILTSTONE interlaminated: lt to med gray, sl micaceous

SHALE: gray, sl micaceous, massive, uniform down to 279.5'; siderite band at 279.5'-279.65'; dk gray, sl micaceous, calcareous, fossiliferous below 279.65'

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GEOLOGICAL BORING LOG: Peabody 10

PROJECT: Management of DGD By-Products in Underground Mines

SURF ELEV: 402. FT

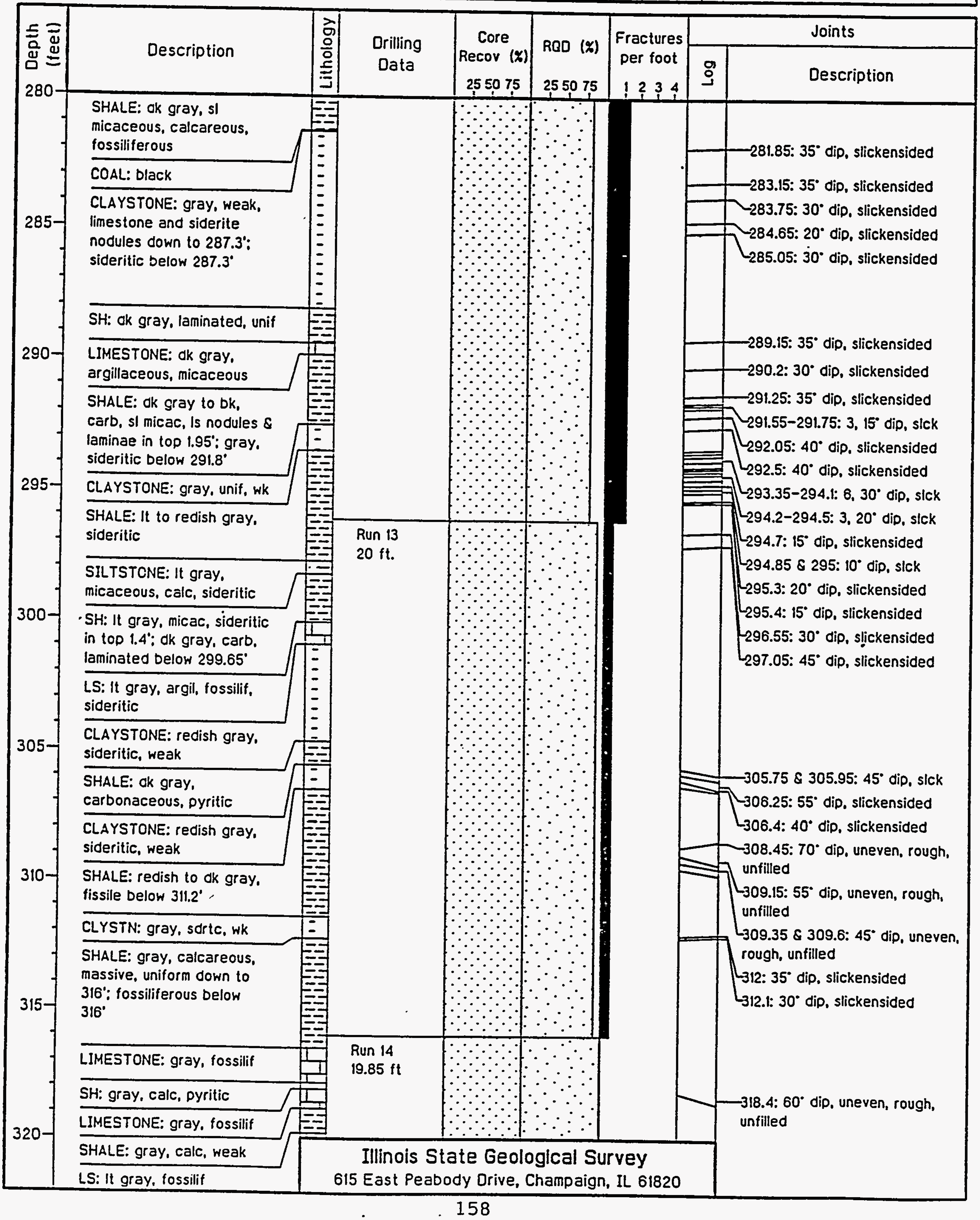
LOCATION: SE1/4, NE1/4, SW1/4, Sec10, T13N, R4W, Christian

TOTAL DEPTH: 355.55 FT

LOGGED BY: N. Kawamura

METHOD: HX-wireline core

DATE DRILLED: 6/21/95 to 6/23/95



SHALE: dk gray, sl micaceous, calcareous, fossiliferous

COAL: black

CLAYSTONE: gray, weak, limestone and siderite nodules down to 287.3'; sideritic below 287.3'

SH: dk gray, laminated, unif

LIMESTONE: dk gray, argillaceous, micaceous

SHALE: dk gray to bk, carb, sl micac, ls nodules & laminae in top 1.95'; gray, sideritic below 291.8'

CLAYSTONE: gray, unif, wk

SHALE: lt to redish gray, sideritic

SILTSTONE: lt gray, micaceous, calc, sideritic

SH: lt gray, micac, sideritic in top 1.4'; dk gray, carb, laminated below 299.65'

LS: lt gray, argil, fossilif, sideritic

CLAYSTONE: redish gray, sideritic, weak

SHALE: dk gray, carbonaceous, pyritic

CLAYSTONE: redish gray, sideritic, weak

SHALE: redish to dk gray, fissile below 311.2'

CLYSTN: gray, sdrtc, wk

SHALE: gray, calcareous, massive, uniform down to 316'; fossiliferous below 316'

LIMESTONE: gray, fossilif

SH: gray, calc, pyritic

LIMESTONE: gray, fossilif

SHALE: gray, calc, weak

LS: lt gray, fossilif

281.85: 35° dip, slickensided
283.15: 35° dip, slickensided
283.75: 30° dip, slickensided
284.65: 20° dip, slickensided
285.05: 30° dip, slickensided
289.15: 35° dip, slickensided
290.2: 30° dip, slickensided
291.25: 35° dip, slickensided
291.55–291.75: 3, 15° dip, slck
292.05: 40° dip, slickensided
292.5: 40° dip, slickensided
293.35–294.1: 6, 30° dip, slck
294.2–294.5: 3, 20° dip, slck
294.7: 15° dip, slickensided
294.85 & 295: 10° dip, slck
295.3: 20° dip, slickensided
295.4: 15° dip, slickensided
296.55: 30° dip, slickensided
297.05: 45° dip, slickensided
305.75 & 305.95: 45° dip, slck
306.25: 55° dip, slickensided
306.4: 40° dip, slickensided
308.45: 70° dip, uneven, rough, unfilled
309.15: 55° dip, uneven, rough, unfilled
309.35 & 309.6: 45° dip, uneven, rough, unfilled
312: 35° dip, slickensided
312.1: 30° dip, slickensided
318.4: 60° dip, uneven, rough, unfilled

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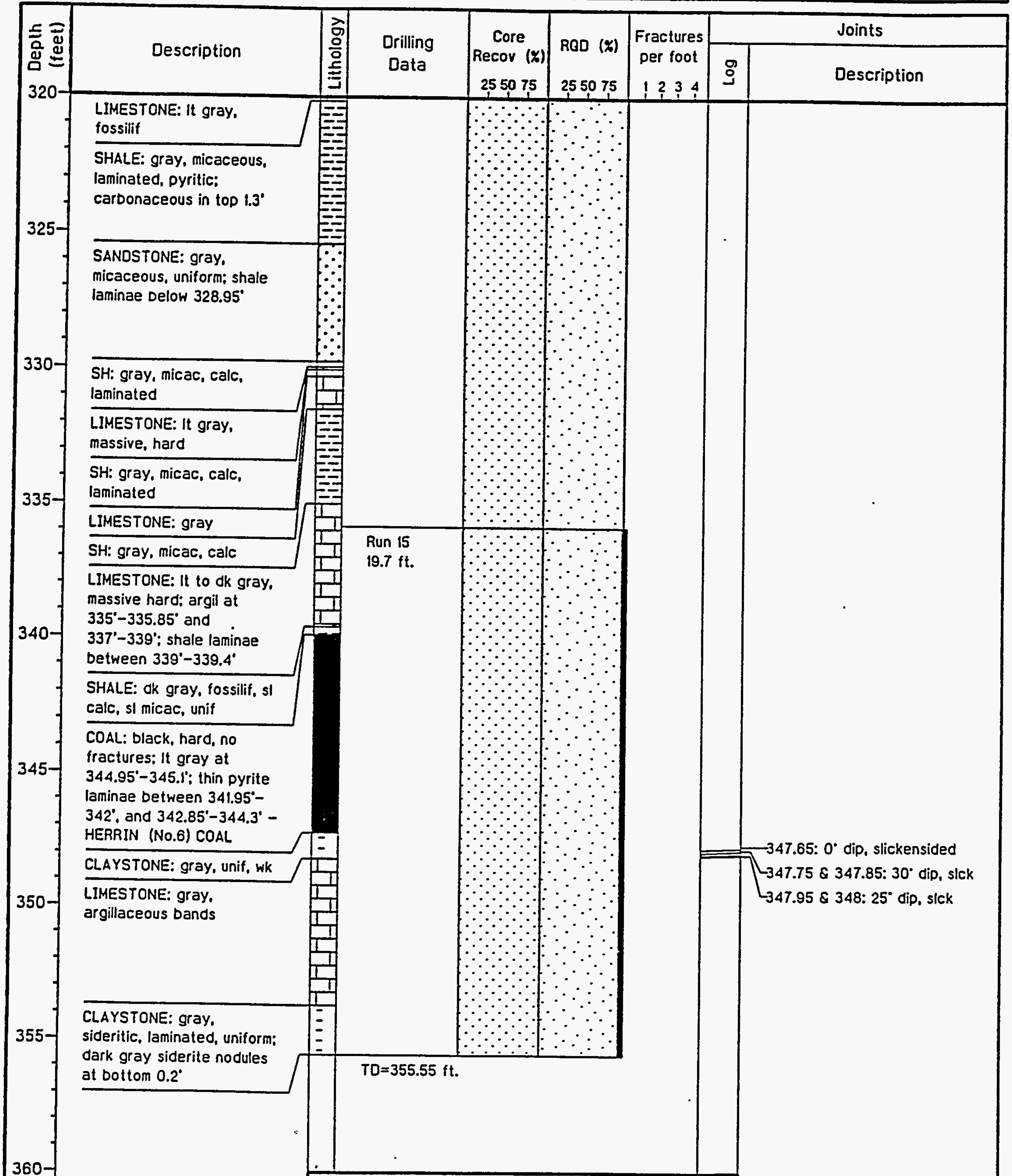
GEOLOGICAL BORING LOG: Peabody 10	*Page 9 of 9*
PROJECT: Management of DGD By-Products in Underground Mines	SURF ELEV: 402. FT
LOCATION: SE1/4, NE1/4, SW1/4, Sec10, T13N, R4W, Christian	TOTAL DEPTH: 355.55 FT
LOGGED BY: N. Kawamura METHOD: HX-wireline core	DATE DRILLED: 6/21/95 to 6/23/95



LIMESTONE: lt gray, fossilif

SHALE: gray, micaceous, laminated, pyritic; carbonaceous in top 1.3'

SANDSTONE: gray, micaceous, uniform; shale laminae below 328.95'

SH: gray, micac, calc, laminated

LIMESTONE: lt gray, massive, hard

SH: gray, micac, calc, laminated

LIMESTONE: gray

SH: gray, micac, calc

LIMESTONE: lt to dk gray, massive hard; argil at 335'-335.85' and 337'-339'; shale laminae between 339'-339.4'

SHALE: dk gray, fossilif, sl calc, sl micac, unif

COAL: black, hard, no fractures; lt gray at 344.95'-345.1'; thin pyrite laminae between 341.95'-342', and 342.85'-344.3' - HERRIN (No.6) COAL

CLAYSTONE: gray, unif, wk

LIMESTONE: gray, argillaceous bands

CLAYSTONE: gray, sideritic, laminated, uniform; dark gray siderite nodules at bottom 0.2'

Run 15
19.7 ft.

TD=355.55 ft.

347.65: 0° dip, slickensided
347.75 & 347.85: 30° dip, slck
347.95 & 348: 25° dip, slck

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ABBREVIATIONS

argil	argillaceous
bk	black
calc	calcareous
carb	carbonaceous
CLYST	CLAYSTONE
DGD	DRY GAS DESULFURIZATION
dk	dark
fossilif	fossiliferous
ls	limestone
LS	LIMESTONE
lt	light
med	medium
micac	micaceous
sdrtc	sideritic
SH	SHALE
sl	slight
slck	slickensided
unif	uniform
wk	weak

APPENDIX J

Packer Test Results

PACKER TEST DATA SHEET

Date: 08/15/95
Page: _1__ of _1_

Tested by: Esling/McDonald/Hemingway: SIU; Mehnert/Restrepo:ISGS;
Project: Peabody Coal Mine SIU/ISGS.

Boring number/description: Peabody #3 (Parking Lot)
Radius [r]: 3.25 inches
Water depth before testing [U]: 11 ft. below ground @ 15:18 hr.

Test#: One
Boring depth: 300 ft.
Gauge height: 17 inches above ground

Interval Tested ft.			Take gal / min.						Pressure psi				Permeability cm/s
From [D]	To	Length [A]	Meter Start	Meter End	Water Loss	Δt (min)	Time	Take [Q]	Gauge (+) [h_1]	Column (+) [h_2]	Pipe Loss (-) [ΔL]	Net [H]	
295	290	5	--	--	--	--	17:10	0	20	--	--	--	Imperm.
295	290	5	--	--	--	--	17:15	0	33	--	--	--	Imperm.
295	290	5	--	--	--	--	17:20	0	47	--	--	--	Imperm.
295	290	5	--	--	--	--	17:25	0	63	--	--	--	Imperm.
295	290	5	--	--	--	--	17:35	0	34	--	--	--	Imperm.

Remarks:
The test was conducted by pumping water through 300 feet of 1 inch rubber hose (some in hole, some on spool). We used rubber hose instead of the 1.5 inch diameter steel pipe. Mud was present inside the packer system before the test. The watch used for the time records was 10 seconds ahead of the CR-10 device. At the end of the day, the system was pulled out the borehole-- the rubber hose was tangled. We decided to use the 1.5" steel pipe instead of the hose to eliminate problem with twisted hose.

PACKER TEST DATA SHEET

Date: 08/16/95
Page: _1__ of _1_

Tested by: Esling/McDonald/Hemingway: SIU; Mehnert/Restrepo: ISGS;
Project: Peabody Coal Mine SIU/ISGS

Boring number/description: Peabody # 3 (Parking Lot)
Radius [r]: 3.25 inches
Water depth before testing [U]: 10 ft. below ground

Test#: Two
Boring depth: 300 ft.
Gauge height: 16.63 inches above ground

Interval Tested ft			Take gal / min.						Pressure psi				Permeability cm/s
From [D]	To	Length [A]	Meter		Water Loss	Δt (min)	Time	Take [Q]	Gauge (+) [h_1]	Column (+) [h_2]	Pipe Loss (-) [ΔL]	Net [H]	
			Start	End									
245	240	5	–	–	–	–	08:38	0	11	–	–	–	Imperm.
245	240	5	–	–	–	–	08:43	0	26	–	–	–	Imperm.
245	240	5	–	–	–	–	08:50	0	52	–	–	–	Imperm.
245	240	5	–	–	–	–	08:56	0	27	–	–	–	Imperm.

Remarks:
The packer system and approximately 200 ft. of steel pipe fell down the borehole (the hydraulic chuck of the drill rig opened). Nobody was injured. Luckily, the packer system and pipe were recovered using the drill rig. The nitrogen gas lines were cut during the recovery process and needed to be fixed before using the packer system again. We decided not to conduct any tests until the next day.

PACKER TEST DATA SHEET

Date: 08/17/95
Page: _1__ of _1_

Tested by: Esling/McDonald/Hemingway: SIU; Mehnert/Restrepo: ISGS;
Project: Peabody Coal Mine SIU/ISGS

Boring number/description: Peabody # 4
Radius [r]: 3.25 inches
Water depth before testing [U]: 40 ft. below ground

Test#: One
Boring depth: 362 ft.
Gauge height: 18.25 inches above ground

Interval Tested ft			Take gal / min.						Pressure psi				Permeability cm/s
From [D]	To	Length [A]	Meter		Water Loss	Δt (min)	Time	Take [Q]	Gauge (+) $[h_1]$	Column (+) $[h_2]$	Pipe Loss (-) [ΔL]	Net [H]	
			Start	End									
222	217	5	–	–	–	5	13:45	0	10	–	–	–	Imperm.
222	217	5	–	–	–	5	13:50	0	25	–	–	–	Imperm.
222	217	5	–	–	–	5	13:55	0	50	–	–	–	Imperm.
222	217	5	–	–	–	5	14:00	0	–	–	–	–	Imperm.
===	===	===	======	======	===	==	===	===	===	===	===	===	======
222	217	5	–	–	–	5	14:15	0	16	–	–	–	Imperm.
222	217	5	–	–	–	5	14:20	0	24	–	–	–	Imperm.
222	217	5	–	–	–	5	14:25	0	50	–	–	–	Imperm.
222	217	5	–	–	–	5	14:30	0	74	–	–	–	Imperm.
===	===	===	======	======	===	==	===	===	===	===	===	===	======
362	217	145	–	–	–	5	14:40	0	10	–	–	–	Imperm.
362	217	145	–	–	–	5	14:45	0	26	–	–	–	Imperm.
362	217	145	–	–	–	5	14:50	0	49	–	–	–	Imperm.
362	217	145	977.56	977.61	0.05	5	15:00	0.010	77	125.38	<0.01	202.38	$9.3x10^{-9}$
362	217	145	977.61	977.62	0.01	5	15:05	0.002	49	125.38	<0.01	174.38	$2.2x10^{-9}$
362	217	145	977.62	977.68	0.06	5	15:10	0.012	20	125.38	<0.01	145.38	$1.6x10^{-6}$
===	===	===	======	======	===	==	18:08	===	===	===	===	===	Start Slug
===	===	===	======	======	===	==	18:24	===	===	===	===	===	Re-start it

Remarks:
No drilling mud was used during drill. The packers were inflated to 300 psi. The flow through the packer system was tested by inflating the bottom packer and flowing water out of the borehole. The injection system was pumping fine. The watch used was 10 seconas ahead of the CR-10 recording device. Please note that the last block of measurements is for a SINGLE PACKER system. The rest of the measurements are for a DOUBLE PACKER system. The length of the rubber hose from the pump to the tip of the well was: 23 ft.

PACKER TEST DATA SHEET

Date: 08/18/95
Page: _1__ of _3_

Tested by: Esling/McDonald/Hemingway: SIU; Mehnert/Restrepo: ISGS;
Project: Peabody Coal Mine SIU/ISGS

Boring number/description: Peabody # 4
Radius [r]: 3.25 inches
Water depth before testing [U]: 40 ft. below ground

Test#: One
Boring depth: 362 ft.
Gauge height: 18.25 inches above ground

Interval Tested ft			Take gal / min.						Pressure psi				Permeability cm/s
From [D]	To	Length [A]	Meter		Water Loss	Δt (min)	Time	Take [Q]	Gauge (+) $[h_1]$	Column (+) $[h_2]$	Pipe Loss (-) [ΔL]	Net [H]	
			Start	End									
222	217	5	986.10	986.10	--	5	08:54	0	17	--	--	--	Imperm.
222	217	5	986.20	986.20	--	5	09:00	0	25	--	--	--	Imperm.
222	217	5	986.45	986.60	0.15	15	09:15	Error	50	--	--	--	Disregard Q
222	217	5	986.60	986.60	--	15	09:30	0	49	--	--	--	Lost Flow
222	217	5	993.21	993.26	0.05	30	10:13	0.0017	71.5	94.82	<0.01	166.32	$2.2\text{x}10^{-3}$

Remarks:
No drilling mud was used during drilling. The packers were inflated at 300 psi. The packer system was tested by inflating only the bottom packer. The injection system was working fine. The watch used was 10 seconds ahead of the CR-10 recording device. The length of the rubber hose from the pump to the tip of the well was: 23 ft. The flow was lost to the pump, so we reprimed it before the last measurement was made.

PACKER TEST DATA SHEET

Date: 08/18/95
Page: 2 of 3

Tested by: Esling/McDonald/Hemingway: SIU; Mehnert/Restrepo: ISGS;
Project: Peabody Coal Mine SIU/ISGS

Boring number/description: Peabody # 5
Radius [r]: 3.2 inches
Water depth before testing [U]: 45 in. below ground
26 in. below top of casing

Test#: One
Boring depth: 362 ft.
Gauge height: 26 inches above ground

Interval Tested ft			Take gal / min.						Pressure psi				Permeability cm/s
From [D]	To	Length [A]	Meter		Water Loss	Δt (min)	Time	Take [Q]	Gauge (+) $[h_1]$	Column (+) $[h_2]$	Pipe Loss (-) $[\Delta L]$	Net [H]	
			Start	End									
350	215	147	1135.34	1135.52	0.18	15	16:50	0.012	24	124.63	<0.01	148.63	$1.5x10^{-8}$
350	215	147	1135.63	1136.00	0.37	15	17:07	0.025	51	124.63	<0.01	175.63	$2.6x10^{-8}$
350	215	147	1136.10	1136.90	0.80	30	17:39	0.027	73	124.63	<0.01	197.63	$2.6x10^{-8}$
350	215	147	1136.90	1136.98	0.08	15	17:57	0.005	50	124.63	<0.01	174.63	$5.3x10^{-9}$
350	215	147	1137.00	1137.10	0.10	15	18:14	0.006	24	124.63	<0.01	148.63	$6.3x10^{-9}$

Remarks:
The packers were inflated at 300 psi. The packer system was tested by inflating only the bottom packer. The watch used was 10 seconds ahead of the CR-10 recording device. Please note that the measurements are for a SINGLE PACKER system.

PACKER TEST DATA SHEET

Date: 08/18/95
Page: 3 of 3

Tested by: Esling/McDonald/Hemingway: SIU; Mehnert/Restrepo: ISGS;
Project: Peabody Coal Mine SIU/ISGS

Boring number/description: Peabody # 5
Radius [r]: 3.2 inches
Water depth before testing [U]: 45 in. below ground
26 in. below top of casing

Test#: Two
Boring depth: 362 ft.
Gauge height: 26 inches above ground

Interval Tested ft			Take gal / min.						Pressure psi				Permeability cm/s
From [D]	To	Length [A]	Meter		Water Loss	Δt (min)	Time	Take [Q]	Gauge (+) $[h_1]$	Column (+) $[h_2]$	Pipe Loss (-) $[\Delta L]$	Net [H]	
			Start	End									
350	232	130	1136.28	1136.28	--	15	19:25	0	51	--	--	--	Imperm.
350	232	130	1136.30	1136.30	--	15	19:40	0	75	--	--	--	Imperm.
350	232	130	1136.29	1136.29	--	15	19:58	0	50	--	--	--	Imperm.

Remarks:
With the bottom packer inflated and the top packer not inflated, pumped 99 gallons of water through the system and back up the brehole to confirm the injection pump was working properly. The bottom packer was then deflated and the top packer inflated to 315 psi. The watch used was 10 seconds ahead of the CR-10 recording device. Please note that the measurements are for a SINGLE PACKER system. Hole was reamed to a depth of 350 feet.

October 23, 1995

Summary of Packer Test Results at the Peabody Mine # 10, near Pawnee, IL

E. Mehnert and M. Restrepo, ISGS

The computations for the single and double packer tests where water flowed into the geological materials are summarized below. Data sheets with the field data are attached at the end of this report.

Table 1. Summary of hydraulic conductivity estimates from packer tests at Peabody #10

Date	Borehole Number	Interval tested (ft)	h1 (psi)	Q (gpm)	K (cm/s)[a]		
					Dagan	Moye	Hvorslev
08/17/95	Peabody #4	217-362	77	0.010	$9.31\text{x}10^{-9}$	$1.5\text{x}10^{-8}$	$8.93\text{x}10^{-9}$
08/17/95	Peabody #4	217-362	49	0.0020	$2.16\text{x}10^{-9}$	$2.44\text{x}10^{-9}$	$2.07\text{x}10^{-9}$
08/17/95	Peabody #4	217-362	20	0.012	$1.56\text{x}10^{-8}$	$1.76\text{x}10^{-8}$	$1.49\text{x}10^{-8}$
08/18/95	Peabody #4	217-222	71.5	0.0017	Not computed	$3.09\text{x}10^{-8}$	$2.23\text{x}10^{-8}$
08/18/95	Peabody #5	215-350	24	0.012	$1.50\text{x}10^{-8}$	$1.70\text{x}10^{-8}$	$1.44\text{x}10^{-8}$
08/18/95	Peabody #5	215-350	51	0.025	$2.65\text{x}10^{-8}$	$3.00\text{x}10^{-8}$	$2.54\text{x}10^{-8}$
08/18/95	Peabody #5	215-350	73	0.027	$2.55\text{x}10^{-8}$	$2.87\text{x}10^{-8}$	$2.44\text{x}10^{-8}$
08/18/95	Peabody #5	215-350	50	0.0050	$5.33\text{x}10^{-9}$	$6.03\text{x}10^{-9}$	$5.12\text{x}10^{-9}$
08/18/95	Peabody #5	215-350	24	0.0060	$6.27\text{x}10^{-9}$	$7.08\text{x}10^{-9}$	$6.01\text{x}10^{-9}$

a: References for methods used to calculate hydraulic conductivity:
Dagan, G., 1978. A note on packer, slug and recovery tests in unconfined aquifers, Water Resources Research, Vol. 14, no. 5, pp. 929-934.

Moye, D.G., 1967. Diamond drilling for foundation exploration, Civil Engineering Transactions, pp. 95-100.

Hvorslev, M.J., 1951. Time lag and soil permeability in groundwater observations, U.S. Army Corps of Engineering, Water Experimental Station, Vicksburg, Ms, Bulletin 36, 56 p.

To assess the sensitivity of the estimates of the hydraulic conductivity to changes in the value of h1 (gauge pressure) and Q (discharge), the following values of K (in cm/s) were estimated using Dagan's method:

Table 2. Estimated hydraulic conductivity for certain flow rates and injection pressures

	Hydraulic conductivity (cm/sec)	
	Q= 0.0005 gpm	Q= 0.00005 gpm
h1 = 25 psi	$6.9x10^{-10}$	$6.9x10^{-11}$
h1 = 75 psi	$5.2x10^{-10}$	$5.2x10^{-11}$

The estimates of hydraulic conductivity require input of the pressure loss in the injection system, which was computed assuming the same system configuration as for the data collected on July 19, 1995. The configuration used included 265 ft of 1.5 inch diameter steel pipe and pipe fittings with an equivalent length of 11.5 ft of steel pipe of the same diameter. Using these values, the pressure losses account for less than 0.01 psi, which is a negligible number.

The equations used are the following:

$$R_{e_D}(Q) = 3244.33Q$$

$$\frac{1}{\sqrt{f}} = -2\log_{10}(2.97x10^{-4}+\frac{2.5}{R_{e_D}}\frac{1}{\sqrt{f}})$$

$$\frac{dp}{dl} = -fQ^2\frac{0.233}{144}$$

where: Q= discharge (gpm)
f= friction coefficient for steel
dp/dl= friction loss (psi/ft)
R_{eD}= Reynolds number

The estimates of hydraulic conductivity in Table 2 can be interpreted as maximum values of hydraulic conductivity for tests where no water flowed into the formation. For example, if the injection pressure were 25 psi and the flow rate into the formation were 0.0005 gpm, then the hydraulic conductivity would be $6.9x10^{-10}$ cm/sec.

APPENDIX K

Monitoring Well Installation Notes

Field Notes for Well Drilling and Well Installation at the Peabody Mine #10, near Pawnee, Illinois

General Notes:

1) All depths recorded with respect to the ground surface
2) Used Type I cement, 94 pound bags, mixed with 4.5 to 5 gallons of water. Starting on 8/28/95, used 6 gallons of water per bag to minimize bridging.
3) One inch PVC tremie is actually 0.25 inches longer than 10 feet.
4) Stainless steel centralizers were added to 2 inch riser pipe at 20 foot intervals, starting at the first joint above the top of the screen.
5) Drillers: Hawkey and Kline of St. Peter, Illinois. Sonny Hawkey drilled P1 and P2. Carroll Kline drilled P2a through P6.
6) Field Personnel: SIUC: Steven Esling, Chris Hemingway, and Tim McDonald. ISGS: Ed Mehnert, Miquel Restrepo, and Nelson Kawamura
7) Borehole Geophysics: Tim Young of the ISGS

Peabody 1

Type: Corehole
Date Drilled: 6/21-6/23/95
Date Logged: 6/23/95
Depth Drilled (feet): 355.5
Depth Logged (feet): 351
Logs Run: Natural Gamma, Caliper, Neutron, Density
Location: Barrier
Status: Abandoned and sealed
Notes: Three inch core retrieved with 20 foot core barrel, nearly 100% recovery. Core description by Nelson Kawamura.

Field Notes for Well Drilling and Well Installation at the Peabody Mine #10, near Pawnee, Illinois

Peabody 2

Type: Geotechnical Monitoring
Date Drilled: 7/1/95
Date Logged: 7/17/95
Depth Drilled (feet): Not recorded
Depth Logged (feet): 317
Logs Run: Natural Gamma, Caliper
Location: Mine
Date Tested with Packer: 7/19/95

Zones of Interest: 300.00-295.00 Fractured Zone
288.00-285.00 Fractured Zone
248.00-243.00 Sandstone/Siltstone

Tested Intervals: 244.50-239.50

Status: Abandoned and sealed 6/23/95
Notes: Six inch hole. Hole caved above Anvil Rock Sandstone. Lost circulation in hole 12 feet from the bottom. Driller said sandstone was thicker in P2 than P1. Hole not suited for monitoring. Driller reamed hole and attempted to plug the bottom, but was not successful. Moved 20 feet to the east and drilled P2A.

Field Notes for Well Drilling and Well Installation at the Peabody Mine #10, near Pawnee, Illinois

Peabody 2A

Type: Groundwater Monitoring
Date Drilled: 8/23-8/24/95
Date Logged: Not Logged
Depth Drilled (feet): >350 feet
Depth Logged (feet):
Logs Run:
Location: Barrier
Date Tested with Packer: Not Tested

Zones of Interest: 345.00-340.00 Coal
248.00-243.00 Sandstone/Siltstone

Screened Intervals: 345.00-340.00 Coal (2 inch PVC)
218.25-213.25 Sandstone/Siltstone (1.25 inch PVC)
80.00-75.00 Upper Bedrock (1.25 inch PVC)

Status: Nested Groundwater Monitoring Wells
Notes:

8/24/95: Set lower screen; added 30 gallons of sand; no water flow through casing to the surface. Resistance on tremie at 323 feet; mixed and pumped 9 bags of cement minus volume of the hose. Flushed with 20 gallons of water.

8/25/95: Resistance on tremie at 218.25 feet; flushed hole with approximately 350 gallons of water. Set middle screen and added 15 gallons of sand, gray water flowed out 1.25 inch pipe, set about 8 feet above grade.

8/28/95: Resistance on tremie at 200.5 feet; mixed and pumped 10 bags of cement, followed by 10 gallons of water. Muddy water flowed out space between the outer steel casing and the PVC pipe. Resistance on tremie at 135.25 feet; mixed and pumped 5 bags of cement; followed by 5 gallons of water. Clear water flowed out of steel casing.

8/29/95: Resistance on tremie at 68 feet, firm at 75 feet. Flushed hole with water. Set upper screen and added 15 gallons of sand. Resistance on tremie at 59 feet; mixed and pumped 4.8 bags of cement.

8/30/95: Set 5 inch outer casing in concrete, within 6 inch steel casing. Cut 2 inch PVC at 348 feet, 1.25 inch PVC at 82 feet and 219 feet.

Field Notes for Well Drilling and Well Installation at the Peabody Mine #10, near Pawnee, Illinois

Peabody 3

Type: Groundwater Monitoring
Date Drilled: 8/14/95
Date Logged: 8/15/95
Depth Drilled (feet): 328 feet
Depth Logged (feet): 308-302 feet
Logs Run: Natural Gamma, Caliper, Neutron
Location: Mine
Date Tested with Packer: See notes

Zones of Interest:
295.00-290.00 Fractured Zone
284.00-279.00 Fractured Zone
222.00-217.00 Sandstone/Siltstone

Screened Intervals:
287.50-282.50 Fractured zone (2 inch PVC)
206.00-201.00 Sandstone/Siltstone? (1.25 inch PVC)

Status: Groundwater Monitoring Wells
Notes:

8/15/95: Packer test over the interval from 284 to 279 feet using rubber hose for packer casing. Detected no flow.

8/16/95: Packer test over the interval from 245 to 240 feet using rubber hose as packer casing. Found hose would kink at depth. Replaced hose with steel pipe.
Lost packer at 3:15 PM; recovered it at 7:15 PM.

8/21/95: Tried to set lower screen, but encountered resistance at 284-279 feet. Pushed pipe with SIUC drill rig. Set screen and added 10 gallons of sand. Resistance on tremie at 282.2 feet. Mixed and pumped 6 bags of cement.

8/22/95: Resistance on tremie at 278.15 feet. Mixed and pumped 5 bags of cement. Waited 30 minutes and added 2 more bags of cement. Added 2.5 gallons of sand.
Resistance on tremie at 238 feet. Added 2 more bags of cement and flushed with 10 gallons of water. Resistance on 1.25 inch pipe at 206.33 feet. Set upper screen and added 15 gallons of sand to 197.8. feet. Added 27 bags of cement.

8/23/95: Cement reached to within 2.75 feet below top of casing.

8/30/95: Set 5 inch PVC pipe inside 6 inch steel casing. 2 inch and 1.25 inch pipe had been previously cut to allow rig to be moved.

Field Notes for Well Drilling and Well Installation at the Peabody Mine #10, near Pawnee, Illinois

Peabody 4

Type: Groundwater Monitoring
Date Drilled: 8/15-8/16/95; squeezed overnight and reamed on 8/23/95
Date Logged: 8/23/95
Depth Drilled (feet): 365; reamed to 361
Depth Logged (feet): 362
Logs Run: Natural Gamma, Caliper, Neutron, Density
Location: Barrier
Date Tested with Packer: See notes

Zones of Interest:	347.50-341.50	Coal
	339.00-335.50	Anvil Rock Sandstone
	331.00-326.00	Limestone
	297.00-292.00	Lower Fractured Zone
	222.50-217.50	Sandstone/Siltstone
	95.00-90.00	Upper Bedrock
Screened Intervals:	348.50-343.50	Coal (2 inch PVC)

Status: Groundwater Monitoring Well
Notes:

8/16/95: Found hole squeezed shut at 308 feet in the morning.

8/17/95: Packer test over the interval from 222 to 217 feet; with no flow at 70 psi. Tested the interval from 217 feet to the base of the hole with trickle flow at 75 psi. Ran slug test overnight; water dropped 69.47 feet at 7:30 AM on 8/18/95.

8/22/95: Resistance on 2 inch PVC at 319 feet; set screen at 299-294 feet. Removed pipe.

8/23/95: Reamed hole; driller adds ASP-700 (adomite, shale stabilizer and viscosifier). Set screen (350 feet of casing and 5 feet of screen) and added 30 gallons of sand. Resistance on tremie at 308 feet. Mixed and pumped18 bags of cement. Water flowed out of 2 inch PVC pipe. Could only pump grout under low pressure or the hose blew off the tremie.

8/24/95: Left 195 feet of tremie in hole overnight; needed 4 people to extract it. Lower 50 feet filled with mud and/or cement. Plumb bobbed to 170 feet. Resistance on tremie at 175 feet. Mixed and pumped 10 bags of cement followed by 20 gallons of water.

8/25/95: Tremie stuck in mud at 55 feet; flushed hole with 300 gallons of water, lowering tremie to 66 feet. Cement found at this depth. Mixed and pumped 7 bags of cement, from 66 to 3 feet.

8/30/95: Cut 2 inch PVC to 1 inch above 352 foot mark. Set 5 inch PVC casing in concrete inside 6 inch steel casing.

Field Notes for Well Drilling and Well Installation at the Peabody Mine #10, near Pawnee, Illinois

Peabody 5

Type: Groundwater Monitoring
Date Drilled: 8/16-8/17/95; reamed 8/23/95
Date Logged: 8/17/95
Depth Drilled (feet): Not Recorded; reamed to 350 feet
Depth Logged (feet): 357
Logs Run: Natural Gamma, Caliper, Neutron, Density
Location: Barrier
Date Tested with Packer: See notes

Zones of Interest:	343.50-337.00	Coal
	335.00-330.00	Limestone
	326.00-321.00	Anvil Rock Sandstone
	220.00-215.00	Sandstone/Siltstone
Screened Intervals:	342.00-337.00	Coal (2 inch PVC)
	220.00-215.00	Sandstone/Siltstone (1.25 inch PVC)
	78.00-73.00	Upper Bedrock (1.25 inch PVC)

Status: Nested Groundwater Monitoring Wells
Notes:

8/18/95: Inflated lower packer and pumped about 100 gallons of water to the surface. Tested the interval with the packer from 362 to 215 feet and from 362 to 232 feet using steel packer casing.

8/23/95: Hole reamed. Resistance on 2 inch PVC at 310 feet; pushed with SIUC drill rig. Set screen and added 27 gallons of sand.

8/24/95: Mixed and pumped 7 bags of cement followed by 20 gallons of water.

8/25/95: Resistance on tremie at 290 feet. Mixed and pumped 7 bags of cement. Resistance on tremie at 235 feet. Set middle screen and added 36 gallons of sand.

8/28/95: Resistance on tremie at 183.5 feet. Mixed and pumped 9 bags of cement followed by 5 gallons of water. Let tremie drain at 85 feet. Muddy water flowed out of steel casing.

8/29/95: Resistance on tremie at 86 feet. Set upper screen and added 20 gallons of sand. Water flowed out of 1.25 inch pipe, first clear, then dirty. Resistance on tremie at 65 feet.

8/30/95: Resistance on tremie at 64 feet (soft). Mixed and pumped 6 bags of cement. Extended 6 inch casing with 4.5 feet length left by drillers. Casing connected with threaded coupling. Turned steel casing pipe with 48 inch and 24 inch pipe wrenches. Placed one bag of concrete inside 6 inch steel casing and cut 2 inch PVC at 343.7 feet and 1.25 inch PVC at 221.67 and 80 feet.

Field Notes for Well Drilling and Well Installation at the Peabody Mine #10, near Pawnee, Illinois

Peabody 6

Type: Groundwater Monitoring
Date Drilled: 8/17-8/18/95
Date Logged: 8/18/95
Depth Drilled (feet): 325; reamed to 320 feet
Depth Logged (feet): 322
Logs Run: Natural Gamma, Caliper, Neutron, Density
Location: Mine
Date Tested with Packer: Not tested

Zones of Interest:	316.00-311.00	Sandstone and Limestone
	294.00-289.00	Fractured Zone
	283.00-278.00	Fractured Zone
	230.00-225.00	Sandstone/Siltstone

Screened Intervals:	316.00-311.00	Limestone/Sandstone (2 inch PVC)
	221.50-216.50	Sandstone/Siltstone (1.25 inch PVC)
	76.00-71.00	Upper Bedrock (1.25 inch PVC)

Status: Nested Groundwater Monitoring Wells
Notes:

8/23/95: Resistance on 2 inch PVC at 300 feet. Pushed pipe with SIUC Drill Rig. Set lower screen and added 30 gallons of sand. Water flowed out of 2 inch PVC, which was about 8 feet above grade.

8/24/95: Resistance on tremie at 296 feet. Mixed and pumped 7 bags of cement. Water flowed out of steel casing.

8/25/95: Resistance on tremie at 263 feet. Mixed and pumped 4 bags of cement. Tagged bottom at 221.5 feet. Set middle screen and added 15 gallons of sand.

8/28/95: Resistance on tremie at 195 feet; felt like sand. Mixed and pumped 10 bags of cement followed by 5 gallons of water. Drained tremie in the hole.

8/29/95: Resistance on tremie at 107 feet. Mixed and pumped cement, but failed to record the number of bags. Resistance on tremie at 92 feet. Mixed and pumped 1 bag of cement. Had to clear plug in bottom of the tremie. Resistance on tremie at 84 feet. Set screen and added 30 gallons of sand. Resistance on tremie at 48 feet, but can push it to 58 feet.

8/30/95: Resistance on tremie at 48 feet. Mixed and placed 4 bags of cement. Set 5 inch outer casing inside 6 inch steel casing with concrete and with wooden shims. Cut 2 inch PVC at 319 feet, 1.25 inch PVC at 79 feet and 220 feet.