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

**Annual Report
October 1993 - September 1994**

Y. P. Chugh
D. Dutta
S. Esling
N. Ghafoori
B. Paul
H. Sevim
E. Thomasson

October 1994

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

MASTER

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Department of Mining Engineering
Carbondale, Illinois 62901

October 1994

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Annual Technical Progress Report -- October 1, 1993 - September 30, 1994

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SECTION I
INTRODUCTION

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

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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 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 in abandoned coal mines, and will assess the environmental impact of such underground residues 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. Steven Esling, Dr. Nader Ghafoori, Dr. Bradley Paul, Dr. Hasan Sevim, and Mr. Edwin Thomasson, all of SIUC. Mr. Jerry Harness was the Contracting Officer's Representative (COR) for the U.S. Department of Energy; however, Mr. Harness was replaced by Mr. Steven J. Bossart as COR late in the year.

The overall program set forth in the cooperative agreement is for a period of four (4) years, beginning September 30, 1993. However, the 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 had been in effect for one full calendar year on September 30, 1994, 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 first full year. Therefore, this report sets forth the technical progress under the agreement for the period September 30, 1993 - September 30, 1994 of Phase I of the overall program.

* * * * *

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

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SUMMARY

At the close of the first year the program was essentially on schedule fiscally, but slightly behind technically. In the management area all sub-accounts have been established by the SIUC General Accounting Office, and the program for monitoring and tracking expenditures and budget balances is in place and functions well in the Office of the Program Manager. A "Kick-Off" meeting of all cooperating parties was held in January, 1994, with the purpose of outlining and agreeing on the role of the various parties. A Steering Committee has been appointed, and a meeting of the committee was held in June, 1994. Program expenditures are within the original budget projections.

Preliminary environmental risk assessment on the FGD by-products to be placed underground is virtually complete. The initial mixes for pneumatic and hydraulic placement have been selected and are being subject to TCLP, ASTM, and modified SLP shake tests as well as ASTM column leaching. Results of these analyses show that the individual coal combustion residues, and the residues mixes, are non-hazardous in character and very unlikely to have any adverse effects on groundwater at the underground placement site.

Based on available information, including well logs obtained from Peabody Coal Company, a detailed study of the geology of the placement site was completed. The study shows that the disposal site in the abandoned underground mine workings at depths of between 325 and 375 feet are well below potable groundwater resources. This, coupled with the benign nature of the residues and residues mixtures, should alleviate any concern that the underground placement will have adverse effects on groundwater resources.

Seven convergence stations were installed in the proposed underground placement area of the Peabody Coal Company No. 10 mine. These stations are essential to establish baseline data for determining the alleviating effect of the underground placement of residues mixtures on surface subsidence. Several sets of convergence data were obtained from the stations.

A study of materials handling and transportation of coal combustion residues from the electric power plant to the injection site has been made. The study evaluated the economics of the transportation of coal combustion residues by pneumatic trucks, by pressure differential rail cars, and by SEEC, Inc. collapsible intermodal containers (CIC's) for different annual handling rates and transport distances.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

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SUMMARY - CONTINUED

The preliminary physico-chemical characteristics and engineering properties of various FBC fly ash - spent bed mixes have been determined, and long-term studies of these properties are continuing.

Dr. Rick Honaker, citing the press of academic and other research responsibilities, withdrew as the Co-Principal Investigator for the Residues Placement area, and Dr. Y.P. Chugh assumed the responsibility for coordinating this area of research and development. Dr. Chugh will be assisted by Dr. Deepak Dutta, a post-doctoral fellow at SIUC. Also, a slight change in the proposed area for underground placement was made by Peabody Coal Company. The new site is more accessible, and thus will require less road and other construction, and will result in less disturbance to prime farm land under cultivation.

A demonstration of the SEEC, Inc. technology utilizing the Collapsible Intermodal Containers (CIC's) for transporting coal combustion residues will be held in November, 1994. Planning for the demonstration is ongoing and nearly complete at the close of the year.

Overall, the program is essentially on schedule, and has encountered only normal problems during the first year. No serious problems are anticipated for the near future.

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MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

Annual Technical Progress Report -- October 1, 1993 - September 30, 1994

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 workings 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.

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

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OBJECTIVES (Continued)

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 I. The demonstration of developed technology in objective 1 will also occur during Phase I. Demonstration of 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 laid during Phase I. This foundation will consist of determining the physical, chemical, engineering, and leachate characteristics of various coal combustion residues and selected residue mixtures; establishing baseline data on groundwater and other environmental characteristics; and developing economic analyses and economic models of the residue 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 coal combustion residues.

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MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

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PROGRAM MANAGEMENT

A sound, efficient, and effective management plan for the overall management of the program has been devised and placed in operation. Soon after the program was initiated, a management plan was written, with input from all the Co-Principal Investigators. The plan detailed a "Management by Objectives" plan for the program, with major objectives and sub-objectives clearly set forth, along with specific time tables and budgets. Use of the "Management by Objectives" concept provides all parties with a clear picture of what is to be accomplished within certain time and budget constraints.

Concurrently with the development and implementation of the management plan, individual budgets for each Co-Principal Investigator were developed, and sub-accounts established in the University General Accounting Office. However, all fiscal authority is retained by the Program Director and the Program Manager. A computer program for tracking expenditures and budget balances was installed in the Office of the Program Manager. This program allows the Program Manager to track expenditures and encumbrances on a daily basis. It should be emphasized that this program is for the use of the program staff in the management of the program. The University General Accounting Office remains the official accounting and fiscal organization within the University.

Also, a Test Plan, detailing the laboratory and field tests to be conducted was prepared and furnished to the Contracting Officer's Representative of DOE. The test plan included detailed quality assurance and quality control (QA/QC) protocols.

Beginning January 1, 1994, the Program Manager initiated a monthly News Letter, designed to keep all interested parties aware of developments of interest within the program. The News Letter has proved to be quite successful, particularly in providing information of a non-technical nature.

Quarterly Technical Progress Reports have been prepared and furnished to the Department of Energy on a timely basis. Copies of such reports have been sent to the Co-Principal Investigators and other cooperating parties as a means of keeping them abreast of all technical developments and progress within the program.

Fiscal management of the program has been sound, with overall expenditures within budget projections. Likewise, technical progress has been within established guidelines, although early in the program delays were experienced. Some of these were due to miscommunication between the program and the University's Central Purchasing Office; others were the result of inclement weather which prevented the timely obtaining of combustion residues samples. These difficulties were soon overcome, and at the end of the first year the program was on target fiscally and only slightly behind schedule technically.

SECTION II

ENVIRONMENTAL CHARACTERIZATION

DR. BRADLEY PAUL

CO-PRINCIPAL INVESTIGATOR

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

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ENVIRONMENTAL CHARACTERIZATION

Overview and Purpose

Environmental characterization studies have the purpose of ensuring that any mixture placed underground as part of the demonstration will be stable and resistant enough to toxic and heavy metals release that groundwater would not be harmed even if contact did occur. The tests and analyses to provide this assurance include the following:

1- TCLP tests were performed on at least two samples and a composite of each combustion residue considered for one of the mixes. These tests give an indication of whether any of the candidate materials might exhibit hazardous characteristics for permitting purposes. The tests also provide an indication of product variability and repeatability of the test procedure and subsequent analysis.

2- TCLP tests were performed on all the basic mix proportions first considered for placement as part of the demonstration. These tests indicate whether any of the candidate combinations exhibit synergistic leaching effects that might prove to be an environmental liability.

3- SLP tests were performed on combustion residues considered for one of the mixes. These tests indicate whether any ions complexes or conditions in the natural groundwater most likely to contact the residues might enhance or produce potentially dangerous leaching characteristics.

4- SLP tests were performed on all the basic mix proportions first considered for placement as part of the demonstration. These tests give an indication of whether any interaction between mixed residues and the groundwater might produce problems with enhanced leaching.

Other tests have and are being performed, and will be discussed under future plans. The Technical Progress Report for the period April 1 - June 30, 1994 discussed the TCLP test results for the pneumatic mixes, and only a summary discussion of those results will appear in this report.

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ENVIRONMENTAL CHARACTERIZATION

Modified Synthetic Leachate Procedure

The Synthetic Leachate Procedure (SLP) is a modification of the standard test developed in North Dakota and carrying that name. The leaching medium used in this test is actual groundwater taken from the Peabody mine site and not a medium prepared from standard chemicals and distilled water. The Peabody mine area where the residues are to be placed is dry; however, there are alluvial aquifers above the mine as well as brine bearing units below the mine. It is the near surface aquifers that have potential use by people, industry, or wildlife, and would thus be of greatest concern. Water from these aquifers is the only water that appears to get into the mine, and is thus the most likely leaching medium should the mine ever flood or become a channel for groundwater movement. A mine shaft is located within about 1/2 mile of the residues placement area. When the mine is closed the shaft is to be backfilled with gravel, but groundwater will likely still be able to move down into the mine. A sump located at the base of the mine shaft and collecting seepage from the aquifers above is the source of the leaching medium used in the tests reported here. The leachate produced with this groundwater is believed to approximate the composition of leachate flowing from the residues in the event of flooding. It provides an indication of the water quality that could penetrate down into deep brines, or if the mine developed sufficient pressure, could flow back up into the near surface aquifers. The hydrogeologist on the project team has indicated that it is very unlikely that the mine will develop sufficient pressure to force leachate back up into the surface aquifers.

Summary of Previously Reported Findings

The potential pneumatic backfill mixes, which are composed of fluidized bed combustion byproducts, were investigated by the TCLP for potential environmental problems. The only potential problems found, both from a standpoint of product variability and mix component interactions were:

1- Levels of arsenic and selenium just exceeding class II groundwater standards. Because most tests gave compliance levels of arsenic and selenium, and those that did not were just over the limit, almost any degree of natural attenuation and dilution will assure compliance concentrations in the aquifers or groundwater.

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ENVIRONMENTAL CHARACTERIZATION

2- High concentrations of dissolved calcium. While calcium is not dangerous or regulated, the calcium alone could exceed total dissolved solids limits. Because cations seldom exist alone in solution, it is likely that the sulfate or chloride anions probably present for a charge balance would also violate standards written specifically either for dissolved chloride or sulfate. This issue will be discussed further in terms of additional findings during the last quarter.

TCLP Tests on Hydraulic Mix Components

Placement of two types of mixes is included in this demonstration. Pneumatic mixes will be composed of dry fluidized bed combustion ash mixed with water sprays at the discharge point to induce setting. The hydraulic mix will be composed of scrubber sludge with additions of fly ash and possibly bottom ash and lime to produce better setting strengths. Table 1 shows the TCLP test results for the three potential component materials. The most conspicuous feature of the table is an almost general absence of problem leachate concentrations for almost all elements. It can be observed that none of the materials alone exhibits hazardous leaching characteristics in terms of the RCRA limits. Asterisks are placed beside any number that would be in violation of class II groundwater standards if the TCLP leachate moved directly into the groundwater with no dilution or attenuation.

As with the pneumatic mixes, arsenic and selenium leaching are potential issues. Two considerations dictate that arsenic and selenium will probably not be a problem at the field site.

1- Dilution and attenuation have not been considered, and dilution factors of 5 or less would assure compliance for even the more serious selenium concentrations.

2- Both arsenic and selenium are "noisy" elements for both AA and ICP analytical techniques, which means that concentrations actually present could easily be over-reported. The ICP background equivalent concentration for selenium is in fact 1 ppm. Should issues arise in terms of the hydrogeology of the site, or the permitting process where it is doubted that a factor of 5 dilution and attenuation can be achieved (since actual factors in the field tend to be 100 or more this is unlikely), key concentrations can be cross-checked by graphite furnace and hydride formation, both of which are also available in the SIUC laboratories.

TABLE 1 - TCLP TEST RESULTS - POTENTIAL COMPONENT MATERIALS

	Unit 3 BA	Unit 3 BA	Scrubber Sludge	Scrubber Sludge	Scrubber Sludge	Scrubber Sludge	Scrubber Sludge	Scrubber Sludge	Scrubber Sludge Comp.	Unit 3 FA	Unit 3 FA	Unit 3 FA Composite
	10/15/93	10/13/93	10/14/93	10/15/93	10/15/93	10/15/93	10/15/93	10/13/93	7/28/94	10/18/93	10/12/93	7/28/94
AG	0	0	0	0	0	0	0	0	0	0	0	0
Al	15.58	22.4	0.59	0.4	0.62	0.43	0.01	0.01	0.01	0.24	0	0.53
AS	0.34	0.18	0.13	0.19	0.2	0.26	0.14	0.14	0.14	0.22	0.15	0.34
B	3.6	3.82	0.16	0	0.27	0.12	P.O	0.37	0.37	48.4	38.7	P.O
Bcl	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.07	0.39	0.37	0.37
Be	0	0	0	0	0	0	0.02	0.02	0.02	0.04	0.02	0.02
Ca	162.7	279.9	826.4	P.O	892.8	842.1	1023	0.02	0.02	899	922	1031
Cd	0	0	0	0	0	0	0	0	0	0.03	0.02	0.02
Co	0	0	0	0	0	0	0	0	0	0	0	0
Cr	0.05	0.02	0	0	0	0	0	0	0	0	0.05	2.02
Cu	0.08	0.07	0.02	0.02	0.05	0.01	0.21	0.03	0.03	0.03	0.06	0.04
Fe	1.28	1.34	1.28	1.28	1.28	1.28	0	23	0	23	0	0
Hg												
K												
Mg	1	7	2	0	2	1	0	0	0	0	0	0
Mn	0.66	1.1	.012	0.12	0.12	0.12	0.34	0.18	0.18	0.18	0.34	0.34
Mo	0.01	0.02	0.02	0.05	0	0.03	0.1	1.18	1.18	1.18	1.45	0.57
Na												
Ni	0.04	0.03	0	0.03	0.03	0	0	0	0	0	0	0
Pb	0	0	0	0	0.02	0	0.04	0	0.04	0	0.35	0
Sb	0	0	0	0	0	0	0	0	0	0	0	0

TABLE 1-(CONTINUED)-TCLP TEST RESULTS-POTENTIAL COMPONENT MATERIALS

	Unit 3 BA	Unit 3 BA	Scrubber Sludge	Scrubber Sludge	Scrubber Sludge	Scrubber Sludge	Scrubber Sludge Comp.	Unit 3 FA	Unit 3 FA	Unit 3 FA Composite
	10/15/93	10/13/93	10/14/93	10/15/93	10/15/93	10/13/93	7/28/94	10/18/93	10/12/93	7/28/94
Se	0.07	0.07	0.16	0.08	0	0.11	0.11	0.27	0.17	0.31
Si	24.58	24.35	0.52	0	0.37	0.71	0	1.94	29	0
Ti	0.59	0.87	0	0	0	0				
V	0.01	0.02	0	0	0	0	0.32	0.32	0.46	0.11
Zn	1.17	0.96	0	0	0	0	0	0	0	0

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The second issue asterisked on Table 1 is the boron concentrations. Boron was not indicated as a potential problem with the pneumatic mixes, and in the table 1 test results is also not reported as a problem for scrubber sludge. The two stabilizing and strengthening additives (fly ash and bottom ash), however, both report problem concentrations, which in the case of the fly ash could require dilution and attenuation factors as high as 24.

A third issue raised by the TCLP test results, even though no asterisks appear, is the calcium concentrations. For the pneumatic mixes, the calcium concentrations alone were a violation of the TDS limits for class II groundwater. None of the calcium concentrations for the hydraulic mix components violate TDS limits, however, when charge balancing anions are assumed, TDS violations are again possible. For bottom ash the calcium and accompanying anions should not violate TDS limits, however, bottom ash has existing market outlets and will be a minor component for purposes of this demonstration. Fly ash concentrations are higher, and scrubber sludge calcium concentrations are the highest and most likely to have heavy sulfate anions for charge balance. (The scrubber sludge is composed mostly of calcium sulfate and limestone. Calcium and TDS issues are probably not a problem for reasons discussed later in this report.

TCLP Tests of Preliminary Hydraulic Mix Compositions

Table 2 reports the TCLP results for a series of preliminary hydraulic mix compositions. One of the important considerations is whether leachate concentrations are additive for any of the components and whether there is a synergy that enhances the amount of leaching when the components are mixed. Enhanced leaching would be present when the values reported for the mixes exceed the values for the components of the mix individually. None of the mixes exhibits hazardous characteristics as defined by RCRA. Only in the case of selenium do concentrations reported for the mixes appear higher than for the components individually, and most of these values are within the values reported for fly ash, which is a component of all the mixes.

While enhanced leaching does not create new or expanded problems in the mixes, the mixes continue to exhibit the leaching problems present in the mix components with respect to calcium, boron, arsenic, and selenium. In general, it seems that the three component mixes have higher concentrations of the elements of concern than the two component mixes where bottom ash is omitted.

TABLE 2-TCLP TEST RESULTS-PRELIMINARY HYDRAULIC MIX COMPOSITIONS

	FA+DA+ Sludge 51.4+8.6 +40	FA+BA+ Sludge 48+12 +40	FA+BA+ Sludge 40+20+40	FA+BA+ Sludge 34.3+5.7 +60	FA+BA+ Sludge 32+8+60	FA+BA+ Sludge 26.7+13.3 +60	FA+BA+ Sludge 16+4+80	FA+BA+ Sludge 13.3+6.7 +80	FA+Sludge 11+89	FA+Sludge 33+67	FA+Sludge 50+50
	6/11/94	6/11/94	6/13/94	6/10/94	6/11/94	6/11/94	6/13/94	6/13/94	6/10/94	5/29/94	5/29/94
AG	0	0	0	0	0	0	0.02	0	0	0	0
Al	1.12	0.87	1.76	3.82	1.42	4.24	2.64	5.22	5.43	0	0
AS	0.05	0.24	0.08	0.21	0.3	0.17	0.2	0.16	0.19	0.14	0.01
B	19.1	23.3	12	16.1	P.O	10.4	P.O	6.45	5.8	P.O	6.66
Bcl	0.37	0.34	0.37	0.39	0.37	0.39	0.37	0.39	0.39	0.37	0.37
Be	0.02	0	0.02	0.04	0.02	0.04	0.02	0.04	0.04	0.02	0.02
Ca	889	955	912	688	1007	740	911	740	737	1032	943
Cd	0.02	0.03	0.02	0.03	0.03	0.03	0.02	0.03	0.03	0.02	0.02
Co	0.07	0	0	0	0	0	0	0	0	0	0
Cr	0.1	0.13	0.07	0.13	0	0.11	0.03	0.06	0.04	0	0.08
Cu	0.16	0.1	0.13	0.12	0.12	0.09	0.09	0.08	0.1	0.06	0.06
Fe	0	3.5	0	2.3	0	23	0	23	23	0	0
Hg											
K											
Mg	0	9.1	0	2	0	0	0	0	0	0	0
Mn	1.22	1.32	1	0.33	0.34	0.57	0.34	0.18	0.18	0.34	0.34
Mo	0.51	0.21	0.37	0.22	0.14	0.14	0.16	0.1	0.13	0.25	0.61
Na											
Ni	0	0	0	0.09	0	0	0	0	0.05	0.15	0
Pb	0.03	0	0	0	0.03	0	0	0	0	0.06	0
Sb	0	0	0	0.47	0	0	0	0	0	0	0

TABLE 2-(CONTINUED)-TCLP TEST RESULTS-PRELIMINARY HYDRAULIC MIX COMPOSITIONS

Se	0.21	0.47	0.64	0.29	0.16	0.28	0.25	0.25	0.21	0.19	0.25
Si	17.19	19.91	15.7	14.88	5.32	11.29	4.55	9.47	7.85	0.28	1.57
Ti		0.28									
V	0	0.06	0	0.08	0	0.17	0.01	0.1	0.1	0	0.05
Zn	0.21	0.82	0.23	0.53	0	0.31	0	0.39	0.26	0	0

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Arsenic and selenium levels appear to have a lot of noise and a clear pattern or trend through the mix components is not found. It can be seen, however, that the worst levels of arsenic and selenium leaching occur in three component mixes that are poor in scrubber sludge. The implication for management of arsenic and selenium leaching is that a mix composition with as much scrubber sludge as possible should be selected, and possibly that omitting bottom ash from the mix might be helpful. As will be discussed further in the next section, arsenic and selenium leaching are not likely to be a major problem, and may not be a prime consideration in selection of the final mix.

The calcium concentrations are similar for all the mixes in terms of the likelihood of creating a TDS problem. As will be discussed in the next section, calcium is also unlikely to be a key problem.

Boron leaching is more of an issue with respect to plant toxicity, but for the hydraulic mix components and compositions there are clear implications as to which mixes should be selected. Boron comes mostly from the fly ash, though some is present in the bottom ash as well. As would be expected, the lower the scrubber sludge percentage, the higher the level of boron leaching. Again the three component mixes appear to have higher levels of boron leaching than the two component mixes. Three component mixes need dilution or attenuation factors of 5 to 14 to reach compliance with class II standards. Two component mixes require dilution factors around 3. The most desirable mix from a leachate management standpoint appears to be the 11% fly ash 89% scrubber sludge mix. Since tests by Mine Systems Design (MSD) indicate that a minimum of 25% fly ash is needed to make a paste, small increases in boron leaching is likely to occur. However, even for a 50-50 mix, the boron concentration was only 6.7 ppm. It would appear desirable to avoid the addition of bottom ash to the mix however.

Background Water Quality

Monitoring wells have not yet been installed at the site, however, water seeping from the overlying aquifers into the sump provides some clue as what the background may be. This water was analyzed in preparation for the SLP tests. The concentrations of elements are reported in Table 3, which will be discussed in greater depth in terms of the leaching behavior of the mixes. Several summary observations can be made about background water quality.

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1- The water is generally free of any heavy metals contamination, particularly with respect to class II groundwater standards.

2- The background water is very hard and has extremely high levels of calcium and other alkali metals. It is almost inconceivable that water of this quality would ever have household application and the selection of class II groundwater standards as a comparison criteria for this program is realistic and justified. If one assumes that the TCLP concentrations for the hydraulic mixes and components represents some equilibrium concentration in solution, then it is unlikely that any of the hydraulic mixes will degrade the water quality. The pneumatic mixes, however, may cause some degrading of the water quality in terms of dissolved solids.

3- No boron was found in the background water. Boron is extremely mobile and less likely to be attenuated by factors other than dilution compared to other elements. The fact that the water background appears boron free means that dilution will likely be more rapid and effective. It also indicates that the hydraulic mixes could potentially add an impurity to the groundwater that is not present in the background. Because of the pristine condition of the groundwater with respect to boron, management of boron leaching may be the key environmental issue for the demonstration as it pertains to groundwater quality.

4- There are indications that the "noisy" behavior of elements such as selenium, and arsenic in ICP analysis may be elevating the concentrations reported for these elements. It can be noted that the arsenic concentration for the background water is 0.14 ppm. The selenium concentration is 0.28 ppm. It is an interesting coincidence that the background equivalent concentration (BEC) of selenium is exactly twice that of arsenic. If concentrations of arsenic and selenium in the baseline water are assumed to be entirely BEC effects and are subtracted from the TCLP test concentrations, the implication is that there are no real violations of selenium and arsenic standards for either pneumatic or hydraulic mixes, though arsenic is really leaching out of the material. Alternately, if the baseline water concentrations are interpreted as entirely real with no BEC effects, and if it is assumed that the TCLP concentrations represent an equilibrium leaching concentration, then the implication is that there will probably be no selenium leaching from the materials. Arsenic leaching will still occur and produce class II violations, but a dilution factor of less than one would be sufficient to cure the problem. Thus, considering the reported baseline water quality with respect to arsenic and selenium, these two elements will probably not be a problem at all at the site.

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SLP Test Results and Hydraulic Mix Selection

Table 3 presents the SLP results for the mix components and the mixes. One of the first concerns in examining the new data is whether the change in leaching medium caused new potential problem elements to appear that were not already identified by the TCLP. Quickly scanning the table will reveal that no new problem elements appear, and that as would be expected the TCLP test may be a much harsher leach than would ever occur in the field with real groundwater. The fact that field conditions may be less harsh in leaching than the TCLP may be particularly important in the case of arsenic. While the data in Table 3 reflects considerable noise (as would be expected from the BEC level), there are few cases where arsenic levels exceed class II groundwater standards, and most of the time, from a comparison to the baseline arsenic concentration in the leaching medium, it appears doubtful that arsenic leaches out at all.

Selenium, like arsenic, shows considerable noise in Table 3 and all the reported concentrations exceed class II groundwater standards. It was suggested in the previous section that given the concentration of selenium already reported for the groundwater, it was doubtful that there would be any net selenium leaching. The SLP test results appear to confirm the earlier hypothesis in that there is little indication that selenium is leaching out of the material in concentrations much above that already identified as a background water condition. The same statement can be made about calcium in that there appears to be little net leaching relative to the existing water baseline.

The element for which there is both leaching occurring and creation of class II groundwater violations that did not previously exist is boron. The boron concentrations, like those of calcium, are well above background interferences present in the analytical technique, and may reflect variability of even material from the same container. Again it is apparent that most of the boron leaching is coming from the fly ash component. The three component mixes may have more boron leaching than the two component mixes, but the difference is less distinct than with the TCLP results. Within the two component mixes, results are erratic and appear to be responding to something other than fly ash scrubber sludge ratio. The concentrations of boron are intermediate between the high boron level of the fly ash and the low boron level of the scrubber sludge. Since there is not a synergistic effect on leaching when the two components are mixed, considerations in limiting the total boron available in the system, would again favor the minimum use of fly ash. Tests done by Mine Systems Design (MSD) the subcontractor on the hydraulic placement system, indicate a minimum of 25% fly ash. It is recommended that the hydraulic mix selection should use no more than 25% fly ash and that consideration be given to use of lime as a setting agent to build strength, rather than to the use of bottom ash.

TABLE 3 - SLP TEST RESULTS - HYDRAULIC MIX COMPONENTS AND MIXES

	FEABODSMP	06/28/94	06/29/94	06/18/94	06/18/94	16/18/94	06/14/94	06/16/94	06/16/94	06/25/94	FA+BA+SLDG (13.3+6.7+80)
	FEABODSMP										FA+BA+SLDG (34.3+5.7+60)
pH	8.33	9.9	7.34	7.34	7.21	7.22	7.21	7.22	7.21	0	
Con	8370	7900	9040	9630	8290	8660	8170	27800			
Cl			1900	1900	1900	1800	1800	1800	1900		
F		0	3.5	5.2	3.9	5.4	5.2	1.8			
Ag	0	0	0.15		0	0	0	0.04			
Al	0	0.08	0		1.34	0.18	0.21	0.07			
As	0.16	P.O	0	22.1	0.06	0.12	0.03	0			
B	P.O	0.37	15.8		6.7	13.1	P.O	13.59			
Ba	0.37	0.02	0.39		0.37	0.39	0.37	0.37			
Be	0.02	1060	0.04		0.02	0.04	0.02	0			
Ca	982	0.02	761		957	745	1013	831.1			
Cd	0.02	0	0.03		0.02	0.03	0.02	0.02			
Co	0	0	0.02		0	0	0	0.01			
Cr	0	0.1	0.03		0	0.07	0.03	0			
Cu	0.09	0	0		0.08	0	0.1	0.02			
Fe	0		2.3		0	2.3	0	0			
Hg		36.6									
K	33.9	50	89.9		84.2	36.1	32.7				
Mg	61	0.34	0		0	0	0	0.11			
Mn	0.34	0.14	0.18		0.34	0.18	0.34	4.96			
Ni	0.14	266.9	0.44			0.47	0.43	0.39			0.25

TABLE 3 - (CONTINUED) - SLP TEST RESULTS - HYDRAULIC MIX COMPONENTS AND MIXES

Na	266.9	0	263.1	1000139	272.8		0	227.3	896.7	
NI	0	0	0					0	0	0.17
Pb	0.42	0	0				0	0	0	0
PbG							0	0	0	0
Sb	0	0	0				0.2	0.36	0.19	0.34
Se	0.13	0.2	0.2				5.55	4.42	3.69	5.36
Si	6.36	2.14	3.67							
Tl							0.17	0.21	0.09	0.11
V	0	0	0.22				0	0	0	0.18
Zn	0	0	0							

TABLE 3 - (CONTINUED) - SLP TEST RESULTS - HYDRAULIC MIX COMPONENTS AND MIXES

	FA+BA+SLDG (16+4+80)	FA+BA+SLDG (17.2+2.8+80)	FA+BA+SLDG (11+89)	FA+BA+SLDG (17+83)	FA+BA+SLDG (25+75)	FA+BA+SLDG (33+67)	FA+BA+SLDG (33+67)
	06/25/94	06/25/94	06/16/94	06/14/94	06/14/94	06/14/94	06/14/94
pl			7.17	10.42	10.88	11.34	11.63
Con			8090	7260	7620	7840	8600
Cl			1800	2000	2000	1800	1900
F			3.2	3	3	52	8.8
Ag		0	0.01	0		0	0
Al		0	0	0		0.23	0
As		0.14	0	0.2		0.12	0.24
B	10.1	7.4	10.77	7.1	16.25	4.8	14.1
Ba		0.39	0	0.39		0.37	0.39
Be		0.04	0	0.04		0.02	0.04
Ca		783	823.2	736		964	736
Cd		0.03	0	0.03		0.02	0.03
Co		0	0	0		0	0
Cr		0.01	0.01	0.02		0	0.06
Cu		0.03	0	0.03		0.08	0
Fe		23	1.61	2.3		0	23
Hg		<.001					
K			34.9	33.7		87.4	
Mg		11	33	7		0	0
Mn		0.18	0.22	0.18		0.34	0.18
Mo		0.17	0.22	0.25		0.43	0.46
Na	1472127	1178120	997.6	221.4	229.4	340.5	227.3

TABLE 3 - (CONTINUED) - SLP TEST RESULTS - HYDRAULIC MIX COMPONENTS AND MIXES

Ni	0	0	0	0	0	0	0
Pb	0	0	0	0	0	0	0
PbG						0	0
Sb	0.49	0.01	0	0	0.13	0.36	
Se	0.25	0.19	0.76		4.01	3.15	
Si	0.39	4.78	3.62				
Ti		0					
V	0.28	0	0.19		0.09	0.2	
Zn	0	0	0		0	0	

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SLP Results and Pneumatic Mix Selection

Results of SLP tests done on the pneumatic mixes are found in Table 4. SLP results contrast with previous TCLP results as follows.

1- TCLP tests had indicated the possibility of periodic incidence of arsenic leaching that could exceed class II groundwater (GW) limits. SLP test results indicate that there will probably be no net arsenic leaching and no class II GW violations.

2- Selenium concentrations were somewhat higher in the SLP tests than in the TCLP, however, the background water quality and the quality of the leaching medium already appeared to violate class II GW standards. Not all samples showed net leaching of selenium, and there was no consistent pattern as to which samples would leach, but where selenium leaching did occur it was on balance less than indicated by the TCLP. The class II GW standards for selenium is 50 parts per billion, and concentrations and differences in concentration reported in the tables are within the 1 ppm background equivalent concentration for ICP. It can be said that there are no large problems with selenium leaching, however any choices that must turn upon the question of whether the required attenuation factor is 3, 5, or 0, will require special analytical techniques to clarify the differences.

3- It was indicated earlier that the pneumatic mixes would likely contribute additional calcium to water that already had very high dissolved solids. Table 4 indicates that calcium leaching above that already in solution will occur for mixes that are high in FBC fly ash. If the water at the site is already held to be in violation of class II GW standards, it is possible that the state could require no deterioration in water quality, not even from small amounts of dissolved calcium. None of the proposed pneumatic mixes could be assured of meeting that requirement based on composition. In this case the hydrogeologic investigations section of this project would become crucial in terms of showing how very unlikely it is that leachate from the pneumatic mixes could ever be back-pressured up into the overlying surficial water bearing units.

4- The TCLP tests had indicated that boron leaching should not be a problem with the pneumatic mixes. The SLP test results seem to indicate that boron leaching will occur and will increase with the percentage of FBC fly ash. Levels as high as 4.255 ppm were found in the 80% FBC fly ash mix. It is interesting to compare this concentration to that found in an earlier field demonstration in which SIUC participated. Two million gallons of acid mine water in a pond was neutralized with 160 tons of pure FBC fly ash. The boron in the

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TABLE 4 - (CONTINUED) - SLP TESTS - PNEUMATIC MIXES

Ni		0.07		0		0	0
Pb		0		0		0	0.14
PbG							
Sb		0.03		0		0	0
Se		0.17		0.35		0.54	0.12
Si		0		1.44		0.61	0
Tl							
V		0		0		0.01	0
Zn		0		0.08		0.09	0

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water after the neutralization was about 4.8 ppm. In comparative testing using the ASTM and TCLP procedures on other projects, research at SIUC has indicated that the TCLP test may suppress boron leachate concentrations. The results of this study appear consistent, and it is likely that boron leaching will occur. If the naturally occurring ratio (70-30) of FBC fly ash to bed ash is used, a boron concentration of 2.6 ppm was found. With less than 1:1 dilution this concentration could be brought into compliance with class II GW standards. Since mixes containing some FBC bed ash appear slightly more desirable from an environmental standpoint it is recommended that mixes be proportional to the naturally occurring percentage of fly and bed ash, less any bed ash for which a market is found.

Future Plans

The work done to date is on the preliminary mix proportions. Based on input from the physical strengths section of the characterization work, and the opinions of the materials handling system subcontractors, a series of no more than four final mixes will be chosen to cover both the pneumatic and hydraulic placement area. When these mix selections are submitted, a series of ASTM column leaching experiments will begin. Each final mix will be subject to 16 days of leaching to gain insight into how much of the higher concentrations are just a transient first flush. One of the deficiencies of the shake tests used to date is that they were done on fresh product. Based on other studies done with combustion residues at SIUC the concentrations reported for fresh material generally exceed significantly the concentrations that would be produced on a sustained basis in the field. A second reason for ASTM column studies is that the solid to liquid ratio more closely approximates what will be realized in the field. In parallel with the ASTM column studies on mixes, each mix component will be subject to ASTM column leaching separately. The selected mix components will be acid digested and subject to ICP analysis to determine total composition. The ICP results will be cross-checked against XRF and neutron activation determinations of the total composition. The amount of an element present in a mix forms an upper bound on the net amount of the element that could leach out of the residues over any length of time.

A final test to be performed is the accelerated aging test. The accelerated aging test will involve columns that can be quickly coupled into, and out of a leaching circuit. Leaching will be done with harsh chemicals to accelerate the physical effects of aging. The leaching apparatus will measure pressure drop and flow rate, which can be used to estimate permeability as a function of time. The test columns will also be subject to freeze thaw cycling. The accelerated aging tests will run a period of 6 months to 1 year, depending on

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the timing and coordination with other aspects of the projects technical direction. The exact test procedure will be developed by a team consisting of a mining engineer with experience in the leachate chemistry of combustion residues, a cement chemist, and a chemist. Detailed development of the test has been held until this time to gain insight from the other contractors and participants in this project as to the exact environmental and placement conditions and procedures at the site. The accelerated aging test has as its primary purpose the determination of permeability as a function of time. If sample can be recovered at the conclusion of the test, strength characteristics will also be examined. Some work will be done on the leachate characteristics, though the background concentration in the leaching medium, and the unusually harsh nature of the leaching medium will have to be weighed closely.

Over the next quarter work will be focused most heavily on preparation of the equipment for the accelerated aging test. Some work must wait until the final candidate mixes are selected. The input and recommendations from environmental characterization are included in the summary to this report.

Summary

All combustion residues considered for the hydraulic and pneumatic mixes have been subjected to replicate TCLP testing and to SLP testing. All preliminary mixes originally proposed for this demonstration have been subject to TCLP and SLP analysis. None of the materials or mixes exhibit hazardous characteristics with respect to leachate toxicity, whether the leaching medium is that specified for the TCLP, or the natural groundwater found at the site. Most parameters meet class II groundwater standards without any dilution and attenuation. Only arsenic, boron, calcium, and selenium remain as possible issues. Given the apparent baseline water quality, it is unlikely that any of the mixes recommended by this report will cause any degradation of the groundwater with respect to arsenic or selenium. Pneumatic mixes could in a worst case cause an increase in TDS because of soluble calcium. The apparently existing groundwater quality is already poor enough with respect to TDS that it is unlikely the already existing water would ever be used for any TDS sensitive application. The groundwater is apparently almost boron free, and the leachates from the mixes will probably not be boron free. Boron is very mobile and may not be chemically attenuated, but could easily be diluted. The recommended mixes appear unlikely to require a dilution factor of more than 3. The ASTM column tests are expected to indicate that the potential problem elements reach higher concentrations only in the first flush and are not likely to be a long term problem.

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From an environmental characterization standpoint, none of the proposed mixes should be rejected as environmentally dangerous to water quality. The recommended mixes appear likely to have more minimal leachate concentrations of the four elements identified as potential issues. Recommendations have been made based on leaching behavior of the mixes and known constraints on mix performance identified by other investigators and subcontractors on the project. For the pneumatic mixes, it is recommended that the FBC fly ash and bed ash be mixed in their naturally occurring proportion at the generating station, less any spent bed ash for which a market outlet is found. For hydraulic mixes, it is recommended that the mix contain a minimum of fly ash (25%) with the balance made up of scrubber sludge with a lime setting agent only if necessary to meet the strength requirements. These recommendations should be regarded as preliminary and subject to change based on future test results and constraints found by other investigators and contractors on the project.

SECTION III

**PHYSICO-CHEMICAL CHARACTERIZATION OF
RAW MATERIALS**

DR. N. GHAFOORI

CO-PRINCIPAL INVESTIGATOR

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

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PHYSICO-CHEMICAL CHARACTERIZATION OF RAW MATERIALS

Introduction

The Physico-Chemical Characterization portion of this program is divided into three phases. The first phase addresses the physico-chemical characteristics of the raw materials in accordance with the applicable ASTM standard testing methods. The second phase is intended to identify the optimized mixes which will be used for pneumatic and hydraulic backfills. In the final phase (phase III), an optimum mixture selected from each series of the Phase II will be subjected to a comprehensive laboratory review of short and long-term properties in order to assess their performance. This annual progress report makes the readers aware of the test results, to date, obtained in the Phase I and II of the investigation.

Sources of Raw Materials

Archer Daniel Midland Corp. (ADM) in Decatur, Illinois was the supplier of the FBC fly ash and spent bed. The PCC by-product residues and scrubber sludge were obtained from City Water Light and Power Company in Springfield, Illinois.

Experimental Procedure

Chemical analyses of all by-product residues were conducted in accordance with ASTM C311. Free lime test (ASTM C25) was used to ascertain the presence of unreacted lime in raw materials. Size gradation, including hydrometer test, was conducted using ASTM C33 and D422. A pH analyzer was used to obtain the pH value of the raw materials. Specific gravity and absorption were determined using ASTM C128. Test on organic impurities was conducted using ASTM C40. The remaining tests relevant to the physical properties of the raw materials were carried in accordance with ASTM C311.

Physico-Chemical Properties of FBC By-Product Residues

Table 1 reports on the chemical properties of the FBC spent bed and fly ash. Both of the by-product residues have low pozzolan oxides. As expected a large quantity of unhydrated calcium oxide and sulfur trioxide were found in the FBC residues, thus, contributing to their high heat of hydration and subsequent thermal expansion. The pH values of 12.0 and 12.20 were obtained for the FBC spent bed and fly ash, respectively.

Table 1: Chemical Test Data of FBC Spent Bed and Fly Ash

Chemical Composition	"FBC" Spent Bed	"FBC" Fly Ash	Fly Ash Specifications (ASTM C618)
Silicon Oxide (SiO_2)	9.70	22.10	_____
Aluminum Oxide (Al_2O_3)	3.69	6.80	_____
Iron Oxide (Fe_2O_3)	2.16	6.67	_____
Total ($SiO_2 + Al_2O_3 + Fe_2O_3$)	15.55	35.57	50.0 Minimum, → Class C 70.0 Minimum, → Class F
Sulfur Trioxide (SO_3)	24.42	15.67	50.0 Maximum
Calcium Oxide (CaO)	53.10*	38.70*	Less than 10%, → Class F More than 10%, → Class C
Magnesium Oxide (MgO)	0.88	1.29	_____
Loss on Ignition (LOI)	0.80	5.46	6.0 Maximum
Free Moisture	0.00	0.11	3.0 Maximum
Water of Hydration	2.65	0.71	_____
Total Na_2O	0.16	0.50	_____
Total K_2O	0.39	1.12	_____
Others ($TiO_2 + P_2O_5 + SrO + BaO$)	2.04	0.83	_____
pH	12.00	12.20	_____

* Mostly unhydrated lime

Table 2: Chemical Test Data of PCC Bottom Ash, PCC Fly Ash, and Scrubber Sludge

Chemical Composition	"PCC" Bottom Ash	"PCC" Fly Ash	Scrubber Sludge	Fly Ash Specifications (ASTM C618)
Silicon Oxide (SiO_2)	50.90	55.90	0.45	—————
Aluminum Oxide (Al_2O_3)	13.60	15.40	BDL*	—————
Iron Oxide (Fe_2O_3)	22.20	16.10	BDL*	—————
Total ($SiO_2 + Al_2O_3 + Fe_2O_3$)	86.70	87.40	0.45	50.0 Minimum, → Class C 70.0 Minimum, → Class F
Sulfur Trioxide (SO_3)	0.41	1.15	58.73	50.0 Maximum
Calcium Oxide (CaO)	6.24	5.06	41.00	Less than 10%, → Class F More than 10%, → Class C
Magnesium Oxide (MgO)	0.72	0.78	BDL*	—————
Total Na_2O	1.01	1.48	BDL*	—————
Total K_2O	1.51	1.93	0.02	—————
LOI	1.09	0.58	0.00	6.0 Maximum
Free Lime**	0.07	0.10	0.23	—————
$CaSO_4$ ***	—	—	99.87	—————
$CaSO_3$	—	—	BDL*	—————
$CaCO_3$	—	—	BDL*	—————
pH	10.30	12.26	7.99	—————

* BDL = Below Detection Level

** % of total available lime

*** $CaSO_4$ is in the dihydrate form

— Not tested for

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Physical properties of the FBC spent bed are shown in Table 3. The oven-dry (OD) and saturated surface dry specific gravity were 1.92 and 2.19, respectively. Such a value of specific gravity is typical for lightweight aggregates. The difference between the OD-specific gravity and the SSD-specific gravity is quite high and is indicative of the ability of the FBC spent bed to absorb a large amount of water. The actual absorption was calculated at 14.6%. Based on ASTM C40, no organic impurities were detected in the FBC spent bed. Size distribution of the FBC spent bed is shown in Figure 1.

Physical properties of the FBC fly ash are reported in Table 4. Water requirement, 7-day compressive strength, No. 325 sieve fineness, and autoclave expansion were all outside the limits required by ASTM C618. Size gradation of the FBC fly ash is illustrated in Figure 2.

Physico-Chemical Properties of PCC By-Products

Table 2 presents the chemical properties of the PCC fly ash and bottom ash. Based on ASTM C618 specifications, these by-product residues are type "F", generated by the burning of bituminous coal. Tables 5 and 6 report on the physical characteristics of the PCC bottom ash and fly ash, respectively. Size distribution of the PCC fly ash is documented in Figure 2.

Physico-Chemical Properties of Scrubber Sludge

Chemical characteristics of the scrubber sludge are shown in Table 2. As expected, the main constituent is dihydrate calcium sulfate ($\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$). Its physical properties and size distribution are reported in Table 7 and Figure 2, respectively.

Engineering Properties Of Trial Mixtures (Phase II)

The mechanical properties of two series of the trial mixtures, relevant to hydraulic and pneumatic backfills, were also investigated. The selection of the matrix proportions were based on the physico-chemical characteristics of the raw materials, the expected production level of the by-product residues, and the economical considerations involved in a large-scale operation.

Table 3: Physical Properties of FBC Spent Bed

Fineness Modulus	Specific Gravity (OD)	Specific Gravity (SSD)	Absorption (%)	Organic Impurities
1.80	1.92	2.19	14.60	None

Table 4: Physical Properties of FBC Fly Ash

#325 Sieve Fineness		Specific Gravity	Autoclaved Expansion			Water Requirement		7-day Compressive Strength* (% of Standard Sample)		
Actual	Limit		Unhydrated	Hydrated	Limit			Actual	ASTM	AASHTO
40.2%	Max. 34%	2.61	0.12%	0.00%	0.80%	96.3%	Max. 105.0	61.20%	Min. 75%	Min. 60%

* Standard Sample Has 7-day and 28-day Compressive Strength of 4,567 and 5,647 psi, Respectively.

Table 5: Physical Properties of PCC Bottom Ash

Specific Gravity (OD)	Specific Gravity (SSD)	Absorption (%)	Organic Impurities
1.93	2.00	3.33	None

Figure 1: FBC Spent Bed Grading

Table 6: Physical Properties of PCC Fly Ash

#325 Sieve Fineness		Specific Gravity	Autoclaved Expansion		Water Requirement		7-day Compressive Strength*			28-day Compressive Strength*		
Actual	Limit		Actual	Limit			Actual	ASTM	AASHTO	Actual	ASTM	AASHTO
	Max.					Max.		Min.	Min.		Min.	Min.
16.70%	34%	2.36	0.06%	0.80%	97.5%	105.0	81.8%	75%	60%	92.6%	75%	60%

Table 7: Physical Properties of Scrubber Sludge

Specific Gravity	Water Requirement		7-day Compressive Strength*			28-day Compressive Strength*		
			Actual	ASTM	AASHTO	Actual	ASTM	AASHTO
		Max.		Min.	Min.		Min.	Min.
2.66	134.7%	105.0	41.27%	75%	60%	44.50%	75%	60%

* Repeat as in Table 4

Figure 2: PCC Fly Ash, FBC Fly Ash and Scrubber Sludge Grading

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Experimental Procedure

For both series, heat of reaction (using a thermometer), immediate demolded bulk density or bulk density at testing, nominal and actual moisture, slump and flow (for the mixtures of hydraulic backfill only), penetrometer index (using a pocket penetrometer), and compressive strength (ASTM C39) were evaluated.

Mechanical Properties of Pneumatic Backfill Trial Mixtures

Mixture Design Details

The devised matrix proportions for the pneumatic backfill are shown in Table 8. For each mix proportion, four different nominal moisture; namely: 25, 30, 35 and 40% by weight of total dry solid; were considered.

Mixing Procedure and Sample Preparation

The dry FBC fly ash and spent bed were first mixed in a counter-current mixer for a period of five minutes. The desired amount of tap water was then slowly added. The mixing continued for an additional three minutes until a homogenous matrix was obtained. The blended material was then placed in a clean container and covered with a plastic sheet. A portion of the matrix was then taken to a drying oven to determine its actual moisture content. Temperature was also taken to determine heat of reaction.

Cylindrical molds (3 x 6 inches) were cleaned and oiled prior to usage. A compression machine was used to densify the matrix. The material was compacted in two layers, each at 50 and 25 psi. The samples were then wrapped in a plastic sheet, while remaining in molds, for a period of nearly three days. Once sufficient strength was obtained, they were extruded and placed in a curing chamber (temperature $90 \pm 10^\circ\text{F}$ and $\text{RH} = 90 \pm 10\%$) for different curing periods.

To measure penetrometer index, samples were compacted in beam-shaped metal molds in two layers, each at 25 psi. They were placed in the curing chamber and the indices were obtained at various curing periods, up to four weeks.

Table 8: Trial Mixes for Pneumatic Backfill

FBC* Fly Ash	FBC Bottom Ash
100%**	0%
80%	20%
70%	30%
50%	50%

* Fluidized Bed Combustion

** Percentage of total dry solid

**Table 9: Properties of Various FBC Fly Ash-Spent Bed Mixtures
Compacted at 50 psi Static Pressure**

Mix Proportions Fly Ash-Spent Bed Ratio	Nominal Moisture (%)*	Actual Moisture (%)*	Heat of Reactions (°F)	Average Demolded Density (pcf)	28-Day Compressive Strength (psi)
100-0	25	9.00	160	66.63	37.2
	30	10.42	160	69.75	45.8
	35	14.10	162	73.86	113.7
80-20	25	8.55	168	70.67	11.2
	30	10.27	-	71.38	92.3
	35	15.20	-	76.36	128.4
	40	18.38	-	81.61	239.5
70-30	25	7.45	164	73.37	3.1
	30	12.14	-	75.17	33.9
	35	14.57	-	81.59	137.1
	40	18.03	-	90.67	302.1
50-50	25	7.45	164	73.37	-
	30	11.40	-	78.77	38.9
	35	16.87	-	83.28	74.5
	40	17.71	-	102.84	442.7

- Temperature not measured
- * Percentage of total dry solid

Figure 3: Average Immediate Density of FBC Spent Bed-Fly Ash Mixtures Compacted at 50 psi

Figure 4: Average 28-Day Compressive Strength of FBC Spent Bed-Fly Ash Mixtures Compacted at 50 psi

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Discussion of Results

The nominal and actual moisture content, average demolded density, and 28-day compressive strength of the test samples compacted at 50 psi pressure are shown in Table 9. Heat of reaction of all matrices were fairly uniform at about 160°F. The average demolded density, as demonstrated in Figure 3, ranged from 66.63 to 102.84 pcf. These values improved with an increase in the moisture content and a decrease in the fly ash content of the matrix. The 28-day compressive strength at various moisture content are illustrated in Figure 4. The optimum strengths are achieved at 25, 30, 35 and 40% for the mixtures containing 100, 80, 70 and 50% fly ash. On the whole, an increase in moisture content provided a steady increase of compressive strength. The strength improvement, with an increase in the spent bed content of the mixture, is attributed to a better matrix gradation and acceleration of the cementitious reactions of the FBC fly ash.

Once the mixtures were observed after mixing by the Program Director, it was concluded that 30% moisture content provides a type of matrix that is ideal for the pneumatic placement of the FBC by-product residues. The Program Director also reconsidered the 50 psi densification effort and, instead, decided to use 25 psi pressure. Subsequently, a new set of test samples were prepared at 30% nominal moisture and 25 psi compaction pressure.

The results of the second set of the FBC spent bed-fly ash trial mixtures are presented in Table 10. As shown in Figure 5, a similar trend to that of the mixtures prepared at 50 psi was obtained. Figure 6 reveals that no significant improvement in the immediate density can be expected as static compaction is changed from 25 to 50 psi. The 28-day and 60-day compressive strength values are demonstrated in Figure 7, whereas the strength comparison at 25 and 50 psi compaction pressure are documented in Figure 8. With the exception of the mixtures containing 100% fly ash, the remaining matrices, prepared at 50 psi pressure, displayed a significantly higher 28-day compressive strength as compared to that of mixtures prepared at 25 psi pressure.

The penetrometer index of various FBC spent bed-fly ash mixtures is shown in Table 11. As can be seen, within four days of curing, all mixtures displayed indices in excess of 4.5 tons/ft².

Table 10: FBC Fly Ash-Spent Bed Mixtures Prepared at 30% Moisture Content and Compacted under 25 psi Pressure

Mix Proportion	Immediate Density * (pcf)	28-Day Unconfined * Compressive Strength (psi)	60-Day Unconfined * Compressive Strength (psi)	Percentage Increase in Compressive Strength
100 ¹ - 0 ²	65.98	50.06	57.65	15.16
80 - 20	73.18	15.60	33.75	116.34
70 - 30	73.85	11.61	20.20	74.00
50 - 50	75.24	5.73	24.54	328.27

¹ FBC Fly Ash

² FBC Spent Bed

* Average of 3 cylinders

Figure 5: Immediate Bulk Density of FBC Spent Bed-Fly Ash Mixtures Compacted at 25 psi under 30% Moisture

Figure 6: Comparison of FBC Spent bed-Fly Ash Mixtures Compacted at 25 and 50 psi under 30% moisture

Figure 7: Compressive Strength of FBC Spent Bed-Fly Ash Mixtures Compacted at 25 psi under 30% Moisture

Figure: 8 Comparison of FBC Spent Bed-Fly Ash Mixtures at 25 and 50 psi under 30% Moisture Content

Table 11: Penetrometer Index* for FBC Fly Ash-Spent Bed Mixes Prepared at 30% Moisture Content and Compacted under 25 psi Pressure

Curing Age	100 - 0	80 - 20	70 - 30	50 - 50
1	4.33	4.5	4.5	2.75
4	4.5 ⁺	4.5 ⁺	4.5 ⁺	4.5 ⁺
7	DO	DO	DO	DO
11	DO	DO	DO	DO
14	DO	DO	DO	DO
18	DO	DO	DO	DO
20	DO	DO	DO	DO
25	DO	DO	DO	DO
28	DO	DO	DO	DO

* Averaged values

⁺ Exceed capacity of the apparatus

DO Same as the previous curing period

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Mechanical Properties of Hydraulic Backfill Trial Mixtures

Mixture Design Details

The devised matrix proportions for the hydraulic backfill are shown in Table 12. For each mixture proportion, three different moisture contents; 26, 28 and 30% by weight of total dry solid; were utilized.

Mixing Procedure and Sample Preparation

In order to achieve a uniform matrix consistency, the use of "dried" scrubber sludge was recommended initially. However, as shown in Table 13 and Figures 9 and 10, the flow properties of the "dried" and "as is" matrices having identical moisture content were quite distinct from each other. For this reason, the "as is" scrubber sludge was used throughout the investigation. The "as is" scrubber sludge was tested daily in order to assess the exact moisture content at the time of mixing. The average deviation from the top and bottom of the drum containing "as is" scrubber sludge was about 3%.

The precalculated mixture components, PCC fly ash and FGD scrubber sludge, were placed in an electric counter-current pan and mixed for five minutes. Next, the measured water was poured slowly into the moving pan and the mixing continued for another three minutes. After mixing; temperature, slump and flow tests were conducted. Upon completion of these tests, a portion of the mixture was placed in an oven (60° Celsius) to determine its moisture content. The remaining matrices were used to fill 3 x 6 in. cylindrical molds (for the purpose of evaluating unit weight and compressive strength) and rectangular-shaped pans (for the purpose of obtaining penetrometer index). There was no densification effort involved and the matrix was placed in molds by action of gravity (pouring). After fabrication, the samples, while in molds, were placed in the curing chamber.

Discussion of Results

The nominal and actual moisture content, heat of reaction, flow diameter, slump, density and 28-day compressive strength are shown in Table 14. The heat of reaction was similar to that expected for class "F" fly ash. Figure 11 provides a good description of the effects on slump caused by water content and matrix proportion. As can be seen, the liquid limit was reached rapidly as demonstrated by the sharp increase in slump. It was observed that the 33-67 mix had the largest overall slump. On the whole, for a given moisture content, there were minimal changes in slump of the various mixture proportions.

Table 12: Trial Mixes for Hydraulic Backfill

PCC* Fly Ash	Scrubber Sludge
50%**	50%
33%	67%
25%	75%
17%	83%

* Pulverized Coal Combustion

** Percentage of total dry solid

9+

Table 13: Flowability Comparison of "Dried" and "As Is" Scrubber Sludge

Nominal* Moisture	"Dried" Scrubber Sludge	"As Is" Scrubber Sludge
20%	See Figure 1	See Figure 1
40%	See Figure 2	Slurry**
60%	ND ***	Slurry**

* Percentage of total dry solid

** Very fluid, flowability could not be measured

*** Due to rapid crystallization of scrubber sludge, a flash set was observed.

Figure 9: Flowability Comparison of Dried and "AsIs" Scrubber Sludge for the Mixes Containing 20% Moisture Content

Figure 10: Flowability of Dried Scrubber Sludge for the Mixes Containing 40% Moisture Content

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Figure 12 illustrates the decrease in density with an increase in the scrubber sludge content of the mix. In most cases, the nominal moisture resulted in an increase of the density. It should be noted that the test results may be easily influenced by the matrix consistency and gravity placement. The consistency of the mixed material with water added acted as a semi solid-liquid. When the matrices were poured in the molds for density reading, there were delayed settlements which differed from one mixture proportion to another.

Figure 13 presents 28-day compressive strength of various fly ash-scrubber sludge mixtures prepared at different nominal moisture contents. Higher mixing water, and subsequently higher slump, enhanced water-cementitious content of the mix, resulting in a reduction of compressive strength. Although strength properties improved with an addition of fly ash, the test matrices displayed low strength values. This behavior can be attributed to the lack of densification effort and a delayed cementitious reactions of fly ash in the absence of a catalyst. However, the strength is expected to improve with curing age and, in addition, it is well documented that class F fly ash contributes mainly to long-term strength development of products.

Table 15 documents the pocket penetrometer index of various fly ash-scrubber sludge for different curing ages. As expected, the penetrometer index increased as the fly ash content of the matrix increased.

Figure 11: Slump for Various Fly ash- Scrubber Sludge Mixtures

Figure 12: Density for Various Fly Ash-Scrubber Sludge Mixtures

Table 14: Properties of Various Fly Ash-Scrubber Sludge Mixtures

Scrubber Sludge/Fly Ash Mix Summary				
Mix Proportions Fly Ash- Scrubber Sludge Ratio	Nominal Moisture (%)	Actual Moisture (%)	Heat of Reactions (F)	Flow Diameter (in.)
50-50	26	17.83	90	9
	28	21.67	89	9
	30	29.70	90	20
33-67	26	20.89	88	9.5
	28	22.77	88	9.25
	30	24.02	82	26
25-75	26	21.32	84	9
	28	-	88	8
	30	24.01	80	23
17-83	26	-	82	8.5
	28	22.65	84	8.5
	30	24.67	79	22
Mix Proportions Fly Ash- Scrubber Sludge Ratio	Nominal Moisture (%)	Density (pcf)	Slump (in.)	28-Day Compressive Strength
50-50	26	109.2	3	15.23
	28	110.1	4	14.90
	30	114.8	10.25	11.56
33-67	26	105.9	4	13.44
	28	104.4	4.25	13.12
	30	111.7	11	11.04
25-75	26	100.9	3.5	12.26
	28	101.1	4	10.0
	30	113.3	8.75	9.12
17-83	26	-	3.5	9.98
	28	101.1	3.5	9.10
	30	110.1	10.25	8.0

Figure 13: 28- Day Compressive Strength of Various Fly Ash-Scrubber Sludge Mixtures at Different Nominal Moistures

**Table 15: Penetrometer Index for Scrubber Sludge-
PCC Fly Ash Mixtures***

Days in Curing Chamber	17-83	25-75	33-67	50-50
2	-	-	4.375	4.5 ⁺
3	0.25	0.583	4.5 ⁺	4.5 ⁺
4	0.416	1.083	-	-
6	-	-	DO	DO
9	-	-	DO	DO
10	2.67	3.25	-	-
12	-	-	DO	DO
15	3.0	3.65	-	-
19	-	-	DO	DO
24	3.25	4.0	-	-
28	3.58	3.83	DO	DO

- * All samples at 28% nominal moisture content
- + Exceed capacity of the testing apparatus
- DO Same as the previous curing age results
- Not tested at that day

SECTION IV

DEVELOPMENT OF A MATERIALS HANDLING SYSTEM FOR DRY FGD BY-PRODUCTS

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Background And Statement Of The Problem

Overview of SEEC's Resource Conservation System

Around the globe, economies are growing at an unprecedented rate. While the fruits of economic growth are clear, there are also difficulties that must be dealt with. Among these problems, one of the most difficult is dealing with the tremendous volumes of waste we produce. As waste streams have grown, the cost of dealing with them has skyrocketed. It has also become clear that problems of waste production and disposal are intertwined with other problems, and cannot be solved in a vacuum. Recognizing this reality, SEEC has developed a holistic resource conservation system that focuses on the value of several waste streams that can be used beneficially instead of being disposed of in landfills.

An overview of SEEC's system explains this approach. First, SEEC addresses the handling and transport problems associated with coal ash, biosolids, and other residuals by encapsulating them in SEEC's patented collapsible intermodal containers (CIC's) that can be carried on railcars, trucks, and barges. SEEC uses the same vehicles that carry coal to utilities to back haul valuable by-products of everyday life to coal mines for use in reclamation. SEEC's system can accomplish

this transport without interrupting the front haul of coal, because CICs are rapidly removed from rail cars upon reaching the mine. This allows rail cars to immediately move to the tippie for filling with coal. CICs that were removed can be handled on site, transported some distance and/or staged on site if necessary, and emptied when convenient. When emptied, CICs are collapsed, stacked, and bundled, then loaded onto flat bed trailers or rail cars for transport back for more waste. Because they are collapsed, empty CICs take little room, and can be transported at relatively low cost.

Handling and Transport of Fly Ash

When dealing with fly ash, one of the most difficult problems faced by the industry is handling and transporting the ash in a manner that is reliable, cost-effective, and that adequately protects human health and the environment. Two major problems are often encountered. First, the fine particle size of fly ash causes severe fugitive dust problems unless the ash is contained throughout transport and handling. Second, many ashes are pozzolanic, which is potentially damaging to handling equipment and transport vehicles.

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One way to suppress dust is to wet the ash for transport, but this adds weight, thus increasing transport costs. Wetting also causes pozzolanic ashes to harden, greatly complicating the subsequent cleaning of coal cars or other transport vehicles. Another approach involves spraying a chemical polymer coating over dry ash that has been placed in empty open top rail cars. This method suffers from fugitive dust problems until polymer is in place, requires a costly ash handling system to be constructed at the destination, and leaves residue of ash and polymer in coal cars. A third method involves pelletization of fly ash, but this has met with very limited success because of technical difficulties in creating consistent pellets, and the tendency of pellets to break up (i.e. become dusty) during transport.

Perhaps the most effective current means of fly ash transport uses pneumatic discharge (PD) rail cars or trucks. The key advantage of PD is that ash remains contained during transport. However, pneumatic transport suffers from high cost and poor versatility. SEEC, Inc., has developed a system that provides many advantages found with pneumatic transport, but with greater versatility and often at a lower cost. This system is built around SEEC's CIC, which is hold fly ash, FGD scrubber sludge, and other residuals that can be beneficially used for mine backfilling and reclamation.

Benefits offered by CICs

The ability to transfer CICs between carriers, to move them intermodally, and to use them when necessary for temporary storage represents a significant advantage in terms of convenience and versatility over other ash handling methods. These and other benefits that CICs offer include:

- The CIC is specifically configured to ride in empty coal cars. One CIC rides in each hopper, so either three or four can be placed in hopper cars, or up to five can be carried in flat bottom gondola cars. Thus, SEEC can use the empty back haul of unit coal trains to transport materials, often at a significantly lower cost than for front haul transport (which PD cars and trucks must pay).
- CICs provide short-term storage so that immediate emptying is not necessary. This means that CICs can be removed from rail cars and staged if necessary, allowing the train to be filled with coal without delay, and the CICs to be emptied when most convenient.

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- By virtue of their intermodality, CICs reduce handling of fly ash and other contents, particularly at transfer points between different types of transportation (e.g. transfer from rail to truck).
- CIC's virtually eliminate fugitive dust problems during filling, emptying, storage, and during transport.
- CICs keep weather and possible contaminants out of the ash, so that it remains clean and dry between the utility and the mine.
- CICs provide a liner that keeps ash and other residuals from "contaminating" the coal cars, thereby eliminating the need for cleaning.
- Empty CICs are collapsed and transported back to the utility by truck or rail, whichever makes the overall system more efficient and cost-effective.
- CICs are easily handled and staged using standard equipment, and in fact were designed with advice from consultants experienced in materials handling, container handling, and railcar engineering.
- CICs are constructed of durable materials, with a 10 year life expectancy under continuous use.
- Connects to gravity, pneumatic, or conveying systems for filling and emptying.

Goals And Specific Objectives

The goal of this project is to develop and demonstrate the CIC as a commercially viable coal combustion residues handling and transport system. The specific objectives are as follows:

1. Assess economics of various handling/transport systems (performed by SIU).
2. Develop the CIC as a coal combustion residues handling and transport system.

This will include the following:

- CIC design
 - Overload test
 - Finite element analyses.
3. Field demonstrate the CIC as a residues handling and transport system.
This should include the following:

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- The general approach to dust-free filling and emptying
- This CIC's capabilities for residues containment while staged and during transport.
- The CIC's capabilities for intermodal transport, primarily the use of CICs to carry cargo in triple hopper cars.

4. Develop a health and safety manual for safe operations using CICs.

Research Methodology

Assess Handling/Transport Systems

SEEC is providing a general description of the Company's residues handling and transport system to the SIU project team for their use in completing this task. This will include information about the economics of SEEC's system as well as a general overview of the benefits of our system for coal-burning utilities, coal mines and others.

Develop Coal Combustion Residues Handling/Transport System - CIC Design

SEEC was originally planning to use a collapsible container that could be provided by Amfuel Corporation. SEEC conducted a demonstration test of these containers, beginning during the fall of 1993 and concluding in January of 1994. The results of this test showed that SEEC's system was viable, but also demonstrated several serious shortcomings of Amfuel's container. Among these were it's small size, small ports, and concerns about it's strength, durability, and stability during transport and handling, as well as railroad concerns about carrying a container that does not fit well in their cars. Because of these concerns, SEEC has been working with several partners to develop a container called a CIC that is designed to meet our specific needs.

The material handling and transport system built around the CIC is relatively complex. This is due to the fact that SEEC's system is designed to handle the container instead of the contents. Therefore, one container must serve the purpose of containing coal combustion residues, while being easy to lift by crane, capable of transport in the uneven bottoms of rail cars and flat bed trailers, and quickly and easily filled and emptied. Once such a container is developed, it offers numerous advantages, but many variables must be factored into the design, and the process requires a lot of time and effort. SEEC's approach has been to identify experts in several key industries (rail, crane, and fabric container manufacturing) to work with us as we try to perfect the CIC design.

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Finite Element Analysis

SEEC contracted with Sims Professional Engineers, a rail car engineering firm, to perform computer-aided finite element analysis. This analysis involved simulation of stresses on filled CICs when lifted, and on triple hopper coal cars carrying three filled (20 ton) CICs. Analyses of coal cars included three scenarios, these being in a stationary rail car, during transport, and after repeated impact.

Overload Test

An "overload" test was conducted with the very first CIC that was manufactured. As suggested by it's name, this test consisted of filling the CIC above it's rated capacity, lifting, and looking for signs of undue stress. In addition, this test provided the opportunity to test the CIC's behavior during emptying of wet sand, which was used as weight in the overload test.

The overload test was performed on the site of Greiner Industries, an industrial manufacturing company in Mount Joy, Pennsylvania. Scaffolding was erected and a hopper was attached to the top of the scaffolding, and sand was delivered into the CIC using a belt conveyor. A 130 ton crane supported the CIC via a spreader bar connected to the lifting brackets. The crane operator positioned the CIC so that the bottom was just resting on the ground. The port was positioned under the hopper by securing two eyebolts to the scaffolding and one eyebolt to a fork lift fork that was raised above the CIC.

The CIC was filled with sand, and then water was added maximize the weight in the container. The CIC was then suspended just off the ground for for 15 minutes and examined.

For emptying, the CIC was placed into a frame inverter that was manufactured by Greiner Industries, secured, and the entire unit was slowly inverted using a 135 ton crane. High pressure water was used to wash the sand out of the CIC.

Field Demonstration Handling/Transport System

Our objective is to demonstrate the capability of SEEC's CICs to perform in an active rail back haul system. This demonstration will include filling of CICs with residues, placing CICs into a triple hopper car, transporting CICs by rail, removal of CICs from the coal car, and emptying the residues by inverting the CICs.

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The CIC is approximately nine feet in diameter and ten feet tall. This CIC has about twice the capacity of the container that SEEC had originally proposed using, with a capacity of approximately twenty tons instead of ten tons. Therefore, instead of using 6 ten-ton bladders as we had originally planned, SEEC will only use three CICs for testing and field demonstrating.

As of September 30, 1994, we were well into the planning process for this demonstration, and have tentatively scheduled the week of November 14 for it's completion. The demonstration will be carried out at Illinois Power's Baldwin Station, with the cooperation and assistance of Illinois Power Company, Peabody Coal, and CSX Railroad. The following is a summary of the demonstration as currently planned:

For this demonstration, we will fill the CIC with coal combustion residues using low-cost and low-technology methods. The approach is to tap into existing ash load out at the Baldwin station. SEEC is having a filling station, consisting of a structural support and an interface that connects the CIC port to the ash load out, fabricated for this demonstration. The interface is a rigid steel plate that physically connects to the CIC port, and provide a support that the fly ash spout can be lowered onto for filling with no other physical connection.

The filling station will be carried on a flat bed trailer on the Baldwin Plant site. A CIC will be placed into the structure and connected to the interface, then the entire unit will be moved into position under the ash spout. The CIC will be filled with fly ash, disconnected from the load out, and moved approximately 100 feet to the rail siding where our triple hopper car is waiting. A crane will lift the CICs and place them into an empty hopper.

Because CICs have never been filled with ash, we will fill two CICs and place them into the rail car prior to the "demonstration." This should allow the demonstration of filling the third CIC to go more smoothly than what might otherwise be expected.

After all three CICs have been placed in the car, they will be transported several hundred miles via Illinois Central and Norfolk Southern rail line, and then transported back to the emptying site at a Peabody Coal mine near Baldwin. There, the CICs will be placed into our frame inverter, which was designed to provide a sturdy but inexpensive means to support the CIC while inverting. CICs will be emptied under permit at the coal mine.

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Health and Safety Issues

SEEC's first priority for this project and any commercial system is the health and safety of everyone who works with and around CICs. Therefore, SEEC is developing an operations, safety and repair manual for working with and around CICs and their contents. This manual will be complete prior to first article testing, and all safety rules will be followed during first article testing and this field demonstration.

Issues of particular concern for this project include:

1. Fugitive dust.
2. Dangers of working around 20 ton filled CICs and the heavy equipment needed to handle these containers.
3. Stability of CICs in railcars during transport.

Specific safety rules have not yet been formulated, but every precaution will be taken to prevent injury of any kind during testing and the field demonstration portion of this project.

Results And Discussion

Planning and coordination

The project kickoff meeting was attended by Lonny Baer to coordinate SEEC's activities with those of other project collaborators. Lonny presented two technical briefings to the meeting attendees, reported on SEEC's estimated time lines for task completion, and had discussions about providing more detailed information as the project progresses.

SEEC submitted a test plan with general time lines for completion of specific tasks. The test plan included SEEC's plans for providing information about our system for SIU's economic analysis, SEEC's development of a residue handling/transport system, field demonstration of that system, and SEEC's plans for dealing with health and safety issues.

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Assess Handling/Transport Systems

SEEC provided information about it's system, including information related to the economics of our system and general comparisons with existing alternatives. Based on this and other information, SIU was able to nearly complete this assessment by September 30, 1994.

Develop Fly Ash Handling/Transport System

CIC Design

Accomplishments during the first year of this project included completion of conceptual CIC design (see Figures 1 and 2), and small scale testing using a model CIC and rail car. Specifications of the CIC 640 are shown in Figure 3.

SEEC has worked with Greiner Industries to design and manufacture a prototype frame inverter for testing that will show how CICs behave while being inverted and emptied.

Finite Element Analysis

Results of these analyses indicate that in no cases will the integrity of either the CIC or the rail car be compromised, and all elements of the CICs and rail car are within AAR limits.

Overload Test

Results of this test indicated that CIC integrity remained uncompromised when subjected to weight that exceeded it's rated capacity. This test was performed at the request of, and in cooperation with, the chief engineer at EFC (the manufacturing company). This engineer was satisfied with the results of the overload test, and gave his approval for further tests of CICs using rated loads and no special precautions.

Field Demonstrate Handling/Transport System

No results to date.

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Health and Safety Issues

SEEC is developing a safety and training manual following examples of similar manuals from related industries. SEEC's manual is subject to ongoing revision and updates, but has been largely completed.

Conclusions

SEEC has successfully performed most of it's work as required under this project, including design and preliminary testing of the CIC. SEEC has addressed function and safety issues, and is now prepared to move ahead and demonstrate the CIC as a commercially viable means of transporting and handling fly ash. Good progress has been made in planning this demonstration, completion of which will fulfill SEEC's obligations for this project.

Figure 1

Figure 2

CIC™ 640

Collapsible Intermodal Containers

Notes:*

1. Designed for coal hopper car
CIC dwg. # SC-H-20-F1
2. Material-
rubber coated aramid and
nylon fabric
polyester webbing
3. Port sizes - (3) - 19", 6", 3"
4. Fill/empty sleeve - rubber fabric
5. Heavy-duty steel lifting bracket
6. Anchor patches used for inverting

CAPACITY *		DIMENSIONS (INCHES)*				
Cu. ft.	640	Filled		Empty		
Gallons	4800	Diameter	Height	Length	Width	Height
Cu. Yards	23.7	110	120	130	94	12
Maximum Load Wt.	20 tons					
Weight Empty	800 lbs.					
GENERAL*						
Operating Temp.						
Max.	+160° F					
Min.	-20° F					
Filling Time	3 min.					
Emptying Time	1.5 min.					
Made in America						

- Weatherproof
- Airtight
- 10 year life expectancy
- Durable
- Collapsible
- Environmentally safe
- Easily handled and staged utilizing standard equipment
- Connects to gravity, pneumatic, or conveying systems

* For further details or specifications outside the ranges consult with SEEC, Inc.
Patent and Patent Pending
"SEEC" and "CIC" are trademarks filed for by SEEC, Inc.

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Figure 3

SECTION V

**MATERIAL HANDLING AND SYSTEM
ECONOMICS**

DR. H. SEVIM

CO-PRINCIPAL INVESTIGATOR

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Introduction And Background

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

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 workings of the underground coal mines in Illinois. Materials handling systems are divided into four modules as seen in Figure 1. The first two modules constitutes the Primary Materials Handling Systems where the storage, handling and loading of the residues at the plant site and their transportation from the plant to the mine site are analyzed. The next two modules constitutes the Secondary Materials Handling Systems where the unloading, storage and handling of the residues at the mine site and their transportation to the injection point are analyzed.

To achieve the above objectives, the research team has undertaken the following activities during the first year of the project.

- 1) Conduct literature survey to investigate the state-of-the-art materials handling systems for residue handling and transportation.
- 2) Identify and assess materials handling systems for dry flue gas desulfurization (FGD) residues. For this purpose, in addition to the literature survey, research affiliates, manufacturing companies, and related institutions were contacted and visited mostly during the second and third quarters of the project. Specifically, the following companies were visited for the purpose of collecting cost and operational data and familiarizing with handling and transportation systems:

Figure 1. Modules in Dry FGD Residue Handling and Transportation

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- Illinois Light and Power - Springfield (source of fly ash and scrubber sludge)
- ADM - Decatur (source of FBC fly ash)
- Illinois Central Railroad - Homewood
- Peabody #10 Mine - Pawnee (proposed injection site)
- Continental Cement Co.

In addition to above visits, the following companies and associations have been contacted for the purpose of obtaining literature and information on handling and transportation equipment:

- Association of American Railroads
- Interstate Commerce Commission
- American Trucking Associations
- American Barge Association
- Department of Transportation
- Fuller-Kovako Co., Laffey Equipment Co., Helm Financial Co. (pneumatic materials handling systems)
- Hi-Vac Co., Heil Trailor Co., Beall Trailor Co., Freuhauf Trailer Co., Summit Trailer Co., McBride Mack Sales (various trucks and trailers)
- Dyna Bulk Inc. (trucking co.)

Based on the above investigations, it is found that the transport methods that can be used for dry FGD residues are: belt conveyors, trucks, rail, barge, pipeline, pneumatic conveyors, collapsible intermodal containers, and aerial trams. A summary analysis of all the methods were given in the Technical Progress Report for the Period of January 1-March 31, 1994. Also, it is determined that the economic feasibility of the mode of transportation of the FGD materials to the mine site from an existing plant depends on, among others, the following factors:

- a) Dusting, corrosion, abrasion, sticking, hardening, and heat generation of the material,
- b) Particle size of the material,
- c) Volume of the material,
- d) Transportation distance,
- e) Profile of the haulage road,

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- f) Possibility of backhauling the material,
- g) Possibility of contracting the haulage,
- h) Presence or absence of waterway facilities,
- i) Presence or absence of railway facilities,
- j) Existing handling facilities at the plant and the mine site, and
- k) Remaining life of the operation.

- 3) Evaluate dry residue transportation alternatives from technical and environmental point of view.

A number of handling and transportation alternatives were examined. Among these, pneumatic trucks, pressure differential rail cars, and collapsible intermodal containers were found to be most promising alternatives.

It is noted that tarped rear-dump trucks are being utilized at least in two fly ash disposal operations in Illinois. The major concern with tarped trucks, however, is the fugitive dust during loading and offloading of the residue. Similarly, open hopper cars have been suggested for residue backhaul. Chemical sprays have been suggested to form a crust on top of these cars to prevent fugitive dust while traveling. Even if the formation of a crust is successful, the major drawback with this alternative is that an expensive baghouse is needed both at the loading and offloading points to prevent serious fugitive dust problems. Also, penetration of moisture through the crust, leading to heating or setting of the residue, may cause severe problems. Another alternative that has been considered is the covered hopper cars (grain cars). These cars can be loaded pneumatically or by gravity and necessary precautions can be taken to minimize fugitive dust. However, they require an undertrack bin to dump in at the mine site. Rehandling of the residue from the undertrack bin is not a desired feature. Offloading from these cars into pneumatic trucks is a possibility but the retrofit of pressurizing systems onto these cars may be costly.

- 4) Develop a computer program to conduct economic analyses of the alternatives. This program is briefly described below.

- 5) Conduct economic evaluations of the transportation alternatives that are found to be technically feasible and environmentally sound. The alternatives that have been economically evaluated are:

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- 1) Pneumatic trucks
- 2) Pressure differential rail cars (PD)
- 3) SEEC Inc's collapsible intermodal containers (CIC)

General information for all three alternatives and their operating scenarios are given after the description of the economic evaluation program.

Economic Evaluation Program

The algorithm developed for this purpose is based on the principle of Net Cash Cost (NCC) analysis which is also known as "After-Tax Cost" analysis. The After-Tax analysis is preferred to "Before-Tax" analysis because, as stated by Stermole (1990), "The effects of tax considerations often vary widely from one investment alternative to another, so it generally is imperative to compare the relative economies of investment alternatives on an after-tax basis to have a valid economic analysis."

In each year of the project life, the NCC is calculated using the following equation:

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

The depreciation allowances are calculated based on the Modified Accelerated Cost Recovery System (MACRS). In the developed model, the "constant dollar" approach is preferred to the "escalated dollar" because it allows the use of "today's dollars" for cost estimations assuming that the cost escalations are equal to the inflation rate throughout the project life. This eliminates the complexity of estimating the future cost escalations.

The NCCs calculated for each year are discounted to time "0" at a predetermined discount rate to obtain the Net Present Cost (NPC) of the project. The NPC is subsequently levelized (or annualized) over the project life using the same discount rate. The levelized cost is then divided by the annual amount of residue transported to obtain the cost-per-ton value. Since this is an after-tax value, the "price-per-ton" to be charged is obtained by dividing the "cost-per-ton" by (1 - tax rate). When the price to be charged is inputted to the model, the project becomes a revenue generating project, and the net present value (NPV) of the project becomes exactly equal to "0" securing a rate of return equal to the discount rate.

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General Information For All Three Alternatives

All three alternatives were evaluated between the loading point at the plant site and the injection point at the mine site. With the current available information, it is decided that the economic evaluation of the alternatives should be conducted excluding the pneumatic (or gravity) ash loading systems at the plant site. The reason for their exclusion is that, each transportation alternative requires specialized loading system at the plant, and reliable technical and economic information on these systems have not yet been obtained. The research team has contacted Fuller-Kovako Corporation, an internationally known company in pneumatic systems, for technical assistance on pneumatic and gravity loading and offloading systems. There was a meeting in late August where Dr. Sevim introduced the project to the Fuller-Kovako engineers. They agreed to cooperate and provide the requested information. Once this information is obtained, the pneumatic loading systems will also be incorporated in the total system evaluation.

It is also noted that, at this time, the cost of injection system is not included in the evaluations. The type, size, and the cost of the injection system will be a function of the quantity of the material handled. The cost of the system will be incremental to each alternative, and this will be added to the total cost when reliable cost information is obtained from the co-principal investigators who are in charge of developing the injection systems.

The three alternatives were evaluated at three levels of production rate and three levels of transportation distance to reveal the relative merits of the alternatives with respect to these variables. The selected production rates were; 100,000, 200,000, and 300,000 tons per year for transportation distances of 30, 100, and 200 miles. Consequently, nine runs were made for each alternative. Selection of a range of 30 to 200 miles is due to the fact that in Illinois most of the underground mines that can serve as residue disposal sites will not be more than 200 miles away from the power plants. Also, it has been found that presently annual residue production from any of the Illinois power plant will be less than 300,000 tons.

In the economic evaluation, project life is assumed to be five years for all three alternatives. Any capital cost item that can not be depreciated over the life of the project is assumed to be liquidated at its book value and accounted as a revenue in the fifth year. A discount rate of 12% is used in all three alternatives after conferring with railroad and trucking operators in Illinois.

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For all three alternatives, it is assumed that the first injection site will be 1 mile away from the rail siding or the main road and that the first injection hole is accessible through an existing stabilized road. It is expected that approximately 10,000 tons of residue will be injected through each borehole covering an area of 300 ft in radius. As the injection operation progresses gradually more boreholes will be drilled, and therefore, the initial 1 mile access road will need to be extended. The length of new road needed in a year is a function of the number of boreholes drilled in that year. For example, 10 injection holes are needed in a year to inject an annual amount of 100,000 tons of fly ash. This will require construction of 6000 ft of stabilized road assuming a distance of 600 ft between two adjacent injection holes. This incremental cost of road construction is incorporated in the evaluation of all three alternatives.

Pneumatic Trucks

Pneumatic trucks, also referred to as bulk tank trucks, are generally used for pneumatic handling of low density dry flowable powder and granules, including plastics and food grade commodities. However, they are also used in the handling of denser materials, such as cement, limestone, and coal combustion residues successfully.

A sketch of a typical pneumatic truck is shown in Figure 2. It is comprised 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 couples to the gate on top of the vessel. The vessel is air tight when the lids of the gates are closed. During offloading, the material inside the vessel is pressurized by the blower (14-15 psig), and the material flows through the piping below the vessel due to pressure difference.

The pneumatic truck operation is schematically shown in Figure 3. These trucks will be loaded from the fly ash bin of the plant either pneumatically or by gravity. They will deliver the ash directly to the injection point at the mine site. There, the blower mounted on the truck will supply the necessary pressure to offload the fly ash into the injection hopper. These trucks are approximately 25 tons in capacity and can offload in about 25 minutes (Freitag et al., 1991).

TYPICAL HOPPER TYPE DRY BULK SEMITRAILER

Figure 2. Typical Hopper Type Dry Bulk Semitrailer

Figure 3. Dry FGD Residues Transportation by Pneumatic Trucks

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The number of trucks in the fleet is a function of the amount of ash to be transported, the distance between the plant and the disposal site, the number of working days per week, the number of shifts per day, and the number of hours per shift. The fleet is sized in such a way that the probability of delivering the daily fly ash production is at least 85%. In the calculation of this probability, it is assumed that the truck availability is 90%. For instance, in a case where four trucks are necessary to deliver the daily fly ash production, the size of the fleet is calculated as follows:

1) What is the probability of four trucks being in service at all times? Using the Binomial probability calculation we obtain 0.656:

$$P(x = 4) = \binom{4}{4} (0.9)^4 (0.1)^0 = 0.656$$

It is clear that four trucks are not sufficient to have an assurance of 85%. Therefore, one must try a fleet of 5 trucks next, and calculate the probability that at least 4 of them are in service. Again, using the Binomial probability calculation we obtain 0.92:

$$P(x \geq 4) = \binom{5}{4} (0.9)^4 (0.1)^1 + \binom{5}{5} (0.9)^5 (0.1)^0 = 0.92$$

Hence, the fleet must have 5 trucks.

Table 1 shows the input data which is composed of three categories: (1) Operational data, (2) Capital Cost items, and (3) Operating cost items. As seen in this table, for 100,000 tons per year production and 30 miles of transportation distance, the truck fleet is composed of 3 units (number of trucks proposed). Also, the total capital cost is \$360,000, and the operating cost per ton is calculated to be \$4.38. When these values are inputted into the economic evaluation model, along with other data as listed in Table 2, we obtain an after-tax cost per ton of \$3.31.

Table 2. Input Data for Pneumatic Truck Alternative (30 miles - 100,000 tons case)

Project Life (years)	= 5
Effective Tax Rate (%)	= 40
Discount Rate (%)	= 12
Operating Cost (\$/ton)	= 4.38
Capital Cost (\$)	= 360,000 (3 trucks)
Residue Production (tons/year)	= 100,000
Depreciation Life of Trucks (years) ..	= 5

Table 1. Input Data for the Economic Evaluation of the Pneumatic Truck Alternative

Distance between the power plant and the mine (miles)	30	100	200	30	100	200	30	100	200
Annual fly ash production (tons)	100000	100000	100000	200000	200000	200000	300000	300000	300000
Project life	5	5	5	5	5	5	5	5	5
Number of working days per year	300	300	300	300	300	300	300	300	300
Area of new road to be constructed(sq.yd)	14000	14000	14000	28000	28000	28000	42000	42000	42000
Truck capacity (tons)	24	24	24	24	24	24	24	24	24
Availability of trucks	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90
Truck loading time (minutes)	5	5	5	5	5	5	5	5	5
Truck unloading time (minutes)	30	30	30	30	30	30	30	30	30
Loaded travel speed (miles per hour)	50	50	50	50	50	50	50	50	50
Empty travel speed (miles per hour)	55	55	55	55	55	55	55	55	55
Number of front tires	10	10	10	10	10	10	10	10	10
Number of rear tires	8	8	8	8	8	8	8	8	8
Front tire replacement frequency (miles)	75000	75000	75000	75000	75000	75000	75000	75000	75000
Rear tire replacement frequency (miles)	150000	150000	150000	150000	150000	150000	150000	150000	150000
Fuel consumption (miles per gallon)	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Fuel cost per gallon	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10
Purchase price of the tractor including sales tax	45000	45000	45000	45000	45000	45000	45000	45000	45000
Purchase price of the tank trailer including sales tax	70000	70000	70000	70000	70000	70000	70000	70000	70000
Purchase price of the blower including sales tax	5000	5000	5000	5000	5000	5000	5000	5000	5000
Total purchase price of tractor & trailers	120000	120000	120000	120000	120000	120000	120000	120000	120000
Tire cost (each)	325	325	325	325	325	325	325	325	325
Insurance cost per tractor and trailer	7500	7500	7500	7500	7500	7500	7500	7500	7500
Licence cost per tractor and trailer	2222	2222	2222	2222	2222	2222	2222	2222	2222
Annual highway tax per tractor and trailer	550	550	550	550	550	550	550	550	550
Maintenance cost per mile	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14
Hourly wage for truck drivers including fringe benefits	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00
Hourly charge of contracted labor at the plant	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00
Overhead cost (% of other operating costs)	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
Cost of road construction(\$/sq.yd)	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Truck Cycle time (minutes)	103.73	284.09	493.18	103.73	284.09	493.18	103.73	284.09	493.18
Number of shifts per day	1.00	1.00	1.00	1.00	1.00	1.00	2.00	2.00	2.00
Duration of each shift (hours)	10.00	10.00	10.00	10.00	10.00	10.00	8.00	8.00	8.00
Number of trips per day	13.89	13.89	13.89	27.78	27.78	27.78	41.67	41.67	41.67
Number of trucks required	2.40	6.11	11.42	4.80	12.23	22.83	4.50	11.48	21.41
Number of trucks proposed	3.00	7.00	13.00	6.00	14.00	27.00	6.00	16.00	29.00
Number of drivers required	3.00	7.00	13.00	6.00	14.00	27.00	12.00	32.00	58.00
Minimum # of trucks imposed for system's availability calc.	2.00	6.00	11.00	5.00	12.00	23.00	5.00	14.00	25.00
System's availability	0.97	0.85	0.87	0.89	0.84	0.87	0.89	0.79	0.84
OPERATING COST									
Insurance	22500	52500	97500	45000	105000	202500	45000	120000	217500
Road construction cost	70000	70000	70000	140000	140000	140000	210000	210000	210000
Licences for tractors and trailers	6666	15554	28888	13332	31108	59994	13332	35552	64438
Highway tax	1650	3850	7150	3300	7700	14850	3300	8800	15950
Tire replacement	15167	50556	101111	30333	101111	202222	45500	151667	303333
Driver wages	192153	432153	792153	384306	884306	1644306	812458	1572458	2820458
Fuel	55000	183333	366667	110000	366667	733333	165000	550000	1100000
Maintenance	35000	116667	233333	70000	233333	466667	105000	350000	700000
Subtotal	398135	924812	1695800	796271	1849225	3463872	1199590	2998477	5431680
Overhead @10% of the subtotal	39814	92481	169580	79627	184922	346387	119959	299848	543168
Total	437949	1017074	1865480	875898	2034147	3810259	1319549	3298325	5974848
CAPITAL INVESTMENT									
Operating cost per ton	4.38	10.17	18.66	4.38	10.17	19.05	4.40	10.99	19.92

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The output corresponding to this run is shown in Table 3. From this table, the calculation of NCC given in Equation (1) can be verified step by step. The capital gain (or loss) calculation shown at the bottom of the table pertains to the undepreciated capital cost items at the end of the project life. For this case, it amounted to \$21,000 and it is added to the NCC of the fifth year as a revenue.

The price to be charged to transport one ton of fly ash from the plant to the injection site is calculated to be \$5.52. If this value is re-entered into the economic evaluation program as the price per unit, the Net Present Value (and not Net Present Cost any more since we have revenue) of the project will be exactly equal to 0. To demonstrate this concept, the program was rerun for this case and the output is listed in Table 4.

The outcomes of nine runs corresponding to the nine combinations of amount and distance for the truck alternative are listed in Table 5 and graphically shown in Figure 4. As seen in this figure, the price increases very steeply as the distance of transportation increases. This is due to the fact that more trucks are needed in the fleet to cope with the increased cycle time. Also, the fact that all three price lines overlap at all distances indicate that the price is insensitive to the tonnage transported.

Figure 4. Prices to be Charged in the Pneumatic Truck Alternative

**TABLE 3. ECONOMIC EVALUATION OF PNEUMATIC TRUCK
ALTERNATIVE (30 MILES - 100,000 TONS CASE)**

Year	0	1	2	3	4	5
Revenue...	0.	0.	0.	0.	0.	0.
-Oper. Costs....	0.	-438.	-438.	-438.	-438.	-438.
-Depreciation...	0.	-72.	-115.	-69.	-41.	-41.
Taxable Income..	0.	-510.	-553.	-507.	-479.	-479.
-Tax.....	0.	204.	221.	203.	192.	192.
Net Income.....	0.	-306.	-332.	-304.	-288.	-288.
+Depreciation...	0.	72.	115.	69.	41.	41.
-Capital Cost...	-360.	0.	0.	0.	0.	0.
Net Cash Flow...	-360.	-234.	-217.	-235.	-246.	-246.

CAPITAL GAIN (OR LOSS) COMPUTATION

Salvage Value.....	21.
Book Value.....	-21.
Capital Gain (or Loss).....	0.
Tax Liability.....	0.
After-Tax Capital Gain (or Loss).....	0.
Book Value.....	21.
After-Tax Cash Flow Due to Cap. Gain (or Loss) ..	21.

AFTER-TAX NET PRESENT VALUE (\$1000).....=	-1193.
AFTER-TAX ANNUAL EQUIVALENT COST (\$1000).....=	-331.
AFTER-TAX COST PER TON	-3.31
BEFORE-TAX PRICE	5.52

**TABLE 4. ECONOMIC EVALUATION OF PNEUMATIC TRUCK
ALTERNATIVE (30 MILES - 100000 TONS CASE) WHEN PRICE TO
BE CHARGED IS CONSIDERED**

Year	0	1	2	3	4	5
Revenue...	0.	552.	552.	552.	552.	552.
-Oper. Costs....	0.	-438.	-438.	-438.	-438.	-438.
-Depreciation...	0.	-72.	-115.	-69.	-41.	-41.
Taxable Income..	0.	42.	-1.	45.	72.	72.
-Tax.....	0.	-17.	0.	-18.	-29.	-29.
Net Income.....	0.	25.	-0.	27.	43.	43.
+Depreciation...	0.	72.	115.	69.	41.	41.
-Capital Cost...	-360.	0.	0.	0.	0.	0.
Net Cash Flow...	-360.	97.	114.	96.	85.	85.

CAPITAL GAIN (OR LOSS) COMPUTATION

Salvage Value.....	21.
Book Value.....	-21.
Capital Gain (or Loss).....	0.
Tax Liability.....	0.
After-Tax Capital Gain (or Loss).....	0.
Book Value.....	21.
After-Tax Cash Flow Due to Cap. Gain (or Loss) ..	21.

AFTER-TAX NET PRESENT VALUE (\$1000)... = 0.
DCFRROR (%) = 12.00

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Table 5. Prices to be Charged in Pneumatic Truck Alternative (\$/Ton)

Amount (tons/year)	Distance (miles)		
	30	100	200
100,000	5.52	12.83	23.63
200,000	5.52	12.83	24.17
300,000	5.17	13.02	23.60

Pressure Differential Cars

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

In this alternative, there will be three sets of rail cars; one set at the power plant, one set in transit, and one set at the mine site. The sequence of operation is shown schematically in Figure 5. The first set of cars is filled pneumatically (or by gravity) at the plant. The second set of cars, which has already been filled and attached to the empty coal unit train, travels to the mine. The third set at the mine site is 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 pushed by a small shunting locomotive and attached to the end of the unit train. Subsequently, the train leaves for the plant. Meanwhile, the full PD cars are pulled in the siding by the shunting locomotive and the process of offloading into pneumatic truck(s) starts. The pneumatic trucks shuttle between the PD cars and the injection point. The number of shuttling trucks is a function of the loading time from the PD cars, traveling time to the injection point, offloading time into the injection hopper, and traveling time back to the siding.

When the coal train arrives at the plant, the empty PD cars are demurred and parked at the junction of the siding. Then, the train pulls over the undertrack bin and dumps the coal. After coal dumping is completed, the full PD cars which have been waiting at the

Figure 5. Dry FGD Residues Transportation by PD Cars

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siding are pushed by a shunting locomotive and attached to the end of the empty coal train. Subsequently, the train leaves for the mine. Meanwhile, the empty PD cars are pulled in the siding by the locomotive and the fly ash loading process restarts.

Table 6 shows the input data for all the cases of PD car alternative. As seen in this table, the number of cars in all nine cases is calculated to be 60. For higher tonnages and longer distances, the need for more cars is compensated by more frequent unit train trips, increased number of shifts, and increased number of offloading pneumatic trucks shuttling between the PD cars and the injection point. Also in this table, a capital cost of \$250,000 is given for the rail siding at the mine site. It is assumed that the siding at the power plant already exists, hence, its cost is not accounted.

The outcomes of nine runs corresponding to the nine combinations of amount and distance for the PD cars alternative are listed in Table 7 and graphically shown in Figure 6. As seen in this figure, the prices remain fairly insensitive to the distance but very sensitive to the amount.

Table 7. Prices to be Charged in PD Car Alternative (\$/Ton)

Amount (tons/year)	Distance (miles)		
	30	100	200
100,000	21.17	21.35	22.45
200,000	13.15	13.27	14.37
300,000	9.62	9.80	10.90

Collapsible Intermodal Containers (CICs)

These containers are basically military type rubber fuel cells. The ones that are envisioned for ash transportation are 120 inches high with a diameter of 110 inches and a loading gate of 19 inches as shown in Figure 7. Assuming a bulk density of 60 lbs per cu. ft. for ash, the capacity of a CIC is approximately 22 tons. These containers are said to be extremely durable and provide for encapsulated transport eliminating fugitive dust problem. The prototype CIC for ash transport has been going through a number of engineering modifications so that they will be handled safely and efficiently.

Table 6. Input Data for the Economic Evaluation of the PD Car Alternative

Distance between the power plant and the mine(miles)	30	100	200	30	100	200	30	100	200
Annual fly ash production(tons)	100000	100000	100000	200000	200000	200000	300000	300000	300000
Distance between the rail siding and the inverter(miles)	1	1	1	1	1	1	1	1	1
Area of new road to be constructed(sq.yd)	14000	14000	14000	28000	28000	28000	42000	42000	42000
Number of working days per year	260	260	260	260	260	260	260	260	260
Number of working days per week	5	5	5	5	5	5	5	5	5
Project life.	5	5	5	5	5	5	5	5	5
Pneumatic truck capacity(tons)	25	25	25	25	25	25	25	25	25
Rail car capacity(tons)	100	100	100	100	100	100	100	100	100
Pneumatic truck cycle time(minutes)	60	60	60	60	60	60	60	60	60
Duration of each shift(hours)	8	8	8	8	8	8	8	8	8
Number of rail cars per unit train	20	20	20	20	20	20	20	20	20
Fuel consumption rate(gal/hour)	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5
Trucks maintenance cost rate(%of purchase price)	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02
Overhead cost rate(%of operating cost)	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Purchase price of pneumatic trucks including sales tax	120000	120000	120000	120000	120000	120000	120000	120000	120000
Tire cost per truck mile(\$/truck/mile)	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06
Hourly charge of contracted labor at the plant site	35	35	35	35	35	35	35	35	35
Hourly wage for pneumatic truck drivers	27	27	27	27	27	27	27	27	27
Fuel cost per gallon	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1
Length of rail siding (yard)	440	440	440	440	440	440	440	440	440
Rail roadcharge per ton per mile	0.0947	0.03	0.02	0.089	0.0277	0.0189	0.0758	0.0243	0.0172
Insurance cost rate(1% of capital cost)	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Cost of road construction(\$/sq.yd)	5	5	5	5	5	5	5	5	5
Total number of truck trips per shift per truck	8	8	8	8	8	8	8	8	8
Total number of trips per shift	16	16	16	31	31	31	24	24	24
Number of pneumatic trucks required per shift	2	2	2	4	4	4	3	3	3
Number of pneumatic rail cars required per trip	20	20	20	20	20	20	20	20	20
Total number of PD cars required for the operation	60	60	60	60	60	60	60	60	60
Time to fill a PD car (hours)	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Total number of shifts per week	5	5	5	5	5	5	10	10	10
Number of shifts per day	1	1	1	1	1	1	2	2	2
Total number of truck drivers required per shift	2	2	2	4	4	4	3	3	3
Number of unit trains per two weeks	2	2	2	4	4	4	6	6	6
CAPITAL INVESTMENT									
Pressure differential rail cars \$75,000 a piece	4500000	4500000	4500000	4500000	4500000	4500000	4500000	4500000	4500000
Pneumatic trucks	240000	240000	240000	480000	480000	480000	360000	360000	360000
Rail siding (\$570/yard)	250800	250800	250800	250800	250800	250800	250800	250800	250800
Total capital investment	4990800	4990800	4990800	5230800	5230800	5230800	5110800	5110800	5110800
OPERATING COST									
Road construction cost.	70000	70000	70000	140000	140000	140000	210000	210000	210000
Rail road charge	284100	300000	400000	534000	554000	756000	682200	729000	1032000
Insurance	49908	49908	49908	52308	52308	52308	51108	51108	51108
Operators wage	121420	121420	121420	242840	242840	242840	364260	364260	364260
Tire cost	499	499	499	998	998	998	748	748	748
Fuel cost	11440	11440	11440	22880	22880	22880	17160	17160	17160
Maintenance	99816	99816	99816	104616	104616	104616	102216	102216	102216
Overhead cost	63718	65308	75308	109764	111764	131964	142769	147449	177749
Total operating cost	630901	648391	758391	1067406	1089406	1311606	1360461	1411941	1745241
Operating cost per ton	6.31	6.48	7.58	5.34	5.45	6.56	4.53	4.71	5.82

Figure 6. Prices to be Charged in the PD Car Alternative

FRONT

Figure 7. Front View of Collapsible Intermodal Container

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The sequence of operation is shown schematically in Figure 8. The coal train arrives at the plant and offloads in to the undertrack bin. When all the coal cars are emptied, the empty CICs are mounted into the rail cars one by one with the help of a crane. There are four CICs in each car, each occupying one of the four bays of the car. It is estimated that mounting a CIC in a coal car will take approximately one minute. The mounted CICs are then filled pneumatically with fly ash. After all the CICs are filled, the train leaves for the mine.

At the mine site, an overhead crane lifts the CICs, one by one, and places them on a concrete ground in two rows. The overhead crane is on rubber tires and travels along the rail track looking inside the hopper cars. 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 which are stacked in two rows along the rail road are then lifted again by the overhead crane one by one and placed on a tote trailer. The tote trailer takes one CIC at a time to the injection point. At the injection site, the CIC is placed into a special inverter and the fly ash is offloaded into the injection system hopper. The empty CIC is placed back on the tote trailer and transported to the rail site where it is stored in a suitable area. After collecting 36 empty CICs, they are loaded into a truck by a forklift and transported to the power plant. In this operation, the number of CICs, tote trailers and trucks are selected in such a way that when the train arrives at the plant there are enough empty CICs for another round of ash delivery. With the arrival of the coal train and reloading of CICs, a new cycle starts.

Table 8 shows the input data for all the cases of CIC alternative. It is calculated that 88 CICs are needed in each trip to the mine. However, to ensure that there is always 88 empties at the plant site when the train arrives, 17 more CICs are added to the system to make a total of 105 CICs. For higher tonnages and longer distances the need for more CICs is compensated by scheduling more frequent unit train trips, by increasing the number of tote trailers, and the trucks transporting the empty CICs.

The outcomes of nine runs corresponding to the nine combinations of amount and distance for the CIC alternative are listed in Table 9 and graphically shown in Figure 9. As seen in this figure, similar to PD car alternative, the prices remain fairly insensitive to distance but very sensitive to amount. The reduction in price is more pronounced for the increase in amount from 100,000 to 200,000 tons when compared for the increase from 200,000 to 300,000 tons.

Figure 8. Dry FGD Residues Transportation by CICs

Table 8. Input Data for the Economic Evaluation of the CIC Alternative

Distance between the power plant and the mine (miles)	30	100	200	30	100	200	30	100	200
Daily fly ash production (tons)	100000	100000	100000	200000	200000	200000	300000	300000	300000
Length of new road to be constructed(sq.foot)	14000	14000	14000	28000	28000	28000	42000	42000	42000
Distance between the rail and the injection site (miles)	1	1	1	1	1	1	1	1	1
Project Life	5	5	5	5	5	5	5	5	5
Number of working days per year *****	260	260	260	260	260	260	260	260	260
Number of unit trains per two weeks	2	2	2	4	4	4	6	6	6
Road charge per ton per mile(\$)	0.0947	0.03	0.02	0.089	0.0277	0.0189	0.0758	0.0243	0.0172
Number of hours per shift	8	8	8	8	8	8	10	10	10
Equipment maintenance cost (2% of capital cost)	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02
Number of shifts per week selected	5	5	5	5	5	5	5	5	5
Number of shifts per day	1	1	1	1	1	1	1	1	1
Number of contracted trailer to take back empty CIC's to plant	1	1	1	2	2	2	2	2	2
Trailer capacity (tons)	22	22	22	22	22	22	22	22	22
Trailer capacity (tons)*****	25	25	25	25	25	25	25	25	25
Circle time of tote trailer	26	26	26	26	26	26	26	26	26
Number of hopper cars that make a unit train	22	22	22	22	22	22	22	22	22
Total number of CICs emptied per shift	18	18	18	18	18	18	23	23	23
Number of CIC's required for the operation.	105	105	105	105	105	105	105	105	105
Number of tote trailers required	1	1	1	2	2	2	3	3	3
Number of MI-JACK cranes required	1	1	1	1	1	1	1	1	1
Number of boom cranes required	1	1	1	1	1	1	1	1	1
Number of inverters required	1	1	1	1	1	1	1	1	1
Total number of tote trailer operators required	1	1	1	2	2	2	3	3	3
Total number of boom crane operators required	1	1	1	1	1	1	1	1	1
Total number of injection system operators required	1	1	1	1	1	1	1	1	1
Purchase price of CIC	15000	15000	15000	15000	15000	15000	15000	15000	15000
Purchase price of boom crane at plant	200000	200000	200000	200000	200000	200000	200000	200000	200000
Purchase price of tote trailer	60000	60000	60000	60000	60000	60000	60000	60000	60000
Purchase price of inverter	600000	600000	600000	600000	600000	600000	600000	600000	600000
Purchase price of a MI-JACK crane.	325000	325000	325000	325000	325000	325000	325000	325000	325000
Purchase price of a Forklift.									
CAPITAL INVESTMENT									
Construction of a storage silo at the plant site(2,000tons)	400000	400000	400000	400000	400000	400000	400000	400000	400000
Storage and staging area at the mine site(65,000square feet)	300000	300000	300000	300000	300000	300000	300000	300000	300000
MI-JACK crane	325000	325000	325000	325000	325000	325000	325000	325000	325000
Boom-cranes	200000	200000	200000	200000	200000	200000	200000	200000	200000
Tote trailers	60000	60000	60000	120000	120000	120000	180000	180000	180000
Forklift to load empty CIC's at the mine.	35000	35000	35000	35000	35000	35000	35000	35000	35000
Inverter	600000	600000	600000	600000	600000	600000	600000	600000	600000
CICs	1575000	1575000	1575000	1575000	1575000	1575000	1575000	1575000	1575000
Total capital investment	3495000	3495000	3495000	3555000	3555000	3555000	3615000	3615000	3615000
Cost of road construction (\$/sq.yd)	5	5	5	5	5	5	5	5	5
Fuel consumption (gallons per hour per unit of equipment)	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5
Fuel cost per gallon	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1
Hourly wage for equipment operators (including 35 % fringe)	27	27	27	27	27	27	27	27	27
Hourly charge of contracted labor at the plant site (4 hrs per train)	35	35	35	35	35	35	35	35	35
Hiring charges for taking back empty CIC's(\$/mile for 35pcs)	5	2	2	5	2	2	5	2	2
OPERATING COST									
Road construction cost	70000	70000	70000	140000	140000	140000	210000	210000	210000
Annual railroad backhaul charge	284100	300000	400000	534000	554000	756000	682200	729000	1032000
Insurance cost @1% of capital cost	34950	34950	34950	35550	35550	35550	36150	36150	36150
Operator wages	231920	231920	231920	295360	295360	295360	443040	443040	443040
Hiring cost of trailer to carry back empty CIC's	18900	25200	50400	37800	50400	100800	56700	75600	151200
Fuel	22880	22880	22880	28600	28600	28600	42900	42900	42900
Maintenance @2% capital investment * # of shifts per day	69900	69900	69900	71100	71100	71100	72300	72300	72300
Subtotal	732650	754850	880050	1142410	1175010	1427410	1543290	1608990	1987590
Overhead @10% of subtotal	73265	75485	88005	114241	117501	142741	154329	160899	198759
Total	805915	830335	968055	1256651	1292511	1570151	1697619	1769889	2186349
Operating cost per ton	8.06	8.30	9.68	6.28	6.46	7.85	5.66	5.90	7.29

Figure 9. Prices to be Charged in the CIC Alternative

Figure 10. Prices for Transporting Dry FGD Residues by Pneumatic Trucks, CICs, and PD Cars

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Table 9. Prices to be Charged in CIC Alternative (\$/Ton)

Amount (tons/year)	Distance (miles)		
	30	100	200
100,000	19.10	19.35	20.73
200,000	11.90	12.08	13.47
300,000	9.47	9.72	11.10

Conclusion

An overall comparison of all three alternatives can be made with the use of Figure 10 which is formed by consolidating Figures 4, 6, and 9. The message from this figure is that, up to about 70 miles, pneumatic trucks can do the job at a lower price than PD cars and CICs. When the production is 300,000 tons, CICs and PD cars can do the job at the same price at all distances. However, when the production is less than 300,000 tons, CICs are more advantageous than PD cars at all distances. This advantage is more pronounced for 100,000 ton production than it is for 200,000 tons.

Plans For The Next Quarter

In the next period, the following tasks will be undertaken:

- 1) Sizing and the costing of the pneumatic loading and unloading systems and silos in cooperation with the Fuller-Kovako Corp.,
- 2) Refinement of costs and system configurations,
- 3) Economic evaluation of the alternatives using the equipment leasing option rather than the buying option.
- 4) Demonstration of the CICs at the Illinois Power's Baldwin Plant.

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SECTION VI

**ENVIRONMENTAL ASSESSMENT AND
GEOTECHNICAL STABILITY**

DR. S. ESLING

CO-PRINCIPAL INVESTIGATOR

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

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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 recover of saturated conditions would minimize leachate production from the residues. Second, deep mines are located generally below potable groundwater resources where the units bounding the mine contain native brine of little economic value. Third, the coal combustion residues have chemical characteristics which may actually ameliorate an environmental problem associated with coal extraction; acid mine drainage. Fourth, the residues harden like cement when wetted which could help prevent subsidence associated with underground mining.

At first, it may seem that disposal into abandoned mines should not create any environmental impacts that do not already exist as a consequence of mining. In effect, the residues are returned to their original environment. Coal combustion residues, however, are more concentrated 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 preliminary work on a large scale field demonstration project on the impact of placing coal combustion residues into underground 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.

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Goals and Specific Objectives

The purpose of the proposed 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 year of the project, preliminary work was begun toward completing the first, third, and fourth 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.

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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 create little environmental impact.

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

Most work planned for the Environmental Assessment portion of the project is scheduled for the second and third years of the project. Preliminary background work developing a conceptual model of site conditions, installing groundwater monitoring wells, and

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determining the hydrologic properties of the strata around the target panels was planned for the first year. Some of this work, however, was delayed because of uncertainty on the status and availability of the proposed site. Specifically, detailed summary of the conceptual model through cross-sections, isopach maps, and structure contour maps were postponed as was monitoring well installation and site hydrologic measurements. Because of this, some work scheduled for the second year of the project was completed instead, including preliminary work on evaluating contaminant transport models. Work completed thus far fall in the following categories:

1. Preparation of a study plan for the proposed work and quality control/quality assurance documents;
2. Literature review of packer testing and analysis, slug test design and analysis, including the development of a code for the analysis of slug test data;
3. Development of certain portions of the conceptual model;
4. Preliminary work toward developing new and evaluating existing flow and contaminant transport models of the site.

This section of the report describes the geologic and hydrologic conditions of the site as well as work completed on modeling.

Conceptual Model

Work began on developing a conceptual model on the geology and hydrology of the study area. This model is based on a preliminary analysis of well logs on file with Peabody Coal Company and the Illinois State Geological Survey as well as a review of the literature on the geology and hydrology of the area around the mine. In addition, individuals working in the area of Environmental Assessment visited the mine on June 10, 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.

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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 will be 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 will be continually refined as additional hydrologic and geotechnical data become available during monitoring well installation and groundwater modeling. 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 first 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.

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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.

Table 1 summarizes a series of six core logs that surround the study area. The Herrin (No. 6) Coal, about 7 feet thick, was mined at depths ranging from 325-375 feet. In most of the underground works, the immediate roof rock is the Anna Shale which is overlain by the Brereton Limestone. The sandstone above the Herrin Coal as indicated on the modified logs (Table 1) is probably the Anvil Rock Sandstone. 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.

Table 1. Summary logs modified from core logs supplied by Peabody Coal Company June 10, 1994.

Drill Hole #179
Elevation 598.87

Thickness		Cumulative Depth		Top Elevation	
feet	inches	feet	feet		
18	0	18.00	598.87	surficial material	
13	0	31.00	580.87	shale	
2	5	33.42	567.87	limestone	
34	7	68.00	565.45	shale	
1	9	69.75	530.87	limestone	
6	3	76.00	529.12	shale	
5	2	81.17	522.87	limestone	
8	6	89.67	517.70	shale	
4	4	94.00	509.20	limestone	
96	0	190.00	504.87	shale	
1	7	191.58	408.87	limestone	
3	5	195.00	407.29	shale	
1	3	196.25	403.87	coal	
35	9	232.00	402.62	shale	
2	1	234.08	366.87	limestone	
27	11	262.00	364.79	shale	
0	6	262.50	336.87	coal	
32	6	295.00	336.37	shale	
11	0	306.00	303.87	sandstone	
30	0	336.00	292.87	shale	
2	9	338.75	262.87	limestone	
7	8	346.42	260.12	Herrin Coal	
7	7	354.00	252.45	shale	
5	3	359.25	244.87	limestone	
24	3	383.50	239.62	shale	

Drill Hole #28
Elevation 602.5 (estimated from topographic map)

Thickness		Cumulative Depth		Top Elevation	
feet	inches	feet	feet		
60	3	60.25	602.50	surficial material	
1	9	62.00	542.25	shale	
1	0.5	63.04	540.50	limestone	
6	5	69.46	539.46	shale	
14	1	83.54	533.04	limestone	
9	0	92.54	518.96	shale	
2	9	95.29	509.96	limestone	
32	10	128.13	507.21	shale	
9	10	137.96	474.38	siltstone	
4	8	142.63	464.54	shale	
16	2	158.79	459.88	siltstone	
20	9	179.54	443.71	shale	
1	7	181.13	422.96	coal	
115	5	296.54	421.38	shale	
0	10	297.38	305.96	siltstone	
6	3	303.63	305.13	shale	
3	3	306.88	298.88	limestone	
6	1	312.96	295.63	shale	
4	10	317.79	289.54	sandstone	
1	9	319.54	284.71	shale	
5	9	325.29	282.96	limestone	
7	2	332.46	277.21	Herrin Coal	
8	5	340.88	270.04	shale	

Table 1. Summary logs modified from core logs supplied by Peabody Coal Company June 10, 1994.

Drill Hole #33
Elevation 602.90

Thickness		Cumulative Depth feet	Top Elevation feet	
feet	inches			
71	11	71.91	602.90	surficial material
6	9	78.66	530.98	limestone
4	4	82.99	524.23	shale
17	0	99.99	519.90	limestone
20	0	119.99	502.90	shale
5	5	125.41	482.90	siltstone
0	7	125.99	477.48	sandstone
35	9	161.74	476.90	siltstone
22	5	184.16	441.15	shale
1	5	185.58	418.73	coal
5	0	190.58	417.32	shale
26	6	217.08	412.32	siltstone
17	2	234.24	385.82	sandstone/ conglomerate
3	7	237.83	368.65	shale
41	9	279.58	365.07	siltstone
45	5	324.99	323.32	shale
2	7	327.58	277.90	sandstone
1	0	328.58	275.32	siltstone
3	5	331.99	274.32	shale
5	5	337.41	270.90	limestone
7	6	344.91	265.48	Herrin Coal
14	8	359.58	257.98	shale
11	6	371.08	243.32	siltstone
35	5	406.49	231.82	shale
13	1	419.58	196.40	siltstone

Drill Hole #175
Elevation 598.30

Thickness		Cumulative Depth feet	Top Elevation feet	
feet	inches			
16	0	16.00	598.30	surficial material
15	0	31.00	582.30	shale
1	4	32.33	567.30	limestone
36	8	69.00	565.97	shale
5	11	74.92	529.30	limestone
4	1	79.00	523.38	shale
5	0	84.00	519.30	limestone
7	0	91.00	514.30	shale
3	8	94.67	507.30	limestone
33	4	128.00	503.63	shale
13	0	141.00	470.30	sandstone
42	5	183.42	457.30	shale
1	6	184.92	414.88	coal
65	1	250.00	413.38	shale
2	7	252.58	348.30	limestone
40	5	293.00	345.72	shale
22	0	315.00	305.30	sandstone
22	9	337.75	283.30	shale
2	3	340.00	260.55	limestone
1	0	341.00	258.30	shale

Table 1. Summary logs modified from core logs supplied by Peabody Coal Company June 10, 1994.

Drill Hole #148
Elevation 599.31

Thickness		Cumulative	Top Elevation	
feet	inches	Depth feet		
20	0	20.00	599.31	surficial material
12	0	32.00	579.31	shale
7	0	39.00	567.31	sandstone
21	8	60.67	560.31	shale
0	4	61.00	538.64	coal
7	1	68.08	538.31	shale
12	4	80.42	531.23	limestone
5	9	86.17	518.89	shale
6	5	92.58	513.14	limestone
87	10	180.42	506.73	shale
1	4	181.75	418.89	coal
29	9	211.50	417.56	shale
2	7	214.08	387.81	limestone
87	1	301.17	385.23	shale
2	5	303.58	298.14	limestone
6	4	309.92	295.73	shale
0	5	310.33	289.39	sandstone
0	4	310.67	288.98	limestone
1	5	312.08	288.64	shale
11	11	324.00	287.23	sandstone
2	4	326.33	275.31	shale
4	9	331.08	272.98	limestone
0	4	331.42	268.23	shale
8	1	339.50	267.89	Herrin Coal
0	7	340.08	259.81	shale

Drill Hole #189
Elevation 602.24

Thickness		Cumulative	Top Elevation	
feet	inches	Depth feet		
35	11	35.91	602.24	surficial material
37	7	73.49	566.32	shale
5	0	78.49	528.74	limestone
109	8	188.16	523.74	shale
1	8	189.83	414.07	coal
41	4	231.16	412.41	shale
5	3	236.41	371.07	limestone
37	0	273.41	365.82	shale
2	2	275.58	328.82	limestone
17	1	292.66	326.66	shale
3	4	295.99	309.57	limestone
17	2	313.16	306.24	shale
1	9	314.91	289.07	limestone
25	0	339.91	287.32	shale
4	1	343.99	262.32	limestone
7	5	351.41	258.24	shale
6	9	358.16	250.82	Herrin Coal
2	8	360.83	244.07	shale

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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 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 original test plan submitted to the DOE may have to be modified with regard to groundwater sampling. The dry mine and the low permeability of the material above the mine suggest that a groundwater monitoring well screened in the coal would not yield groundwater samples. The nearest horizon that may have sufficient permeability to yield groundwater samples is probably a sandstone located just under 10 feet above

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the top of the Herrin Coal (Anvil Rock Sandstone?). This unit does not contain potable groundwater. Test drilling, planned next year is necessary to determine if the sandstone above the Herrin Coal has sufficient permeability to yield groundwater samples.

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 than 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.

Further work on the conceptual model must wait for a final word on the site status. Data collected during installation of geotechnical instruments and groundwater monitoring wells are also necessary. The Environmental Assessment team, including representatives of the Illinois State Geological Survey, Southern Illinois University, and Peabody Coal Company met to coordinate future drilling activities.

Contaminant Transport Model

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

As originally proposed, the Environmental Assessment will include simulations of groundwater flow and contaminant transport away from the coal combustion residues with existing analytic and numeric algorithms within the public domain or available commercially including models produced by the Electric Power Research Institute

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(EPRI) and those described by Yeh (1981) or Esling and Jones (1992). Work was completed this past year on tasks necessary to complete the modeling. The Illinois State Geological Survey (ISGS) developed utility software to reformat binary files from MODFLOW, a groundwater flow code, for use with RAND3D, a three-dimensional particle tracking code. Edward Mehnert, of the ISGS, attended an EPRI seminar on the use of their models, including MANAGES, MYGRT, and ROAM. MYGRT is a code for modeling solute transport analytically. The EPRI seminar was sponsored by Illinois Power Company and held in Decatur, Illinois. Arrangements for a copyright license agreement between EPRI and the Illinois State Geological Survey to use PC TRANS, a numerical solute transport code, were completed. In addition, work began this past year on a new numeric algorithm for simulating groundwater contaminant transport away from the target panels; a program that offers some features unavailable in the currently available algorithms.

An analytic model of contaminant transport may be sufficient because of the relatively simple boundary conditions associated with the mine panel. In most analytic transport models, however, the rate of transport is limited by factors such as retardation and properties of the porous media such as porosity, hydraulic conductivity, and dispersivity. For the case of injecting coal combustion residues into an underground mine, the source material is a solid that reacts with native groundwater to produce a leachate. Thus the rate of transport is also limited by source depletion and the rate of reaction between the solid source and native groundwater. Esling and Jones (1992) described a new contaminant transport model that handles this case under more realistic assumptions than existing analytic transport models. The new model allows depletion in the solid source and transport limited by reaction between the non-aqueous and aqueous phases. The analytic model, however, cannot handle the case where certain constituents react as groundwater flows through the residues. Therefore, in addition to a series of simulations with analytic transport codes, this project will also evaluate more flexible and versatile numeric transport models.

Paul and Esling (1993) presented the results of column studies of groundwater flow through combustion residues that indicate relatively high initial concentrations of contaminants, rapidly decreasing with time. This research also indicated that laboratory time is compressed. A scale-up based on three different column sizes suggested a first flush period of eleven months for the residues studied. Information from column studies such as that by Paul and Esling (1993) could provide relationships between the groundwater chemistry and the residue chemistry that could provide more realistic simulations of contaminant transport.

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Prickett and others (1981) developed a popular computer program (Random Walk) for simulating a large class of solute transport problems. Their code included routines for handling advection, dispersion, and chemical reactions. It is a fairly straight forward process to modify this code to simulate the chemical reactions between coal combustion residues and native groundwater. Random Walk simulates transport of contaminants by moving particles of constant mass by advection and simulating dispersion through a stochastic process. Prickett and others simulated a loss of a particular chemical species by randomly deleting particles. Therefore, Random Walk treats contaminants as a discrete mass. If reactions remove a particle or a new particle is released at the source, the total mass in the system changes by a discrete amount. For example, suppose a model contained 500 particles of 50 mg each. The total mass would be 25,000 mg. If a particle would leave the flow system, the total mass would drop by 50 mg, to a value of 24,950. In reality, mass of a particular chemical species within the groundwater will adjust in a continuous fashion as it moves through the problem domain. The preliminary random walk numeric model developed for this project is based on the theory presented by Prickett and others (1981), but modified to simulate three-dimensional transport. In addition, the mass of each particle is allow to change as a result of reactions. This model promises to be a powerful tool for simulating contaminant transport from sources such as coal combustion residues.

Calibration of the transport models is not possible if no contaminants reach one of the monitoring wells in less than one year of monitoring, a very real possibility. Because of this, the purpose of the contaminant transport modeling is to investigate different scenarios for contaminant migration away from the target panels based on a range for each input parameter, including nominal, minimum, and maximum values. This range of values will be justified based on field data or published information. Contaminant transport model scenarios will cover time periods of up to 500 years or until a steady state plume develops.

Summary

The Environmental Assessment portion of the project had four tasks scheduled during the first year; 1) development of a preliminary conceptual model of site geology and hydrology; 2) installation of groundwater monitoring wells, 3) preparation of geotechnical logs of the boreholes, and 4) initial measurements of the hydrogeologic characteristics of the geologic units bounding the mine. Only two of these tasks were scheduled to be completed in the first year; monitoring well installation and the development of the preliminary conceptual model. The uncertain status and availability of the original mine site, however, has delayed monitoring well installation and postponed some of the work on the conceptual model. To

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make productive use of their time, individuals involved with Environmental Assessment started work on groundwater flow and contaminant transport modeling, a task scheduled for the second and third years of the project.

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.

The Environmental Assessment test plan may have to be modified. As originally planned, the screens for the six monitoring wells would be placed in the coal, or the roof or floor rock adjacent to the target panels. These horizons would not yield groundwater samples. The Anvil Rock Sandstone, about 10 feet above the target panels, may provide groundwater samples. A final decision on the design of the monitoring wells must wait for test drilling at the site.

The Environmental Assessment team also familiarized themselves with existing analytic and numeric transport codes appropriate to this project. In addition, some work was completed on a new transport model specifically developed for the transport of contaminants from coal combustion residues.

Plans

During the next quarter, (October 1 - December 31, 1994) geologic cross-sections, structure contour maps and isopach maps of major geologic units will be prepared. These graphical displays of the conceptual model will be evaluated by those individuals involved with the Environmental Assessment of the proposed site, including the Illinois State Geological Survey. Additional plans for drilling will also be prepared, based in part on the developing conceptual model.

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Plans for the Environmental Assessment over the next year include installing the groundwater monitoring wells and geotechnical instrumentation, continued work on the conceptual model, measurement of basic hydrologic properties, developing the grid for the groundwater flow model, conducting a well survey of groundwater users in the area, and evaluating contaminant transport models.

Drilling boreholes for the groundwater monitoring wells and geotechnical instrumentation will begin in the early Spring, 1995, with an expected completion by June, 1995. Logs collected during drilling will be used to improve the conceptual model. Permeability measurements in the open boreholes will be completed through double packers immediately after the borehole is drilled, prior to the installation of the casing and sampling pumps for the monitoring wells. Additional single well tests will be conducted on the cased bore holes during the Summer, 1995. One groundwater sample from each of the six monitoring wells to represent ambient groundwater quality conditions will be collected in the Fall, 1995 and analyzed before the end of the second year of the project.

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 before the end of the second year of the project. Preliminary work, however, should show advantages and limitations of the different programs.

By the end of the second year, the Environmental Assessment portion of the project should be back on schedule as described in the original proposal and study plan.

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SECTION VII
GEOTECHNICAL STUDIES

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CO-PRINCIPAL INVESTIGATOR

MANAGEMENT OF DRY FLUE GAS DESULFURIZATION BY-PRODUCTS IN UNDERGROUND MINES

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Background And Problem Statement

The adoption of advanced combustion and flue gas desulfurization (FGD) technologies, as mandated in the 1990 Clean Air Acts, will result in a significant increase of FGD by-products. The management of this increasing FGD by-products in facilities near power plants, in conventional land fills, and in surface mines is costly and environmentally disadvantageous. Underground disposal of these residues will overcome most of the disadvantages of surface disposal. Also, underground disposal offers significant potential for reducing the likelihood of future subsidence caused by underground coal mining as well as potentially controlling acid mine drainage.

The risk of mine subsidence over abandoned coal mines in Illinois is well recognized. More than one million acres (1,000,000) of land have been undermined by about two thousand six hundred and sixty (2,660) mines where the mined out area could be outlined (Treworgy et al., 1991). About three hundred (300) claims are filed each year with the Illinois Mine Subsidence Insurance Program, and about 17% of these claims are actually attributed to mine subsidence. Over four thousand (4,000) incidents of subsidence are on record. More than two hundred and twenty (220) mine subsidence events were investigated during the period of 1975 to 1990 by the Illinois Abandoned Mine Lands Reclamation Council and cooperating parties.

A major cause of subsidence incidents over abandoned mines is the weakening and failure of floor strata due to wetting which in turn leads to pillar and roof failures (Chugh et al., 1990). If the mine workings can be backfilled with coal combustion by-products without the use of a significant amount of water, the weakening of floor strata due to wetting and possible later flooding and associated failure of roofs and coal pillars can be minimized. Subsidence events over abandoned mines can then be significantly reduced. Thus, blind backfilling of abandoned underground mines with FGD by-products to minimize future mine subsidence events is of significant interest to Illinois Basin States and mine operators.

In this cooperative agreement, minimizing future subsidence is to be achieved through controlled blind backfilling of mine voids using hydraulic or pneumatic backfilling of appropriate mixtures of coal combustion by-products.

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Specific Objectives

Hydraulic placement of CCR (coal combustion residues) and FGD by-products into abandoned mine workings may negatively impact 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. Resulting short-term subsidence due to wet backfilling can cause damage to surface structures and impact land use patterns. In addition, underground bulkhead stability may be negatively impacted due to hydraulic and 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) and, therefore, the objective of these studies are:

1. Analyze the stability of abandoned mine workings prior to and after disposal of combustion by-products,
2. Estimate the surface subsidence movements and their characteristics due to wet disposal of by-products,
3. Assess the stability of isolation structures such as bulkhead to withstand pressures due to the disposal of FGD by-products,
4. Monitor 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.

Tasks Worked On During The Year

The proposed hydraulic backfill area in Peabody No. 10 mine is shown in Figure 1. An extensive underground investigation was conducted in the proposed area (Figure 1) to identify inaccessible and completely filled entries and cross-cuts. The inaccessible areas are those areas where roof falls have occurred but are open for the placement of backfill materials. Figure 1 shows the details of inaccessible and completely filled areas. This investigation is required for designing the best locations of the boreholes through which the FGD by-products will be placed underground.

Seven convergence stations were installed in the proposed area (see Figure 1) to monitor convergence of the openings prior to backfilling. A convergence rod was used to measure the roof to floor convergence with an accuracy of 0.001 inch. Three sets of readings were taken over a period of 50 days. From these readings, the average roof-to floor convergence over a year was determined.

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To study the effects of the height and strength of the backfill materials in the openings on the roof convergence and floor heave, a cross section of the proposed area is taken to conduct preliminary 2-D finite element analysis. More extensive time-dependent 2-D and possibly 3-D analysis will also be conducted in the next quarter.

Two sets of samples were prepared using different proportions of fly ash, FBC, water and fly ash, scrubber sludge, water to conduct geotechnical testings, such as free swelling pressures, moduli of elasticity, ultimate compressive strengths, moisture contents, and stress-strain curves.

Modeling, Testing Procedures, Results and Discussion

Underground Investigations

Figure 1 shows a section of Peabody #10 mine (room-and-pillar) where backfilling is proposed. Detailed underground investigations revealed that #8 and #7 entries are completely filled. The cross-cuts between #5 and #4 entries have permanent ventilation stoppings which will prevent the flow of backfill materials. Four cross-cuts between #7 and #6 entries and four cross-cuts between #6 and #5 entries are completely filled. A portion of #6 entry is also completely filled while a portion of #5 entry is inaccessible. Completely filled areas will prevent the flow of backfill materials whereas inaccessible areas will allow flow of backfill materials. This information will be used in designing the locations of boreholes through which backfill materials will be pumped underground.

Roof to Floor Convergence Measurements

The initial baseline measurement of roof to floor convergence was conducted on May 23, 1994. After 17 days, (on June 9, 1994) a second set of readings of roof to floor convergence was taken. On June 23, 1994, a third set of readings was taken. The fourth set of convergence readings was taken on July 13, 1994.

Results and Discussions

Table 1 shows the roof to floor convergence for all the seven convergence monitoring stations on June 9, June 23, and July 13, 1994. The maximum convergence after 17 days was 0.033 inch. The average convergence of all the points after 17 days was 0.028 inch which is equivalent to 0.6 inch of convergence per year. After 30 days (on June 23, 1994),

Figure 1 Map of the proposed backfill area

Table 1 Measured roof to floor convergence (in inches)

Dates when measurements were taken	Convergence stations						
	#1	#2	#3	#4	#5	#6	#7
6/9/94	0.012	0.008	0.300	0.033	0.080	0.030	0.006
6/23/94	0.020	0.030	0.052	0.106	0.180	0.050	0.020
7/13/94	0.080	0.060	0.070	0.156	0.200	0.070	0.050

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more than 0.1 inch of convergence occurred at convergence stations 4 and 5. This is either due to the measurement errors or local roof movements. The average roof to floor convergence of all the measurement stations after 30 days was 0.065 inch which is equal to 0.8 inch of convergence per year. Roof to floor convergence readings on July 13, 1994 revealed stations 4 and 5 moved 0.05 and 0.02 inches, respectively, between June 23 and July 13 (over a period of 20 days) which amounts to an average of 0.64 inches per year. If roof to floor convergence data for measurement stations 4 and 5 are discarded, the average of the rest of the measurement stations is 0.066 inch (over a period of 50 days) which is equal to 0.48 inch of convergence per year. If measurement stations 4 and 5 are included, the average roof to floor convergence of all the points is 0.72 inch per year.

Finite Element Modeling

To study the effects of the strength and height of the backfill materials on the floor heave and roof convergence, preliminary finite element analyses were carried out using 2-D, plane strain models. A typical geologic log as shown in Figure 2 was used for modeling.

The extraction height of the coal seam is seven feet and the immediate floor is 10 feet thick claystone (weak floor). The immediate roof is four feet thick grayshale overlain by a eight feet thick limestone stratum. In the model, 68 feet 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 feet beneath the coal seam is used for modeling.

Figure 3 shows the finite element mesh and geometries used in the models. The length of the model is 1020 feet which includes two complete barrier pillars of 80 feet length and two half-barrier pillars (40 feet length) at two extreme ends. A five-entry panel configuration with 20 feet wide entries and 40 feet wide pillars is used in the model. The model is loaded on the top by uniform vertical loads to simulate 282 feet of overburden which corresponds to 350 feet of overburden thickness (282 feet of loading and 68 feet 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 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 limestone bed.

Table 2 shows material properties used for different strata and backfill materials for modeling purposes. Modeling is done for six feet (one foot void from the roofline to the top surface of the backfill material) and seven feet (complete backfilling of the voids) of backfilling with two different properties (Young's modulus) of backfill materials. The program ADINA (Automatic Dynamic Incremental Nonlinear Analysis), a general purpose finite element program, is used for modeling.

Figure 2 Geologic log used for finite element modeling

Figure 3 Finite element mesh used for modeling

Table 2 Material properties used for finite element modeling

Materials	Young's Modulus, psi	Poisson's Ratio	Unit Weight, pcf
Limestone	700,000	0.30	180
Grayshale	250,000	0.25	150
Coal	50,000	0.30	75
Claystone	25,000	0.35	160
Backfill materials	7,400 15,000	0.40	70

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Modeling Procedure

The modeling process involves two sets of steps—one without any backfill and one with two different thicknesses of backfill with two different properties of backfill materials. The first set of steps involve:

1. Apply pre-mining stresses (vertical and horizontal) to the model without any openings.
2. Excavate openings to simulate coal extraction. Redistribution of stresses occur and vertical displacements in the roof and floor are determined due to the creation of openings.
3. Reduce modulus of floor strata to 20,000 psi (from 25,000 psi) to simulate the degradation of floor with time.
4. Floor heaves and roof sags are determined from vertical displacements.

In the second set of steps, after the excavation, the openings are backfilled and the floor modulus is degraded at the same time. Floor heaves and roof sags are determined from the vertical displacements.

Figures 4 and 5 show the floor heave and roof convergence, respectively, in the middle entry for different situations. In the legends of Figures 4 and 5, FL 25k and FL 20k indicate the Young's moduli of floor as 25,000 psi and 20,000 psi, respectively. In the same legends, FI 7k and FI 15k indicate the Young's moduli of backfill materials as 7,400 psi and 15,000 psi, respectively.

Results and Discussions

Figure 4 shows that the maximum floor heave at the center of the entry is 0.65 inches (without any backfilling) when the modulus of the floor is reduced to 20,000 psi to simulate the degradation of the floor. When the void is completely backfilled, the floor heave at the center of the entry is 0.48 inches for a backfill material whose Young's modulus is 7,400 psi and 0.39 inches for a backfill material whose Young's modulus is 15,000 psi. However, when the void is not completely backfilled (six feet of backfilling in a void of seven feet height), the reduction in the floor heave at the center of the entry is 0.04 inches for a weaker backfill material (Young's modulus is 7,400 psi) and 0.09 inches for a stronger backfill material (Young's modulus is 15,000 psi).

Figure 4 Vertical floor movements in the middle entry

Figure 5 Vertical roof movements in the middle entry

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The maximum amount of pillar settlement into the floor without backfilling is 0.18 inches which is reduced to 0.06 inches for a weaker backfill material and 0.04 inches for a stronger backfill material when the void is completely filled. For the partial filling of the void, the reduction in the maximum pillar settlement into the floor is 0.01 inches.

Figure 5 shows that the roof sag at the center of the entry without backfilling is 0.38 inches which can be reduced to 0.21 and 0.15 inches with complete backfilling by weaker and stronger backfill materials, respectively. The compression at the pillar edge can be reduced to 0.07 inches from 0.23 inches by using a stronger backfill material and complete backfilling. However, for incomplete backfilling, the compression at the pillar edge reduces to 0.20 inches with a stronger backfill materials.

The preliminary finite element analyses indicate that a stronger backfill material (preferably stronger than the floor stratum) with complete backfilling will significantly reduce floor heaves, roof sags and pillar compressions. These reductions will result in long term pillar, floor and roof stability. However, extensive time dependent analysis is needed to conclude the effects of partial backfilling.

Laboratory Testing of Samples of Backfill Materials

Two sets of samples of backfill materials are chosen for laboratory testing to determine the effects of water on the backfill materials. One set comprises of samples made from different proportions of fly ash, FBC and water and the second set of samples are made from different proportions of fly ash, scrubber sludge and water.

The following laboratory tests are performed on the samples:

1. Determination of free swelling pressures of the samples over a period of five to seven days.
2. Determination of pH of water in contact with the samples over a period of five to seven days.
3. Determination of temperature rise of water in contact with the samples over a period of five to seven days.
4. Determination of moisture content of the samples after five to seven days the samples are immersed in water.
5. Determination of the ultimate compressive strength and Young's modulus of saturated samples (after five to seven days the samples are immersed in water).

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Laboratory Testing Procedure

Figure 6 shows a schematic of the testing apparatus used in the laboratory. The proving ring is calibrated so that the load on the sample can be determined from the dial gage (load) readings. The sample dimensions (length and diameter) are measured and the sample is put inside the water jacket (a PVC cylinder with a closed bottom). A pressure distribution plate is placed on the top of the sample and the water jacket is placed in between the loading platens of the apparatus. A known load is applied to the sample. The temperature and pH of tap water is measured and then the water is poured inside the water jacket. The pH and temperature of water in the water jacket and the dial gage (load) readings are taken at certain time intervals for a period of five to seven days. Before terminating the test, the load-deformation values of the sample in water are recorded by using two dial gages (load and displacement--see Figure 6). After the sample fails, the moisture content of the sample is determined by using the standard ASTM procedure. Besides, cyclic loading and unloading test was also performed on one dry sample having 70% fly ash, 30% FBC and 30% water.

Results and Discussions

Seven samples of different proportions of fly ash, FBC, and water were tested to study the effects of water on the samples and the quality of water in contact with the samples. Two samples had 100% fly ash (identified as 100-0-30-1 and 100-0-30-2), two samples had 80% fly ash and 20% FBC (identified as 80-20-30-1 and 80-20-30-2), one sample had 70 % fly ash and 30% FBC (identified as 70-30-30-1), and two samples had 50% fly ash and 50% FBC (identified as 50-50-30-1 and 50-50-30-2). All the samples had 30% water. The samples were tested after 30 days of curing.

Temperature Rise

The temperature of water in contact with the samples was never above the room temperature. Most of the time, the water temperature was 0.5 to 1 degree celsius below the room temperature. The temperature of the tap water, which is always 2 to 3 degrees celsius below the room temperature, increased by 1.5 to 2.5 degrees celsius within 1 to 3 hours of pouring water into the water jacket.

Figure 6 Experimental setup for laboratory testing of samples

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Increase in pH

Figure 7 shows the increase in pH of water (in the jacket and in contact with the samples) for the first six hours from the time when water was poured into the water jacket. The pH of tap water was in between 9.3 to 9.6 which increased sharply to 10.2 to 11.7 within the first half hour. After the first four hours, the pH was around 12.0. Figure 8 shows the pH of water after 6 hours. For six samples, the pH stabilized at 12.2. For one sample (sample 50-50-30-2) the pH stabilized at 12.3. Figures 7 and 8 do not show any relationship between pH and the amount of fly ash in the sample.

Free Swelling Pressure

Figure 9 shows the change in swelling pressure for the first six hours of the testing period. All the samples show a sharp decrease in the swelling pressure immediately after adding water. This is due to the radial expansion of the samples resulting in axial contraction. However, all the samples except one (100-0-30-1) show either relatively constant swelling pressures or slightly increasing swelling pressures within the first six hours. Figure 9 also shows that the drop in swelling pressure immediately after adding water depends on the amount of fly ash in the sample. As the amount of fly ash in the sample decreases, the drop in swelling pressure increases (which means more radial expansion of the sample).

Figure 10 shows the variations in swelling pressures after six hours. Except for one sample (100-0-30-2), rise in swelling pressures for the rest of the samples is very low. For sample 100-0-30-2, swelling pressure reached a maximum value of 9 psi after 256 hours.

Unconfined Compressive Strength and Young's Modulus

Figures 11 and 12 show the unconfined compressive strength and Young's modulus of seven wet samples. Both the figures show no relationships between the mechanical properties and the amount of fly ash in the samples. The variation in the values of Young's modulus for samples having the same amount of fly ash is very high, which suggests that more samples need to be tested to conclude the characteristics of the samples.

Moisture Content

Figure 13 shows the post-testing moisture content of the samples. For the samples having 100% fly ash, the average moisture content is 70%. For the rest of the samples, the moisture content lies within 54 to 59 percentages.

Figure 7 Increase in pH of water within the first six hours of testing

Figure 8 pH of water after six hours of testing

Figure 9 Change in swelling pressure within the first six hours of testing

Figure 10 Change in swelling pressure after six hours of testing

Figure 11 Unconfined Compressive Strength of Wet Samples

**Figure 12 Elastic Modulus of Wet Samples
(50% of Unconfined Compressive Strength)**

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GEOTECHNICAL STUDIES

Cyclic Loading and Unloading Test

Cyclic loading and unloading test was performed on a dry sample (70-30-30-2) to obtain the stress-strain curve of the sample (Figure 14). The ultimate strength of the sample was 14.9 psi. In the first cycle, the sample was loaded to 5 psi (33% of the ultimate strength) and the strain observed was 0.00442. The unrecoverable strain was 0.00249 (56% of 0.00442). In the second cycle, the sample was loaded to 6.4 psi (43% of the ultimate strength) and the strain recorded was 0.00531. The unrecoverable strain after the second cycle was 0.00370 (70% of 0.00531). In the third cycle, the sample was loaded to 9.3 psi (62% of the ultimate strength) and the strain measured was 0.00740. The unrecoverable strain after the third cycle was 0.00555 (75% of 0.00740).

Two samples of different proportions of fly ash, scrubber sludge, and water were put in water in an attempt to study the effects of water on the samples. Both the samples failed after the first four minutes of submersion in water. Upon failure, the sample was reduced to a mud-like consistency, completely losing its original shape. The consistency was like that of pancake batter, allowing the sample to flow and take the shape of the container. This is due to the presence of a high percentage of calcium sulfate in the scrubber sludge.

Plans For The Next Year

More extensive time dependent finite element analyses will be carried out to simulate the effects of backfilling. This will involve time dependent degradation of floor for two different situations--without backfilling and with backfilling. Data from time dependent plate loading tests of weak floors of Illinois Coal Basin will be used to establish an empirical creep function for weak floors. The empirical creep function will be used to model the time dependent behavior of weak floors. Geotechnical properties obtained from the laboratory tests of samples of backfill materials will be used in the modeling. The model will be fine-tuned with field observed roof to floor convergence data to simulate the actual situation.

Subsidence monuments will be installed on the surface over the proposed area of backfilling and two baseline survey measurements will be taken. Besides, two or three measurements will be taken to determine surface movements.

Borehole instrumentation will be done in two boreholes to study the subsurface movements. A borehole TV camera will be used to observe underground mining condition from the surface. Geotechnical tests will be conducted on cores from one borehole.

Figure 13 Post-testing moisture content of the samples

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GEOTECHNICAL ASSESSMENT

Cyclic Loading and Unloading Test

Cyclic loading and unloading test was performed on a dry sample (70-30-30-2) to obtain the stress-strain curve of the sample (Figure 14). The ultimate strength of the sample was 14.9 psi. In the first cycle, the sample was loaded to 5 psi (33% of the ultimate strength) and the strain observed was 0.00442. The unrecoverable strain was 0.00249 (56% of 0.00442). In the second cycle, the sample was loaded to 6.4 psi (43% of the ultimate strength) and the strain recorded was 0.00531. The unrecoverable strain after the second cycle was 0.00370 (70% of 0.00531). In the third cycle, the sample was loaded to 9.3 psi (62% of the ultimate strength) and the strain measured was 0.00740. The unrecoverable strain after the third cycle was 0.00555 (75% of 0.00740).

Two samples of different proportions of fly ash, scrubber sludge, and water were put in water in an attempt to study the effects of water on the samples. Both the samples failed after the first four minutes of submersion in water. Upon failure, the sample was reduced to a mud-like consistency, completely losing its original shape. The consistency was like that of pancake batter, allowing the sample to flow and take the shape of the container. This is due to the presence of a high percentage of calcium sulfate in the scrubber sludge.

Plans For The Next Year

More extensive time dependent finite element analyses will be carried out to simulate the effects of backfilling. This will involve time dependent degradation of floor for two different situations--without backfilling and with backfilling. Data from time dependent plate loading tests of weak floors of Illinois Coal Basin will be used to establish an empirical creep function for weak floors. The empirical creep function will be used to model the time dependent behavior of weak floors. Geotechnical properties obtained from the laboratory tests of samples of backfill materials will be used in the modeling. The model will be fine-tuned with field observed roof to floor convergence data to simulate the actual situation.

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70% Fly Ash, 30% Fluidized Bed Combustion Residue, 30% Initial Water, Dry Test

Figure 14 Stress-strain curve of a dry sample

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GEOTECHNICAL STUDIES

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