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LAND APPLICATION USES FOR DRY FGD BY-PRODUCTS

Phase 2 Report

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ABSTRACT

The Clean Air Act, as amended in 1992, spurred the development of flue gas desulfurization (FGD) technologies. Several FGD processes produce a dry, solid by-product material consisting of excess sorbent, reaction products containing sulfates and sulfites, and coal fly ash. Generally, dry FGD by-products are treated as solid wastes and disposed in landfills. However, landfill sites are becoming scarce and tipping fees are constantly increasing. Provided the environmental impacts are minimal, socially acceptable beneficial uses of dry FGD by-products can provide economic benefits to both the producer and the end user of the FGD by-products.

A study was initiated in December, 1990 to demonstrate large volume beneficial uses of FGD by-products. A Phase 1 report provided results of an extensive characterization of chemical, physical, mineralogical and engineering properties of 58 dry FGD by-product samples. The Phase 1 report concluded that high volume beneficial reuses will depend on the economics related to their ability to substitute for existing materials for various types of applications (e.g. as an agricultural liming material, soil borrow for highway embankment construction, and reclamation of active and abandoned surface coal mine lands). Phase 2 objectives were (1) to conduct laboratory and greenhouse studies of FGD and soil (spoil) mixtures for agronomic and engineering applications, (2) to initiate field studies related to high volume agronomic and engineering uses, and (3) to develop the basic methodological framework for estimation of the financial and economic costs and benefits to society of several FGD reuse options and to make some preliminary runs of economic models.

Laboratory studies were conducted to investigate time-dependent changes in solution chemistry and solid phase composition resulting from the forced leaching of mine spoil-FGD mixes (FGD applied at 0, 10, 20 and 40 wt %). Water quality of the leachate was primarily a function of FGD by-product type, application rate and reaction time. Spoil type had little effect on pore water quality except at the lowest FGD application rate. FGD by-products containing free lime (CaO or $\text{Ca}(\text{OH})_2$) initially yielded leachate pH values above 12 but the pH rapidly decreased with time. Leachates were generally dominated by Ca, S and Cl. If the sorbent used to scrub the flue gases contained Mg, then Mg concentrations were also found to be greatly increased in the leachate of the treated spoil. In contrast, Al and Fe concentrations, which often reach toxic levels in acid mine spoil, were decreased to near instrument detection limits when FGD was applied at rates of 10% or greater. Arsenic, Cd, Cr, Cu and Ni concentrations were below the maximum contaminate levels (MCL) defined for drinking water. The MCL for Se was exceeded in 78% of the samples in the first leachate, but in only 25% of the samples after 26 days of reaction. Geochemical modeling of the leachate chemistry suggested that ettringite ($\text{Ca}_6\text{Al}_2(\text{SO}_4)_3(\text{OH})_{12}\cdot 6\text{H}_2\text{O}$) forms as a reaction product in FGD by-products containing free lime.

In greenhouse experiments, FGD by-products were found to be highly effective in neutralizing spoil and soil acidity and ameliorating phytotoxic conditions. When applied

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soils or minespoils in amounts needed to neutralize active and potential acidity, plant growth was greatly improved and concentrations of Al, Fe, Mn, Ni and Zn in leachates and in plant tissues decreased relative to plants grown in unamended spoil. At rates in excess of the amount needed to neutralize acidity, FGD by-products containing oxides and hydroxides of Ca promoted ettringite formation. Spoil layers cemented by ettringite were impermeable to plant roots. FGD by-products containing Mg also initially caused poor plant growth due to high salt concentrations.

FGD obtained from a spray dryer was successfully used for construction of a truck unloading ramp. Laboratory tests performed on samples which had been compacted to 100% of Standard Proctor density indicated that the FGD by-product possessed the necessary strength and had acceptable permeability and swelling to be a satisfactory material for truck ramp construction. FGD by-product from a pressurized fluidized bed combustor was used to construct a base at a cattle feedlot. Visual inspection and laboratory analyses of samples obtained from the feedlots have indicated satisfactory performance over a three-year period.

Economic studies revealed that use of FGD by-product as an agricultural limestone substitute or for construction of cattle feedlot bases would offer only a small market. Assuming an average financial cost per ton of FGD as \$26.10, 90% of FGD would continue to be landfilled, 9% would be applied to cropland and 1% to surface coal mine land. However, landfilling itself suffers from an additional economic liability due to the reduced housing prices that result in the immediate vicinity of a landfill.

High volume beneficial reuses of dry FGD by-products have been successfully demonstrated. Adverse environmental impacts have been negligible. Although few sources of dry FGD by-products currently exist in Ohio and the United States there is potential for smaller coal-fired facilities to adopt SO₂ scrubbing technologies that produce dry FGD material. Also much of what we have learned from studies on dry FGD by-products is applicable to the more prevalent wet FGD by-products. The adaptation of the technologies demonstrated in this project seem to be not only limited by economic constraints, but even more so, by the need to create awareness of the market potential of using these FGD by-products.

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EXECUTIVE SUMMARY

Development of flue gas desulfurization (FGD) processes in the state of Ohio, as well as other coal producing states in the eastern United States, has been spurred by the 1990 amendments to the Clean Air Act. Much of the coal mined in the eastern United States contains sulfur levels that, when burned, necessitate some type of scrubbing of exhaust gasses to remain in compliance with SO₂ emission standards. The U.S. Department of Energy, the Electric Power Research Institute (EPRI), the Ohio Coal Development Office and various industries (mostly utilities) have been cooperating on the development and evaluation of new sulfur dioxide removal technologies that can be retrofitted onto existing coal burning systems. Several of these technologies produce a dry, solid by-product material consisting of excess sorbent, reaction products containing sulfates and sulfites, and coal fly ash. Because of the lack of basic information, FGD by-products are generally treated as solid wastes and are landfilled. However, landfill sites are becoming scarce and tipping fees are constantly increasing. Provided the environmental impacts are minimal and socially acceptable, beneficial uses of dry FGD by-products are highly desirable and can provide economic benefits to both producers and end users of the FGD by-products.

In December of 1990, a project was begun to demonstrate high volume uses of FGD by-products which may substitute for other materials now being used for land reclamation (e.g. reclamation of active and abandoned mined land, agriculture, soil stabilization and other engineering uses. This project was divided into three phases. Phase 1 involved FGD by-product sample collection and characterization and collection of baseline economic data. This phase has been completed and a report is available from EPRI (Stehouwer et al., 1995). Phase 2 objectives were to (1) conduct laboratory and greenhouse studies of mixtures of FGD by-products with soil or spoil, (2) to initiate field studies related to high volume agronomic and engineering uses, and (3) to develop the basic methodological framework for estimation of the financial and economic costs and benefits to society of several FGD reuse options, and to make some preliminary runs of economic models. Plans for Phase 3 include field demonstrations of high volume uses of FGD by-product and development of engineering, economic and environmental models for use with FGD applications.

This report provides a description of project activities completed during Phase 2. Three dry FGD by-products and two acidic mine spoils were collected from eastern Ohio and used in a laboratory study. The FGD by-products were from the lime injection multi-stage burner (LIMB), the pressurized fluidized bed combustion (PFBC), and the spray dryer (SPD) scrubbing processes. The by-products were mixed with the spoils at rates of 0, 10, 20 and 40 wt % and sequentially extracted with deionized water over a four month period. Concentrations of elements in the leachates were measured at each sampling event. After 132 days the mixtures containing 40% FGD by-products were sampled for mineralogical analyses using x-ray diffraction.

Leachate composition was primarily a function of FGD by-product type, application rate and reaction time. Spoil type had little effect except at the 10% application rate. The LIMB and SPD by-products initially increased the pH above 12. There was a tendency for pH to decrease with time for all three by-products and this was most noticeable at the lowest application rate. Calcium, S, and Cl dominated the chemistry of the leachates from the LIMB and SPD amended spoils and the PFBC yielded significant amounts of Mg in addition to the previously mentioned elements. Sulfur concentrations were greater in leachates from the LIMB and PFBC amended spoils due to the presence of more soluble forms of sulfate in these materials. The PFBC leachates contained Mg because the PFBC process used dolomite ($\text{Ca,Mg}(\text{CO}_3)_2$) as the sorbent. Leachate concentrations of Fe and Mn decreased to near detection limits (0.03 and 0.001 mg L^{-1} respectively) at all FGD application rates, and Zn and Al decreased with increasing amendment rates. Reductions in soluble Al concentrations are important because this element is commonly at phytotoxic levels in acid spoils. Arsenic, Cd, Cr, Cu and Ni concentrations were below the maximum contaminant levels (MCL) defined for drinking water for the duration of the study. Leachate concentrations of Pb were below the detection limit (0.020 mg L^{-1}) in all samples except in two amended spoil samples. Selenium concentrations exceeded the MCL levels in 78% of the samples at the first leaching and in 26% of the samples after 26 days of reaction.

Geochemical modeling of the leachate chemistry suggested that ettringite ($\text{Ca}_6\text{Al}_2(\text{SO}_4)_3(\text{OH})_{12}\cdot 6\text{H}_2\text{O}$) formed as a reaction product in the LIMB and SPD amended materials while gypsum ($\text{CaSO}_4\cdot 2\text{H}_2\text{O}$) was the solid phase controlling Ca and S solubilities in the PFBC-amended mixtures. Observed changes in solid phase composition supported the weathering scenario predicted by modeling. In the LIMB mixes, CaSO_4 and CaO rapidly hydrated to form gypsum and portlandite ($\text{Ca}(\text{OH})_2$). With longer reaction times, gypsum was converted to ettringite. With the PFBC mixtures gypsum, rather than ettringite, was the major Ca phase formed.

FGD by-products and soil-FGD by-product mixtures were characterized as geotechnical materials using test procedures typically used for natural soils. Four separate laboratory test programs were conducted on the dry FGD by-products: swelling, durability, strength and permeability. These tests were conducted using equipment that is generally readily available in a commercial or state department of transportation geotechnical laboratory. American Society for Testing and Materials (ASTM) standard procedures were employed. Results revealed that amending soils with FGD offers the opportunity to modify the behavior of a soil in an engineered structure. Two important engineering properties, strength and permeability, were found to be affected by the ratio of soil to FGD by-product and curing time. In all cases studied, the strength of the amended soils was higher compared to the strength of the soil alone. Additionally, permeability increased with addition of FGD to soil with the highest values recorded for approximately equal amounts of soil and FGD by-product.

Greenhouse studies revealed that FGD by-products (LIMB and PFBC) were highly effective in neutralizing spoil and soil acidity and ameliorating phytotoxic conditions.

When applied in amounts needed to neutralize active and potential acidity, plant growth was improved and concentrations of Al, Fe, Mn, Ni, and Zn in leachates and in plant tissues were all reduced compared to unamended spoil or soil. Leachate concentrations of Ca, Mg and S were increased indicating some potential for downward movement and amelioration of phytotoxic conditions that may exist below the treated layer of spoil or soil. When applied at rates that greatly exceeded the neutralizing needs of the spoil, FGD by-products containing oxides and hydroxides of Ca caused initially high pH (in the range of 8.5 to 11) and, in some cases, cementing due to ettringite formation. This cementing reaction inhibited plant growth by restricting root development. Also at the same high rates, FGD by-products containing Mg caused high salt levels which initially suppressed plant growth, but then dissipated with time. Application of FGD in excess of that required for acid neutralization in the treated layer may be warranted if it leads to increased leaching of Ca, Mg and S and improvement in soil or spoil conditions below the amended layer.

Economic analyses of recycling FGD by-products on cropland, for feedlot surfaces, for stripmine reclamation, and for highway construction were of two types; financial costs and returns, and social costs and benefits. Both types of analyses were utilized to develop conceptual models and to derive empirical estimates useful for supplier and users of FGD by-products as well as for public policy formulation.

Under current regulations, assumptions, and technologies regarding FGD by-product use or disposal, the average financial cost for disposal per ton in 1994 was \$26.10. Approximately 90% of the expected production of FGD would continue to be placed in landfills, 9% would be used on cropland and 1% for surface coal mine reclamation. The demand for ground agricultural limestone (ag-lime) is more affected by prices of other agricultural inputs used in production and by crop prices than by the price of the ag-lime itself. Therefore, the addition of an alternative source, albeit a cheaper source, of ag-lime would have little impact on overall ag-lime use. These findings suggest the market for an ag-lime substitute, such as FGD by-product, would not be greatly affected by cost and availability of the FGD by-product. Use of FGD by-product on Ohio farmer feedlots offers some potential for reduction in costs. However, the magnitude of FGD use is likely to be quite small because of the relative cost disadvantage generally faced by producers in Ohio and the Midwest compared to cattle producers in western states.

Economic studies also indicated that housing prices decreased as the distance to a landfill decreased. Therefore, the use FGD by-product to pave feedlot surfaces, or for other beneficial reuses, could have substantial indirect benefits to communities. If beneficial reuses of FGD by-products are not utilized, landfills would impose "external" costs on communities, which are reflected in the depressed market value of real estate surrounding the landfills.

Soil (spoil) mixed with FGD by-products were characterized for chemical, mineralogical and engineering properties. These data were used to design greenhouse and field studies and to evaluate potential environmental impacts. Several

high volume beneficial uses of dry FGD by-products were successfully demonstrated during Phase 2. Adverse environmental impacts were negligible. Although few sources of dry FGD by-products currently exist in Ohio and the United States there is potential for smaller coal-fired facilities to adopt SO₂ scrubbing technologies that produce dry FGD material. Also much of what we have learned from studies on dry FGD by-products is applicable to the more prevalent wet FGD by-products. The adaptation of the technologies using dry FGD by-products demonstrated in this project seem to be limited not so much by economic constraints, but by a need to create awareness of the market potential of using these FGD by-products.

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1 INTRODUCTION

1.1 Background and Project Description

The Clean Air Act as amended in 1992 has confirmed the need to develop and implement processes for sulfur dioxide (SO₂) removal from flue gasses of coal burning utility generating units. Some of these flue gas desulfurization (FGD) processes generate dry, solid by-product material consisting of calcium-based excess sorbent, reaction products containing sulfates/sulfites and fly ash (Batelle, 1989).

Recent studies indicate that as much as 7,000 to 11,000 MW of Ohio Electric Utility generating capacity may be required to install some type of FGD system (Ohio Electric Institute, 1986). A portion of the newly installed FGD systems will be "dry" type systems, but the number of such dry systems and the tons of FGD by-product to be produced each year is yet to be determined by market forces. Presently FGD by-product materials are treated as solid waste and must be landfilled. However, landfill sites are becoming scarce and tipping fees are increasing. Provided the environmental consequences of FGD by-product materials reuse are socially acceptable, it is highly desirable to find beneficial uses for these materials.

In December of 1990, this project was initiated to demonstrate high volume uses of dry FGD by-product materials which will substitute for other materials now being used for land reclamation (e.g. reclamation of abandoned acid mine lands), agriculture, and soil stabilization. The work for the project is being performed by The Ohio State University with cooperation from the United States Geological Survey. Dravo Lime Company is the prime contractor with additional direction being provided by a steering committee made up of representatives from funding organizations, industry, state regulatory agencies and a citizen's group (Appendix 6.1).

Project objectives were developed over an 18-month period during extensive discussions with personnel from industry, regulatory agencies, and research institutions. The specific objectives of the project are as follows:

1. To characterize the material generated by dry FGD processes,
2. To demonstrate the utilization of dry FGD by-products as a soil amendment material on agricultural lands and on abandoned and active surface coal mines in Ohio,
3. To demonstrate the use of dry FGD by-product as an engineering material for soil stabilization,
4. To determine the quantities of dry FGD material that can be utilized in each of these applications,
5. To determine the environmental and economic impact of utilizing the material, and
6. To calibrate environmental, engineering, and economic models that can be used to determine the applicability and costs of utilizing these processes at other sites.

The study, originally planned to be completed in approximately 4½ years but since extended to 5½ years, is being conducted at The Ohio State University main campus (Columbus), at The Ohio Agricultural Research and Development Center (Wooster), at The Ohio State University branch stations, at field-scale active and abandoned coal mine sites, and at various other sites in Ohio. The project is divided into three phases. The first phase focused on characterization of FGD by-products and collection of baseline economic data. Results obtained during the first phase of the project are reported in the Phase 1 report (Stehouwer et al., 1995). The second phase focuses on laboratory and greenhouse studies of mixtures of FGD by-products and soil or spoil, and the development of economic models. The third will focus on field demonstrations of various high volume uses for FGD by-products and the development of environmental, engineering and economic models.

1.2 Phase 2 Objectives

The specific objectives for Phase 2 of this project are listed below.

1. Determine the chemical, mineralogical, physical, and engineering properties of dry FGD by-products which have been combined with soils or minespoils at several ratios, and determine how these properties vary with time.
2. Conduct greenhouse studies to investigate the efficacy and potential environmental impact of using dry FGD by-products as alkaline amendments for acidic minespoils and acidic agricultural soils.
3. Determine the potential environmental impacts and changes in material characteristics which result from exposed storage of a dry FGD by-product.
4. Determine the feasibility of using FGD by-products for soil stabilization and base construction of cattle feedlots.
5. Conduct detailed, premining characterization of the geology and ground-water flow and quality of a mine site, and evaluate the acid producing potential of the postmining spoils.
6. Study the economic feasibility of recycling or disposing of coal-fired electric power plant FGD by-products on crop land, reclaimed coal surface mines, highway construction projects, and other uses.

1.3 Approach

The phase 2 objectives were accomplished by conducting laboratory and greenhouse studies on FGD mixes with either minespoil or agricultural soil, by field experiments involving various applications of FGD materials, and by development of economic models and data bases for beneficial use of FGD in various applications.

1.3.1 Materials Used in Laboratory and Greenhouse Studies

Most of the laboratory and greenhouse studies involved experiments using FGD by-products as admixtures to minespoil or soil materials. The FGD by-products used in these studies represented five different FGD processes and by-products (described

below). Characteristics of FGD by-products used in the chemistry/mineralogy and greenhouse studies are given in Table 1.

Lime Injection Multistage Burners (LIMB). Samples were collected from Ohio Edison's Edgewater plant in Lorain, Ohio. In the LIMB process a calcium-based sorbent is injected into the boiler where it calcines to CaO , and reacts with SO_2 and O_2 in the combustion gasses to produce CaSO_4 . Temperatures at the point of injection approach $2,300^\circ\text{F}$ (1260°C) and the reaction occurs between this temperature and about $1,600^\circ\text{F}$ (870°C). The reaction product and unspent sorbent are collected with the fly ash in electrostatic precipitators. This material was used in all laboratory and greenhouse studies and in the construction of a temporary stockpile (section 2.3.3).

Pressurized Fluidized Bed Combustion (PFBC). Samples were collected from the full-scale commercial demonstration of PFBC at American Electric Power's Tidd plant in Brilliant, Ohio. In this system the boiler is pressurized and, in addition to steam generation, pressurized gasses are used to power a gas turbine. In the PFBC system dolomite and crushed coal are introduced together into the boiler bed where they are "fluidized" or suspended by jets of air. The fluidizing process intimately mixes the coal and sorbent. Reaction of the sorbent with SO_2 occurs in the boiler at temperatures generally somewhat less than $1,600^\circ\text{F}$. These temperatures are high enough to calcine MgCO_3 to MgO , but are not consistently high enough to cause much calcination of CaCO_3 . Two FGD by-product streams result; the heavier, granular bed ash material (PFBC_{bed}), and smaller particles suspended in the flue gas which are removed with the particulate emission control system ($\text{PFBC}_{\text{cycl}}$). Both by-product streams contain reaction products and unspent sorbent as well as conventional bed or fly ash. The $\text{PFBC}_{\text{cycl}}$ material was used in laboratory studies, both $\text{PFBC}_{\text{cycl}}$ and PFBC_{bed} were used separately in the greenhouse studies, and a blend of the two was used in the construction of cattle feed lots (section 2.3.2).

Spray Dryer. Samples of spray dryer FGD by-products were collected from The Ohio State University McCracken Power Plant in Columbus, Ohio. In this spray dryer system, flue gasses enter a separate scrubber vessel located downstream of the boiler and air heater where the sorbent, a slurry of hydrated lime, is injected. The scrubber vessel allows for increased residence time of flue gas and therefore lengthens the time period for injected sorbent to react with SO_2 in the flue gas. Temperature in the scrubber vessel is usually around 300°F (150°C). The reaction with SO_2 produces a mixture of calcium sulfate and calcium sulfite. Reaction products and unspent sorbent are removed together with the fly ash in the particulate emission control system. This material was used in laboratory studies and in the construction of a truck unloading ramp (section 2.3.1).

Coolside. Coolside by-product was used in the engineering laboratory studies. Samples were collected from a full-scale commercial demonstration of the Coolside process at Ohio Edison's Edgewater plant in Lorain, Ohio. In the Coolside process sorbent (hydrated lime) is injected into the flue gas as it enters a humidification chamber in the ductwork downstream of the boiler and air heater. Here, injection of a

Table 1. Characterization of FGD amendments used in chemical, mineralogical, and greenhouse experiments.

Parameter	LIMB	PFBC _{cycl}	PFBC _{bed}	SPD
Mineralogy (%)				
Anhydrite (CaSO ₄)	25	22	36	nd
Calcite (CaCO ₃)	15	11	27	17
Dolomite (CaMg(CO ₃) ₂)	nd [†]	23	nd	nd
Lime (CaO)	21	nd	4	nd
Portlandite (Ca(OH) ₂)	5	nd	1	29
Periclase (MgO)	nd	13	27	nd
Coal ash	30	32	10	21
CaCO ₃ equivalent (%)	59.4	60.3	60.0	
pH (1:1, water)	12.5	10.5	12.2	
Total chemical analysis				
Ca (%)	35.96	17.53	24.44	33.1
Mg (%)	0.60	10.64	16.28	0.94
S (%)	5.77	5.21	8.64	10.4
Al (%)	3.52	3.93	2.75	1.6
Si (%)	6.58	7.24	45.25	2.6
Fe (%)	5.56	5.17	2.08	2.3
Na (%)	0.33	1.03	na	na
P (%)	0.02	0.02	0.01	94
K (%)	0.91	0.50	0.13	0.24
Ba (%)	0.03	0.02	0.14	0.01
As (mg kg ⁻¹)	55.1	75.0	46.7	43.2
B (mg kg ⁻¹)	233.1	171.2	206	315
Cd (mg kg ⁻¹)	1.0	1.9	3.5	1.2
Cr (mg kg ⁻¹)	28.0	36.9	17.5	13.0
Cu (mg kg ⁻¹)	21.0	52.5	912.9	18.5
Pb (mg kg ⁻¹)	16.0	16.0	28.0	24.1
Mo (mg kg ⁻¹)	5.9	6.6	4.8	4.4
Ni (mg kg ⁻¹)	31.1	52.1	21.2	17.6
Se (mg kg ⁻¹)	8.1	5.6	2.3	9.1
Zn (mg kg ⁻¹)	86.0	74.0	245	90

[†]not analyzed[‡]not detected

fine water mist lowers the gas temperature from about 300°F (150°C) to about 140 to 150°F (60 to 65°C) and raises the humidity of the flue gas to a 20 to 25F° (11 to 14C°) approach to saturation. The sorbent reacts with the SO₂ in the flue gas ductwork and produces a mixture of calcium sulfate and calcium sulfite. Dry FGD by-product is removed from the flue gas stream in an electrostatic precipitator and contains the reaction products, fly ash, and unspent sorbent.

Wet FGD. Wet FGD by-product was used in the engineering laboratory studies. Wet FGD samples were collected from American Electric Power's Conesville plant near Coshocton, Ohio. In this scrubber a mist of hydrated lime slurry is injected into the flue gas stream downstream from particulate removal. The SO₂ reaction product is primarily calcium sulfite which is separated from the slurry by a belt filter. The filter cake is then combined with dry fly ash and 2-3% lime to improve handling characteristics and strength development for landfilling. Engineering laboratory studies were conducted using both the filter cake (relatively pure calcium sulfite) and with the combined filter cake/fly ash/lime material. The combined material was also used in construction of cattle feedlots (section 2.3.2).

Three eastern Ohio mine spoil materials were chosen for the preparation of admixtures with dry FGD by-products (Table 2). Minespoil and underclay materials were collected from the Fleming abandoned mine site in Tuscarawas County, Ohio. Mining activities began on this site in 1940 and continued intermittently until the late 1970's. The site was characterized by high erosion rates and discharge of acid mine drainage arising from the oxidation of pyrites contained in the mining overburden derived from marine deposits associated with the #5 and #6 coals (middle and upper Kittaning). The Fleming spoil was a highly weathered, extremely acidic, nutrient poor material which was assumed to be a mixture of strata removed to access the coal. At the Fleming site exposed underclay (underlaid the #5 coal) was sampled and used in the greenhouse experiments. The R&F spoil (also associated with the #5 and #6 coals) was an unweathered, moderately acidic, nutrient poor material collected from an active mining site in Coshocton County, Ohio. This overburden sample was collected in 1991 after being exposed to weathering conditions for approximately one year.

Two soil materials were also used in the greenhouse studies (Table 3). The R&F topsoil was collected from a segregated stockpile of topsoil at the same time and from the same location as the R&F spoil. The Wooster soil (Aquic Fragiudalf) was collected from the 0 to 20 cm depth of a field on an OARDC experiment farm located in Wayne County, Ohio.

Table 2. Characterization of coal mine materials used in preparation of admixtures with dry FGD by-products for laboratory and greenhouse column studies.

Parameter	Fleming spoil	Fleming underclay	R&F spoil
Particle size (%)			
Sand (0.05-2mm)	17.3	5.0	14.4
Silt (2-50µm)	34.7	61.0	54.9
Clay			
0.2-2µm	40.8	28.9	26.0
<0.2µm	7.2	5.2	4.7
Extractable cations (1N ammonium acetate)			
Ca (mg kg ⁻¹)	50	130	1180
Mg (mg kg ⁻¹)	27	38	370
K (mg kg ⁻¹)	65	76	181
pH (1:1, water)	3.1	3.4	5.2
Potential acidity (cmol _e kg ⁻¹)	33.6	5.3	5.3
Mineralogy [†]	Q,M,K,F,J		Q,M,K,F,G
Total Chemical Analysis			
Al (%)	8.7	9.5	9.2
Si (%)	17.6	27.2	23.5
S (%)	1.02	0.27	1.06
Ca (%)	<0.1	<0.1	<0.1
Mg (%)	<0.1	<0.1	0.18
Fe (%)	2.5	<0.01	1.3
K (%)	2.0	1.3	2.3
Ba (%)	0.05	0.03	0.04
As (mg kg ⁻¹)	46.3	14.4	13.7
Cd (mg kg ⁻¹)	0.8	0.8	0.8
Cr (mg kg ⁻¹)	94.4	95.6	79.8
Cu (mg kg ⁻¹)	26.8	37.3	79.0
Pb (mg kg ⁻¹)	78.0	35.0	30.5
Mo (mg kg ⁻¹)	14.0	1.0	<0.9
Ni (mg kg ⁻¹)	28.5	33.2	65.1
P (mg kg ⁻¹)	1027	351	371
Se (mg kg ⁻¹)	4.5	5.6	4.3
Zn (mg kg ⁻¹)	<0.3	<0.3	<0.3

[†]Minerals are coded as given below and in the table are listed in order of estimated abundance.

F = Feldspar [(K,Na)AlSi₃O₈]

G = Gypsum (CaSO₄·2H₂O)

J = Jarosite [KFe₃(SO₄)₂(OH)₆]

K = Kaolinite [Al₂Si₂O₅(OH)₄]

M = Mica [(K,Na)Al₂(AlSi₃O₁₀)(OH)₂]

Q = Quartz (SiO₂)

Table 3. Characterization of soil materials used in greenhouse experiments.

Parameter	Unit	R&F topsoil	Wooster silt loam
Lime requirement [†]	Mg ha ⁻¹	10.6	20.6
pH		4.4	4.6
Bray P ₁	mg kg ⁻¹	7	9
Exchangeable K	mg kg ⁻¹	63	103
Exchangeable Ca	mg kg ⁻¹	260	430
Exchangeable Mg	mg kg ⁻¹	60	85
Cation exchange capacity	cmol _c kg ⁻¹	8.0	15.7

[†] Amount of agricultural limestone required to raise soil pH to 7 according to SMP buffer test (Shoemaker et al., 1962).

1.3.2 Laboratory Studies

This study was designed to evaluate the potential impact of dry FGD by-products on water quality in mined land reclamation by following time-dependent changes in solution chemistry and solid phase composition resulting from the forced leaching of artificial mine spoil-FGD mixes. Water soluble components and reaction products of mixtures of two eastern Ohio acid mine spoils (section 1.3.1) and three dry FGD by-products were examined. The highly alkaline FGD by-products selected for study were from three different scrubber processes (section 1.3.1). The mixtures (0, 10, 20, and 40 wt.% by-product) were incubated at approximately field capacity for the duration of the study. The samples were sequentially extracted with deionized water for six sampling events over a 4-month period. The concentrations of elemental constituents released to solution were measured at each sampling event. After 132 days the solid residues of the highest FGD by-product mixing rate (40 wt.%) were collected for mineralogical analysis. The analytical procedures used are described in section 2.1.2 and results and discussion are presented in section 2.1.3.

1.3.3 Greenhouse Studies

Greenhouse studies were conducted on spoil and underclay materials collected from an abandoned mined land area, on topsoil and spoil from an active coal mine, and on an agricultural soil (section 1.3.1). These materials, which were all acidic, were amended with various amounts of LIMB and PFBC by-products. Columns of the FGD-amended materials were periodically leached to determine the mobility and solubility of FGD constituents and the potential impact of FGD amendments on surface water,

ground water, and soil quality. Plant growth was measured to determine the effectiveness of the FGD materials as alkaline amendments for mine reclamation and as limestone substitutes for acidic agricultural soils. Plant uptake of salts and trace elements was measured to determine the potential impact of FGD amendments on plant forage quality and the potential for introduction of trace elements into the food chain.

1.3.4 Economic Studies

The "economics" of recycling or disposing of coal-fired electric power plant FGD by-products on crop land, reclaimed coal surface mines, highway construction projects, or other uses was based on two perspectives or accounting stances. The first was financial analysis of cost and returns from the perspective of the supplier (i.e., electric power plant) and the user (i.e., farmer, surface mine owner, highway construction entity). In this case, actual costs and returns incurred by an organization in pursuing the FGD recycling options determined the "economics" of alternative options.

The second approach was economic analysis of benefits and costs from a societal perspective. It recognized that externalities or unpriced environmental services can result in large gaps between private and social costs and benefits. Examples in the case of FGD waste recycling included the impacts of reclaiming surface mines or landfill sites on adjacent property values.

1.3.5 Field Studies

Several field studies were initiated as part of the Phase 2 activities of this project. These studies are described briefly below.

LIMB stockpile. A stockpile for temporary storage of LIMB FGD by-product material was constructed to study the effects of exposed storage on an FGD by-product. The purpose of this study was to determine if changes in the chemical, physical, and engineering properties of the FGD material would impact its potential for subsequent beneficial uses. The stockpile was also monitored to assess the potential for environmental impact from exposed storage.

Truck-unloading ramp. A truck-unloading ramp was constructed using OSU spray-dryer by-product material. The purpose of this study was to assess the suitability and stability of such materials for road-bed base material.

Cattle feedlots. Two FGD by-product materials, PFBC and wet FGD, were used to stabilize soils and to construct pads for cattle feedlots

Active mine site reclamation. This study was designed to supply a detailed characterization of the pre-mine ground-water-flow regime and pre-mine water quality. A thorough understanding of pre-mine conditions is required to differentiate the effects of mining and reclamation versus the effects of FGD by-product application on water

quality. Site characterization was initiated by a search of data bases maintained by the USGS and state and local agencies. Geologic mapping, rock-coring and sampling, lithologic and geophysical logging, triaxial chamber tests for permeability of underclay samples, and geochemical analyses of rock cores were conducted to enhance definition of site characteristics. The acid-producing potential of the post-mine spoils was evaluated with geochemical data obtained from rock cores by use of acid-base accounting techniques (Sobek et al., 1978).

2 REPORTS ON PHASE 2 TASKS

2.1 Laboratory Studies of FGD plus Soil/Spoil Mixes

Laboratory studies were conducted on several mixtures of FGD by-products with either soils or minespoils. These mixtures were analyzed using a variety of procedures and techniques in order to assess the chemical, mineralogical, physical, and engineering properties of the mixtures and changes in these properties when the mixes were subjected to various treatments under controlled laboratory conditions.

2.1.1 Procedures Used in Studies

The following sections describe the various materials, procedures, and QA/QC measures used in conducting the chemical, mineralogical, physical, and engineering laboratory studies.

2.1.1.1 Materials Used for Chemical and Mineralogical Studies

The three dry FGD by-products (LIMB, SPD, and PFBC) used as admixtures for two mine spoil materials (Fleming and R&F) in these studies are described in section 1.3.1.

2.1.1.2 Chemical Methods and QA/QC Measures

The methods for all chemical analyses employed in this study are listed in Table 4.

Total Chemical Analysis. Total chemical analysis of mine spoils and dry FGD by-products were determined according to the method of Bernas whereby 100 mg samples were placed in Teflon® decomposition vessels and wetted with an aqua-regia-HF mixture. The vessels were then placed in stainless steel digestion bombs and heated at 110°C for 40 min. The digested samples were mixed with excess H_3BO_4 and diluted to 100 mL total volume with distilled H_2O . For all elemental analyses, sample concentrations represent the means of duplicate digestions and analyses.

The digests were analyzed for Al, Ba, Be, Ca, Fe, K, Li, Mg, Mn, Mo, P, Si, Sr, V, and Zn (mine spoils only) using inductively coupled plasma emission spectrometry (ICP) with a Leeman PS2000 instrument. The ICP was subjected to the following operating procedures and QA/QC measures. The sample induction system was checked and adjusted for optimum operating conditions at the beginning of each operating period. Pump tubing was inspected and replaced when necessary. Instrument peaking routines were performed to optimize the torch placement and to align the optics. The instrument was programmed to recheck and peak the optics every 20 min thereafter. Standardization was performed using a four-point calibration curve. Standards consisted of one blank and three solutions containing different concentrations of each element analyzed. Intermediate standard solutions served as check standards and were analyzed between every ten samples. The low and high

Table 4. Experimental protocols for determination of chemical properties of mine spoils, dry FGD by-products and spoil-FGD mixes.

Laboratory Procedure	Reference
Total chemical analysis	Bernas, 1968
Total boron	Custom Method (USGS)
Total neutralizing power	ASTM C 602-90 ¹
Suspension pH	Method 8C1a ²
Solution pH	ASTM D 1293-84 ³
Eh	ASTM D 1498-76 ³
Potential acidity	Sobek et al., 1978
Reaction products study	Custom Method
Geochemical modeling	Sposito and Coves, 1988 Allison et al., 1990 Sposito and Traina, 1987 Lindsay, 1979
Anion analysis	Method 429 ⁴
Specific conductance	Method 205 ⁴

¹1990 Annual Book of ASTM Standards, Vol. 4.01.

²U.S. Department of Agriculture, Survey Laboratory Methods Manual, Report No. 42, 1992.

³1991 Annual Book of ASTM Standards, Vol. 11.01.

⁴American Public Health Association, 1985.

standards were also analyzed between every ten samples and used to update the standardization. Two standard fly ashes (NIST 2691 and 1633a) were also analyzed.

The concentrations of As, Cd, Co, Cr, Cu, Ni, Pb, and Se in the aqua-regia-HF digests were measured with a Perkin-Elmer 4100-ZL, graphite-furnace atomic absorption spectrophotometer equipped with a Zeeman background corrector and L'vov-platform atomization. Electrode-discharge lamps were used for As and Se analysis, and hollow cathode lamps were used for all other elements. Matrix-modifier composition and the temperature program of the graphite furnace were optimized for each element. The graphite pyrolysis tubes were inspected daily and replaced when necessary. Standardization was performed using a four-point calibration curve. Standards consisted of one blank and three solutions containing different concentrations of each element analyzed. Intermediate standard solutions served as

check standards and were analyzed between every eight samples, after which standard resloping was initiated. An aqua-regia-HF digest of a standard fly ash, NIST-1633a, was also analyzed. Analytical matrix blanks were run for each element analyzed. Automatic sample dilution was initiated by the auto-sampler whenever the total absorbance from the analyte exceeded the linear response range.

Total S was determined with a Leco Model 521 Induction Furnace equipped with a semi-automatic titrator by placing a small sample (15-100 mg) in a crucible containing accelerator metal. The sample was heated to 3000°C where, in the presence of O₂, S in the sample was oxidized to SO₂. The SO₂ produced was bubbled through a blue, standardized solution containing KIO₃, KI, HCl, and starch which faded as SO₂ was added. The color of the solution was monitored by a photoelectric cell and was back titrated to the original blue color. Percent S was based on the amount of titrant (KIO₃) used and was read directly from the calibrated buret. Standardization was performed by using a three-point calibration curve derived by analysis of Leco metal rings containing known amounts of S. Individual samples were duplicated. Two standard reference materials (NIST coal fly ash 2690 and 2691) were also analyzed during the operating period.

Total B contents were determined by a custom method (USGS-sodium peroxide sinter procedure) whereby 100 mg samples were placed in Pt crucibles and mixed with 0.4 g of Na₂H₂O₂. The crucibles were heated at 445°C for 30 min and allowed to sinter. The samples were then removed from the furnace and cooled rapidly. Fourteen mL of distilled H₂O were added to the sinter cakes and stirred for 10 min. Six mL of 6M HCl were then added to the solution and stirred until the sinter cakes were completely dissolved. The solutions were diluted to 100 mL total volume with distilled H₂O and heated in a drying oven for 1 hr at 100°C. The solutions were analyzed for B by ICP with a Leeman PS 2000 instrument. Calibration and standardization of the instrument were performed as described previously in this section. Individual samples were duplicated. Two standard reference materials (CANMET Sy-2 and Sy-3 rock samples) were analyzed.

Zn contents in the by-products were determined by atomic absorption using a Varian Techtron Atomic Absorption Spectrophotometer AA6 instrument. The instrument was calibrated using a 5mA lamp current, 213.9 nm wavelength, 0.2 nm spectral band pass, and an air-acetylene flame. Standardization was performed using a three-point calibration curve from standards within the optimum working range (0.4 - 1.6 µg Zn mL⁻¹). The low, intermediate, and high standards were analyzed between every six samples and used to update the standardization. A standard reference material (NIST coal fly ash 1633a) was analyzed during the operation period.

Results of the total elemental analyses of standard fly ash samples (NIST coal fly ash 1633a and 2691) are presented in Tables 5, 6, 7, and 8. Boron analyses were compared to standard samples (CANMET Sy-2 and Sy-3 rock samples) obtained from the Canadian Center for Mineral and Energy Technology and are presented in Table 7. The values reported by OSU are means from a minimum of four replicate

Table 5. Certified elemental composition of NIST standards 2691 and 1633a and as measured by ICP analysis of HF-aqua-regia digests.

	Al	Ba	Be	Ca	Fe	K	Li
NIST 2691	-----%-----		mg kg ⁻¹	-----%-----			mg kg ⁻¹
OSU	10.14 ± 0.04	0.57 ± 0.00	5 ± 0	18.58 ± 0.27	4.04 ± 0.04	0.34 ± 0.01	43 ± 1
Certified	9.81 ± 0.39	0.66*	NR†	18.45 ± 0.32	4.42 ± 0.03	0.34 ± 0.01	NR†
NIST 1633a	-----%-----		mg kg ⁻¹	-----%-----			mg kg ⁻¹
OSU	15.5 ± 0.2	0.13 ± 0.00	14 ± 0	1.17 ± 0.01	9.0 ± 0.0	1.84 ± 0.01	186 ± 2
Certified	14.3 ± 1.0	0.15*	12*	1.11 ± 0.01	9.4 ± 0.1	1.88 ± 0.06	NR†
	Mg	Mn	Mo	P	Si	Sr	V
NIST 2691	-----%-----		mg kg ⁻¹	-----%-----			mg kg ⁻¹
OSU	3.26 ± 0.03	0.02 ± 0.00	13 ± 0	0.54 ± 0.01	14.58 ± 0.32	0.31 ± 0.00	229 ± 2
Certified	3.12 ± 0.08	0.02*	NR†	0.51 ± 0.02	16.83 ± 0.12	0.27*	NR†
NIST 1633a	%	-----mg kg ⁻¹ -----		-----%-----		-----mg kg ⁻¹ -----	
OSU	0.487 ± 0.005	178 ± 0	37 ± 1	0.17 ± 0.00	21.5 ± 0.5	814 ± 3	300 ± 6
Certified	0.455 ± 0.010	179 ± 8	29*	NR†	22.8 ± 0.8	830 ± 30	297 ± 6

*Value not certified by NIST, †Value not reported by NIST. The uncertainty (OSU values) is expressed as one standard deviation.

Table 6. Graphite furnace analysis of NIST 1633a.

Source of result	As	Cd	Co	Cr	Cu	Ni	Pb	Se
<hr/> mg kg ⁻¹ <hr/>								
OSU	146±31	1.10±0.10	32±4	191±11	109±6	122±7	73.6±2.0	12.2±0.2
Certified	145±15	1.00±0.15	46 [†]	196±6	118±3	127±4	72.4±0.4	10.3±0.6

[†]Value not certified by NIST. The uncertainty (OSU values) is expressed as one standard deviation.

Table 7. Concentration of B in SY-2 and SY-3 standard materials.

Sample	Source	Reported value	OSU analysis [†]
<hr/> mg kg ⁻¹ <hr/>			
SY-2	CANMET	85	87±4
SY-3	CANMET	110	100±5

[†] Uncertainty for OSU results are expressed as one standard deviation.

Table 8. Concentration of Zn in NIST 1633a.

Reported value	OSU analysis [†]
<hr/> mg kg ⁻¹ <hr/>	
220±10	267±30

[†] Uncertainty for OSU results are expressed as one standard deviation.

measurements obtained by ICP or a minimum of 15 replicate measurements obtained using the graphite furnace atomic absorption spectrophotometer. There was good agreement between the analytical values measured by OSU and the certified values reported by NIST and CANMET.

Total Neutralizing Power. Total neutralizing power of the FGD samples was determined using ASTM C 602-90. Samples were heated with excess standard HCl and back-titrated with standard NaOH. Analyses were performed in triplicate and reagent grade CaCO_3 was used as a standard reference material.

Suspension pH. Suspension pH of the mine spoils and the FGD by-products was determined by placing equal volumes of sample and distilled water into a 70-mL cup. The mixture was stirred and allowed to equilibrate for 30 min. The stirring and equilibration steps were then repeated. Measurements were conducted with an Orion Research Expandable Ion Analyzer, model EA920. The meter was calibrated at the beginning of each operating period using two commercially-prepared buffer solutions. Buffer solutions were used once and discarded. The electrode was inspected weekly for scratches, cracks, salt crystal build-up, or membrane/junction deposits. The reference chamber was drained, flushed, and filled weekly with fresh Ross Filling Solution. Once a month the electrode was soaked in 0.1M HCl for 0.5 hr and then placed in storage solution for 1 h. Sample pH's were measured in duplicate.

Potential Acidity. Potential acidity of the mine spoils was determined by a method whereby 1 g samples were placed in beakers and mixed with 20 mL of 30% H_2O_2 . The samples were heated until the solution temperature reached 40°C. Samples were removed from the heat and allowed to react completely as indicated by cessation of bubbling. An additional 10 mL of H_2O_2 was then added and allowed to completely react. The solutions were heated at 95°C until all dissolved CO_2 was removed, cooled to room temperature, and titrated with 0.05 M NaOH (free of CO_2 and protected from the atmosphere) to pH 7.0 using a pH meter.

Long-term Equilibrium Studies. This study examined water soluble ions and solid-phase reaction products of two mine spoils (R&F and Fleming) and three FGD by-products (EDG-LIMB-14, OSU-SPD-03, and TID-PFBC-03) mixed at four rates and incubated at approximately field capacity for 132 days. In order to simulate field conditions, the mass of spoil remained constant in all samples while the quantity of FGD added was varied. The final solid to solution ratios ranged from 1:5 (0 wt.% samples) to 1:3 (40 wt.% samples).

Ten g of each spoil was mixed with the appropriate quantity of FGD by-product to yield mixtures containing 0, 10, 20, and 40 wt.% by-product. Duplicate samples were placed in 60-mL polypropylene extraction tubes containing 2 g of polypropylene wool as a filter pulp. All samples were sequentially extracted with 50 mL deionized water for six sampling events over a 4-month period. The concentrations of elemental constituents released to solution were measured at each sampling event (1, 7, 14, 26, 62, and 132 days). After the final leaching event (132 days) residues from the spoil

treated with 40 wt.% FGD were quick frozen with liquid N₂ and lyophilized for subsequent X-ray diffraction analysis following the 132-day sampling event .

The solutions from each sampling event were analyzed for Ag, Al, As, B, Ba, Ca, Cd, Co, Cr, Cu, Fe, K, Li, Mg, Mn, Mo, Na, Ni, P, Pb, S, Se, Si, Sr, and Zn by ICP using a Leeman PS2000 instrument. Calibration and standardization were performed as described in section 2.1.1.2. All solutions were analyzed after a 1:3.5 dilution with H₂O.

The aqueous concentrations of Cl⁻, SO₃²⁻, and SO₄²⁻ were determined according to Method 429 from Standard Methods for the Examination of Water and Waste Water (1985). Analyses were performed on a Dionex 2000i ion chromatograph equipped with an AS4A anion separator column and an AG4A guard column. Before each operating period, eluant flow rate and the detector were adjusted to achieve the peak separation outlined in the method. The system was then allowed to reach equilibrium as indicated by a stable baseline. A standard containing a mixture of Cl⁻ and SO₄²⁻ was injected to determine anion retention times. Three different concentrations of each of the anions were injected and a calibration curve was derived by plotting peak area against concentration. The instrument was re-calibrated whenever the detector setting was changed. Due to problems with peak resolution and standard stability, only a semi-quantitative measurement of SO₃²⁻ concentration could be obtained. This estimate was derived from a relative peak area analysis of SO₃²⁻/SO₄²⁻ over the range of 0-100 mg SO₄²⁻ L⁻¹.

Solution pH. Solution pH values were determined by electrometric measurement using a glass electrode as described in ASTM D1293 . Measurements were conducted with an Orion Research Expandable Ion Analyzer, model EA920. Calibration and standardization were performed as described in section 2.1.1.2.

Oxidation-reduction potential. The oxidation-reduction potentials (E_h) of the extracts were determined according to the method described in ASTM D1498. The measurements were conducted using an Orion Research Expandable Ion Analyzer, model EA920, fitted with a Pt electrode coupled with a Ag-AgCl reference electrode. The meter was calibrated at the beginning of each operating period using an Fe(II)-Fe(III) reference solution. The Fe (II)-Fe(III) solution was discarded and replaced every 30 d. The electrodes were inspected weekly for scratches, cracks, or deposits. In addition, the electrodes were cleaned at the beginning of each operating period. The E_h of the Fe (II)-Fe(III) reference solution was measured between every 10 samples. If the measured value was different from the published value for the reference solution by more than 2%, the meter was recalibrated and all samples measured after the previous calibration were repeated.

Solution Conductivity. Conductivity was determined according to Method 205 of Standard Methods for the Examination of Water and Waste Water (1985). Measurements were conducted with a YSI Specific Conductance Meter (Model 32) fitted with a YSI 3403 conductivity cell. The meter was calibrated at the beginning of

each operating period using a YSI calibration solution (1000 $\mu\text{mhos/cm}$). The calibration solution was used once and discarded. The electrode was inspected weekly for scratches, cracks, and deposits. The electrode was rinsed thoroughly between measurements and was kept immersed in distilled water when not in use.

Geochemical Modeling. An aqueous-phase geochemical speciation of the reaction products data for sample solutions collected at 1, 62, and 132 days was calculated with the computer programs, Soilchem (Sposito and Coves, 1988) and MinteqA2 (Allison et. al, 1990). The former model contains an extended Debye-Huckle expression which allows for accurate calculation of single ion activity coefficients, and accurate predictions of mineral solubilities in solutions with ionic strengths (I_s) $\leq 2 \text{ mol L}^{-1}$ (Sposito and Traina, 1987). In the present study, molar solution composition data were input into the Soilchem model and the chemical species that would occur under open conditions (reaction with atmospheric CO_2 permitted) and with precipitation prohibited were predicted. The predicted equilibrium solution concentrations of free cations and anions (based on the Soilchem results) were recorded. The total solution composition data were then input into MinteqA2. The solution concentrations of free cations and anions from the Soilchem output were used as the initial "guesses" for the "free" ion concentrations in the MinteqA2 calculations. All MinteqA2 calculations were conducted with precipitation prohibited. Log Saturation Index values (log SI) were calculated by MinteqA2 with the following equation

$$\log \text{SI} = \log \text{IAP} - \log K_{\text{sp}}$$

where IAP is the activity product of the solution concentrations of "free" cations and anions present in a given solid, and K_{sp} is the solubility product for that solid. Given a general solid, $\text{A}_a\text{B}_b(\text{s})$, the K_{sp} for each solid is given by:

$$K_{\text{sp}} = (a\text{A}^{b+})(b\text{B}^{a-})$$

where () denotes ion activities. The ion activity product is given by:

$$\text{IAP} = (a\text{A}^{b+})(b\text{B}^{a-}).$$

Equality between the K_{sp} and the IAP indicate that a solution may be at equilibrium with a given solid. This is denoted by a log SI value equal to 0.

Values of log SI for the solid, ettringite, were calculated manually using the aqueous activities of the cations and anions obtained from the geochemical computer model. A value of -111.32 was used for the log K_{sp} of ettringite (Myneni et. al, 1993).

2.1.1.3 Mineralogical Methods and QA/QC Measures

A variety of analytical methods were employed to derive semi-quantitative measures of mineralogical composition from the mine spoils, dry FGD by-products and

the spoil-FGD mixtures. These methods and references are listed in Table 9.

Table 9. Experimental protocols for determination of mineralogical properties of mine spoils, dry FGD by-products and spoil-FGD mixtures.

Laboratory Procedure	Reference
X-Ray Diffraction	Custom Method
Thermal Analysis	ASTM STP 997 ¹
Scanning Electron Microscopy	Custom Method
Calcite and Dolomite	Dreimanis, 1962
Ash Content	Custom Method

¹Earnest, C.M. 1988. Compositional Analysis by Thermogravimetry, ASTM STP 997.

X-ray Diffraction. X-ray diffraction (XRD) patterns from the mine spoils, dry FGD by-products, and spoil-FGD mixtures were obtained from randomly-oriented powder mounts using Cu K α radiation and a Phillips PW 1216/90 wide-range goniometer equipped with a theta-compensating slit and a graphite monochromator. Diffraction patterns were recorded from 5 to 70°2 θ with a step interval of 0.05°2 θ and a counting time of 4 sec per step. The instrument was calibrated using both low (cholesterol) and high angle (NIST SRM 640b Si powder) diffraction standards. Quartz, a ubiquitous phase in coal fly ash, was employed as an internal standard to verify the accuracy of diffractometer readings from sample materials. Crystalline phase assignments were based on published literature, files of the International Centre for Diffraction Data (Newton Square, PA), and comparative analyses of reference mineral samples.

Thermal Analysis. Thermal studies were conducted on the dry FGD by-products using a Seiko SSC5020 instrument that provided simultaneous thermogravimetric (TGA) and differential thermal analysis (DTA). Samples were heated from 105 to 1125°C at a rate of 20°C/min under a continuous flow (200 mL/min) of dry N₂. Calibration of the temperature signal was achieved using the melting points of In and Sn. Calibration of the thermal balance was performed using a reference weight provided by the instrument manufacturer. Thermal events observed with heating of sample materials were assigned to phase transitions based on published literature and analyses of standard mineral samples. A reference calcite (CaCO₃) was analyzed to assure instrument standardization. Weight losses induced by thermal decomposition of the standard were maintained within $\pm$ 5% of calculated (ideal) values.

Calcite and Dolomite. Calcite and dolomite could not be consistently distinguished by thermal analysis. When both phases were present in a by-product sample (TID-PFBC-03), contents were determined by the gasometric method of Dreimanis (1962) using a

Chittick apparatus. The amount of CO_2 evolved from the sample following the addition of 20 mL of 6M HCl containing 0.6 g of FeCl_2 was measured after 30 sec (for calcite) and 30 min (for dolomite). After volume corrections were made for temperature and pressure, regression equations developed from the results of Dreimanis (1962) were used to calculate the percent calcite and dolomite.

Scanning Electron Microscopy. Air-dried samples from the 40 wt.% spoil-FGD mixtures were fixed on Al stubs and C-coated with a planar magnetron sputter coater. Micrographs were obtained using a JEOL JSM-820 scanning electron microscope (SEM). Filament current was increased until the beam was saturated ($<250 \mu\text{A}$). Accelerating voltage, probe current, spot size, and working distance were adjusted to optimize image qualities. The accelerating voltage was set to 15kV or less. Spot size was controlled by the aperture settings and ranged from 50 to 110 μm . Working distances of 15 to 30 mm were used to manipulate resolution and depth of field.

Ash Content. Ash contents of the dry FGD by-products were determined by mixing weighed samples (10 - 20 g) of FGD by-product with Na acetate-acetic acid buffer (pH 5.0) to form thick slurries. The slurries were then treated with 30% H_2O_2 to convert $\text{CaSO}_3 \cdot \frac{1}{2}\text{H}_2\text{O}$ to the more soluble $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$. Following oxidation and heating at 60°C to decompose excess H_2O_2 , the samples were quantitatively transferred to cellulose dialysis bags and submerged in Na acetate-acetic acid buffer (pH 5.0) for 4-6 d to remove acid-soluble materials. The samples were then dialyzed against distilled H_2O to remove excess salt and to dissolve residual $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$. Removal was judged to be complete when a negative test for dissolved SO_4^{2-} was obtained with BaCl_2 . The samples were then quantitatively transferred to tared beakers, dried at 100°C , and weighed to determine the residual ash contents.

2.1.1.4 Physical Methods and QA/QC Measures

Mixes were constructed using the three FGD by-products (LIMB, SPD, and PFBC) and two mine spoils (Fleming and R&F) described in section 1.3.1. The FGD materials were individually combined with each of the two spoils at rates of 0, 10, 20 and 40% by weight using a 2 liter twin shell mixer (Patterson Kelly Co., East Stroudsburg, PA). This resulted in 20 individual mix treatments. During blending, the mixes were moistened to 15% gravimetric water content and subsequently stored in sealed containers for no less than 30 days.

Soil Physical measurements conducted on the mixes included the compression index, compressed bulk density, the water retention curve and saturated hydraulic conductivity. The experimental protocol for these determinations were as described in the Phase I report. With the exception of the water retention curve, 3 replications were conducted for each measurement and the data was analyzed as a completely randomized design with a hanging control group for each soil. The water retention measurements were conducted using two replications and the data from these measurements were pooled. The Van Genuchten equation was subsequently fit to the pooled water retention measurements using STASTICA (StatSoft, 1994).

2.1.1.5 Engineering Methods and QA/QC Measures

It was assumed during the Phase 2 test program of engineering properties that the FGD by-product and soil mixtures could be characterized as geotechnical materials having specific engineering properties. Based on the evaluation of these properties, construction practices that are typically employed with natural soils could be followed with confidence when building with FGD by-products. The laboratory experiments performed were conventional tests using equipment that should be readily available in a quality commercial or state department of transportation geotechnical laboratory. American Society for Testing and Materials (ASTM) Standard procedures were employed to obtain the desired engineering properties in all cases where an appropriate standard existed. Detailed procedures for each test employed can be found in ASTM (1990) and Lin and Evett (1990). The test procedures used are presented in Table 10.

Table 10. Laboratory procedures used in testing engineering properties.

Laboratory procedure	Reference
Optimum moisture and density (standard proctor compaction)	ASTM D698-78
Unconfined compression	ASTM D2166-85
One-dimensional swell	ASTM D4546-85
One-dimensional consolidation	ASTM D2435-80
Falling head permeability	ASTM D2445-80

2.1.2 Results and Discussion of Laboratory Studies

The following sections present the findings of laboratory studies conducted to assess the chemical, mineralogical, physical, and engineering properties of FGD by-products and mixtures of FGD by-products with either soil or spoil.

2.1.2.1 Chemical and Mineralogical Results and Discussion

Chemical and mineralogical studies were conducted to characterize the FGD by-products and soil and spoil materials used in the long-term laboratory equilibration studies.

2.1.2.1.1 Characterization of FGD By-products

Mineralogy

Mineralogical analyses of the dry FGD by-products were initiated with the identification of crystalline phases using X-ray diffraction. A qualitative identification of all phases was essential in order to reliably interpret other forms of data leading to mineral quantification. Representative X-ray diffraction patterns from the three by-products are presented in Figs. 1 to 3. Patterns such as these provided "fingerprints" of the crystalline phases present.

Scrubber residues from the spray dryer sample (OSU-SPD-03) consisted primarily of Ca(OH)_2 (portlandite) and $\text{CaSO}_3 \cdot \frac{1}{2}\text{H}_2\text{O}$ (calcium sulfite hemihydrate) (Fig. 1). Both phases are indicative of low reaction temperatures in the scrubbing process. By comparison, the LIMB by-product (EDG-LIMB-14) contained significant amounts of CaO (lime), CaSO_4 (anhydrite), and CaCO_3 (calcite) (Fig. 2). In the LIMB process, the sorbent is injected directly into the upper portion of the boiler which results in much higher reaction temperatures. At these temperatures, the sorbent calcines to form CaO which then reacts with SO_2 and O_2 in the flue gas to form CaSO_4 . The presence of portlandite indicates partial re-hydration of the FGD by-product. Likewise, CaCO_3 reflects re-carbonation of Ca(OH)_2 by reaction of Ca(OH)_2 with atmospheric CO_2 . Analyses of the cyclone ash (TID-PFBC-03) from the Tidd facility indicated that in addition to CaSO_4 , the ash was enriched with $\text{CaMg(CO}_3)_2$ (dolomite), CaCO_3 , and (MgO) periclase (Fig. 3). Periclase was a calcination of the dolomite used as a sorbent in the PFBC process.

Mineralogical analyses are normally semiquantitative, at best, because reference minerals that closely match the physical, chemical and structural characteristics of sample materials are difficult to obtain. Furthermore, no single analytical technique was adequate for "quantifying" the full range of mineral phases present in the dry FGD by-products. Data from multiple procedures were used, whenever possible, to support or verify results from the analytical procedure of choice.

Figure 1. X-ray diffraction pattern of OSU-SPD-03.

Figure 2. X-ray diffraction pattern of EDG-LIMB-14.

Figure 3. X-ray diffraction pattern of TID-PFBC-03.

Thermal analysis was found to be an excellent tool for quantifying those compounds that were susceptible to phase transitions with heating. Standard reference materials were used to develop a basic understanding of the thermal reactivity of each phase, but the thermal characteristics of the FGD by-products were never identical to standard specimens. Variations in thermal behavior resulted from differences in particle size, crystallinity and the presence of other phases. Spiked samples, selective dissolution, X-ray diffraction analysis, and the technical literature were used to guide interpretations.

Weight losses resulting from phase transitions due to calcination, dehydroxylation, or dehydration were used to quantify the mineral phases present in the by-products (Table 1). The thermal events associated with decomposition of $\text{Ca}(\text{OH})_2$ and $\text{CaSO}_3 \cdot \frac{1}{2}\text{H}_2\text{O}$ could be defined with little interference from other thermal transitions. The content of either CaCO_3 or $[\text{CaMg}(\text{CO}_3)_2]$ could also readily be determined by the weight loss associated with CO_2 evolution. The analysis was straightforward when only one of the two carbonate phases was present; however, both calcite and dolomite were present in the TID-PFBC-03 sample, and the weight loss event resulting from dolomite decomposition was not completed prior to the onset of calcite decay. In this sample, calcite and dolomite were quantified using the gasometric Chittick technique.

The contents of CaSO_4 , CaO , and MgO (Table 1) were not measured by thermogravimetric analysis. CaSO_4 contents were calculated from total S values while CaO was determined by first allocating elemental Ca from total chemical analyses to all other Ca species present in the samples and then converting the remaining Ca to CaO . Likewise, dolomite in TID-PFBC-03 was quantified by using the Chittick apparatus, and a corresponding amount of total Mg was allocated to dolomite based on a 1:1 Mg to Ca ratio. The remaining Mg was then assigned to MgO .

Fly ash contents of the FGD by-products ranged from 21 to 32 wt% (Table 1). Although fly ash was a significant component, it was "hidden" in X-ray diffraction patterns from the whole by-product specimens because of its mostly X-ray amorphous character. No attempt was made to quantify the fly ash mineralogy; however, X-ray diffraction analyses of the water and acid-insoluble residues showed that all were composed of glass, quartz (SiO_2), hematite (Fe_2O_3), mullite ($\text{Al}_6\text{Si}_2\text{O}_{13}$) and a magnetic phase with the characteristics of partially oxidized magnetite (Fe_3O_4). A representative X-ray diffraction pattern from the fly ash component of EDG-LIMB-14 is shown in Fig. 4.

Figure 4. Representative X-ray diffraction pattern for fly ash component of FGD by-products.

Total neutralizing potential and pH. Total neutralizing potential (TNP), expressed as percent CaCO_3 equivalency (CCE), was similar for all three by-products, ranging from 59.4 to 64.7 (Table 1). The sources of alkalinity in the samples varied according to the FGD technology and sorbents used and included CaO , Ca(OH)_2 , CaCO_3 , $\text{Ca,Mg(CO}_3\text{)}$, and MgO (Table 1). The PFBC by-product contained mostly calcite and dolomite and yielded a suspension pH of 10.4. The higher contents of CaO and Ca(OH)_2 in the SPD and LIMB by-products is reflected in high suspension pH's (12.5) when these materials were mixed with H_2O . Long-term exposure of these by-products to moisture and atmospheric CO_2 should cause conversion to CaCO_3 with an associated decrease in the equilibrium pH to around 8.3.

All three by-products appear to contain sufficient acid neutralizing potential to warrant their use as alkaline amendments. The fine particle size of these materials, as well as the reactive nature of CaO , Ca(OH)_2 , and MgO , suggests that they will react rapidly to neutralize spoil acidity.

Total Elemental Analysis. The total elemental concentrations in the FGD by-products selected for use in the spoil-FGD mixtures are listed in Table 1.

The bulk chemical composition of the FGD by-products was dominated by Mg (PFBC), Ca, S, Si, Fe, and Al (Table 1). The abundance of Ca, Mg, and S reflected the amounts of excess sorbent and scrubber residue in the various by-products. On the other hand, Fe, Si, and Al are primarily associated with the fly ash component (acid-insoluble residue) and would tend to increase in abundance as the by-products are weathered in an open environment. Chemical analyses of acid-insoluble residues from the FGD by-products (data not given) showed this fraction to also contain many of the trace elements associated with the by-products. Elements showing partitioning primarily into the fly ash were Al, As, Ba, Be, Co, Cr, Cu, Fe, K, Li, Mn, Ni, P, Pb, Si, and V. In contrast, Ca, S, and Mg showed an inverse relationship with the concentration of acid insoluble residue. Intermediate elemental partitioning was exhibited by B, Cd, Mo, Se, and Sr indicating these elements occurred in both the acid-soluble and insoluble fractions. Total trace element concentrations of the acid-insoluble residues varied between sources and samples suggesting a dependency primarily on coal composition and secondly, but of lesser importance, on operating conditions within a given coal-fired facility.

2.1.2.1.2 Characterization of Mine Spoils

Mineralogy. The bulk mineralogies of the two mine spoils were similar. Both consisted primarily of quartz (SiO_2), mica [$(\text{K,Na})\text{Al}_2(\text{AlSi}_3\text{O}_{10})(\text{OH}_2)$], kaolinite [$\text{Al}_2\text{Si}_2\text{O}_5(\text{OH})_4$], and feldspar [$(\text{K,Na})\text{AlSi}_3\text{O}_8$] (Figs. 5 and 6); however, the proportions of various constituents in the samples could only be estimated from the relative X-ray diffraction peak intensities (Table 2). X-ray diffraction analysis also detected jarosite [$\text{KFe}_3(\text{SO}_4)_2(\text{OH})_6$] in the Fleming spoil while the R&F spoil contained a small amount

Figure 5. X-ray diffraction pattern of Fleming spoil.

Figure 6. X-ray diffraction pattern of R&F spoil.

of gypsum ($\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$). Jarosite is a common oxidation product when iron sulfides are brought to the surface during mining. When exposed to surface weathering conditions, FeS_2 (pyrite) becomes unstable and oxidizes to form Fe^{2+} and sulfuric acid. When the Fe^{2+} is further oxidized and hydrolyzed, jarosite and other mineral phases may form with the release of additional acidity. The gypsum in the R&F spoil most likely results from the reaction of CaCO_3 with sulfuric acid formed by similar processes. Gypsum is relatively soluble and is a transient phase in the weathering process.

Suspension pH and Potential Acidity. The spoils were extremely to moderately acid with pH values of 3.0 and 5.2 (Table 2). The extremely acidic nature of the Fleming spoil arises from the oxidation of iron disulfide minerals that are commonly found in coal and associated underclays. The higher pH and lower potential acidity of the R&F spoil reflects a lower content of these materials (Table 2).

Total Elemental Analysis. The total concentrations of Al, As, B, Ba, Be, Ca, Cd, Co, Cr, Cu, Fe, K, Li, Mg, Mn, Mo, Ni, P, Pb, S, Se, Si, Sr, V and Zn in the mine spoils selected for use in the spoil-FGD mixtures are listed in Table 2. The chemical composition of the mine spoils is dominated by Si, Al, Fe, K, and S. This chemistry is consistent with the mineralogy as discussed in section 2.1.2.1.1. Trace element concentrations are similar (within an order of magnitude) to those observed in the FGD by-products (Table 2).

2.1.2.1.3 Chemistry and Mineralogy of Leachates from Spoil-FGD Mixtures

Leachate composition was primarily a function of FGD-by-product type, application rate, and equilibration time (Appendix Table 46). Spoil type, however, had little influence on pore water chemistry except at the lowest FGD application rate.

All rates of FGD by-products increased the solution pH significantly (Appendix Table 46 and Fig. 7). There was a tendency in most mixtures for the pH to decrease with time, and this effect was usually most pronounced at the lowest application rate (Appendix Table 46). After 132 days the LIMB and SPD-amended spoils attained mean pH levels of 11 while the PFBC mixtures attained a mean pH of 8.2. The lower final pH of the PFBC mixes can be attributed to the PFBC process which uses dolomite as a sorbent and which produces little CaO .

Compared to the PFBC material, the LIMB and SPD mixes released greater quantities of soluble salts to solution as evidenced by the leachate specific conductance (Appendix Table 46 and Fig. 8). In addition, conductance increased with an increase in by-product application rate. (Appendix Table 46 and Fig. 9). Dolomite is relatively insoluble and accounts for the lower conductance of the PFBC-spoil mixes.

In most instances, addition of FGD by-products resulted in initial decreases in leachate E_H but a recovery of E_H was observed with time (Appendix Table 46 and Fig's. 10 and 11). The lowest values were observed for samples amended with the SPD by-products due to the presence of CaSO_3^{2-} in the by-product. However, E_H values

Figure 7. Effect of FGD by-products on leachate pH values from mine spoils after 132 days of reaction.

Figure 8. Effect of by-product source on leachate conductance from mixtures of FGD and Fleming spoil (40 wt. %).

Figure 9. Effect of application rate on leachate conductance from mixtures of LIMB and Fleming spoil.

Figure 10. Effect of by-product source on leachate E_H from mixtures of FGD and Fleming spoil (40 wt. %).

Figure 11. Effect of application rate on leachate E_h from mixtures of LIMB and Fleming spoil.

remained positive for the duration of the study.

Leachates from the LIMB and SPD amended spoils were dominated by Ca, S and Cl, whereas the PFBC mixes yielded significant amounts of dissolved Mg in addition to the aforementioned ions (Appendix Table 46). The concentrations of dissolved Ca, S, and Cl were independent of spoil type. They were more related to the characteristics of the by-products themselves (Appendix Table 46). FGD amendments resulted in initial increases in leachate Ca followed by decreases in solution concentrations with time (Fig. 12). FGD amendments also increased S in the leachates from both spoils (Fig. 13). Concentrations were greater in the LIMB and PFBC amended spoils than the SPD mixes (Fig. 13), presumably because the SPD materials were composed primarily of low solubility SO_3^{2-} reaction products while the LIMB and PFBC materials contained more soluble SO_4^{2-} forms of S. Initial leachate concentrations of Cl tended to increase with increased FGD amendment rates (Appendix Table 46); however, all leachates exhibited low solution concentrations of Cl after the second leaching (14 d) (Fig. 14). Significant quantities of dissolved Mg were measured only in leachate samples from the PFBC-spoil mixtures due to the presence of dolomite and MgO (Fig. 15). Leachate Mg concentrations in the PFBC mixes also showed a dependence on spoil type. At any given time, Mg concentrations were approximately 4 times greater in mixtures with the Fleming spoil as compared to the R&F spoil. This result probably reflects the lower initial pH of the Fleming spoil and the relatively slow dissolution of dolomite (as compared to CaO , Ca(OH)_2 , or CaCO_3) to counter this acidity.

Leachate concentrations of Al, Fe, Mn, and Zn decreased with additions of FGD amendment. The solubility of these metals is greatest under acid conditions and decreases rapidly with increasing pH (Bohn et al., 1985). Concentrations of Fe and Mn decreased to near or at detection limits with the addition of 10 wt. % by-product. Aluminum and Zn concentrations, however, showed continual concentration decreases as by-product rates increased from 10 to 40 wt %. Reductions in soluble Al are particularly significant because this element is commonly present at phytotoxic levels in acid spoils.

Leachate concentrations of B followed similar trends in both the R&F and Fleming spoils, exhibiting fairly low solution concentrations after 14 days of reaction (Fig. 16). However, initial solution concentrations in the LIMB and SPD mixes were somewhat higher than those in the PFBC mixes (Fig. 16). The unweathered PFBC by-products contained the least total B and the PFBC mixes also had the lowest pH values (7.8 to 8.9). Boron solubility and mobility decrease as pH decreases from the alkaline range (Sposito, 1989). These factors most likely resulted in the lower B values.

Appendix Table 47 gives leachate concentrations of seven elements (Hg not available, Co not regulated) regulated by the Ohio Environmental Protection Agency with respect to drinking water standards (Table 11). In the unamended mine spoils, most trace element concentrations were below maximum contaminant levels (MCL) with respect to drinking water standards for the duration of the study (Appendix Table 47).

Figure 12. Effect of FGD by-product additions (40 wt. %) on leachate Ca from mine spoils.

Figure 13. Effect of FGD by-product additions (40 wt.%) on leachate S from mine spoils.

Figure 14. Effect of FGD by-product additions (40 wt.%) on leachate Cl from mine spoils.

Figure 15. Effect of FGD by-product additions (40 wt. %) on leachate Mg from mine spoils.

Figure 16. Effect of FGD by-product additions (40 wt. %) on leachate B from mine spoils.

Table 11. Ohio EPA Maximum Contaminant Level (MCL) for drinking water standards.¹

Element	Concentration
	(mg L ⁻¹)
As	0.05
Cd	0.005
Cr	0.1
Cu	1.3 ²
Ni	0.1
Pb	0.015 ²
Se	0.05

¹Ohio Administrative Code , Rule 37454-81-11

²Action Level

However, in the 132 day leachates, As and Se exceeded MCL's in the Fleming spoil whereas Pb and Se concentrations were greater than defined levels in the R&F spoil (Appendix Table 47).

In the leachates obtained after 132 days of reaction of the 40 wt. % amended spoils, As, Cd, Co, and Cr were below detection for all the mixtures (Appendix Table 47). Solution concentrations of Cu were also below detection except for the SPD-amended Fleming spoil. Leachate concentrations of Ni in the amended spoils ranged from below detection to 0.050 mg L^{-1} . These concentrations, however, were lower than the MCL for drinking water. Lead was detected in the SPD and PFBC amended R&F spoil at levels of 0.026 and 0.023 mg L^{-1} , respectively. These solution concentrations are approximately 65% greater than the action level defined by drinking water standards. In all other leachates, Pb was below the detection limit.

The leachate concentrations of Se in the amended spoils were variable ranging from below detection to 0.421 mg L^{-1} . Seventy-eight percent of the leachates exceeded the MCL for Se at day one (Fig. 17). By comparison, less than 30% of the leachates exceeded the MCL for Se after 26 days of reaction (Fig. 17). In addition, no correlation between leachate Se concentrations and FGD amendment rate could be found.

It should be noted that the 40 wt. % amended mixtures represented a "worst case scenario". A more realistic field application of 10-12.5 wt. % ($100\text{-}125 \text{ tons acre}^{-1}$) by-product would result in lower solution concentrations for many elements (Appendix Tables 46 and 47).

Table 12 contains values of log SI for selected solids corresponding to the sampling events at 1, 62, and 132 days. Values of log SI > 0 indicate that the sample solutions were supersaturated with respect to the specified mineral, so that precipitation of the solid could occur. Values of log SI < 0 indicate undersaturation with respect to a given solid, generally precluding its precipitation. Table 13 provides a comparison of mineral phases predicted by geochemical modeling to those phases observed by X-ray diffraction after 132 days of reaction (40 wt. % rate). Although the leachate samples from the 40 wt. % rate at 132 days were not equilibrium solutions, geochemical equilibrium modeling did indicate that the LIMB and SPD amended spoils were undersaturated with respect to gypsum while the PFBC amended spoils were near equilibrium to slightly oversaturated at 132 days (Table 12). Gypsum was not detected by X-ray diffraction in the LIMB and SPD amended spoils whereas the PFBC sample contained gypsum as evidenced by X-ray diffraction and scanning electron microscopy (Figs. 18 and 19).

Saturation index calculations indicated the LIMB-amended Fleming spoil to be supersaturated with respect to ettringite while predicting the LIMB amended R&F spoil to be undersaturated (Table 12). Conversely, the SI values indicated undersaturation with respect to ettringite in the SPD-amended Fleming spoil and supersaturation in the SPD amended R&F spoil (Table 12). However, ettringite was observed by X-ray diffraction and scanning electron microscopy in the LIMB and SPD amended mixtures

Figure 17. Percentage of leachates exceeding Se drinking water MCL by sampling date.

Table 12. Values of log SI for selected solids after 1, 62, and 132 days of reaction of mine spoil-FGD mixtures.

SAMPLE	Time	Mineral								
	days	Anhydrite	Brucite	Calcite	Dolomite	Ettringite	Gypsum	Lime	Periclase	Portlandite
Fleming Spoil	1	-2.86	-15.09	-11.33	-22.79	-60.84	-2.65	-30.92	-19.81	-20.80
	62	-3.98	-14.56	-11.03	-21.96	-64.45	-3.77	-30.17	-18.93	-20.50
	132	-3.75	-14.22	-10.58	-21.16	-61.52	-3.54	-10.99	-8.24	-20.05
Flem-LIMB(10%)	1	0.12	-3.52	0.21	-8.05	-6.57	0.33	-10.99	-8.24	-0.87
	62	-0.34	-13.91	-8.31	-18.58	-46.18	-0.12	-27.90	-18.63	-17.78
	132	-0.67	-4.12	-1.08	-3.80	-11.98	-0.46	-18.42	-8.83	-8.30
Flem-LIMB(20%)	1	0.28	-2.94	0.24	-8.35	-0.50	0.49	-10.05	-7.65	0.07
	62	-0.63	-5.32	0.15	-7.43	-0.53	-0.42	-13.55	-10.04	-3.43
	132	-0.41	-1.64	0.11	-2.72	-1.52	-0.20	-14.65	-6.36	-4.53
Flem-LIMB(40%)	1	0.15	-2.79	0.24	-8.40	-0.55	0.36	-9.87	-7.51	0.26
	62	-0.19	-2.75	0.22	-8.25	-2.02	0.03	-10.03	-7.47	0.09
	132	-0.82	-2.97	0.18	-7.91	4.20	-0.61	-10.65	-7.68	-0.53
Flem-SPD(10%)	1	-0.24	-3.93	0.20	-7.74	4.34	-0.03	-11.74	-8.65	-1.62
	62	-0.35	-2.32	-0.24	-2.27	-5.54	-0.14	-16.49	-7.04	-6.37
	132	-0.06	-3.87	-0.96	-3.37	-18.44	0.15	-18.37	-8.59	-8.24
Flem-SPD(20%)	1	-0.08	-3.17	0.25	-8.17	7.46	0.13	-10.46	-7.89	-0.34
	62	-0.36	0.40	0.17	-2.54	1.74	-0.15	-12.67	-4.32	-2.55
	132	-0.28	-0.59	0.17	-2.23	-0.05	-0.07	-13.99	-5.31	-3.87
Flem-SPD(40%)	1	0.05	-2.91	0.25	-8.32	-1.21	0.26	-10.05	-7.63	0.07
	62	-1.13	-2.60	0.21	-8.25	4.10	-0.92	-9.88	-7.32	0.24
	132	-1.11	-3.39	0.24	-7.91	-6.53	-0.90	-10.96	-8.11	-0.84

Table 12. continued

SAMPLE	Time	Mineral								
	days	Anhydrite	Brucite	Calcite	Dolomite	Ettringite	Gypsum	Lime	Periclase	Portlandite
Flem-PFBC(10%)	1	0.04	-4.63	-2.32	-4.43	-13.22	0.25	-20.80	-9.35	-10.67
	62	-1.73	-2.74	-1.52	-2.94	-25.05	-1.52	-18.80	-7.46	-8.68
	132	-2.31	-3.68	-2.09	-4.02	-19.79	-2.10	-19.78	-8.39	-9.66
Flem-PFBC(20%)	1	0.11	-3.30	-1.52	-2.91	-10.00	0.32	-19.40	-8.02	-9.28
	62	-0.54	-2.96	-1.33	-2.72	-21.17	-0.33	-18.86	-7.68	-8.74
	132	-1.71	-4.75	2.72	5.46	-29.41	-1.57	-20.69	-9.44	-10.60
Flem-PFBC(40%)	1	0.24	-0.93	-0.42	-0.81	-14.62	0.45	-16.91	-5.64	-6.79
	62	-0.13	-3.74	-1.38	-3.26	-20.40	0.08	-19.20	-8.46	-9.08
	132	-0.14	-4.15	-1.73	-3.73	-12.69	0.07	-19.85	-8.87	-9.72
R&F Spoil	1	-0.65	-9.81	-6.16	-12.35	-33.73	-0.44	-25.73	-14.53	-15.61
	62	-1.95	-7.46	-4.00	-8.22	-36.00	-1.74	-23.19	-12.18	-13.07
	132	-1.69	-6.41	-3.33	-6.79	-22.78	-1.48	-22.24	-11.13	-12.12
R&F-LIMB(10%)	1	0.22	-2.80	0.24	-8.41	8.95	0.43	-9.86	-7.51	0.26
	62	-1.12	-4.87	0.08	-7.26	-1.42	-0.91	-13.40	-9.58	-3.28
	132	-0.80	-1.78	0.03	-2.79	-3.45	-0.59	-14.89	-6.50	-4.77
R&F-LIMB(20%)	1	0.15	-2.79	0.23	-8.37	8.20	0.36	-9.92	-7.51	0.20
	62	-1.43	-2.60	0.20	-8.24	3.21	-1.22	-9.90	-7.31	0.22
	132	-1.58	-3.94	0.08	-7.30	-0.49	-1.37	-12.44	-8.66	-2.32
R&F-LIMB(40%)	1	0.26	-2.80	0.25	-8.47	9.32	0.47	-9.79	-7.52	0.33
	62	-0.04	-2.64	0.23	-8.41	-0.91	0.17	-9.73	-7.36	0.39
	132	-0.34	-2.62	0.21	-8.32	-2.07	-0.13	-9.83	-7.34	0.29

Table 12. continued

SAMPLE	Time	Mineral								
	days	Anhydrite	Brucite	Calcite	Dolomite	Ettringite	Gypsum	Lime	Periclase	Portlandite
R&F-SPD(10%)	1	-0.14	-2.89	0.21	-8.18	-2.33	0.07	-10.25	-7.61	-0.13
	62	-2.14	-3.91	0.06	-7.24	-3.36	-1.93	-12.52	-8.63	-2.39
	132	-1.36	-1.46	-0.09	-2.17	-8.10	-1.15	-15.44	-6.18	-5.32
R&F-SPD(20%)	1	0.07	-2.91	0.23	-8.29	7.46	0.28	-10.11	-7.63	0.01
	62	-1.71	-2.60	0.21	-8.25	-6.33	-1.50	-9.88	-7.31	0.24
	132	-1.71	-2.95	0.20	-7.93	1.67	-1.50	-10.57	-7.67	-0.45
R&F-SPD(40%)	1	0.08	-2.78	0.23	-8.37	9.12	0.29	-9.92	-7.50	0.20
	62	-1.53	-2.60	0.21	-8.27	-5.71	-1.32	-9.86	-7.32	0.26
	132	-1.84	-2.71	0.20	-8.12	3.08	-1.63	-10.14	-7.43	-0.02
R&F-PFBC(10%)	1	-0.11	-0.78	-0.22	-0.80	-14.62	0.10	-16.38	-5.50	-6.26
	62	-1.24	-4.08	-2.07	-4.13	-26.03	-1.03	-20.04	-8.79	-9.92
	132	-2.29	-3.44	-2.18	-3.98	-29.08	-2.08	-19.78	-8.16	-9.66
R&F-PFBC(20%)	1	0.09	0.82	0.07	-0.43	-10.40	0.31	-14.57	-3.90	-4.45
	62	-0.59	-3.15	-1.14	-2.90	-20.54	-0.38	-18.48	-7.87	-8.36
	132	-0.65	-3.53	-1.60	-3.34	-22.57	-0.44	-19.35	-8.25	-9.23
R&F-PFBC(40%)	1	0.23	1.10	0.16	-0.69	1.05	0.44	-13.85	-3.62	-3.73
	62	0.01	-0.03	0.17	-1.38	-0.81	0.22	-14.27	-4.75	-4.15
	132	-0.03	-2.22	-0.51	-1.85	-16.25	0.18	-17.34	-6.94	-7.22

Table 13. Mineral phases predicted and observed after 132 days of reaction (40 wt.%).

Sample	Anhydrite		Calcite		Dolomite		Ettringite		Gypsum		Portlandite	
	pred. [†]	obs. ^{††}	pred.	obs.	pred.	obs.	pred.	obs.	pred.	obs.	pred.	obs.
Fleming-LIMB(40%)	✓		✓	✓			✓	✓	✓		✓	
Fleming-SPD(40%)			✓	✓				✓				✓
Fleming-PFBC(40%)	✓			✓		✓			✓	✓		
R&F-LIMB(40%)			✓	✓				✓	✓		✓	✓
R&F-SPD(40%)			✓	✓			✓	✓			✓	✓
R&F-PFBC(40%)	✓		✓	✓		✓			✓	✓		

[†] Predicted by geochemical modeling (log SI > -1.0)

^{††} Observed by X-ray diffraction

Figure 18. X-ray diffraction pattern of Fleming-PFBC (40 wt.%) after 132 days of reaction.

Figure 19. SEM photomicrograph of monoclinic gypsum crystals (top, center) in Fleming spoil with PFBC (40 wt%) after 132 days of reaction.

of both spoils (Figs. 20, 21, 22 and 23). The PFBC amended spoils were highly undersaturated with respect to ettringite, and no ettringite was detected by X-ray diffraction in these samples. The precipitation of gypsum in the PFBC-amended samples and ettringite in the LIMB and SPD-amended samples is likely responsible for the decrease in Ca^{2+} and SO_4^{2-} concentrations in the leachate solutions with time (Appendix Table 46). The geochemical modeling suggests that in the absence of significant recarbonation, ettringite will form as the final reaction product in the LIMB and SPD amended materials while gypsum is the solid phase controlling Ca and S solubilities in the PFBC amended materials.

Changes in solid phase composition in the FGD-spoil mixes were independent of the spoil. All major mineralogical components of the spoil were still detectable by X-ray diffraction after 132 d of weathering (Table 13). The final reaction products were more strongly influenced by the characteristics of the by-products themselves. In the LIMB mixes, CaSO_4 and CaO hydrated rapidly to form $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ and $\text{Ca}(\text{OH})_2$, respectively (Fowler et al., 1992). With longer reaction time the conversion of $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ to ettringite $[\text{Ca}_6\text{Al}_2(\text{SO}_4)_3(\text{OH})_{12} \cdot 26\text{H}_2\text{O}]$ was observed (Fig. 20). Ettringite was also observed in X-ray diffraction patterns of the SPD mixes, though in lower quantities than in the LIMB-amended samples. As in previous studies (Fowler et al., 1992), precipitation of $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ does not necessarily precede ettringite formation in SPD materials. Ettringite formation in the SPD materials may be limited by the oxidation of SO_3^{2-} from $\text{CaSO}_3 \cdot \frac{1}{2}\text{H}_2\text{O}$ to SO_4^{2-} . However, oxidation may also not be necessary, as McCarthy et al., (1992) have described the synthesis and characterization of a SO_3^{2-} ettringite. A distinction between the two phases could not be made with the present data. In the PFBC mixes, Ca weathering reactions resulted in the precipitation of $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ as a major Ca phase with no conversion of the $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ to ettringite detected at 132 days (Fig. 18). The conditions necessary for ettringite formation include sources of soluble (or solubilizable) Ca, Al, and S, pH values between 11.5 and 12.5, and excess water (McCarthy et al., 1992). The significantly lower solution pH in the PFBC amended spoils (7.8 to 8.9) most likely inhibited ettringite formation.

In the 40 wt.% LIMB and SPD amended mixtures, pH values were 12.2 or higher and ettringite was present at 132 days. Ettringite forms and remains fully stable only in highly alkaline conditions. In a laboratory study, Myneni et al. (1993), observed that synthetic ettringite was a metastable mineral dissolving incongruently to gypsum and gibbsite below a pH of 10.7. In addition, McCarthy et al. (1992), equilibrated ettringite with deionized water with solution pH controlled between 0 and 13. As pH decreased to below 11, gypsum and poorly crystalline $\text{Al}(\text{OH})_3$ were observed in the solid fraction. At pH's between 6 and 9, ettringite dissolved incongruently into Ca and oxyanion species and noncrystalline Al rich solids. At pH's below 5, significant concentrations of Al were measured with complete dissolution of ettringite at a pH of about 2. The stability of ettringite as a final reaction product in these weathering environments is questionable. Thus, the rate of which this dissolution occurs would appear to be dependent on the acidity of the soil/spoil material, the pH buffering capacity of the FGD material, and the ratio of the soil/spoil-FGD mixture.

Figure 20. X-ray diffraction pattern of Fleming-LIMB (40 wt.%) after 132 days of reaction.

Figure 21. X-ray diffraction pattern of R&F-SPD (40 wt.%) after 132 days of reaction.

Figure 22. SEM photomicrograph of needle-like ettringite crystals in Fleming spoil with LIMB (40 wt%) after 132 days of reaction.

Figure 23. SEM photomicrograph of needle-like ettringite crystals in R&F spoil with SPD (40 wt%) after 132 days of reaction.

2.1.2.3 Physical Characterization Results and Discussion

The Fleming spoil contained 42% sand, 49% silt and 9% clay; while the R&F spoil contained 50% sand, 34% silt and 17% clay. From Phase 1 testing, the percent (by weight) of the 3 FGD materials finer than 0.025 mm diameter was 98.5%, 86.5% and 56.5% for the LIMB, SPD, and PFBC materials, respectively. Thus, addition of these fine textured FGD materials to either of the two spoils would create a finer texture of the mix relative to that of the unamended spoils.

A comparison of the compression index for the two spoils indicates that the R&F spoil is more compressible as compared with the Fleming spoil (Table 14). The relative similarity in sand contents between these two spoils would suggest that the decreased compressibility of the Fleming spoil was due to a higher aggregate strength. In terms of FGD addition to the spoils, the compression index generally decreased for all mixing rates and FGD sources as compared with the native spoils. The LIMB material mixed with the Fleming spoil yielded the most dramatic decrease in compression index at 10% addition while addition of PFBC yielded the least change at this amendment rate. Subsequently higher rates of the LIMB material produced little further change while higher rates of PFBC produced a relatively large decrease in mix compressibility. A similar response was observed for the R&F-FGD mixes both in terms of the incorporation rate and FGD source effects on compression index.

Bulk densities of the individual spoils and the spoil-FGD mixes generally exhibited a similar relationship as compared with the compression index values (Table 15). This is not surprising since the bulk densities were determined in a compressed state and a spoil with a higher compressibility generally exhibits higher compressed bulk densities. The R&F spoil had a high bulk density relative to the Fleming spoil. The LIMB material mixed with either spoil yielded the most dramatic decrease in bulk density at 10% addition while addition of PFBC yielded the least change at this incorporation rate. Subsequently, higher rates of the LIMB material produced little further change while higher rates of PFBC produced a relatively large decrease in mix bulk density for the R&F spoil.

Decreasing α and increasing n values of the Van Genuchten equation yield water retention curves showing that samples retained water near saturation at progressively more negative matric potentials. This progression was evident over the range of 10 to 40% addition of all FGD source materials mixed with the Fleming spoil (Table 16). Thus, increasing amounts of FGD translated into increased water retention in the Fleming spoil-FGD mixes. This was not, however, the case for 10% incorporation of LIMB and SPD relative to unamended Fleming spoil where the FGD amendment yielded less water retention. The R&F spoil retained water to a much greater extent as compared with the Fleming spoil. This is likely due to the increased compaction experienced by the R&F spoil. All FGD materials added at 10%, however, exhibited increased α and decreased n values characteristic of decreased water retention as compared with the unamended R&F spoil. In this regard, the LIMB yielded the greatest effect while the PFBC the least. Increasing rates from 10 to 40% for the LIMB, however,

Table 14. Mean compression index of the spoil/FGD mixes.

Rate	Compression index		
	LIMB	SPD	PFBC
wt %	Fleming spoil		
0	0.091	0.091	0.091
10	0.054	0.072	0.082
20	0.044	0.046	0.075
40	0.051	0.054	0.050
LSD _{0.05}		0.0075	
	R&F spoil		
0	0.113	0.113	0.113
10	0.051	0.073	0.115
20	0.055	0.059	0.080
40	0.062	0.045	0.051
LSD _{0.05}		0.0074	

Table 15. Mean compressed bulk density of the spoil/FGD mixes.

Rate	Bulk Density		
	LIMB	SPD	PFBC
wgt %	Mg m ⁻³		
	Fleming spoil		
0	1.32	1.32	1.32
10	1.12	1.19	1.32
20	1.14	1.11	1.28
40	0.98	0.98	1.29
LSD _{0.05}		0.03	
	R&F spoil		
0	1.82	1.82	1.82
10	1.16	1.18	1.59
20	1.18	1.12	1.46
40	1.07	0.97	1.10
LSD _{0.05}		0.09	

Table 16. α and n values from fitting the Van Genuchten equation to water retention data of the spoil/FGD mixes.

Rate	α			n		
	LIMB	SPD	PFBC	LIMB	SPD	PFBC
wgt %	kPa^{-1}					
Fleming spoil						
0	0.015	0.015	0.015	1.37	1.37	1.37
10	0.054	0.043	0.014	1.25	1.30	1.27
20	0.012	0.034	0.002	1.39	1.31	1.72
40	0.002	0.002	0.002	2.90	2.29	2.12
R&F spoil						
0	0.009	0.009	0.009	1.70	1.70	1.70
10	5.500	0.700	0.072	1.23	1.19	1.21
20	0.005	1.100	0.780	1.69	1.18	1.15
40	0.003	0.003	0.500	2.09	1.98	1.36
Mean c.v.(%)		43.0		8.7		
Mean $r^2 = 0.92$						

progressively increased water retention while for the PFBC, increasing rates over this range progressively decreased water retention. For the SPD material, the mix returned to a highly retentive state comparable to the unamended R&F spoil at the 40% incorporation rate.

Saturated hydraulic conductivity of the R&F spoil was nearly one order of magnitude less than that for the Fleming spoil (Table 17). Addition of LIMB at 10% and 40% did not change, and at 20% decreased the conductivity relative to the unamended Fleming spoil. Further, SPD did not affect the conductivity at all incorporation rates relative to the unamended Fleming spoil. Addition of 10% PFBC decreased the conductivity relative to the unamended spoil, while PFBC at 40% yielded equivalent conductivity values. Addition of LIMB to the R&F spoil at 10% addition dramatically increased hydraulic conductivity. Higher rates of LIMB, however, served to decrease conductivity values. The highest conductivity for the R&F-SPD mixes occurred at 20% addition with all incorporation rates exhibiting higher conductivities relative to the unamended R&F spoil. Incorporation of PFBC at 10% yielded no change in conductivity relative to the unamended spoil. At the 20 and 40% rates, however, an increased conductivity was observed.

Table 17. Mean $\log_{10}$ saturated hydraulic conductivity of the spoil/FGD mixes.

Rate	Saturated conductivity		
	LIMB	SPD	PFBC
wgt %	$\log_{10} \text{ cm s}^{-1}$		
	Fleming spoil		
0	-4.45	-4.45	-4.45
10	-4.55	-4.57	-5.54
20	-4.90	-4.57	-5.00
40	-4.27	-4.41	-4.42
LSD _{0.05}		0.25	
	R&F spoil		
0	-5.41	-5.41	-5.41
10	-1.02	-2.56	-5.35
20	-2.29	-1.63	-1.43
40	-4.08	-3.79	-1.32
LSD _{0.05}		0.33	

2.1.2.4 Engineering Results and Discussion

Both dry and wet FGD materials were studied in Phase 2. The corresponding test results are analyzed in sections 2.1.2.4.1 and 2.1.2.4.2, respectively.

2.1.2.4.1 Dry FGD

During Phase 2, three separate laboratory tests were conducted on dry FGD material.

Swelling - A determination of the amount of swelling that could be expected in a fill constructed of compacted FGD is a necessary step in the process of establishing the suitability of these materials in engineered structures. In order to evaluate the swelling potential of the FGD by-products, laboratory tests which were begun during the Phase 1 study were continued through Phase 2 so that the long term swell in these materials could be evaluated. The swell test is summarized in Table 18. The duration of some tests has exceeded three years. Swelling continued throughout the testing period although at a rate that decreased with time. Figures 24 through 27 show measured volume increases for four types of dry FGD by-product.

Durability - The strength of the FGD, when subjected to alternating cycles of freezing and thawing can be used as a measure of the suitability of the material as a road base or sub-base.

Samples of PFBC ash from the Tidd plant were compacted at two moisture contents. The first moisture content was only slightly wetter than the optimum water content determined for this material in the Phase 1 studies. The second moisture content was approximately 10% higher than the optimum value. The laboratory tests presented in Table 19 describe the thermal conditioning each of the six samples received prior to being subjected to dynamic loading. The conditioned samples were loaded using the waveform depicted in Fig. 28.

The results of dynamic load tests (summarized in Figs. 29 through 33) indicate that the freeze-thaw cycling affected the measured stiffness (represented in these results by the Resilient Modulus, M_r). In tests of two samples prepared at the two different water contents (Fig. 29) the effect of water content on Resilient Modulus is apparent. Much lower values for M_r were recorded for the sample with the high water content. In general the same behavior seen for samples tested after seven days of curing (Fig. 29) persisted throughout the test program (Fig. 30). The effects of repeated thermal cycling (Fig. 31) illustrate that the modulus decreased with increasing temperature cycles. The cycling of the temperature, however, does not appear to have a greater effect on the measured values of M_r than does a constant below freezing temperature (Fig. 32). For example a significant drop in modulus occurred when a sample was frozen before being tested. This drop was similar to that observed when temperatures were cycled. It is clear from the results of several freeze-thaw tests (Figure 33) that the range of recorded values is great. For the parameters tested, the

Table 18. Swelling Test Results

Sample	Plant	FGD Process	Duration (days)	Total Swell (%)
CLS-01A	Edgewater	Coolside	1180	14
LIM-06	Edgewater	LIMB	1171	0.8
OSU-03A	McCracken	Spray dryer	1041	2.4
LIM-10A	Edgewater	LIMB	1013	5.8
OSU-03B	McCracken	Spray dryer	893	2.8
TID3+TID4(40:60)	Tidd	PFBC	888	1.7
LIM-14A	Edgewater	LIMB	881	95.
STOCKPILE#4A	Edgewater	LIMB	522	0.3
STOCKPILE#9-1	Edgewater	PFBC	507	8
FEEDLOT#1-1	Tidd	PFBC	470	0.04
FEEDLOT#3-1	Tidd	PFBC	450	-0.02
STOCKPILE#9-2	Edgewater	LIMB	237	7.1
(x) HST-05	EPRI HST	Spray dryer	595	1
(x) LIM-07	Edgewater	LIMB	425	2.5
(x) LIM-08A	Edgewater	LIMB	425	2.5
(x) LIM-08B	Edgewater	LIMB	404	1.3
(x) CLS-CA1	Edgewater	Coolside	249	6.1
(x) CLS-CA2	Edgewater	Coolside	238	41
(x) LIM-10B	Edgewater	LIMB	184	3.2
(x) LIM-14B	Edgewater	LIMB	184	12
(x) CLS-01B	Edgewater	Coolside	169	11
(x) CLS-01C	Edgewater	Coolside	64	0.1

Note: (x) means the test on that sample has been terminated.

Figure 24. Swelling vs. time: Coolside by-product.

Figure 25. Swelling vs. time: LIMB by-product.

Figure 26. Swelling vs. time : OSU Spray Dryer by-product.

Figure 27. Swelling vs. time: Tidd by-product.

Table 19. Freeze-thaw tests on dry FGD (Tidd PFBC ash) samples.

Days to Cure	Water Content = 30%			Water Content = 22%		
	Sample 1	Sample 2	Sample 3	Sample 4	Sample 5	Sample 6
	----- Number of freeze-thaw cycles prior to cyclic testing -----					
1	0.*	0	NT.**	0	0	NT
7	0	4	NT	4	0	NT
14	0	8	NT	8	0	NT
21	0	12	NT	12	0	NT
28	0	16	0	16	0	0
35	4	20	4	20	4	4
42	8	24	8	24	8	8
49	12	28	12	28	12	12

*0 = Test performed before any freeze-thaw cycles.

**NT = Not tested

Note: Samples 3 and 6 were frozen for the first 28 days.

Figure 28. Resilient Modulus test load pattern.

Figure 29. Resilient Modulus of samples after 7 cycles of Freeze-Thaw.

Figure 30. Resilient Modulus vs time for Freeze-Thaw cycling.

Figure 31. Resilient Modulus as a function of thermal cycling.

Figure 32. Effect of freeze thaw cycling on Resilient Modulus.

Figure 33. Resilient Modulus as a function of curing time before beginning of thermal cycling.

samples that were compacted near the optimum water content had consistently higher values of M_r . A comparison of the measured M_r for the Tidd samples with published values obtained for commonly used base materials is presented in Figure 34. In this figure, the M_r value is given in terms of two material parameters (K_1 and K_2) where:

$$M_r = K_1 \theta^{K_2}$$

θ = sum of the principal stresses ($\sigma_1 + \sigma_2 + \sigma_3$).

The values for the PFBC samples are somewhat lower than those of the more conventionally used materials but could be considered as an alternative material in a number of roadway applications.

Soil/ FGD Mixtures -When the engineering properties of on-site soils can be improved, the need to replace unsatisfactory local soils with select borrow material is lessened potentially resulting in substantially reduced construction costs. Therefore a possible beneficial use of FGD materials would be as a soil amendment. Typical of the types of projects that could benefit from an FGD-modified soil are earth embankments and slopes and backfills for retaining walls. In these applications, two of the most important engineering properties are strength and permeability.

To evaluate the capability of dry FGD by-products to improve the engineering properties of natural on-site soils, laboratory experiments were performed on samples of mixed soil and FGD ash which had been compacted according to ASTM D 698-78.

Strength - Unconfined compression tests were performed according to ASTM Standard D 2166-85 on compacted samples consisting of equal parts (by weight) soil and FGD by-product. Three different soils (kaolin clay, a red clay taken from near Marietta, Ohio and a brown silty clay from the OSU campus) were mixed with two different FGD by-products (Coolside ash from the Edgewater plant in Lorain, Ohio and spray dryer ash from the power plant on the OSU campus). The tests were conducted on the samples at three different times, the first being immediately after compaction (0 days) and then at 7 days and 28 days after compaction.

In previous testing of FGD alone, strength gains were observed as a function of time (Stehouwer et al., 1995). Therefore three test times were included to identify whether or not the soil/FGD mixture would also gain strength with time. The results of these tests (Figures 35 through 37) are shown along with the measured strengths of the unamended soil so that a determination can be made of the usefulness of modifying soil with FGD. It is clear from these figures that substantial strength improvement is generally observed (the OSU-SPD/ Marietta Clay strength increase is apparent only after seven and 28 days) when the soil and FGD are mixed. The use of FGD by-products for soil strength improvement thus seems warranted.

(A: Silty Sands; B: Sand Gravels; C: Sand Aggregate Blends; D: Crushed Stone; E: Limerock; F: Slag; G: Dry FGD)

Figure 34. Comparison of resilient modulus (M_r) values for FGD by-product and standard base course materials.

Figure 35. Stress-strain relationships for kaolin clay and Coolside/kaolin mixture allowed to react for 0, 7, or 28 days after compaction.

Figure 36. Stress strain relationships for samples of Marietta clay and OSU spray dryer mixture.

Figure 37. Stress - strain relationships for OSU clay and OSU-SPD/OSU clay mixture.

Permeability - Falling head permeability tests were performed on soil/FGD mixtures in accordance with ASTM Standard Procedure D 2434-74. The FGD by-product used in this testing program was the ash obtained from the OSU spray dryer. The soil was the OSU silty clay studied in the strength tests described in the previous section. The different conditions investigated were changes in the proportions of soil to FGD, curing time and confining pressure.

Figure 38 presents the permeability of the mixes as a function of relative percentages of FGD to soil and sample curing periods. The soil/SPD mixtures were found to be more permeable than either the soil or the SPD alone. Permeability of the soil/SPD mixtures changed as a function of time.

Amending soil with FGD offers the designer the opportunity to modify the behavior of a soil in an engineered structure. By combining FGD by-product with local soils two of the most important engineering properties, strength and permeability, become a function of the ratio of soil to FGD by-product and curing time. In all cases studied, the strength of the amended soil was greater than the strength of the soil alone. Additionally, soil permeability was increased with the addition of FGD with the highest values recorded for approximately equal measures of soil and by-product.

Figure 38. Effect of curing time on permeability of mixtures of soil/OSU-spray dryer material.

2.1.2.4.2 Wet FGD

Laboratory tests on wet FGD by-products were begun during Phase 2. The by-products were collected from Conesville and Zimmer plants (see section 1.3.1) as either filter cake, which contained only the spent sorbent, or stabilized by-product, which consisted of spent sorbent, flyash and additional lime. Bottom ash, boiler slag and flyash from the Conesville plant, and flyash from the Zimmer and the Mountaineer plants were also collected for testing. The laboratory program, although not as extensive as had been the case for the dry FGD ashes, employed the same test procedures. The water contents of the wet FGD (whether filter cake or stabilized FGD) were too high when received in the laboratory to be properly compacted. Therefore the first step in testing these materials was to reduce the water content. This was usually accomplished by mixing additional flyash with the FGD and air drying this new material for a period of several days. Lime was typically added to the mixture to facilitate the drying process. Tables 20 through 23 present the results of unconfined compression tests (ASTM D 2166-85) performed on samples of wet FGD compacted at optimum moisture content according to ASTM D 698-78. Tables 20 and 21 present results obtained from samples of stabilized Conesville ash and Conesville filter cake respectively, when mixed with Conesville flyash. The results shown in Table 22 are for mixtures of the stabilized Conesville product and Mountaineer flyash. Table 23 shows the strength measurements made on samples made from mixtures of Zimmer Station filter cake and flyash. A program consisting of eight different mixtures of by-product, ash and lime was performed. The relative percentages on a dry weight basis of the various constituents of each sample is designated by the following notation:

Mixture No: FGD / Flyash / Boiler Slag / Bottom Ash / Lime1 / Lime2

where FGD refers to either filter cake or stabilized ash. The percentages of FGD, flyash, boiler slag and bottom ash combine to total 100%. The stated percentages of lime added are based on the dry weight of the sum of the ash components. Lime1 refers to the percentage of lime added at the time the flyash, boiler slag and/or bottom ash were added to the FGD prior to air drying. Lime2 refers to the lime added after the mixture was sufficiently dried and immediately prior to compaction.

It is clear that the strength of the mixtures is strongly affected by the curing time and the lime additives (Tables 20 through 23). Strength gain appears to be relatively uniform with time for all samples treated with at least 5% added lime. No significant gain in strength over time was observed in the samples of stabilized FGD to which no additional lime was added prior to strength testing. The strongest samples were those to which lime was added just prior to compaction. In those samples where the FGD was filter cake the increase in strength resulting from the addition of Lime2 was substantial. For our limited sampling area, there does not appear to be any significant effect on the measured strength attributable to the source of the FGD or the flyash.

Table 20. Unconfined compression tests on stabilized wet Conesville FGD.

Sample ¹	Water content	Dry density	Unconfined compressive strength			
			0 days	7 days	28 days	60 days
	%	pcf ²	psi ³			
Mixture 1: 50/50/0/0/0/0	41.0	74.9	32.6	54.6	77.2	80.2
Mixture 2: 50/50/0/0/5/0	43.5	70.5	52.0	117.5	198.7	372.1
Mixture 3: 50/50/0/0/5/5	39.2	71.2	46.5	135.6	245.3	417.5
Mixture 4: 50/50/0/0/0/5	36.0	78.0	27.6	171.1	427.3	428.9
Mixture 5: 75/0/25/0/5/5	54.0	67.4	61.5	88.5	129.6	236.7
Mixture 6: 75/0/25/0/0/5	52.5	64.0	60.9	97.8	140.5	160.3
Mixture 7: 33/33/17/17/5/5	26.0	87.2	59.7	153.7	391.2	471.6
Mixture 8: 33/33/17/17/0/5	32.7	83.0	50.6	131.7	256.1	364.5

¹ Mixture No: weight ratio of FGD / Flyash / Boiler Slag / Bottom Ash / Lime1 / Lime2
where: FGD refers to either filter cake or stabilized ash

² Pounds per cubic foot

³ Pounds per square inch

Table 21. Unconfined compression tests on Conesville filter cake with laboratory-added fly ash.

Sample	Water content	Dry density	Unconfined compressive strength			
			0 days	7 days	28 days	60 days
	%	pcf ¹	psi ²			
Mixture 1	44.8	85.4	59.4	33.7	21.0	24.8
Mixture 2	54.5	80	47.5	67.3	52.7	46
Mixture 3	53.2	82.2	68.2	74.5	158.4	233.6
Mixture 4	42.9	83.4	80.9	98.5	187.2	209

¹ Pounds per cubic foot

² Pounds per square inch

Table 22. Unconfined compression tests on stabilized wet Conesville FGD + laboratory-added Mountaineer fly ash.

Sample	Water Content	Dry Density	Unconfined Compressive Strength			
			0 days	7 days	28 days	60 days
	%	pcf ¹	psi ²			
Mixture 1	29.0	69	28.1	34.6	42.3	51.9
Mixture 3	45.3	64.5	36.9	97.1	319.3	465.9
Mixture 4	29.9	71.5	35.5	152.7	237.3	276.9

¹ Pounds per cubic foot

² Pounds per square inch

Table 23. Unconfined compression tests on Zimmer Station filter cake + laboratory-added fly ash.

Sample ¹	Water content %	Dry density pcf ²	Unconfined compressive strength			
			0 days	7 days	28 days	60 days
			psi ³			
Mixture2*	57.9	82.0	48.9	53.3	56.3	62.0
Mixture3*	56.7	78.8	68.5	120.0	167.5	321.4
Mixture3**	58.7	79.7	64.7	123	159.8	347.6

¹ Mixture2*, Lime 3%; Mixture3*, Lime 3%; Mixture3**, Lime1 3%, Lime2 10%

² Pounds per cubic foot

³ Pounds per square inch

Durability - Because the strengths obtained for the lime treated wet FGD samples compared satisfactorily with those for several of the dry FGD materials, it was felt that if the durability of the wet FGD could be established, it might also be viewed as an alternative to conventional structural fill materials. Therefore, a test program designed to measure the effects of freeze-thaw cycling on the static strength of compacted wet FGD samples was conducted.

Stabilized FGD was collected at two different times from the Conesville plant, combined with additional flyash and lime as described in the preceding section to make Mixture #4 (50/50/0/0/5). Samples were allowed to cure for various lengths of time before being subjected to 12 cycles of 24 hours of freezing temperature followed by 24 hours of thaw. The procedure described in ASTM D 560-89 was followed except that, because we felt the condition to be more representative of average climactic conditions in Ohio, the low temperature used was 0°F (-18°C) rather than the -10°F (-23°C) specified in the Standard. Table 24 lists the strength of each sample as a function of conditions at compaction, cure time and final conditions.

The strengths after freeze-thaw were strongly dependent on water content at failure. Higher water contents significantly reduced the strength. However, variability in water content decreased dramatically with curing time which resulted in more consistent strength results. A minimum curing time before freezing temperatures are expected should be required in any specification allowing the use of stabilized wet FGD in embankment construction. With adequate curing, however, wet FGD strengths compare satisfactorily with those obtained for samples not subjected to cyclic temperature changes.

Table 24. Unconfined compression tests on wet FGD samples subjected to freeze-thaw.

Curing time prior to freeze thaw cycling	Water content at compaction	Water content at failure	Unconfined compressive strength
days	%	%	psi ¹
7	35.2	34.5	226
7	30.4	32.1	250
7	34.7	43.6	20
7	31.8	35.6	234
7	33.8	37.2	297
7	34.0	39.2	215
7	34.0	48.4	32
28	35.2	41.0	163
28	34.7	43.0	72
28	33.8	35.0	342
28	35.2	41.7	149
28	34.7	36.4	365
28	30.4	40.6	133
28	31.8	33.6	274
28	33.8	33.8	538
60	35.2	33.9	434
60	34.7	35.5	485
60	30.4	31.1	426
60	30.4	33.0	316
60	31.8	31.3	421
60	31.8	nr	495
60	33.8	33.5	558
60	34.0	36.1	377
60	34.0	37.4	300

¹ Pounds per square inch

2.2 Greenhouse Studies of FGD and Soil/Spoil Mixes

Greenhouse pot studies were conducted to determine the impact of acid soil and spoil amendment with dry FGD by-products on soil/spoil and leachate water quality, and on plant growth and quality.

2.2.1 Procedures used in Greenhouse Studies

Three greenhouse pot experiments were conducted using dry FGD by-products as alkaline amendments: (1) amendment of spoil and underclay from the Fleming abandoned, mined land site, (2) amendment of spoil and topsoil from the R&F active coal mine site, and (3) amendment of an agricultural soil.

The FGD by-products used in the greenhouse experiments were LIMB and PFBC which are described in section 1.3.1. By-products from the PFBC cyclone and bed ash streams were used separately in these experiments and are denoted as PFBC_{cycl} and PFBC_{bed} respectively. Chemical and mineralogical characteristics of the LIMB and PFBC materials used in these experiments are given in Table 1.

2.2.1.1 Fleming Abandoned Mined Land Experiment

Spoil and underclay materials used in these experiments were described in section 1.3.1. These materials were air-dried and passed through a 10-mm sieve prior to use in the greenhouse experiments. Characteristics of the Fleming materials are given in Table 2.

Reclamation research on acid spoil materials has shown use of organic amendments, such as sewage sludge, produces superior revegetation results (Sutton and Dick, 1987). For this reason digested sewage sludge (from Rahway, NJ) (Table 25), was used together with varying amounts of the FGD by-products as an amendment for the Fleming spoils. The sewage sludge was air-dried and passed through a 10-mm screen before being used.

The spoil (3 kg) and underclay (4 kg) materials from the Fleming site were each mixed with sewage sludge (6% by weight) and with each of the two dry FGD by-products at rates of 0, 3, 6, 12, and 24% by dry weight. These rates were based on a preliminary incubation study with the Fleming materials and FGD by-products which indicated a 6% amendment rate would raise the overburden pH to 7. Rates in excess of 6% were investigated because in mine reclamation it is desirable to provide excess pH buffering capacity and to encourage downward leaching of alkaline amendments. In addition to the treatments with sewage sludge, overburden material without sewage sludge was combined with the two dry FGD by-products at rates of 0 and 12% by dry weight. This was done both to determine the effects of FGD amendment alone on vegetative growth and to investigate interactions between FGD by-products and sewage sludge.

Table 25. Total chemical analysis of sewage sludge amendment used in the Fleming greenhouse study.

Element	Concentration
Major Elements (%)	
Organic C	31.2
Ca	2.76
Mg	0.34
S	1.41
Al	3.40
Fe	1.24
Na	0.07
N	3.8
P	1.78
K	0.15
Ba	0.01
Minor Elements (mg kg⁻¹)	
As	<0.03
B	31.1
Cd	6.3
Cr	315.2
Cu	1174
Pb	16.1
Mo	11.2
Ni	166.4
Se	<0.3
Zn	1494

Phosphorus (0.64 g P₂O₅ as triple super phosphate) and K (0.25 g K₂O as KCl) fertilizer was mixed with each pot containing sewage sludge. Nitrogen (0.15 g N as NH₄NO₃), P (0.75 g P₂O₅, and K (0.74 g K₂O) fertilizers were mixed with the no sewage sludge treatments. Each pot with no sewage sludge also received a surface application of N (0.18 g N), P (0.46 g P₂O₅), and K (0.23 g K₂O) after each harvest to replace plant removal.

All materials were mixed in an air-dry condition and poured into PVC columns (30-cm tall, 15 cm diameter). The columns were mounted on flat PVC plates with a nipple in the center to allow for leachate collection. A complete factorial experimental design was used with all treatments replicated four times and arranged in randomized complete blocks.

After filling the columns, sufficient deionized water was added to each column to produce between 150 and 200 mL of leachate. Following this initial leaching, approximately 5 g of soil/spoil was collected from the surface 1 cm of each column for

pH determination

Immediately following the first leaching columns were weighed to determine gravimetric moisture content, which was assumed to represent field capacity. All pots were planted with 30 seeds of Kentucky 31 tall fescue (*Festuca arundinacea* Schreb.). Artificial lighting was used to provide a minimum of 14 hours of light per day ($400 \mu\text{mol m}^{-2} \text{sec}^{-1}$) and temperature was maintained on a 25/20°C day/night cycle.

Following an initial 3-month growth period, fescue was harvested once each month for a total of six harvests. Columns were leached a second time after the second harvest, and a third time following the sixth harvest. Leaching procedures used were the same as described for the first leachate collection. Between these leachings, columns were watered with only enough deionized water to replace evapotranspiration losses. Soil moisture was monitored by weighing the columns weekly and adjusting moisture content to approximately 75% of field capacity on a gravimetric moisture basis. Following the final harvest and leachate collection, the columns were sampled at various depth intervals for pH determination, and selected samples were analyzed for crystalline mineral phases by x-ray diffraction.

Leachates were analyzed for pH, electrical conductivity, for As, B, Ba, Be, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Li, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Se, Si, and Zn by inductively coupled plasma emission spectrometry, for SO_4^{2-} by ion chromatography, and for dissolved organic C using a carbon analyzer.

Harvested plant tissue was dried at 60 C for 48 hours, weighed, and digested by heating in concentrated HNO_3 and HClO_4 . Digests were analyzed for As, B, Ba, Be, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Li, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Se, and Zn by inductively coupled plasma emission spectrometry. Separate samples were analyzed for total N using a nitrogen analyzer. Following the final harvest, topsoil and spoil in the pots were sampled at various depths and analyzed for root density.

Data analysis for each experiment was conducted using analysis of variance and single degree of freedom orthogonal contrasts. Regression analysis was used to assess the responses to amendment rate for each spoil and FGD by-product.

2.2.1.2 R&F Active Mine Experiment

The R&F spoil and topsoil used in this experiment were described in section 1.3.1 and Tables 2 and 3. Spoil and soil materials were air-dried, and passed through a 10-mm sieve prior to use in this experiment. Spoil and topsoil were combined with the FGD by-products to simulate their placement during coal mine reclamation. Spoil (6.2 kg) was mixed with the two dry FGD by-products at rates of 0, 4, 8, 16, and 32% by weight. The air-dry mixes were poured into PVC columns (60 cm tall, 15 cm diameter) constructed as previously described. Depth of the FGD-amended overburden ranged from 25 to 36 cm depending on the rate and type of FGD by-product. Topsoil (5.4 kg) was mixed with LIMB (0.4%) and PFBC (0.8%) and placed in the PVC columns above

overburden treated with the same FGD by-product. This resulted in a 20-cm layer of FGD-amended topsoil in each column above the FGD-amended overburden. Rates of FGD amendment to the topsoil were based on a preliminary study with LIMB and PFBC which indicated these rates would raise the topsoil pH to 7. Topsoil in each pot was fertilized with N (0.27 g N), P (1.23 g P_2O_5), and K (0.71 g K_2O). These pots also received surface applications of N (0.18 g N), P (0.05 g P_2O_5), and K (0.19 g K_2O) after each harvest to replace plant removal.

A complete factorial experimental design was used with all treatments replicated 4 times and arranged in randomized complete blocks.

After being filled with FGD-amended spoil and topsoil, the columns were leached, planted with Kentucky 31 tall fescue (*Festuca arundinacea* Schreb.), watered, and harvested as was described for the Fleming columns. The only exception was that these columns were leached twice rather than three times, once immediately after filling, and again after the sixth harvest. Analyses of leachates and harvested plant tissue was also conducted as described for the Fleming experiment.

2.2.1.3 Agricultural Soil Experiment

Soil material used in this experiment was described in section 1.3.1 (Table 3). Soil material was air-dried and passed through a 10-mm sieve prior to being used in the experiment.

Soil (4.8 kg) was mixed with LIMB, PFBC_{cycl}, and PFBC_{bed} by-products at rates of 0, 0.35, 0.7, 1.4, and 2.8% by dry weight of soil. These rates correspond to approximately 0, 1/4, 1/2, 1, and 2 times the amount of alkaline material recommended by the Ohio State University Agronomy Guide (1988) to raise soil pH to 7. The air-dry mixtures were placed in the same type of columns as were used in the Fleming experiment. Alfalfa (*Medicago sativa* L.) and Kentucky 31 tall fescue (*Festuca arundinacea* Schreb.) were planted in separate pots of each soil-FGD mixture. A complete factorial experimental design was used with soil-FGD-crop treatment combinations replicated 3 times and arranged in randomized complete blocks. After being filled with FGD-amended soil, the pots were leached, planted, watered, and harvested as was described for the Fleming experiment. The only exception was that pots were leached twice rather than three times; once immediately after filling, and again after the sixth harvest. All pots were fertilized according to soil test recommendations at the beginning of the experiment with P (1.00 g P_2O_5 /pot) and K (0.42 g K_2O /pot). Pots with fescue were also fertilized with N (0.24 g N/pot). These pots also received surface applications of N (0.18 g N), P (0.05 g P_2O_5), and K (0.19 g K_2O) after each harvest to replace plant removal.

Analyses of leachates and harvested plant tissues were conducted as described for the Fleming experiment.

2.2.2 Results and Discussion of Greenhouse Studies

Three greenhouse pot experiments were conducted using dry FGD by-products as alkaline amendments: (1) amendment of spoil and underclay from abandoned, mined land, (2) amendment of spoil and topsoil from an active coal mine site, and (3) amendment of an agricultural soil. The following sections present the findings of these experiments.

2.2.2.1 Abandoned Mined Land Spoil Study

The results of this experiment are presented in three sections which focus on spoil chemistry, leachate/soil solution chemistry, and plant growth and quality.

2.2.2.1.1 Spoil Chemistry

Both LIMB and PFBC FGD by-products were effective in rapidly raising the pH of the spoil and underclay materials as well as leachates collected from these materials (Figs. 39 and 40). Because of its lime (CaO) and portlandite (Ca(OH)_2) content the LIMB by-product initially raised both spoil and leachate pH to higher levels than the PFBC by-product, in which alkalinity was due primarily to Ca and Mg carbonates with some periclase (MgO) (Table 1). With both FGD amendments, higher pH values were observed in the underclay than in the spoil material. This was because the spoil had a higher potential acidity than the underclay due to its higher S content. Much of the spoil S was present as FeS_2 which oxidizes to produce acidity in the form of H_2SO_4 (Caruccio and Geidel, 1978; Hill, 1978).

At application rates of 6% the FGD by-products, both with a neutralization potential of about 60% CaCO_3 equivalency, increased minespoil pH to the range 7 to 7.5 and sustained this pH throughout the 8-mo experiment. Application of alkaline materials in excess of that needed to neutralize acidity in the treated zone may be desirable if downward leaching will increase the pH and base status of the subjacent spoil. Although the LIMB and PFBC materials used in this study have similar CaCO_3 equivalencies, it should be noted that dry FGD by-products vary considerably in their total neutralization potential (Stehouwer et al., 1995). Therefore, the amount of material needed to neutralize acidity in a given spoil will also vary from one type of FGD by-product to another.

Application of FGD by-products at rates greater than 6% initially raised the pH of most leachates and spoils above 8. These initially high pH's tended to decrease with time, with the largest decreases occurring near column surfaces (Fig. 39). The decrease was apparently the result of carbonation of lime, portlandite, and periclase since the pH was tending toward that of free carbonates in equilibrium with atmospheric CO_2 . Carbonation was limited in columns with the highest FGD application rates because high pH and soluble salt concentrations inhibited both root and microbial activity in these columns and thus reduced introduction of CO_2 from respiratory activity.

Figure 39. pH distribution with depth after 8 mo of reaction time of Fleming spoil and underclay amended with LIMB and PFBC. Inset figures show beginning (time zero) surface pH.

Figure 40. pH, electrical conductivity (EC), and dissolved organic C (DOC) of leachates collected from Fleming spoil and underclay amended with LIMB and PFBC.

Another factor which limited carbonation at the higher FGD amendment rates was the formation of secondary minerals. With LIMB amendment of 24%, conditions of high pH, high Ca and SO_4^{2-} concentrations (Fig. 41), and adequate moisture led to the formation of the mineral ettringite $[(\text{Ca}_6\text{Al}_2(\text{SO}_4)_3(\text{OH})_{12}\cdot 26\text{H}_2\text{O})]$ (Figs. 42 and 43). Ettringite is highly expansive and cementitious, and thus severely restricted both water movement and root extension below the depth at which it occurred. At the 24% LIMB amendment rate, extensive ettringite formation occurred below 15 cm in the Fleming overburden and below 8 cm in the Fleming underclay. With PFBC amendment of 24%, weak cementation occurred in the underclay, but not in the more acidic spoil. The cementation was apparently due to formation of gypsum.

2.2.2.1.2 Leachate Chemistry

The first leachates were collected from previously air-dry spoil and underclay materials. These leachates are indicative of element mobility immediately after application of FGD and sewage sludge, and are not representative of equilibrium conditions in the spoil+amendment mixes. The first leachates tended to have somewhat higher maximum concentrations of Ca (with LIMB) and lower concentrations of Mg and S (with PFBC) than occurred in the second and third leachates. In general, concentrations of most other elements were somewhat lower in the first leachates than in the later leachates. The second leachates, and even more so the third leachates, provide a better indication of equilibrium solution chemistry because they were collected after several months of reaction and equilibration between the spoil, the amendments, and the pore waters. For this reason we have decided to focus this discussion of solution chemistry on the third leachates, those collected after 8 mo of equilibration.

Leachate chemistry and element mobility were found to be affected by the type and rate of FGD amendment, the type of minespoil material, the presence of sewage sludge, and interactions between each of these components. To a large extent, spoil pH was the factor which governed the interactions between these components and their effects on leachate chemistry.

Leachate dissolved organic C (DOC) increased with increasing LIMB and PFBC amendment in the Fleming spoil and underclay materials. Leachate DOC levels were also increased by 6% sewage sludge amendment (Fig. 40). As with pH, LIMB amendment produced higher leachate DOC concentrations than PFBC amendment, and Fleming underclay gave higher DOC than Fleming overburden. In fact the increases in DOC were strongly correlated with increases in pH above 7 ($r^2=0.82$). These results are expected since it is well established that the solubility of organic matter increases substantially as pH increases above the neutral range (Stevenson, 1982).

Figure 41. Ca, Mg, and S concentrations of leachates collected from Fleming spoil and underclay amended with LIMB and PFBC.

Figure 42. X-ray diffraction tracings of 15 to 20 cm depth samples of Fleming spoil amended with 24% LIMB and PFBC.

LIMB

PFBC

Figure 43. Scanning electron micrographs of 15 to 20 cm depth samples of Fleming spoil amended with 24% LIMB and PFBC showing needle-like ettringite crystals in LIMB-amended material and monoclinic gypsum crystals in PFBC amended material.

Increasing FGD amendment increased leachate EC with the largest increases resulting from PFBC amendment (Fig. 40). Specific conductance is an indicator of dissolved salt concentrations. Similar patterns were seen in the FGD amendment effects on concentrations of Ca, Mg, and S, the major elements in the leachates (Fig. 41). In all leachates, greater than 90% of the total S was in the form of SO_4^{2-} . Concentrations of Ca and S in LIMB amended minespoils appeared to be controlled by the solubility of gypsum when ettringite did not form. Because ettringite is much less soluble than gypsum, Ca and S concentrations decreased in leachates where ettringite was present even though the total amount of Ca and S in the columns increased. With PFBC amendment, leachate Ca appeared to be controlled by the solubility of calcite, while Mg and S were likely controlled by the solubility of epsomite ($\text{MgSO}_4 \cdot 6\text{H}_2\text{O}$). Leachate S and Mg could go to much higher levels with PFBC amendment than leachate S and Ca with LIMB amendment because epsomite is approximately 300 times more soluble than gypsum. Thus the presence of Mg in the PFBC by-product gives it a greater potential for excessive salt loading than the LIMB by-product. Leachate concentrations of Al, Fe, Mn, and Zn, elements which are frequently toxic in acid spoils, all decreased with FGD amendment (Figs. 44 and 45). This was expected since the solubility of each of these elements is greatest under acid conditions and decreases rapidly with increasing pH (Bohn et al., 1985).

Amendment of the Fleming spoil with sewage sludge gave large decreases in leachate Al and Fe even though the corresponding pH increase was small (from 2.98 to 4.05). This reflects the ability of organic C to form strong complexes with these elements (Stevenson, 1982), and the inverse relationship between soil organic matter and Al^{3+} solubility and toxicity (Kamprath, 1970; Hue et al., 1986). The same effect was not observed with Mn and Zn because concentrations of these metals were relatively high in the sewage sludge (Table 25). This is evidence of how organic and alkaline amendments function synergistically to ameliorate toxic conditions in acid minespoils and supports the use of FGD with an organic amendment for minespoil reclamation.

Figures 45 and 46, and Appendix Tables 48, 49, and 50 give leachate concentrations of elements which are of environmental concern and have been regulated with respect to land application of sewage sludge (U.S. Environmental Protection Agency, 1993). Not listed are Hg and Pb. Mercury was below detection limit ($<0.04 \text{ mg L}^{-1}$) in all of the column leachates and was also not detected ($<0.0002 \text{ mg L}^{-1}$) in Toxicity Characteristic Leaching Procedure (Environmental Protection Agency, 1991) extracts of LIMB or PFBC by-product. Lead was also below the detection limit of 0.04 mg L^{-1} . Leachate Cd showed no change in concentration with increasing FGD amendment (Appendix Table 50). Thus, with respect to these metals there appears to be little potential for environmental contamination as a result of using FGD by-products as minespoil amendments.

Leachate Cu and Ni either remained constant or decreased when amended with FGD by-products, except for the highest amendment rates. These results are consistent with most studies of trace metal behavior in soil which have found that trace metal solubility and mobility decreases as pH increases above the acid range (Chaney

Figure 44. Al, Fe, and Mn concentrations of leachates collected from Fleming spoil and underclay amended with LIMB and PFBC.

Figure 45. Cu, Ni, and Zn concentrations of leachates collected from Fleming spoil and underclay amended with LIMB and PFBC.

Figure 46. As, B, and Se concentrations of leachates collected from Fleming spoil and underclay amended with LIMB and PFBC.

et al., 1987; Woodbury, 1992). Increases in leachate Cu and Ni with increasing FGD amendment occurred only where spoils were amended with sewage sludge and where pH was also high. This was most noticeable with LIMB-amended underclay, where high pH increased the solubility of organic matter in the sewage sludge. The first leachates from the 24% LIMB amended spoil and underclay had pH near 12 (Fig. 40) and relatively high concentrations of Cu (14.03 mg L^{-1} from spoil, 57.87 mg L^{-1} from underclay,) and Ni (3.77 mg L^{-1} from spoil, 8.96 mg L^{-1} from underclay). Copper and Ni were present at much higher concentrations in the sewage sludge than the FGD by-products (Table 25); thus, a relatively constant amount was added to all the Fleming columns. Apparently, organic matter to which these metals were bound was solubilized and the metals were held in solution through complexation with dissolved organic C. Organic ligands will form relatively stable complexes with these metals (Stevenson, 1982).

Leachate concentrations of the oxyanions As, B, and Se generally tended to increase with increasing FGD amendment (Fig. 46). These elements were present in the FGD by-products (Table 1); thus, some of the increase in leachate As, B, and Se can be attributed to the increasing amounts of these elements added with the FGD by-products. Some of the increase may also be attributable to increased competition by sulfate and organic ligands for anion sorption sites. This was particularly evident with PFBC amendment where increases in leachate As, B, and Se with amendment rate, and during the course of the experiment, tended to coincide with increases in leachate S. It should be noted that in spite of the increases, leachate As and Se from the 12% or less FGD amendment rates remained near or below drinking water maximum contaminant levels.

Concentrations of all three elements (As, B, and Se) decreased sharply where extensive ettringite formation had occurred (third leaching of 24% LIMB amended underclay). Because of its expansive and cementitious properties, the formation of ettringite would have limited the contact between leachate water and As, B and Se present in the FGD by-products, thereby decreasing their concentrations in the leachate. There is evidence that these elements may substitute for SO_4^{2-} during the precipitation of ettringite which would also decrease their leachate concentrations (Hassett et al., 1990).

Leachate As concentrations showed large changes among the three leachates (Fig. 46). Most notable were the large decreases in leachate As from the first to the second leaching of LIMB amended spoil. This was followed by a smaller increase in the third leaching. The reasons for these changes are not readily apparent.

Boron is of concern because of the potential for the development of soil solution concentrations which are phytotoxic. Although there was some increase in leachate B with PFBC amendment, much larger increases were observed with LIMB amendment up to the 12% rate. This is not surprising since total B in the LIMB by-product was much higher than in the PFBC by-product. The most phytotoxic B species are relatively water soluble (Woodbury, 1992). In an actual mine reclamation site, leaching due to

natural precipitation may rapidly move soluble B below the root zone, thereby decreasing the potential for B phytotoxicity.

As was noted with As, changes in leachate Se concentrations appeared to be related to competitive interactions with SO_4^{2-} , OH^- , and organic ligands for anion sorption sites. The high levels of Se in the first leachates of LIMB amended underclay were associated with high pH (high OH^-) and likely high DOC (DOC was not measured in the first or second leachates). These levels then decreased in the second and third leachings as did pH. Likewise Se in leachates from PFBC treatments increased during the course of the experiment in conjunction with increases in SO_4^{2-} concentrations. Due to their weak acid character, SeO_4^{2-} and SeO_3^{2-} should be held more strongly by soils than SO_4^{2-} (Bohn et al., 1985); thus, high SO_4^{2-} concentrations would be necessary to exchange with SeO_4^{2-} and SeO_3^{2-} . These experiments do not indicate what fraction of the mobilized Se is from the FGD and what is from the spoil or underclay.

Overall, at intermediate FGD application rates (6 to 12%) leachate concentrations of most trace elements remained at very low levels. Most in fact, were below drinking water maximum contaminant levels. In most cases, elevated concentrations were due to conditions of excessively high pH and high DOC caused by FGD application of greater than 12%. These undesirable conditions can be avoided by limiting application rates or by providing sufficient moisture and reaction time to carbonate lime and portlandite prior to FGD application.

2.2.2.1.3 Fescue Growth

Fescue yield was affected by harvest interval, type of minespoil material, type of FGD by-product, and addition of sewage sludge. Yields tended to decline over the last 3 harvests (Fig. 47). This decline was likely due to the intensive harvest schedule, rather than a FGD treatment effect.

No fescue could be grown on the unamended spoil (Fig. 47). Fescue seeds planted in the unamended spoil germinated, but seedlings died within 2 to 3 days of emergence. Addition of sewage sludge or either FGD by-product permitted fescue growth on the otherwise phytotoxic spoil. Sewage sludge amendment caused a substantial decrease in soluble Al with little change in pH, while FGD by-products increased pH and decreased soluble Al (Figs. 40 and 44). Fescue growth on sludge-amended spoil was further increased by FGD amendment. Growth response to PFBC was similar at all amendment rates, while with LIMB there was growth suppression at the highest rate. This growth suppression was greatest at the beginning of the experiment and had disappeared by the end of the experiment. It was associated with the initially high spoil pH of 9.6 which decreased with time to pH 7.8 (Fig. 40). The early growth of fescue was also suppressed by 12% LIMB amended spoil without sewage sludge. Comparison of 12% LIMB amended spoil with and without sewage sludge shows some of the negative effects of high pH were ameliorated by the addition of sewage sludge. This again points out the benefit of using FGD by-product in conjunction with an organic amendment for mine reclamation.

Figure 47. Yield of fescue grown on Fleming spoil and underclay amended with LIMB and PFBC.

With the underclay material all LIMB amendment rates initially suppressed growth of fescue. By the end of the experiment, however, the lower LIMB amendment rates increased fescue yield, while the yield suppression persisted at the 24% LIMB rate. A similar response was seen with PFBC amendment in that initially negative effects on yield changed to positive effects in the last two harvests. These effects on yield were also associated with gradual decreases in the initially high pH caused by FGD amendment (Figs. 39 and 40). Similar growth suppression of peanuts with 10 to 25% application rates of AFBC has been attributed to high pH and salinity (Terman et al., 1978).

Root growth was also affected by FGD amendments. In the highly toxic Fleming spoil, rooting density at the 10-15 cm depth was increased by FGD amendment up to the 12% rate. With the 24% amendment rate, however, root growth was concentrated in the upper 10 cm of the pots. Below this depth the combination of cementation and high pH restricted root growth. In the less toxic underclay, LIMB amendment above the 3% rate decreased root growth. With 24% LIMB amendment, a thick mat of roots grew in the upper 10 cm while below this depth no root growth was possible because formation of the secondary mineral ettringite cemented the underclay into a solid, highly impermeable mass. With 24% PFBC amendment gypsum formation caused less extensive cementing, thus root growth was not completely inhibited.

While tissue composition of fescue grown on Fleming spoil and underclay was affected by FGD and sewage sludge amendment, concentrations of most elements remained within sufficiency levels and below toxic levels (Figs. 48 to 52, Appendix Table 51). No toxicity symptoms were observed on the growing fescue plants. Differences in the Ca and Mg content of the FGD by-products were reflected in tissue concentrations, with LIMB amendment producing higher Ca, and PFBC producing higher Mg tissue concentrations. Magnesium toxicity has been reported in plants grown on high sulfate soils amended with dolomite (Evangelou and Thom, 1984). In spite of leachate Mg concentrations as high as 7,000 mg L⁻¹ (Fig. 41), tissue Mg remained within a normal range and there were no signs of Mg toxicity.

Tissue S concentrations were largely unaffected by the large increases in spoil S due to FGD amendment (Fig. 48). This was likely because soluble SO₄²⁻ concentrations in the spoil and underclay were sufficient to meet plant needs. The decrease in tissue S with LIMB amended underclay was caused by the S being physically unavailable to fescue roots because of cementing in the lower portion of the pots. There was a tendency for PFBC amendment to produce higher tissue S concentrations which reflected the much higher SO₄²⁻ solubility in the PFBC material. When grown on 12% FGD amended spoil, fescue tissue S concentration was increased by sewage sludge amendment even though leachate S concentrations were similar (Fig. 41).

Although tissue concentrations of the three major plant nutrients N, P, and K showed some effects from FGD amendment, the decreases in tissue P would be of the greatest concern in establishing and maintaining vegetative cover on a reclamation site

Figure 48. Concentrations of Ca, Mg, and S in fescue shoot tissue grown on Fleming spoil and underclay amended with LIMB and PFBC.

Figure 49. Concentrations of N, P, and K in fescue shoot tissue grown on Fleming spoil and underclay amended with LIMB and PFBC.

Figure 50. Concentrations of Al, Mn, and Ni in fescue shoot tissue grown on Fleming spoil and underclay amended with LIMB and PFBC.

Figure 51. Concentrations of Cu, Mo, and Zn in fescue shoot tissue grown on Fleming spoil and underclay amended with LIMB and PFBC.

Figure 52. Concentrations of As, B, and Se in fescue shoot tissue grown on Fleming spoil and underclay amended with LIMB and PFBC.

(Fig. 49). The LIMB amendment resulted in tissue P levels falling below sufficiency levels at the highest amendment rates. These decreases were associated with high soil pH, high soil Ca and Mg, and cementing. A number of factors known to decrease the availability of soil P occurred with FGD amendment. These included hydrolysis of exchangeable Al, and increasing pH above 7 with concurrent increases in Ca, Mg, and carbonates (Tisdale et al., 1985). Much of the P added in the sewage sludge was in an organic form and P mineralization was likely limited by high pH, thereby further reducing P availability. Finally, as was noted with tissue S, cementing in the lower portions of the pots also made much of the P physically unavailable by preventing root extension.

Tissue Al and Mn concentrations were decreased from near phytotoxic levels to normal levels by FGD amendment of the Fleming spoils. Similar responses were seen with Cu, Ni, and Zn. Each of these responses was associated with decreases in leachate concentrations of the same elements (Fig. 45). The large increases in leachate Cu and Ni which occurred with 24% LIMB amendment, however, were not seen in plant tissue concentrations. This again was likely due to roots being concentrated in the upper portion of the pots while much of the soluble Cu and Ni occurred in the higher pH region deeper in the pot.

The trace elements As, Cd, and Cr, which are of concern because of their potential toxicity to higher animals, showed decreasing plant tissue concentrations with increasing FGD amendment (Fig. 52 and Appendix Table 51), while no changes in tissue Hg or Pb concentrations were observed (Appendix Table 51). Selenium also showed decreasing tissue concentrations with increasing FGD amendment, except for an increase at the highest application rate at the second harvest only. While the reasons for the increased Se uptake are not clear, root growth in these pots was restricted to a small portion of the pot due to ettringite cementation.

Tissue B concentrations were unaffected by PFBC, but increased with increasing LIMB amendment to levels in excess of 100 mg kg⁻¹ (Fig. 52). Boron toxicity symptoms were not observed in the fescue plants. More susceptible plant species, however, would likely be adversely affected by similar B concentrations (Marschner, 1986).

As was noted for S, fescue tissue concentrations of P, Ca, Mg, Mn, B, Cd, Cu, Ni, Se, and Zn were higher with than without sewage sludge amendment. The reasons for increased uptake with sludge amendment are not obvious, particularly since many of these elements were present in the sludge at very low concentrations and leachate concentrations of most of these elements were not significantly different with or without sludge. It is clear in this experiment, however, that fescue uptake of trace elements cannot be predicted from the leachate (soil solution) concentrations of the same elements.

2.2.2.2 R&F Spoil Study

The results of this experiment are presented in three sections which focus on spoil chemistry, leachate/soil solution chemistry, and plant growth and quality.

2.2.2.2.1 Spoil Chemistry

Both FGD by-products were effective in raising spoil and topsoil pH (Fig. 53). Because of its lime and portlandite content the LIMB by-product initially raised leachate pH to higher levels than the PFBC by-product (Fig. 54), in which alkalinity was due primarily to Ca and Mg carbonates with some periclase (MgO) (Table 1).

At application rates of 4%, the FGD by-products, both with a neutralization potential of about 60% CCE, increased minespoil pH to around 7 and sustained this pH throughout the 8-mo experiment. Application of alkaline materials in excess of that needed to neutralize acidity in the treated zone may be desirable if downward leaching will increase the pH and base status of the subjacent spoil. Although the LIMB and PFBC materials used in this study have similar CCE's, it should be noted that dry FGD by-products vary considerably in their total neutralization potential (Stehouwer et al., 1995).

Application of FGD by-products at rates of 4% and above initially raised leachate pH to near 12 with LIMB and from 8.5 to 9.5 with PFBC. These initially excessively high pH's decreased with time. The largest decreases occurred with LIMB amended spoil and was apparently due to conversion of lime and anhydrite to ettringite which is stable in the pH range 9.5 to 10. Evidence for ettringite formation is provided in X-ray diffraction patterns and electron micrographs of material collected for 32% LIMB amended spoil (Fig. 55). With LIMB amendment, conditions of high pH, high Ca and SO_4^{2-} concentrations, and adequate moisture promoted the formation of ettringite. Ettringite is highly expansive and cementitious, and thus severely restricted both water movement and root extension below the depth at which it occurred. Ettringite formed throughout the spoil section of the columns at LIMB amendment rates of 16 and 32%.

With PFBC amended spoil the decrease was apparently the result of carbonation of periclase since the pH was tending toward that of free carbonates in equilibrium with atmospheric CO_2 . Cementation occurred only at the 32% rate with the PFBC amendment. There was no indication of ettringite formation and cementation was apparently due to the conversion of anhydrite to gypsum.

2.2.2.2.2 Leachate Chemistry

The leachate chemistry of the R&F spoil columns was very similar to that of the Fleming spoil columns. Thus while results will be noted in this section, reference is made to section 2.2.2.1.2 for much of the discussion and interpretation of the results.

Figure 53. pH of R&F topsoil and spoil 8 months after amendment with LIMB and PFBC.

Figure 54. pH, electrical conductivity (EC), and concentrations of dissolved organic C (DOC), Ca, Mg, and S in leachates from columns of R&F topsoil and spoil amended with LIMB and PFBC.

Figure 55. X-ray diffraction patterns of R&F spoil 8 months after amendment with LIMB and PFBC.
(C=calcite, D=dolomite, E=ettringite, G=gypsum, K=kaolinite, L=lime, M=mica, Q=quartz)

Leachate chemistry and element mobility were found to be affected by the type and rate of FGD amendment. To a large extent, spoil pH was the factor which governed the interactions between these components and their effects on leachate chemistry.

Leachate dissolved organic C (DOC) increased with increasing FGD amendments, with LIMB giving the greatest increase. Maximum DOC concentrations, however, were much lower than those of the Fleming spoils with sewage sludge amendment.

In the first leachates LIMB amendment caused a small increase in EC while in the second leachates EC was decreased by LIMB. By contrast, with PFBC amendment had little effect on EC of the first leachates, but caused a large increase in the second leachates (Fig. 54). Similar patterns were seen in the FGD amendment effects on concentrations of Ca, Mg, and S, the major elements in the leachates (Fig. 54). In all leachates, greater than 90% of the total S was in the form of SO_4^{2-} (Appendix Table 52).

In the first leachates from LIMB amended R&F spoil, Ca was apparently released as lime and portlandite reacted with acidity in the topsoil and spoil. After 8 months of equilibration between spoil and 4% LIMB, leachate Ca and S remained elevated because ettringite did not form and concentrations appeared to be controlled by the solubility of gypsum. At greater than 4% LIMB amendment leachate Ca and S were reduced due to the formation of ettringite. Because ettringite is less soluble than either gypsum or calcite, Ca and S concentrations decreased in leachates where ettringite was present even though total Ca and S in the columns increased. It should be noted that ettringite is only stable at elevated pH (>9). If the pH of the spoil should decrease below this level it is expected that ettringite would dissolve, thereby releasing Ca and S.

With PFBC amendment, final leachate Ca appeared to be controlled by the solubility of calcite, while Mg and S were likely controlled by the solubility of epsomite. Leachate S and Mg could go to much higher levels with PFBC amendment than leachate S and Ca with LIMB amendment because epsomite is approximately 300 times more soluble than gypsum. Thus the presence of Mg in the PFBC by-product gives it a greater potential for excessive salt loading than the LIMB by-product. The decrease in leachate S and Mg (and consequently in EC) at the 32% PFBC amendment rates was apparently caused by gypsum cementing the overburden column.

Concentrations of Fe and Mn were decreased by FGD amendment in both the first and final leachates (Fig 56). This was expected since the solubility of each of these elements is greatest under acid conditions and decreases rapidly with increasing pH (Bohn et al., 1985). With 16% LIMB amendment there was an increase in final leachate Al (Fig. 56), which reflects the increased solubility of Al at high pH's. Leachate Al was lower with the 32% LIMB amendment rate even though pH was as

Figure 56. Concentrations of Al, Mn, Fe, B, Mo, and Se in leachates from columns of R&F topsoil and spoil amended with LIMB and PFBC.

high or higher than with 16% LIMB. This decrease was likely the result of Al being immobilized due to ettringite formation.

Leachate concentrations of the oxyanions B and Se were decreased by FGD amendments in the first leaching. First leachate concentrations of Mo, however, were increased by LIMB and unaffected by PFBC. For Mo essentially the same response was seen in the second leachates. For B and Se, however, second leachates showed increased B at 16% LIMB and increased Se at intermediate PFBC amendment rates. Both B and Se concentrations decreased at the highest amendment rates due to cementation in the spoil columns. Similar results were observed in the Fleming experiment (section 2.2.2.1.2).

Selenium concentrations were much higher in the R&F than in the Fleming spoil leachates. In the R&F spoil leachates, Se was decreased by LIMB amendment, although it remained above the solid waste limit (0.30 mg L^{-1}). With PFBC amendment, however, Se concentrations increased at all amendment rates except 32% where concentrations decreased, apparently due to secondary mineral formation. Total Se concentration in the PFBC by-product was less than the LIMB by-product (Table 1); thus, it does not appear that the PFBC material was the source of the Se. Therefore, it appears that Se was mobilized from the minespoil materials. Possible mechanisms for Se behavior are discussed in section 2.2.2.1.2.

Figures 56 and 57 and Appendix Tables 52 and 53 give leachate concentrations of elements which are of environmental concern, some of which have been regulated with respect to land application of sewage sludge (Environmental Protection Agency, 1993). Mercury was below detection limit ($<0.04 \text{ mg L}^{-1}$) in all of the column leachates and was also not detected ($<0.0002 \text{ mg L}^{-1}$) in Toxicity Characteristic Leaching Procedure (Environmental Protection Agency, 1991) extracts of LIMB or PFBC by-product. Lead was detected at a level of 0.12 mg L^{-1} only in final leachates from the unamended spoil. In all other leachates Pb was below detection limit (0.04 mg L^{-1}). Thus, with respect to these metals there appears to be little potential for environmental contamination as a result of using FGD by-products as mine spoil amendments.

Both LIMB and PFBC amendment caused a decrease in Cd, Ni, and Zn concentrations in the first and final leachates (Fig. 56). Leachate Cu remained relatively constant except for a small increase with 32% LIMB.

2.2.2.2.3 Plant Growth

In the R&F experiment, FGD by-product treatments were applied to overburden material which was then covered by a 20-cm layer of topsoil amended with either LIMB (0.4%) or PFBC (0.8%). Thus the topsoil layer had a larger effect on fescue growth and tissue composition than the underlying overburden. There was very little effect on fescue yield from PFBC amendment. Increasing LIMB amendment, however, did cause a decline in fescue yield (Fig. 58). This response was attributed to high pH (initially in

Figure 57. Concentrations of Cd, Cr, Cu, Ni, and Zn in leachates from columns of R&F topsoil and spoil amended with LIMB and PFBC.

Figure 58. Yield of fescue grown on R&F topsoil and spoil amended with LIMB and PFBC.

excess of 12) and cementing in the overburden at LIMB amendment rates of 16 and 32%.

Root growth was most extensive in the topsoil layer and was not affected by the FGD treatments in the underlying spoil. Roots were able to grow into the unamended overburden and root mass density at 20 to 25 cm was not affected by PFBC amendment. With 32% PFBC amendment root growth below 25 cm was severely restricted due to cementation and high salt concentrations. With LIMB amendment, root mass density at 20 to 25 cm was increased up to the 8% rate and declined at higher rates due to cementation and high pH. Cementing was most extensive with 32% LIMB where it limited root growth to the upper 1 cm of the treated overburden layer. While there was only a slight yield depression due to these root restricting layers in this experiment, changes in rooting volume could have a much greater impact in the field. Mine reclamation sites are often drought prone and treatments which increase or decrease rooting volume would consequently decrease or increase drought susceptibility of vegetation growing on the site.

Elemental composition of fescue tissue remained within sufficiency limits for most elements and no toxicity symptoms were observed during the experiment. The effects of FGD by-product amendment on fescue tissue composition were similar to those observed in the Fleming experiment. This again reflects the fact that root growth was concentrated in the topsoil layer. Restricted root growth into the overburden at the higher LIMB amendment rates further limited the by-product's effect on plant composition.

Concentrations of N, P, and K were unaffected by FGD amendment. Differences between the two by-products were evident in tissue Ca, Mg, and S concentrations; Ca was higher with LIMB amendment while Mg and S were higher with PFBC amendment (Fig. 59). The increase in soil and spoil pH caused by both FGD by-products decreased fescue Mn and Ni while LIMB also decreased fescue Cu and Zn (Figs. 60 and 61). First harvest tissue concentrations of Al and Fe were elevated with 32% LIMB amendment, when pH in the treated spoil was very high, but before ettringite had formed. The effect no longer occurred at the sixth harvest.

As was seen in the Fleming experiment, fescue B was increased by LIMB, but not by PFBC amendment. The increases, however, were much less than occurred in the Fleming experiment even though application rates were higher. This again may be due to the roots being concentrated in the topsoil where solution B was undoubtedly much lower than in the underlying spoil. Finally with the exception of Cr which was increased by PFBC amendment, all other trace elements either decreased or showed no response to increasing FGD amendment rates.

Figure 59. Concentrations of N, P, K, Ca, Mg, and S in leaves of fescue grown on R&F topsoil and spoil amended with LIMB and PFBC.

Figure 60. Concentrations of Al, Fe, Mn, B, and Mo in leaves of fescue grown on R&F topsoil and spoil amended with LIMB and PFBC.

Figure 61. Concentrations of Cr, Cu, Mo, Ni, and Zn in leaves of fescue grown on R&F topsoil and spoil amended with LIMB and PFBC.

2.2.2.3 Agricultural Soil Study

The results of this experiment are presented in three sections which focus on spoil chemistry, leachate/soil solution chemistry, and plant growth and quality.

2.2.2.3.1 Soil Chemistry

The three FGD by-products tested as agricultural limestone substitutes, LIMB, PFBC_{cycl}, and PFBC_{bed}, each were effective in raising the pH of the Wooster silt loam soil from approximately 4.5 up to 7.5 (Fig. 62). Although the three materials had similar neutralization capacity, and after 8 months of equilibration all three materials were equally effective in raising soil pH, the initial reactions of the materials differed. The LIMB by-product reacted very rapidly due to its very fine texture and its content of lime and portlandite. These two minerals are both more soluble and more reactive than carbonate minerals, which are the primary source of alkalinity in the PFBC materials. They also caused LIMB amendment to initially raise soil pH to somewhat higher levels than did the PFBC materials. During the course of the experiment, however, these minerals were converted to carbonates by reaction with atmospheric and soil CO₂ thereby decreasing soil pH. The PFBC_{bed} by-product was the most slowly reacting material because of its very coarse particle size. Initial leachates from PFBC_{bed}-amended soil were also much lower in pH than were the final leachates, again indicating the PFBC_{bed} material is less soluble and more slowly reacting than the other two materials (Fig. 63).

2.2.2.3.2 Leachate Chemistry

The EC of leachates showed a pattern similar to what was seen in the minespoils amended with FGD by-products. In the first leachates there were relatively small increases in EC for all three materials, and with LIMB there was little increase in EC during the course of the experiment (Fig. 63). By contrast the two PFBC materials showed large increases in EC during 8 months of equilibration, with the PFBC_{bed} giving the largest increase. This larger increase is due to the higher concentration of S and Mg in the bed material than in the cyclone material. The changes in EC parallel changes in Ca, Mg, and S concentrations (Fig. 64). These results suggest (as was also seen in the minespoil experiments) that Ca and S concentrations in LIMB amended soil are controlled by the solubility of gypsum. With PFBC amendment, however, Ca concentration is controlled by calcite solubility, and Mg and S concentrations are controlled by epsomite.

What is striking with respect to final leachate EC, Ca, Mg, and S concentrations is that they were all much lower in leachates from fescue pots than from alfalfa pots. Although at first glance this would seem to indicate that fescue reduced soluble salt concentrations in the soil, further investigation showed this was not the case. While an estimate of harvest removal of these elements indicated little difference between the crops, extractions of soil sampled in 5 to 8 cm depth increments at the conclusion of the experiment showed that a different depth stratification of these elements had occurred

Figure 62. pH of Wooster silt loam soil following amendment with LIMB and PFBC.

Figure 63. pH, electrical conductivity (EC), and dissolved organic C (DOC) in leachates from columns of Wooster silt loam soil amended with LIMB and PFBC.

Figure 64. Concentrations of Ca, Mg, and S in leachates from Wooster silt loam soil amended with LIMB and PFBC.

with each crop (data not shown). With alfalfa, Ca, Mg, and S concentrations were reduced near the surface and concentrations increased with depth. With fescue concentrations were more uniform with depth. The cause of this differential depth stratification is not known. Possibly alfalfa roots took up more water from deeper in the columns, while fescue roots took up water more uniformly throughout the columns. This would have promoted downward transport of soluble constituents in the alfalfa columns. If this crop effect on depth stratification was not an artifact of the greenhouse environment and the restricted rooting volume, these results suggest that transport of FGD materials in the field could be affected by the crop.

The FGD by-products were also effective in reducing initial leachate Al, Mn, and Zn concentrations (Fig. 65). The reductions paralleled, and undoubtedly resulted from, the pH increase. In most cases Al, Mn, and Zn concentrations were further decreased during 8 months of equilibration in the pots. These elements, as well as Fe, are frequently phytotoxic in acid soils. The Wooster soil is particularly noted for Mn toxicity, and the decrease in Mn was likely the main factor in increasing alfalfa yield with FGD amendment.

As was seen with the major elements, there were notable differences between crops in the final leachate concentrations of Mn, and Zn. Fescue apparently reduced soluble concentrations of these elements in the untreated soil. If this is the case it would suggest growing fescue in a mixture with less acid tolerant crops could reduce toxicity for those species.

Leachate B was increased by FGD amendment with the largest increases from the LIMB by-product (Fig. 66). While B was increased only slightly in the first leachates, a much larger effect was observed in the final leachates. In the final leachates from LIMB amended soil, B concentration went up to 3 mg L⁻¹ from 0.24 mg L⁻¹. In the minespoil experiments where up to 10 times as much LIMB was added the highest leachate B concentrations were similar at 3.5 to 4 mg L⁻¹. Thus in LIMB amended soils and spoils it appears B solubility is limited. While some plant species may be adversely affected by soil solution B concentrations in this range, no evidence of B phytotoxicity was observed in these experiments with alfalfa and fescue.

The only trace element besides B to show a measurable increase in any of the leachates was Cu. In the final leachates Cu was increased up to 0.1 mg L⁻¹ at the highest application rate of PFBC_{bed}. Again this was observed with alfalfa and not with fescue. It should be emphasized, however, that this concentration is still ten times lower than the secondary drinking water MCL for Cu.

2.2.2.3.3 Plant Growth

Growth of alfalfa was increased by all three FGD by-products (Fig. 67). Yields of the first two harvests were increased by low rates of FGD application (<0.7% for LIMB, <1.4% for PFBC), but higher rates suppressed yields. This effect was greatest for LIMB and was likely the effect of high pH (>8). The early suppressive effect on alfalfa

Figure 65. Concentrations of Al, Mn, and Zn in leachates from columns of Wooster silt loam soil amended with LIMB and PFBC.

Figure 66. Concentrations of B in leachates from columns of Wooster silt loam soil amended with LIMB and PFBC.

Figure 67. Yield of alfalfa and fescue grown on Wooster silt loam soil amended with LIMB and PFBC.

growth was less with the PFBC by-products where pH did not go as high as with LIMB, and was likely due in part to high soluble salts suppressing growth of the young plants. From the third to the sixth harvest, there was no longer any suppressive effect of FGD application rates up to 2.8%. This was due to decreased pH in the case of the LIMB by-product and greater tolerance for salts of the older plants in the case of PFBC by-products. These results indicate that application of FGD should be done 2 to 3 months before planting, such as fall application followed by a spring seeding or spring application followed by fall seeding. This would allow sufficient time for pH to be decreased by carbonation and for salts to be flushed from the seeding zone. Fescue was much less responsive than alfalfa to FGD amendment. This is not surprising given the hardiness and tolerance of fescue for acid growing conditions. Early growth of fescue (harvests 1 through 3) was increased by low FGD application rates (0.35 to 1.4%). The positive effect was greatest for LIMB and less for the PFBC by-products. The later harvests of fescue, however, showed no benefit to fescue growth from FGD application. Given the general trend for yields to decrease during the experiment, it appears the lack of fescue yield response in later harvests is not due to decreasing beneficial effects of FGD, but to improving fescue growth in the unamended soil. This is further evidenced by comparison with alfalfa yield on the unamended soil which declined during the experiment. Final leachate pH was slightly higher, and Mn was much lower in fescue pots than in alfalfa pots without FGD amendment. These differences may account for the improved growth of fescue.

The lack of a negative effect on later alfalfa and fescue growth at the 2.8% FGD rate (approximately two times the lime requirement of the soil) suggests it may be possible to apply FGD by-products at rates greater than required for neutralization of surface soil acidity. Such over-application would be warranted if it could be demonstrated that downward leaching of Ca, Mg, and SO_4^{2-} would be encouraged and would ameliorate phytotoxic conditions in subsoils.

Alfalfa and fescue tissue concentrations of the major nutrients, N, P, and K were relatively unaffected by FGD application with the exception of sixth harvest alfalfa N which increased with increasing FGD (Fig. 68). Concentrations of these elements in plants grown with FGD amendment were all within normal concentration ranges.

Tissue concentrations of Ca, Mg, and S reflected the leachate concentrations of these elements (Fig. 69). Calcium, particularly in alfalfa, was elevated in LIMB amended soil. Tissue Mg and S were elevated with PFBC by-products with PFBC_{bed} producing the largest increase. In spite of these responses the concentrations of these elements were largely within normal ranges and there was no evidence of toxicity.

Tissue concentrations of Al, Mn, Ni, and Zn were all reduced by FGD application (Figs. 70 and 71). This pattern was also seen in the leachate data. The reduction in solubility and uptake of these elements which are often toxic in acid soils, likely accounts for the beneficial effect of FGD on plant growth.

Figure 68. Concentrations of N, P, and K in leaves of alfalfa and fescue grown on Wooster silt loam soil amended with LIMB and PFBC.

Figure 69. Concentrations of Ca, Mg, and S in leaves of alfalfa and fescue grown on Wooster silt loam soil amended with LIMB and PFBC.

Figure 70. Concentrations of Al, Mn, and Ni in leaves of alfalfa and fescue grown on Wooster silt loam soil amended with LIMB and PFBC.

Figure 71. Concentrations of B, Mo, and Zn in leaves of alfalfa and fescue grown on Wooster silt loam soil amended with LIMB and PFBC.

The only trace elements which showed any response to FGD amendment were B and Mo. Boron uptake by fescue was increased in the first harvest from LIMB amended soil. The effect was much less in the sixth harvest even though leachate B increased during this period (Fig. 66). Alfalfa B uptake with 2.8% LIMB was also increased slightly at the sixth harvest. Boron concentrations, however, remained below toxic levels and no B toxicity symptoms were observed. Amendment with PFBC by-products had no effect on alfalfa or fescue B uptake. Thus there is no evidence from this experiment to suggest that FGD amendment would cause B toxicity problems even at application rates equivalent to twice the soil's lime requirement.

All three FGD by-products caused increased Mo uptake in both alfalfa and fescue (Fig. 71). Molybdenum is taken up as the molybdate anion and its uptake has been shown to be suppressed by competition with high concentrations of the sulfate anion. In this experiment there were certainly high concentrations of sulfate present, thus this result is somewhat surprising. Even the highest tissue Mo concentrations, however, were not high enough to have caused any nutritional problems for animals feeding on these plants.

2.3 Field Studies

Field work conducted as part of Phase 2 included engineering studies on the use of FGD by-products in a truck unloading ramp and in cattle feedlots, studies on changes which occur during temporary storage of FGD by-products, and site characterization and evaluation for hay bale storage pads.

2.3.1 Truck Unloading Ramp

A truck ramp was constructed on the OSU campus using a dry flue gas desulfurization by-product generated at the OSU McCracken power plant (section 1.3.1). The ramp was designed by University employees in the Department of Physical Facilities. It was intended for use as a temporary hard trash unloading facility serving the needs of operating units on the central campus. The planned dimensions of the facility were 17 meters long by 7.5 meters wide by 1.2 meters high at its highest point. A schematic of the ramp plans are shown in Fig. 72.

2.3.1.1 Procedures

The purpose of the demonstration project was to document the performance of the OSU spray dryer material in a constructed facility and to illustrate the potential for using this FGD material as a substitute for select borrow in a structural fill. Construction of the ramp was performed entirely by University maintenance personnel using University owned equipment. No special measures were taken either in the design or in the construction of the truck ramp.

The design specifications were based on the results of laboratory strength and leachate tests performed on samples of the spray dryer by-product (Table 26). In all the laboratory tests the spray dryer by-product had been compacted to 100% of Standard Proctor density. At the appropriate density the spray dryer by-product possessed the necessary strength, and had acceptable permeability and swelling characteristics to be a satisfactory replacement for the originally specified select borrow material.

The by-product was delivered to the construction site by the contract haulers and dumped next to the ramp site. The geomembrane and underdrain system were installed by Civil Engineering graduate students who then observed the construction of the ramp by OSU staff. The ramp was constructed during July and August, 1992 whenever there was a break in the work schedule of the University maintenance people. Density measurements made in the fill as it was being placed indicated that the average in-place density was only about 90% of Standard Proctor density. Although the plan had been for a density of 100% of Standard Proctor to be achieved in order to be consistent with the laboratory studies, an increase in density did not appear to be possible with the equipment the crew had available. Therefore density measurements were taken, but no material was removed. In addition, heavy rains in excess of 1 inch in early July meant that optimum moisture conditions could not be maintained for all remaining construction. In spite of the unplanned addition of large amounts of water,

Table 26. Engineering properties of the OSU FGD by-product.

Sample ID	Opt. density	Opt. moisture	Compressive strength			Permeability coefficient		
			Curing time, days					
			0	7	28	0	7	28
	g/cm ³	%	psi ¹			cm/sec x 10 ⁻³		
OSU-01	0.909	50	170	152	360	3.6	1.6	0.76
OSU-02	0.833	68	292	258	491	0.17	*	*
OSU-03	0.845	63	300	262	352	0.74	0.77	0.15
OSU-11	1.056	40	460	399	605	0.42	0.39	0.27

Sample ID	Duration	Swell
	days	%
OSU-03A	423	1.97
OSU-03B	272	2.56
OSU-03C	82	1.60

¹ Pounds per square inch

* not enough of this material remained to perform the test

the material was still fairly easy to work with and the construction of the ramp was allowed to continue. Approximately 181 metric tons of by-product were placed in the ramp. A wearing surface of crushed limestone was placed after the ramp was brought up to within 15 centimeters of the final grade.

Core samples from the ramp have been collected at various times and tested for water content. Samples of the water from both the underdrains and the surface of the ramp have been collected periodically since the completion of construction.

2.3.1.2 Results and Discussion

In spite of the fact that the light duty equipment used to place the by-product was not capable of achieving the densities used in the laboratory study, the ramp has performed as designed and, more than three years after construction, there is no evidence of physical deterioration that could be attributed to the FGD fill.

2.3.2 Cattle Feed Lot Test Pads

Two field projects were conducted at an Ohio State University research farm located near Belle Valley, Ohio. Both wet and dry FGD materials were used to stabilize the soil bases in cattle feedlots. PFBC by-product from American Electric Power's Tidd plant in Brilliant, Ohio was placed in three lots each designed to accommodate approximately fifty animals. Stabilized wet FGD scrubber sludge from AEP's Conesville plant near Coshocton, Ohio was placed at two hay bale storage and winter feeding sites. The plots were visually inspected and samples were collected for laboratory tests several times during the months since the FGD bases were installed.

2.3.2.1 Dry FGD By-Products

The Ohio State University operates the Eastern Ohio Resource Development Center (EORDC) near Belle Valley, in Noble County in the southeastern portion of the state. One of the primary activities at the center is to conduct long term genetic selections using either direct or indirect measures of productive traits in beef cattle. Twice per year, bull calves are kept in feed lots on the farm where they are fed and monitored for 84 days to observe the effects of several factors on their growth. The feed lots, which are not covered, have always had problems with too much water getting into the soil floors and causing the bulls to sink into the resulting mud. This lack of solid footing has caused increases of as much as 35% in the energy an individual bull must expend just to move to and from the feed and water troughs. As a result, the Center has experienced significantly higher feed costs as well as reduced weight gains when compared with expected performance under ideal conditions. By stabilizing the floors of feedlots, one important uncontrolled variable would be removed from the study. Furthermore, the performance of the animals would be improved with more uniform energy requirements.

The site chosen for the dry PFBC by-product demonstration consisted of three adjacent pens with a total area of 1300 m² (Fig. 73). The existing soil floor had high water and organic material contents resulting in very low bearing strength. This low strength made it necessary to stabilize the natural material even before the FGD wearing surface could be placed. The stabilization of the natural soil was accomplished by field mixing the in-place soil with PFBC cyclone ash. This was done by blending the dry by-product into the top 20 cm of the in-place soil and compacting the mixture. Once each pen was treated, strength gains in the amended soil were fairly rapid. This procedure produced a suitable platform for the placement of the surface course. A cover of 20 to 30 cm of blended cyclone and bottom ash by-products was placed over the stabilized base. All the field placement and compaction activities were performed by EORDC personnel using standard farm equipment. Weather conditions and equipment problems experienced during the construction phase of the project resulted in compacted densities of only about 85 to 90% of the Standard Proctor density with moisture contents ranging from 8% below to 10% above the laboratory determined optimum condition. None of these difficulties, however, was attributable to the by-product itself.

Construction of the pads in the feed lot took approximately one week. This duration was slightly longer than expected and, coupled with the late arrival of the by-product, caused construction difficulties. More importantly, because of the delays experienced in the construction process, the EORDC managers were forced to place cattle in the pens only seven days after the FGD was compacted rather than the 28 days the laboratory investigation showed would be desirable. In spite of these rather significant changes in our project plans, only minor failures were observed in isolated areas in two of the three pens after the first cycle of animals was completed. These problem areas were repaired before the second cycle of animals was brought in. After the second cycle, some minor spalling was found at the joint between the FGD by-product base and a concrete apron on which the feed bins are located.

Figure 73. Dry FGD Test Pads at EORDC Tie Barn

2.3.2.2 Wet By-Products

Due in large part to the success of the PFBC demonstration, two additional 1200 m² pads for storing hay bales were constructed at EORDC (Fig. 74). The construction of these sites began in early August, 1993 with the stripping of the ground cover and was completed by the end of September.

Figure 74. Wet FGD Test Pads at EORDC Research Farm

Approximately 1200 metric tons of wet scrubber sludge and flyash from AEP's Conesville plant arrived on the site with a moisture content as high as 66% (all measurements were made on a dry weight basis). The optimum moisture content for this material was determined in the laboratory to be close to 40%. Because the material would not compact properly at the delivered water contents, an attempt was made to lower the water content of the material down to workable moisture levels by allowing some additional drying on site. After drying, when the stabilized sludge was ready for placement, lime was added and the material was compacted and placed. On average, 5% by weight additional lime was added to the stabilized FGD. The mixing was done on site using only equipment found on the farm.

When the FGD was finally compacted, the water contents were still high and the dry densities of the field samples ranged from 0.54 g cm^{-3} to 0.79 g cm^{-3} , which was significantly lower than the laboratory optimum value of 0.85 g cm^{-3} . The low densities actually achieved and the wide range of densities and moistures measured is indicative of how difficult it was to insure uniform conditions of these materials in the field using available equipment.

Probably because of the non-uniformity of the mixed material as placed, the laboratory compressive strengths measured on samples of the FGD recovered from the feedlots have shown considerable variability (location of core samples is shown on Figure 74). Although the measured properties were variable, based on pad performance, strengths of all samples have been satisfactory. Also, in the tests conducted in the laboratory, the samples typically retained a substantial portion of their compressive strength (approximately 1,100 to 1,300 kPa) until well after the peak strength had been passed. This means that failures that might result from overloading would likely remain localized thus making any necessary repairs simpler. The pads have been used for bale storage and feeding during winter and in spite of the difficulties encountered during construction, the pads have performed adequately. Laboratory tests performed on the material recovered from the storage pads and feedlots are provided in Table 27.

Table 27. Test results for EORDC demonstration plot samples.

Sample no.	Date sampled	Water content	Density	UC ¹ Strength	Date Tested ²	H/D ³	pH	Notes
		%	g cm ³	psi				
1A	9/20/93	58.0	1.6					
2A	9/20/93	62.2	1.63					
3A	9/20/93	61.4	1.62	276.0	10/8/93	2.11		
4A	9/20/93	66.7	1.62					
5A	9/20/93	57.1	1.56	277.0	10/28/93	2.09		
6A	9/20/93	53.1	1.49					
7A	9/20/93	52.5	1.51					
8A	9/20/93	62.2	1.48					
1B	10/6/93	51.3	1.64					
2B	10/6/93	52.1	1.65	348	10/28/93	2.14		
3B	10/6/93	61.8	1.56					
4B	10/6/93	58.4	1.53					
5B	10/6/93	56.0	1.51					
6B	10/6/93	55.0						
7B	10/6/93	58.5	1.45					
8B	10/6/93	58.6	1.56					
1C	10/29/93	51.8	1.67	238	11/ 8/93	1.72	9.4	
2C	10/29/93	42.0						
3C	10/29/93	45.9	1.55					
4C	10/29/93	50.9	1.71	192	11/ 8/93	1.67		
5C	10/29/93	48.0						
6C	10/29/93	53.5	1.57	88.9	11/ 8/93	1.98	9.4	cracks in sample
7C	10/29/93	59.0	1.43					
8C	10/29/93	60.2						
9C	10/29/93	53.6	1.53	84.8	11/ 8/93	1.67		Cracks
10C	10/29/93	65.8	1.54	57.8	11/ 8/93	1.59		Cracks
1D	11/18/93	69.7	1.59					
2D	11/18/93	72.7	1.55					
3D	11/18/93	44	1.59					
4D	11/18/93	69.2	1.48					
5D	11/18/93	71.2	1.55					

Table 27. continued

Sample No.	Date Sampled	Water Content	Density	UC Strength	Date Tested ¹	H/D ²	pH	Notes
		%	g cm ³	psi				
7D	11/18/93	58.3	1.60	50	12/7/93	1.17		
1E	12/28/93	29.8						
2E	12/28/93	32.1						
3E	12/28/93	25.0						
4E	12/28/93	34.1						
5E	12/28/93	79.6						
6E	12/28/93	67.6						
7E	12/28/93	44.5						
8E	12/28/93	80.5	1.36					
9E	12/28/93	85.6	1.47					
1G	12/28/93	31.3	1.79	8.7	1/8/94	2.17		Many cracks after freeze/thaw cycling
2G	12/28/93	57.4	1.54					Broken after freeze/thaw
3G	12/28/93	52.9	1.48					Broken after freeze/thaw
4G	12/28/93	68.9	1.52	74.8	1/8/94	1.85		Many cracks after freeze/thaw
5G	12/28/93	76.8	1.40	22.4	1/8/94	1.64		Many cracks after freeze/thaw
6G	12/28/93	29.3	1.86	212.4	1/8/94	1.48		Clayey
7G	12/28/93	60.1	1.49	29.8	1/8/94	1.66		Cracks
8G	12/28/93	65.1	1.50					Broken
1H	6/14/94	57.0	1.46					Block sample
2H	6/14/94	67.1	1.51					Block sample

¹ Unconfined compressive strength² Date sample tested (if different from recovery date)³ H/D = height to diameter ratio of samples tested for strength

2.3.3 LIMB Stockpile

The use of an FGD by-product in construction, land reclamation, or other large-volume applications will require delivery of large volumes of FGD material to the job site in a short period of time. To insure timely delivery of material, in many cases it will be necessary to temporarily store or stockpile the material. The aging and weathering processes which occur during exposed storage may alter the chemical, mineralogical, and physical properties of FGD materials, thereby altering engineering and neutralization properties. To study these processes and their effects, a 1,500 ton stockpile of LIMB by-product material was constructed at the Central Fuel Company, New Philadelphia, Ohio. The stockpile was constructed in July, 1991, and monitored during the following 3 years.

2.3.3.1 Procedures

The LIMB by-product was conditioned with water at the power plant while being loaded into dump trucks. The objective was to add enough water to bring the LIMB to the optimum moisture content to achieve optimum compacted density (OMC), which is in the range 40 to 50% water added on a wet weight basis. This amount of water is also very near the amount added during normal plant operations for fugitive dust control, so major adjustments in operations were not necessary. Water added was determined by measurement of total volume of water added to a truck-load and net weight of the conditioned ash immediately after filling the truck. Residual moisture content of the LIMB material after being placed in the stockpile averaged 17% (Table 28). The water loss between 45% water added and the 17% residual moisture is due to hydration of lime (CaO) and anhydrite (CaSO_4) contained in the LIMB, loss of water as steam, and drying during trucking and handling at the stockpile site. The total amount of LIMB material hauled to the stockpile site was 1,500 tons including the added water. Characteristics of the LIMB by-product used to construct the stockpile are given in Table 28. The LIMB material used in the pile is most similar to the LIMB sample denoted EDG-LIM-14. Complete chemical, physical, and mineralogical characterization of this sample is given in the Phase 1 report of this project.

The stockpile was constructed using a front-end loader to place the LIMB material, with minimal compaction, in a pile on a pad of recompacted clay. Dimensions of the completed pile were approximately 60 by 100 feet, and 10 feet high. Prior to piling the LIMB material, 4-in diameter PVC drain tile was placed on the surface of the pad to collect any water which might percolate through the pile. The drainage tile directed flow to a single collection point outside a recompacted clay berm which surrounded the pile. All surface flow water was retained within the clay berm and directed to a collection point outside the berm. Valves at the two collection points were kept closed between sampling to insure no loss of leachate or runoff between sampling times.

The pile was monitored for leachate and runoff after every significant rainfall event through the end of 1992. A rain gauge was installed at the stockpile site and a record of all precipitation events was kept (Table 29.). Samples of surface runoff

Table 28. Characteristics of LIMB by-product used to construct stockpile.

	Water added	Moisture content	Slaking temperature	Free CaO	Calcium carbonate equivalent	Particle density	Blaine fineness	Bulk density
	%	%	C° 15 min ⁻¹	% CaO	% CaCO ₃	g cm ⁻³	cm ² g ⁻¹	Mg m ⁻³
Mean	43.5	17.5	8.2	21.4	46	2.84	10,030	0.72
Range	31.0-53.5	7.5-35.2	6.5-13.7	11.4-25.5	27.3-48.2	2.83-2.87	9,790-10,460	0.71-0.73
Number of samples	22	8	16	24	9	5	5	3

Table 29. Monthly precipitation at the stockpile site from September, 1991 to December, 1995.

Year	Month	Precipitation inches
1991	September	4.01
	October	0.74
	November	1.24
	December	2.14
1992	January	2.25
	February	1.40
	March	4.15
	April	0.85
	May	1.50
	June	1.30
	July	7.80
	August	3.75
	September	3.75
	October	1.77
	November	4.41
	December	2.81
Total Precipitation: 9/91-12/92		43.87

(there was no leachate during the sampling period) were analyzed for pH, electrical conductivity, for 28 elements by ICP, and for F^- , Cl^- , SO_3^{2-} , and SO_4^{2-} by ion chromatography (IC).

Samples of aged material were collected 3, 6, 9, 18, and 30 months after placement in the stockpile. Samples were analyzed for pH, total neutralization potential, free lime, particle density, specific surface area, permeability, and bulk density. Mineralogical analyses were conducted on selected samples.

2.3.3.2 Results and Discussion

Environmental effects. The LIMB material exhibited a large capacity for absorption and retention of water. During the 30-month monitoring period there were 19 storms with at least 1.3 cm of rainfall. However no water entered the leachate collection system during this period, and there was very little runoff from the pile. During 30 months, moisture content of the stockpiled material increased from 17 to 62% in the upper 5 cm, and to 52% at 30-50 cm (Table 30). In addition to the free water held by the LIMB material, continuing hydration reactions, in particular the formation of ettringite with 32 molecules of water (discussed below), contributed to the large water retention capacity. Hydration reactions (conversion of anhydrite to gypsum and formation of ettringite) stabilized the pile surface by forming a crust firm enough to support a person walking on the pile. This crust also prevented the loss of any airborne dust from the pile even during periods of no rainfall when the pile surface became dry. The crust was relatively water permeable as there was relatively little surface runoff even following storms of greater than 2.5 cm of rainfall. The crust also served to stabilize the pile so that there was very little sloughing or erosion of material from the steep sides of the pile.

There was very little rainfall during the first six months after construction of the stockpile, thus no runoff samples were obtained. Analyses of eight runoff samples collected during the second year of observation are presented in Table 31. The near neutral pH of these samples, in spite of the very high pH of the fresh LIMB material (around 12), resulted from two factors. The first and most significant factor is that reaction of atmospheric CO_2 with CaO and $Ca(OH)_2$ in the surface of the stockpile resulted in the formation of $CaCO_3$. This reduced the surface layer pH to approximately 8.3 (the pH of free carbonates in equilibrium with atmospheric CO_2). Thus, the runoff water interacted with material of a lower pH than that of fresh LIMB by-product or the LIMB material deeper in the pile. Secondly, the runoff water also would have reacted with the more acidic clay material used to construct the pad and berm which surrounded the stockpile. The major elements in the runoff samples are Ca and S (predominantly in the form of SO_4^{2-}) as was expected given the large concentration of gypsum in the LIMB material. It should be noted that concentrations of all trace elements in the runoff samples were very small, less than drinking water standards and/or below detection limits.

There appears to be a very limited potential that any adverse impact on air, water, or soil quality could occur during short-term stockpiling of LIMB material. Because of the LIMB material's large capacity to absorb and retain water, significant amounts of

Table 30. Properties of LIMB by-product material after various periods of aging in an exposed stockpile.

Sample ID	Depth	Moisture Content	Free CaO	CCE	Particle Density	Permeability	Bulk Density
	cm	%	%	%CaCO ₃	g cm ⁻³	cm sec ⁻¹	lb ft ⁻³
3 months storage							
STO-01	0-5	25			1.24		
	50	33					
STO-02	0-70	10					
	70	12					
6 months storage							
STO-03	0-15	43					
STO-04	0-15	73					
STO-05	0-5	70	12.4	51			
	15	49	1.2	49			
9 months storage							
STO-06	150	62			0.70	1.4 x 10 ⁻⁴	
STO-07	0		1.6	29.6	0.98	4.9 x 10 ⁻⁶	
	90	47	7.7	34.3		(compacted)	
	150		19.1	50.5			
STO-08	30	19			0.66		
18 months storage							
STO-09	0-5	62					
	30	102	6.5	22.2	0.6	4.0 x 10 ⁻⁴	58.6
30 months storage							
STO-10	0-5	62			0.6		
	30-50	52	2.0	26	0.5		

Table 31. Chemical composition of LIMB stockpile runoff samples.

ID	Date	Vol L	pH	EC mmho cm ⁻¹	Al	As	B	Ba	Be	Ca	Cd	Co	Cr
mg L ⁻¹													
SPRO-001	05/20/92	8	7.2	2.32	0.09	<0.08	2.36	0.02	<0.001	686.6	<0.003	<0.009	<0.004
SPRO-004	07/13/92	20	7.7	1.51	0.09	<0.08	2.04	0.01	<0.001	395.7	<0.003	<0.009	0.007
SPRO-007	07/30/92	40	7.2	0.94	<0.05	<0.08	1.22	0.01	0.001	237.2	<0.003	<0.009	<0.004
SPRO-008	08/28/92		7.4		0.14	<0.08	0.67	0.04	<0.001	143.4	<0.003	<0.009	<0.004
SPRO-010	09/23/92		7.8		0.10	<0.08	2.22	0.16	<0.001	655.0	<0.003	<0.009	0.010
SPRO-011	10/16/92		7.6		0.23	<0.08	0.33	0.07	<0.001	91.8	<0.003	<0.009	<0.004
SPRO-012	11/13/92	60	7.2	1.43	1.73	<0.08	1.01	0.00	<0.001	328.1	<0.003	<0.009	<0.004
SPRO-014	12/11/92		7.0		0.24	<0.08	0.35	0.16	<0.001	89.2	<0.003	<0.009	<0.004

ID	Cu	Fe	Hg	K	Li	Mg	Mn	Mo	Na	Ni	P	Pb	S
mg L ⁻¹													
SPRO-001	0.012	<0.01	<0.04	45.8	0.30	34.61	<0.001	0.13	11.55	0.03	<0.09	<0.04	500.6
SPRO-004	<0.007	<0.01	<0.04	23.3	0.06	10.97	0.001	0.11	4.54	0.02	0.44	<0.04	281.3
SPRO-007	<0.007	<0.01	<0.04	16.1	0.04	5.94	<0.001	0.05	2.49	<0.01	<0.09	<0.04	184.7
SPRO-008	<0.007	0.07	<0.04	11.3	0.03	8.38	0.009	0.03	16.67	<0.01	<0.09	<0.04	133.8
SPRO-010	0.010	0.06	0.05	49.9	0.09	18.81	0.010	0.10	16.31	<0.01	0.09	<0.04	496.9
SPRO-011	<0.007	0.17	<0.04	6.1	0.01	3.65	0.004	0.02	12.32	<0.01	<0.09	<0.04	81.1
SPRO-012	<0.007	<0.01	<0.04	16.7	0.04	9.58	<0.001	0.06	4.10	<0.01	<0.09	<0.04	272.1
SPRO-014	<0.007	0.35	<0.04	6.5	0.01	4.01	0.005	0.01	13.54	<0.01	<0.09	<0.04	79.9

ID	Sb	Se	Si	Sr	V	Zn	F	Cl	SO ₃	SO ₄
mg L ⁻¹										
SPRO-001	<0.165	<0.27	5.69	1.50	0.007	0.056	1.2	39.9	11.1	1363
SPRO-004	<0.165	<0.27	6.40	0.55	<0.006	0.006	<0.1	10.3	<0.1	809
SPRO-007	<0.165	<0.27	3.06	0.40	0.017	<0.005	<0.1	4.8	<0.1	498
SPRO-008	<0.165	<0.27	2.43	0.28	<0.006	0.060				333
SPRO-010	<0.165	<0.27	12.80	0.97	<0.006	2.000				1564
SPRO-011	<0.165	<0.27	1.76	0.16	<0.006	0.034				213
SPRO-012	<0.165	<0.27	4.88	0.51	<0.006	0.064	2.2	13.1	<0.1	764
SPRO-014	<0.165	<0.27	1.83	0.17	<0.006	0.038				208

water would runoff a pile only during very heavy storm events. The rapid runoff which would occur during such events would limit interaction with the stockpiled material and result in water of a similar or better quality than that reported here. The self-stabilizing nature of the LIMB material greatly reduced the possibility of material spreading onto surrounding soil or eroding into waterways and also eliminated dust formation during dry periods.

Aging Impacts on Beneficial Uses. Significant changes occurred in the stockpiled material during 42 months of exposure. Both free lime and TNP decreased, with the largest decreases occurring in the surface layers. The loss of free lime and TNP apparently was due to carbonation to form CaCO_3 , and to the formation of ettringite $[(\text{CaO})_6\text{Al}_2(\text{SO}_4)_3 \cdot 32\text{H}_2\text{O}]$ and thaumasite $[\text{Ca}_6[\text{Si}(\text{OH})_6]_2 \cdot (\text{CO}_3)_2 \cdot 24\text{H}_2\text{O}]$. The presence of these minerals was confirmed in X-ray diffraction analyses of material collected after both 30 and 42 months of exposure in the stockpile (Figs. 75 and 76). Both ettringite and thaumasite are highly expansive minerals, due to the large number of hydrating water molecules. At each sampling time separate X-ray diffraction tracings were conducted on loose (unconsolidated) and hard (consolidated) fractions of the LIMB material. There appears to be little difference in the proportion of ettringite relative to thaumasite in the loose vs. hard fractions of the 30-month samples (Fig. 75). In the 42-month samples there appears to be a decline in the amount of thaumasite relative to ettringite and perhaps a slight increase in the amount of calcite (Fig. 76). Furthermore, gypsum is clearly present in the loose fraction. From these changes, one might speculate that as weathering and leaching continue to lower the pH of the pore waters, ettringite and thaumasite may degrade to form calcite and gypsum. The gypsum would certainly disappear with time. However, it should also be noted that these samples were collected from different sections of the stockpile, and differences may simply reflect natural heterogeneity in the stockpiled material.

The transformations of lime, portlandite and anhydrite to calcite, ettringite, thaumasite, and gypsum, have important implications for post-storage beneficial use of stockpiled LIMB material. The loss of TNP would indicate a decreased value as an alkaline amendment. Placement of the material in an acid environment, however, should result in the decomposition of ettringite and thaumasite to form calcite as well as gypsum. Thus the apparent loss in TNP should be recovered and the value of the material as an alkaline amendment not greatly diminished.

The formation of ettringite and thaumasite are the primary causes for the undesirable expansion of LIMB and similar FGD materials when used in engineering applications. By allowing these expansive reactions to occur during temporary storage, future problems with expansion after placement and compaction may be avoided. At the same time it is likely that these same reactions caused a loss in the desirable cementitious nature of the LIMB material. This is because the primary cementing agents, lime (CaO) and portlandite $[\text{Ca}(\text{OH})_2]$, were consumed through hydration, carbonation, and formation of ettringite and thaumasite.

Figure 75. X-ray diffraction tracings of unconsolidated (A) and consolidated (B) LIMB stockpile material sampled after 30 months of exposure.

Figure 76. X-ray diffraction tracings of unconsolidated (A) and consolidated (B) LIMB stockpile material sampled after 42 months of exposure.

2.3.5 Active Coal Mine Reclamation

In 1991, the U.S. Geological Survey (USGS) began investigation of the effects on ground water of applying flue-gas desulfurization (FGD) by-products to the recontoured overburden at an active surface coal mine. During Phase 1 of the project, an active-mine site was located and a literature search was done. The search for an active-mine site involved extensive investigation of active surface coal mines in Ohio and resulted in the selection of four sites. Site-selection criteria and methods are described in the Phase 1 Final Report (Stehouwer et al., 1995). A site in Coshocton County, Ohio, was selected from these four, and Phase 2 premining site-characterization work was begun. Phase 2 work was not completed, however. Six months after the beginning of Phase 2, the mining company terminated the project to protect itself from potential liability. The USGS and cooperating agencies were unable to provide indemnification against this potential liability. Attempts to secure another active-coal-mine site resulted in similar responses from other mining companies and landowners.

The purpose of this chapter is to present the results of Phase 2 at the Coshocton County site before the termination of the study. Phase 2 was designed to (1) supply a detailed characterization of the site that included a quantitative description of the geology and the premining ground-water-flow regime, and (2) determine premining water quality. A thorough understanding of premining conditions would document the water quality and assist in differentiating the effects of mining and reclamation from the effects of FGD by-product application on water quality. Site characterization began with a search of data bases maintained by the USGS and State and local agencies. The USGS was responsible for geologic mapping, rock coring and sampling, lithologic and geophysical logging, triaxial chamber tests for permeability of underclay samples, and geochemical analyses of rock cores. Evaluation of the acid-producing potential of the postmining spoils was based on geochemical data obtained from rock cores by use of acid-base accounting techniques (Sobek et al., 1978).

2.3.5.1 Site Description

The study site is in eastern Coshocton County, approximately 3 miles east-southeast of West Lafayette, Ohio (Fig. 77). The active-mine site (also referred to as "the study area"), lies within the unglaciated part of the Appalachian Plateaus Physiographic Province. The site encompasses more than 1,000 acres wherein the Lower and Middle Kittanning coals (locally known as the number 5 and 6 coals, respectively) have been mined. The lithology beneath the site consists of relatively flat-lying strata of the Pennsylvanian Allegheny Group consisting of shales, sandstones, trace limestones, coal, and clay. Correlation of stratigraphic units between outcrops and (or) core holes is generally possible by reference to major coal seams, but such correlation can be difficult because overburden lithology varies greatly over

Figure 77. Study area and outcrops of coals, Coshocton County, Ohio.

small distances. Coal beds commonly overlie a soft, plastic clay layer, known as an underclay. The coal/underclay sequences are horizontally continuous throughout much of the coal-bearing area of eastern Ohio. Relief in the study area is as much as 300 feet. Surface drainage is to the south into the Muskingum River, which drains to the Ohio River.

The coal beds were being mined by a 60-cubic-yard dragline that made cuts oriented nearly north-south. At the time of the study, mining was progressing from west to east. As new overburden was removed from the east, the pit that was left to the west was filled. The number 5 coal is not persistent or of sufficient quality to warrant mining throughout the site; therefore the number 5 coal was actively mined only in the southeast part of the permit area.

2.3.5.2 Sources of Available Data

The primary source of information on the study area was the file of mine-permit applications submitted to the Ohio Department of Natural Resources (ODNR). Additional sources of data included geophysical well logs and water-quality data bases maintained by the ODNR Division of Geological Survey and Division of Water and the USGS.

2.3.5.2.1 Ohio Department of Natural Resources Mining Permits

The ODNR Coal Mining Permits (permit numbers D-689 and D-689-1) contain extensive information about the site and include mining-plan maps that display the location of the coal outcrops, seeps, springs, surface- and ground-water sampling sites, entrances to underground mines, and exploratory core holes (Figs. 77 and 78).

2.3.5.2.2 Site Map

The mining-plan map was digitized by use of a geographic information system (GIS). This facilitated reproduction of the map at a selected scale and size (for example, Fig. 77); more importantly however, use of the GIS allowed for spatial analyses such as area and volume calculations used in the acid-base accounting analyses discussed later in this chapter. Additionally, the GIS allowed for clear visual presentation of selected map elements, such as the location of all springs or previously drilled core holes. Each map feature from the mining-plan map was digitized into a separate GIS data layer, and attribute data were added. Examples of attribute data are core-hole number, depth to coal beds, and total depth of the core hole.

2.3.5.2.3 Core Data

The data contained in the ODNR permit files were used to investigate the stratigraphy and lithology of rocks at the mine site. Locations of selected core holes drilled by the mining company are shown on Figure 78, and a summary of the corresponding surface elevation, overburden thickness, and total sulfur content of the

Figure 78. Selected core holes and water-quality sites, Coshocton County, Ohio.

coal is listed in Table 32. ODNR permitting regulations require collection of samples from each lithologic unit in a core hole for analysis of total sulfur concentration and neutralization potential (NP, expressed as tons CaCO_3 per 1,000 tons of rock, hereafter referred to as "tons CaCO_3 /1,000 tons"). These data are used to calculate the calcium carbonate (CaCO_3) deficiency (the net neutralization potential or NNP) of the rock. The NNP data are used to evaluate the acid-producing potential of overburden by means of acid-base accounting techniques (Sobek et al., 1978).

The ODNR data indicate that the average thickness of the number 6 coal within the study area is 3.2 feet (0.98 meters), and the average total sulfur concentration for 23 analyses is 3.26 weight percent (range = 1.32 to 6.99 weight percent). The average thickness of the number 5 coal is about 1.5 feet (0.45 meters) and the average total sulfur concentration for 11 analyses is 4.46 weight percent (range = 2.07 to 7.22 weight percent). The ODNR core data indicate that the highest total sulfur concentrations are found in the coal beds and in shale or underclay adjacent to the number 6 coal. ODNR regulations governing premining overburden analysis require that only one sample be collected for each lithologically distinct unit penetrated during coring, regardless of the thickness of each individual unit. Therefore, only one sample is required for a 3-foot sandstone interval as well as for a 30-foot sandstone interval. Although this sampling method may result in a general view of the acid-producing potential of the rocks, sampling over large intervals may obscure definition of thin acid-producing strata. Two types of core-sampling strategies were used by USGS personnel to provide a more accurate measure of the acid-producing potential of overburden at the study area; these are discussed in the "Methods of Site Investigation" section. Because the mining company was interested only in general lithologic characteristics of the overburden and coal beds, such as overburden thickness, acid-producing capacity, coal thickness and quality, detailed information on the lithologic and hydrogeologic properties of the overburden was not available from the ODNR permit applications.

2.3.5.2.4 Geophysical Logs

The ODNR Division of Geological Survey, Subsurface Section, maintains records of geophysical logs from water wells and oil and gas wells. Most of these logs were generated for exploration of petroleum in rock formations that are much deeper than the surficial rocks of the coal mine. However, geophysical logs were available for two wells (permit numbers 6006 and 6120) shown in Figure 78. These logs included natural gamma-ray logs, which measure the radioactivity of strata intersected by the borehole, and neutron logs, which indicate the amount of water in the formation and are therefore a basis for estimating porosity of the rocks. The logs are not included in this report, but are available for inspection at the USGS and ODNR offices, in Columbus, Ohio. These logs provided USGS investigators with an indication of what to expect when natural gamma-ray logs were later run on core holes specifically drilled for this study. The mining company drilled many additional exploratory core holes throughout the study area, but they did not do any geophysical logging of these holes.

Table 32. Summary of overburden and total sulfur data from coal seams from permits D-689 and D-689-1, Coshocton County, Ohio.

[all data from Ohio Department of Natural Resources permit files; ft, feet; number 6 coal, Middle Kittanning Coal; number 5 coal, Lower Kittanning Coal; —, not available; location of core holes denoted by asterisk (*) are shown in Fig. 78]

Core hole number	Surface elevation	Overburden thickness (depth to number 6 coal)	Total sulfur	
			number 6 coal	number 5 coal
		feet	weight percent	
4988 *	—	30	1.71	4.88
5343	—	34	3.33	—
5344 *	1,033	90	3.03	—
5347	—	49	1.71	—
5351	—	29	1.32	—
8479 *	1,072	117	4.00	5.19
8480 *	1,020	59	2.89	5.10
8481 *	1,000	46	1.64	3.83
9011 *	1,023	77.1	3.80	2.07
9012 *	1,007	67.6	6.99	6.86
9013 *	1,013	65.8	4.38	3.55
9014 *	1,004	54.3	5.52	—
9015 *	980	20.5	1.38	3.98
9016 *	1,030	70.3	2.32	—
9017 *	1,098	132	1.58	7.22
9018	1,021	60.3	4.89	2.98
9019	988	29.2	2.95	—
9020	1,035	85.6	6.47	—
9021	1,034	41.6	3.62	—
9751 *	1,048	73	3.03	—
9752	1,037	84	2.00	—
9753	1,111	138	3.59	—
11177	1,109	135.8	—	—
11191	1,021	69.5	—	—
11198	1,114	166.5	—	—
11199	1,164	191.0	—	—
11290	1,047	78.9	—	—
11292	1,085	115.4	—	—
11293	1,069	99.8	—	—
11294	1,088	123.5	—	—
11295	992	44.0	—	—
14285	1,164	187	2.85	3.44
		Mean	3.26	4.46

2.3.5.2.5 Water-Quality Data

ODNR permitting regulations require that the mining company sample and analyze surface water collected from streams, springs, and seeps found on the mine site. Water-quality data are also submitted for ground-water samples collected from domestic or municipal wells near the mine site. Water-quality samples for determination of selected properties and constituents were collected in and near the study area and are listed in Table 33.

The surface-water-quality data show that, with the exception of some acidic seeps exiting an old drift mine (site DM-109 and an unnamed mine, Fig. 78), acid mine drainage is unusual at the study site. Acidic water sampled at seeps exiting the abandoned drift mine (site labeled DM-109, Fig. 78) and stream-sampling sites near an unnamed drift mine (D-111 and U-110, Fig. 78) have chemical characteristics similar to those of mine-drainage waters in other coal-mining areas of the eastern United States (pH < 4 and elevated concentrations of iron, manganese, and sulfate). Water-quality data for three domestic wells adjacent to the site (W-50, W-51, and W-130) also are listed in Table 33, and the physical characteristics of the wells are listed in Table 34. The domestic wells generally yield less than 10 gallons per minute. Mining-permit data for these wells are incomplete; however, the wells appear to be screened near the number 5 coal at an approximate elevation of about 925 to 935 feet above sea level. The water-quality data indicate that the well waters do not exceed limits set by U.S. Environmental Protection Agency drinking water regulations for dissolved constituents, but they may contain high concentrations of dissolved solids, iron, and sulfate. Two springs near the permit areas are used as water supply: DS-138 for agricultural uses, and DS-177 for domestic supply (Fig. 78). The permit notes that, in general, water quality is related to the lithology of the formation from which the spring or well produces.

2.3.5.2.6 Other Data Sources

The USGS ground-water data base, the Ground-Water Site Inventory (GWSI), did not contain records of any wells within 1 mile of the site. Previous water-quality studies by the USGS in the coal-mining areas of Ohio (Cunningham and Jones, 1990; Razem, 1984) documented water quality at sites with similar topography, land use, and lithology and can be used as an indicator of premining water quality. Additionally, the USGS has published numerous reports on coal hydrology in eastern Ohio (Sedam and Francy, 1993; Sedam, 1991; Jones, 1988; Hren, 1986; Eberle and Razem, 1985; Razem and Sedam, 1985; Hren et al., 1984; Engelke et al., 1981; Pfaff et al., 1981). Reports on the chemistry of coal and related rock formations have been published as a series of circulars by the ODNR (Botoman and Stith, 1988, 1986, 1981, 1978). These reports list chemical analyses of coals by coal seam and county.

Table 33. List of selected water-quality analyses from permit areas D-689 and D-689-1, Coshocton County, Ohio.

[Abbreviated units of measure: ft³/s, cubic feet per second; μ S/cm, microsiemens per centimeter at 25 degrees Celsius; mg/L, milligrams per liter. Prefix before site identification number: U, upstream surface-water sample; D, downstream surface-water sample; S, spring; DS, developed spring; W, drilled well; DM, drift mine; WI, water impoundment sample. Suffix after site identification number: H, sample collected during high flow; L, sample collected during low flow. Lithologic units codes: 1, shallow sandstones and shales within undifferentiated Freeport Sandstone; 2, Middle Kittanning Coal (number 6); 3, Lower Kittanning Coal (number 5); 4, Clarion Sandstone. --, not applicable or not measured]

Site identi- fication number	Sample date	Dis- charge	Litho- logic unit	pH	Specific con- ductance	Acidity	Alka- linity	Man- ganese, total	Sulfate, total	Iron, total	Iron, dis- solved	Sus- pended solids, total	Hard- ness, total
		ft ³ s ⁻¹		standard units	μ S cm ⁻¹	mg L ⁻¹ as CaCO ₃				mg L ⁻¹			mg L ⁻¹ as CaCO ₃
U-70 H	5/18/85	0.05	--	7.15	170	6.84	22.42	0.12	27.0	0.47	<0.04	2.0	--
U-70 L	8/26/86	0.01	--	6.94	270	6.65	58.33	<0.02	63.0	<0.04	<0.04	2.0	84.0
U-71 H	5/18/85	0.14	--	7.46	320	10.26	57.00	0.28	60.0	0.25	0.07	1.0	--
U-71 L	8/26/86	0.003	--	6.98	609	8.74	98.23	1.35	143.0	<0.04	<0.04	19.0	194.0
U-80 H	5/18/85	0.08	--	7.33	320	7.60	32.49	0.85	53.0	3.68	0.08	25.0	--
U-80 L	8/26/86	<0.001	--	7.15	357	4.37	84.36	0.03	48.0	<0.04	<0.04	21.0	103.0
U-81 H	5/18/85	0.25	--	7.71	347	8.36	51.30	0.16	54.0	0.18	<0.04	3.0	--
U-81 L	8/26/86	0.15	--	7.32	431	7.41	156.94	<0.02	73.0	<0.04	<0.04	11.0	135.0
U-110 H	5/18/85	0.03	--	6.82	200	8.55	19.00	1.37	51.0	2.02	0.10	46.0	--
U-110 L	8/26/86	0.01	--	3.84	1260	51.30	0.00	5.15	740.0	1.112	0.33	25.0	486.0
D-1 H	5/18/85	0.33	--	7.71	250	11.02	70.87	0.39	39.0	1.44	0.08	30.0	--
D-1 L	8/26/86	0.16	--	6.63	515	5.51	151.43	<0.02	59.0	<0.04	<0.04	4.0	149.0
D-4C H	5/18/85	1.0	--	7.78	195	9.69	50.73	0.21	36.0	0.31	0.05	1.0	--
D-4C L	10/8/86	0.5	--	8.27	220	8.57	30.40	0.03	31.0	0.09	<0.04	16.0	56.0
D-82 H	5/18/85	1.0	--	7.65	347	9.88	71.82	0.11	53.0	0.20	<0.04	4.0	--
D-82 L	8/26/86	0.07	--	6.95	467	8.55	150.67	<0.02	67.0	<0.04	<0.04	3.0	147.0
D-111 H	5/18/85	0.16	--	4.15	250	23.18	0.0	0.77	64.0	0.34	0.05	5.0	--
D-111 L	8/26/86	0.04	--	3.34	966	90.63	0.0	3.50	430.0	1.27	1.05	1.0	328.0
U-123 H	2/21/91	4.0	--	6.97	190	4.56	27.55	<0.02	37.0	0.12	--	11.0	76.0
S-100 H	2/21/91	0.3	--	7.14	130	5.89	24.32	<0.02	24.0	0.07	--	19.0	54.0
S-101 H	2/21/91	3.0	--	7.38	80	2.28	13.49	<0.02	15.0	0.04	--	2.0	32.0
S-103 H	2/21/91	5.0	--	7.28	100	8.55	22.61	<0.02	14.0	0.04	--	1.0	32.0
S-104 H	2/21/91	1.0	3	7.23	90	5.89	15.77	<0.02	24.0	0.05	--	7.0	27.0

Table 33. continued.

Site identi- fication number	Sample date	Dis- charge	Litho- logic unit	pH	Specific con- ductance	Acidity	Alka- linity	Man- ganese, total	Sulfate, total	Iron, total	Iron, dis- solved	Sus- pended solids, total	Hard- ness, total
		ft ³ s ⁻¹		standard units	$\mu\text{S cm}^{-1}$	mg L ⁻¹ as CaCO ₃				mg L ⁻¹			mg L ⁻¹ as CaCO ₃
S-106 H	2/21/91	5.0	1	7.32	90	6.08	23.56	<0.02	21.0	0.09	--	17.0	36.0
S-113 H	2/21/91	1.0	1	7.06	120	3.42	18.24	<0.02	34.0	0.06	--	20.0	44.0
S-120 H	2/21/91	0.5	3	6.66	115	6.27	9.31	<0.02	31.0	0.06	--	5.0	30.0
S-121 H	2/21/91	0.5	4	6.60	120	4.37	8.93	0.13	36.0	0.39	--	21.0	48.0
S-124 H	2/21/91	0.5	4	7.39	190	8.17	30.21	<0.02	34.0	0.10	--	16.0	88.0
D-123 H	2/21/91	0.15	--	6.98	180	7.22	27.74	0.03	37.0	0.18	--	10.0	63.0
D-125 H	2/21/91	0.25	--	6.84	270	8.74	29.07	0.04	22.0	0.22	--	7.0	110.0
S-126 H	2/21/91	0.5	4	7.54	155	5.51	50.54	<0.02	30.0	0.07	--	15.0	71.0
DS-111 H	2.21/91	2.0	2	7.28	100	4.37	18.05	<0.02	21.0	0.05	--	23.0	28.0
DS-118 H	2/21/91	2.0	2	7.38	150	7.03	17.86	<0.02	34.0	0.07	--	19.0	88.0
W-50 H	2/21/91	--	3	8.36	357	0.0	16.34	<0.02	40.0	0.05	--	<1.0	180.0
W-51 H	2/21/91	--	3	8.33	399	0.0	209.76	<0.02	27.0	0.09	--	<1.0	203.0
DM-102 H	2/21/91	1.0	2	7.72	90	9.69	24.13	0.02	12.0	0.10	--	2.0	36.0
DM-109 H	2/21/91	0.5	2	3.89	220	31.92	0.00	0.06	60.0	0.44	--	4.0	50.0
DM-115	2/21/91	5.0	2	7.02	100	2.28	30.97	<0.02	16.0	0.06	--	3.0	31.0
DM-122 H	2/21/91	2.0	2	7.63	180	10.07	39.52	0.03	44.0	0.50	--	18.0	78.0
S-128 H	2/25/91	1.5	3	7.83	120	4.18	19.00	<0.02	23.0	0.06	--	6.0	56.0
S-132	2/25/91	0.3	4	7.63	260	6.27	68.40	<0.02	16.0	0.11	--	9.0	112.0
S-133 H	2/25/91	1.4	4	7.55	240	6.46	52.82	0.05	50.0	0.73	--	14.0	94.0
S-136	2/25/91	0.4	2	7.01	120	26.60	11.02	<0.02	17.0	0.42	--	10.0	44.0
S-139 H	2/25/91	0.5	2	7.45	140	2.09	21.66	<0.02	34.0	0.25	--	9.0	48.0
S-140 H	2/25/91	3.0	2	6.95	110	2.85	8.36	<0.02	23.0	0.12	--	7.0	32.0
S-143	2/25/91	0.5	4	6.94	90	2.09	8.93	<0.02	25.0	0.23	--	15.0	28.0
S-144	2/25/91	0.3	4	6.83	80	3.80	9.31	0.03	22.0	0.08	--	8.0	29.0
S-148	2/25/91	0.4	2	7.10	120	3.80	25.46	0.02	17.0	0.20	--	20.0	44.0
S-150	2/25/91	2.0	4	7.39	130	7.79	31.54	<0.02	33.0	0.07	--	4.0	62.0
S-153 H	2/25/91	0.4	3	7.18	190	6.84	44.08	0.32	57.0	0.29	--	2.0	94.0
S-155	2/25/91	2.0	3	7.35	170	8.36	33.44	0.03	46.0	0.62	--	6.0	73.0

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Table 33. continued.

Site identi- fication number	Sample date	Dis- charge	Litho- logic unit	pH	Specific con- ductance	Acidity	Alka- linity	Man- ganese, total	Sulfate, total	Iron, total	Iron, dis- solved	Sus- pended solids, total	Hard- ness, total
		ft ³ s ⁻¹		standard units	$\mu\text{S cm}^{-1}$	mg L ⁻¹ as CaCO ₃				mg L ⁻¹			mg L ⁻¹ as CaCO ₃
DS-138 H	2/25/91	2.00	4	8.06	451	10.07	100.13	<0.02	115.0	0.07	--	1.0	221.0
D-151 H	2/25/91	0.25	--	7.83	150	9.12	27.74	0.02	27.0	0.16	--	3.0	54.0
W-130 H	2/25/91	--	4	7.92	525	18.81	293.74	0.13	<10.0	0.10	--	2.0	126.0
D-6 H	3/11/91	0.20	--	6.89	150	31.73	26.79	0.04	38.0	0.18	--	8.0	42.0
D-7 H	3/11/91	7.00	--	7.04	160	8.55	22.42	0.13	34.0	0.24	--	12.0	49.0
D-9 H	3/11/91	3.00	--	7.43	230	11.21	36.86	0.03	13.0	0.17	--	9.0	74.0
U-152A H	3/11/91	0.70	0	7.32	150	5.89	20.33	0.07	30.0	0.16	--	10.0	51.0
S-160 H	3/11/91	1.50	2	6.48	110	7.98	11.78	<0.02	13.0	0.06	--	2.0	24.0
S-170 H	3/11/91	2.00	2	6.63	120	4.18	12.35	<0.02	13.0	0.04	--	4.0	25.0
S-186 H	3/11/91	1.00	2	6.85	150	12.92	25.46	<0.02	<10.0	0.04	--	18.0	37.0
WI-1	3/11/91	--	--	7.97	200	6.65	63.65	0.09	34.0	0.43	--	4.0	77.0
WI-2	3/11/91	--	--	7.39	260	7.03	47.69	0.02	17.0	0.15	--	3.0	98.0
DS-177	3/11/91	5.00	4	7.52	200	10.83	55.29	<0.02	36.0	0.09	--	2.0	86.0
DM-166	3/11/91	2.50	2	7.38	130	3.61	10.64	0.03	31.0	0.04	--	16.0	39.0
W-130L	6/17/91	--	4	7.99	588	11.02	312.40	0.14	<10.0	<0.04	--	1.0	114.0
U-152A L	6/17/91	0.05	--	7.48	240	4.56	34.40	<0.02	27.0	0.07	--	2.0	85.0
DS-138 L	6/17/91	1.50	4	7.48	903	27.36	219.40	<0.02	180.0	0.39	--	1.0	447.0
DS-177 L	6/17/91	2.00	4	7.79	368	8.36	106.60	<0.02	40.0	0.21	--	3.0	140.0
DM-166 L	6/17/91	0.40	2	6.41	200	8.74	9.80	0.03	45.0	<0.04	--	6.0	65.0
U-123 L	6/10/91	0.50	--	7.73	280	9.31	62.40	0.08	32.0	0.36	--	5.0	102.0
D-123 L	6/10/11	4.00	--	7.80	290	7.03	104.40	0.07	42.0	0.33	--	2.0	128.0
D-125 L	6/10/91	0.05	--	7.87	290	9.50	112.40	0.55	41.0	0.58	--	3.0	131.0
DM-109 L	6/10/91	<0.10	2	3.73	290	50.54	0.00	0.16	88.0	1.37	--	12.0	90.0
DM-122 L	6/10/91	<0.10	2	7.96	290	7.22	103.80	0.05	45.0	0.43	--	16.0	142.0
DS-111 L	6/10/91	1.50	2	7.37	120	6.27	31.00	0.05	27.0	0.38	--	17.0	48.0
DS-118 L	6/10/91	0.50	2	7.32	220	7.60	55.60	<0.02	31.0	0.05	--	2.0	89.0
DM-102 L	6/10/91	0.50	2	7.62	190	5.51	61.60	0.10	<10.0	1.08	--	8.0	72.0
S-100 L	6/10/91	<0.10	2	7.05	200	6.08	64.40	0.19	24.0	0.29	--	9.0	96.0

Table 33. continued.

Site identi- fication number	Sample date	Dis- charge	Litho- logic unit	pH	Specific con- ductance	Acidity	Alka- linity	Man- ganese, total	Sulfate, total	Iron, total	Iron, dis- solved	Sus- pended solids, total	Hard- ness, total
		ft ³ s ⁻¹		standard units	$\mu\text{S cm}^{-1}$	mg L ⁻¹ as CaCO ₃				mg L ⁻¹			mg L ⁻¹ as CaCO ₃
S-101 L	6/10/91	<0.10	2	6.74	130	3.42	19.40	0.09	26.0	0.19	--	6.0	58.0
S-103 L	6/10/91	0.40	2	7.64	261	14.44	100.00	0.04	41.0	0.06	--	8.0	125.0
S-104 L	6/10/91	<0.10	3	7.44	150	4.75	51.80	0.06	14.0	0.20	--	5.0	62.0
S-106 L	6/10/91	<0.10	1	7.42	140	3.42	37.60	0.17	<10.0	1.40	--	2.0	62.0
S-113 L	6/10/91	0.20	1	7.37	170	7.98	39.40	0.04	43.0	0.75	--	13.0	76.0
S-120 L	6/10/91	0.15	3	7.43	190	5.32	48.40	0.03	40.0	0.73	--	1.0	88.0
S-121 L	6/10/91	0.10	4	6.57	180	10.83	45.00	0.62	33.0	1.70	--	27.0	77.0
S-124 L	6/10/91	0.10	4	6.84	280	24.51	120.80	1.43	24.0	11.30	--	37.0	149.0
S-126 L	6/10/91	0.30	4	8.12	410	10.83	192.40	0.29	53.0	1.83	--	14.0	212.0
S-128 L	6/10/91	<0.10	3	7.12	130	3.99	20.80	0.04	18.0	0.39	--	2.0	48.0
S-153 L	6/10/91	<0.10	3	7.66	504	11.78	95.00	0.09	127.0	0.11	--	3.0	249.0
S-140 L	6/17/91	<0.10	2	6.88	130	5.70	28.20	0.07	13.0	0.11	--	8.0	44.0
S-160 L	6/17/91	<0.10	2	6.94	120	2.66	21.00	0.11	13.0	0.35	--	12.0	50.0
S-170 L	6/17/91	0.40	2	7.19	120	7.03	35.00	<0.02	13.0	0.56	--	7.0	43.0
S-186 L	6/17/91	<0.10	2	7.63	170	5.32	49.60	<0.02	13.0	0.94	--	2.0	72.0
W 50 L	6/17/91	--	3	7.86	399	16.72	182.60	<0.02	38.0	0.07	--	1.0	193.0
W 51 L	6/17/91	--	3	7.92	473	15.77	223.80	<0.02	39.0	0.07	--	<1.0	215.0
D-6 L	6/17/91	5.00	--	7.37	240	11.21	48.60	0.11	28.0	0.39	--	5.0	80.0
D-7 L	6/17/91	1.00	--	7.58	240	4.94	45.80	0.04	42.0	1.08	--	7.0	74.0
D-9 L	6/17/91	0.50	--	7.52	420	13.30	81.60	0.56	13.0	0.61	--	10.0	137.0
D-151 L	6/17/91	0.05	--	7.72	310	6.65	94.20	0.10	40.0	0.27	--	3.0	128.0
S-133 L	6/17/91	<0.01	4	7.73	441	11.97	120.40	<0.02	78.0	1.86	--	6.0	178.0
S-139 L	6/17/91	0.40	2	7.46	180	2.47	40.20	0.06	15.0	0.25	--	8.0	59.0

Table 34. List of physical characteristics of domestic wells located adjacent to permits D-689 and D-689-1, Coshocton County, Ohio.

Map designation	Surface elevation above sea level	Depth of well below land surface	Approximate water level		Lithology of screened interval
			below land surface	above sea level	
-----feet-----					
W-50	995	70	38	957	Number 5 coal
W-51	1,010	75	17	993	Number 5 coal
W-130	805	--	--	--	--

2.3.5.3 Methods of Active Mine Site Investigation

In this section, methods used by USGS personnel to characterize the active-mine site are described. Preliminary site characterization was done by the mining company, whose main concern was the thickness of overburden and the quality and extent of the coal seams. Using this information, the USGS did additional investigations to (1) define the stratigraphy above and below the coal seams, (2) determine the hydrogeologic characteristics of the overburden rock and coal underclays, and (3) determine the bulk geochemistry of overburden that would become postmining spoils.

2.3.5.3.1 Mapping

Geologic mapping was done by the mining company to comply with ODNR mining permit requirements. Additional site work was done by USGS personnel to define site stratigraphy and hydrogeology in greater detail. The mining map prepared by the coal company was used as a base map for verification and refinement of site hydrogeology. Location of features shown in Figures 77 and 78 were verified as a result of this work. Lithology of the main stratigraphic units in the study area is expressed topographically; that is, strata composed of "soft" rocks susceptible to weathering (such as shales) formed gentle surface slopes whereas strata composed of "hard" resistant rocks (such as sandstones) formed steeper slopes. Numerous springs and seeps (which are commonly related to coal beds) were located by use of the mining company's map and helped to delineate the surface exposure of the coals (Figs. 77 and 78).

2.3.5.3.2 Rock Coring

Rock cores were collected to define the lithologic distribution of the overburden, coal beds, and underlying aquifers and to obtain samples for geochemical analysis. Three core holes were drilled to an approximate depth of 300 feet each within the area that was to contain the postmining FGD by-product application plots (CS-158, CS-159, and CS-160; Fig. 78). The cores were collected by means of a rotary drill rig operated by the ODNR, Division of Geological Survey, that produced 2-inch-diameter core sections 10 feet in length.

The elevation at the base of the core holes corresponded approximately to the elevation of the two small streams draining the study area. The coal seams were known to lie at an elevation ranging from 930 to 960 feet above sea level (corresponding to approximately 120 to 200 feet below land surface, depending on surface topography). Determination of the extent and nature of the underlying aquifers was also an important aspect of this work; therefore, hole CS-159 was drilled to allow estimation of the dip of the rocks between holes and to evaluate the horizontal distribution of rocks between the holes (approximately 3,300 feet apart). Detailed lithologic logs were compiled during the drilling, and composite core samples were collected for geochemical analysis. A detailed description of the sampling techniques can be found in section 2.3.5.3.6 "Sampling and Analysis of Rock Cores."

2.3.5.3.3 Lithologic Logging of Core Holes

Detailed lithologic logs were constructed during the coring process. Notes were taken regarding color, texture, mineralogy, rock type, thickness, bedding structures, nature of lithologic contacts (sharp, gradational), general hydrogeologic properties (porosity), and the presence or absence of fractures. Natural fractures were distinguished from drilling-induced fractures by the presence of iron-oxide staining or other secondary mineralization on the fracture faces. Fractures produced during drilling would not exhibit such features. Water levels were recorded several days after coring was completed to determine the approximate depth to the saturated zone. These water levels did not represent true water levels for any given aquifer intersected by the core hole; instead, they represented a composite water level for the entire open-hole interval.

2.3.5.3.4 Geophysical Logging of Core Holes

Geophysical logs recorded in each open core hole included temperature, electric, caliper, and natural gamma-ray logs (Keys and MacCary, 1985). Temperature logs were recorded beneath the water table to detect slight differences in water temperature which may indicate a source or sink of water to or from of the core hole. Three electric logs were recorded; self-potential, single-point resistance, and resistivity. A self-potential log documents the natural potential difference, or voltage, created by two strata of different lithology such as shale and sandstone. A single-point resistance log measures the drop in current between a source of electric current placed in the core

hole and a surface electrode; it differs from a resistivity log in that the resistivity probe and the receiving electrode are both placed in the core hole. A resistivity log, produced when an electric current is passed through the rocks, measures the resistivity of solid and liquid materials between two measuring points. The recorded resistivity is related in a general way to the porosity and hydraulic conductivity of the rocks but is also affected by the composition of pore water. Interpretation of the lithology by use of resistivity logs requires the assumption that either the water quality remains constant as the lithology changes or the lithology remains constant as the water quality changes. For the purposes of this study, the water quality was assumed to remain constant. A caliper log records variations in the diameter of the core hole. The core-hole diameter is rarely constant, owing to variable lithology and "washouts" where the formation has been eroded by drilling fluid. Caliper logs are useful in the interpretation of other logs and can aid in locating fractures and clay layers--broken rock and clay are especially susceptible to washouts. A natural gamma-ray log records the relative radioactivity of geologic materials adjacent to the core hole. Shales and clays, because of their high potassium concentrations, emit relatively high levels of natural gamma radiation, whereas coals and limestones emit low levels of radiation (Keys and MacCary, 1985). Natural gamma logs are ideal for use in coal-bearing strata because the gamma log of a coal/underclay sequence has a clearly visible low/high response.

2.3.5.3.5 Permeability Testing of Underclay Samples

Samples of underclays beneath the number 5 and 6 coals were subjected to triaxial chamber tests to determine coefficients of permeability because the presence of a poorly permeable formation such as the underclay may prohibit vertical flow of ground water. Samples were obtained from the core barrel and were immediately wrapped in plastic wrap and aluminum foil to prevent desiccation. The samples were then sent to the laboratories of The Ohio State University (OSU) Civil Engineering Department (under the direction of Dr. William Wolfe) for testing. Samples were placed in a chamber that subjected the sample to a predetermined confining pressure and hydraulic gradient that approximated field conditions. A value for coefficient of permeability was then measured. If field conditions were unknown, estimated ranges of confining pressures and hydraulic gradients were used to determine whether these variables had an effect on the measured coefficient of permeability. A flow rate, or hydraulic conductivity, was measured by recording the amount of time it takes a given amount of water to pass through the sample of known thickness. Because flow through clay and other confining units is predominantly vertical, the samples were tested for vertical hydraulic conductivity only.

2.3.5.3.6 Sampling and Analysis of Rock Cores

Two methods were used to sample the three rock cores collected at the site. During drilling, composite samples of the cores were collected to characterize the major- and trace-element geochemistry of overburden materials and coal underclays. These data were used to estimate possible concentrations of trace elements in

postmining spoil materials and to estimate the potential acidity of these materials (see section 2.5.3.4.4 "Acid-Base Accounting").

In the composite samples, strata in the upper 150 feet of each 300-foot core were divided into seven sampling intervals based on hydrologic and lithologic factors. The composite sampling intervals were labeled A through F and were defined as follows: (A) the weathered strata in the unsaturated zone at hole CS-159 only (because of its higher surface elevation), (B) the strata in the rest of the unsaturated zone, (C) the strata in the transitional zone between the unsaturated and saturated zones (the exact depth to the water table was unknown; therefore, a range of 50 feet was assumed to be the transitional zone), (D) the strata between the saturated zone and the top of the number 6 coal bed, (E) the number 6 underclay, and (F) the strata between the number 6 and number 5 coal beds. Coal seams were not sampled because most of the coal would be removed during mining; however, data for coal from other sources were used in subsequent analyses. Additionally, strata beneath the number 5 coal were not sampled because the mining company was only mining down to the number 5 coal; therefore the rock beneath the number 5 coal would not be incorporated into the postmining spoils.

For each composite sampling interval A through F, 1 to 2 inches of the continuous core was sampled within every 5-foot length. These subsamples were then combined into a single sample bag for that interval and shipped to the laboratory for analysis. This sampling strategy was intended to reveal clearly defined differences in lithologies and geochemical conditions such as saturated versus unsaturated, or reducing versus oxidizing.

The composite samples were analyzed by laboratories operated by the USGS Branch of Geochemistry (BOG) Denver, Colorado, for forms of sulfur and carbon and for more than 40 major and trace elements by inductively coupled plasma (ICP) scans using techniques described in Arbogast (1990). Owing to limitations in the ICP analytical technique, arsenic and selenium were analyzed in replicate by means of the atomic absorption hydride method. Major-, minor-, and trace-element data reported by the BOG laboratory for the composite core samples were originally reported as total element concentrations. Because major- and minor-element data for rock samples are traditionally reported in terms of oxide components, elemental data for the composite core samples were converted to their equivalent oxide component concentrations by multiplying the elemental concentration by the appropriate stoichiometric factor. For example, to convert Na to Na₂O, the reported Na concentration is multiplied by 2.7. Major- and minor-oxide component data were then summed with total C and total S concentration data to estimate the total silica concentration of the composite core samples.

Examination of the data indicated that the composite sampling method may not have provided adequate resolution for accurate definition of potentially acid-producing horizons. Therefore, the cores were resampled according to standard acid-base accounting (ABA) techniques (Sobek et al., 1978). Resampling involved removing a

quarter of the entire length of core in 3-foot intervals. If discrete changes in lithology were noted, smaller sampling intervals were used. The quartered core samples were ground to a powder less than 100 mesh and were then analyzed for total sulfur and neutralization potential (total carbonate) at Geochemical Testing, in Somerset, Pennsylvania, by use of techniques outlined by Sobek et al. (1978). "Fizz," a qualitative measure of calcium carbonate reaction with dilute acid, also was measured by use of methods of Sobek et al. (1978).

2.5.3.4 Preliminary Hydrogeologic and Geochemical Characterization

Data and results of the Phase 2 research activities are presented in this section. These include lithologic logs, geophysical logs, results of permeability tests, and geochemical analyses of core materials. Additionally, an evaluation of the acid-producing potential of overburden at the site is given.

2.5.3.4.1 Lithology

Lithologic logs of cores produced from three exploratory holes drilled at the study site indicate that the stratigraphy of the site is complex (Table 35). Major sandstone and shale units above the number 6 coal generally can be correlated, although variations in thickness of the individual units are considerable. The number 5 coal thins northward and is absent at core holes CS-158 and CS-159; this observation is consistent with exploration data from the ODNR mining permits. The elevation of the number 6 coal bed at the three core sites indicates that strata in the study area strike roughly east-west and dip slightly to the north at approximately 0.015 feet/foot. This strike and dip may differ locally, however, because of (1) paleotopographic relief on bedding planes and the coal and underclay surfaces and (2) low-amplitude folding or other geologic structures.

Numerous fractures were seen in each core; depths of these fractured intervals are listed in Table 35. These fractures probably formed as erosion removed surficial rocks, thereby lessening the confining stresses and allowing underlying units to expand. Fractures may significantly affect the direction and magnitude of ground-water flow. Composite water levels measured in each of the holes are near or at the same elevation as fractures noted in the cores, an indication that fractures may be the principal sinks or sources of water to the premining aquifer system.

The geophysical logs for core holes CS-158, CS-159, and CS-160 are shown in Figures 79a-c and include water levels and lithologic interpretations of the logs that were made in conjunction with the geologic logs ran during coring. The temperature log shows small fluctuations in water temperature, presumably due to continuous flow of water into the core hole. A clearly defined jump in temperature is shown in CS-158 at 950 feet. This depth does not correspond to a logged fracture, sandstone, or coal bed, but does correlate with a slight change in lithology from gray shale to dark-gray shale that may imply flow of water along a bedding plane. The three electric logs clearly show the presence of coal and sandstone (coal is high in resistance relative to shale and

Table 35. Lithologic logs of three core holes drilled by the U.S. Geological Survey, Coshocton County, Ohio

Lithologic description	Thick- ness	Depth below		Elevation above sea		Fractured intervals, depth from land surface
		From	To	From	To	
feet						
Hole CS-158: latitude: 401427; longitude: 0813930; surface elevation: 1,092 ft above sea level.						
Soil	3.0	0.0	3.0	1,092.00	1,089.00	
No recovery	7.0	3.0	10.0	1,089.00	1,082.00	
Brown sandstone	25.0	10.0	35.0	1,082.00	1,057.00	
Gray sandstone	11.0	35.0	46.0	1,057.00	1,046.00	
Blue-gray clay	2.0	46.0	48.0	1,046.00	1,044.00	
Gray shale	3.0	48.0	51.0	1,044.00	1,041.00	
Gray sandstone	5.0	51.0	56.0	1,041.00	1,036.00	
Gray shale	45.5	56.0	101.5	1,036.00	990.50	
Gray sandstone	15.5	101.5	117.0	990.50	975.00	103.5
Dark-gray shale	15.5	117.0	132.5	975.00	959.50	
Coal, no. 6	4.0	132.5	136.5	959.50	955.50	
Gray clay	3.5	136.5	140.0	955.50	952.00	
Gray shale	10.0	140.0	150.0	952.00	942.00	
Dark-gray shale	25.5	150.0	175.5	942.00	916.50	168.0-170.0
Light-gray sandstone	1.5	175.5	177.0	916.50	915.00	
Greenish-white	6.0	177.0	183.0	915.00	909.00	
Dark-gray shale	30.0	183.0	213.0	909.00	879.00	
Dark gray-black shale	14.0	213.0	227.0	879.00	865.00	226.0
Dark-gray limestone	1.7	227.0	228.7	865.00	863.30	
Coal	0.8	228.7	229.5	863.30	862.50	
Gray shale	1.5	229.5	231.0	862.50	861.00	
Light-gray shale	4.0	231.0	235.0	861.00	857.00	
Dark-gray shale	18.1	273.0	291.1	819.00	800.90	
Dark-gray limestone	2.2	291.1	293.3	800.90	798.70	
Coal	0.9	293.3	294.2	798.70	797.80	
Gray shale	1.3	294.2	295.5	797.80	796.50	
Dark-gray shale	7.5	295.5	303.0	796.50	789.00	

Table 35. continued

Lithologic description	Thick- ness	Depth below		Elevation above sea		Fractured intervals, depth from land surface
		From	To	From	To	
feet						
Brown sandstone	25.7	8.0	33.7	1,155.00	1,129.30	
No recovery	1.3	33.7	35.0	1,129.30	1,128.00	
Gray clay	2.0	35.0	37.0	1,128.00	1,126.00	36.0-41.0
Dark-gray shale	16.0	37.0	53.0	1,126.00	1,110.00	
Calcareous dark-gray	8.0	53.0	61.0	1,110.00	1,102.00	
Dark-gray shale	18.0	61.0	79.0	1,102.00	1,084.00	63.0-66.0; 77.0-79.0
Dark-gray limestone	0.3	79.0	79.3	1,084.00	1,083.70	
Gray sandstone	1.0	79.3	80.3	1,083.70	1,082.70	
Gray limestone	0.3	80.3	80.6	1,082.70	1,082.40	
Gray shale	15.4	80.6	96.0	1,082.40	1,067.00	
Brown sandstone	67.3	96.0	163.3	1,067.00	999.70	125.0-127.0; 151.0
Gray sandstone	12.7	163.3	176.0	999.70	987.00	169.0-171.0
Dark-gray shale	3.0	176.0	179.0	987.00	984.00	
Gray sandstone	6.9	179.0	185.9	984.00	977.10	183.0-185.0
Dark-gray shale	20.1	185.9	206.0	977.10	957.00	
Coal, no. 6	3.0	206.0	209.0	957.00	954.00	
Blue-gray clay	2.0	209.0	211.0	954.00	952.00	
Dark-gray shale	3.0	224.0	227.0	939.00	936.00	
Hole CS-160: latitude: 401354; longitude: 0813919; surface elevation: 1,086 ft above sea level						
Soil	3.0	0.0	3.0	1,086.00	1,083.00	
No recovery	7.0	3.0	10.0	1,083.00	1,076.00	
Brown sandstone	33.0	10.0	43.0	1,076.00	1,043.00	
Gray shale	12.5	43.0	55.5	1,043.00	1,030.50	45.0-47.0
Gray clay	0.5	55.5	56.0	1,030.50	1,030.00	
Brown sandstone	0.7	56.0	56.7	1,030.00	1,029.30	
Light-gray shale	2.3	56.7	59.0	1,029.30	1,027.00	
Dark-gray shale	5.0	59.0	64.0	1,027.00	1,022.00	60.5
Gray shale	7.0	64.0	71.0	1,022.00	1,015.00	
Dark-gray shale	41.5	71.0	112.5	1,015.00	973.50	75.0; 98.0-101.0; 106.0
Coal, no. 6	3.0	112.5	115.5	973.50	970.50	
Blue-gray clay	3.5	115.5	119.0	970.50	967.00	

Table 35. continued

Lithologic description	Thick- ness	Depth below		Elevation above sea		Fractured intervals, depth from land surface
		From	To	From	To	
		feet				
Gray clay	5.5	136.0	141.5	950.00	944.50	
No recovery	5.5	141.5	147.0	944.50	939.00	
Light-gray shale	6.5	147.0	153.5	939.00	932.50	
Greenish-gray	6.5	153.5	160.0	932.50	926.00	155.0
Dark-gray shale	29.0	160.0	189.0	926.00	897.00	
Coal, trace	0.1	189.0	189.1	897.00	896.90	
Dark-gray shale	21.5	202.3	223.8	883.70	862.20	
Brownish-gray shale	2.2	223.8	226.0	862.20	860.00	
Dark-gray shale	0.0	226.0	246.0	860.00	840.00	
Light-gray shale	5.0	246.0	251.0	840.00	835.00	
Dark-gray shale	15.0	251.0	266.0	835.00	820.00	
Dark-gray limestone	2.6	266.0	268.6	820.00	817.40	
Coal	0.3	268.6	268.9	817.40	817.10	
Gray shale	5.1	268.9	274.0	817.10	812.00	
Dark-gray shale	15.0	274.0	289.0	812.00	797.00	
Dark-gray limestone	7.8	289.0	296.8	797.00	789.20	289.0
Coal	0.5	296.8	297.3	789.20	788.70	
Gray shale	2.7	297.3	300.0	788.70	786.00

Figure 79a. Geophysical log responses and lithologic interpretation, core hole CS-158, Coshocton County, Ohio.

b CS-159

c CS-160

Figure 79b, c. Geophysical log responses and lithologic interpretation, core hole CS-159 and CS-160, Coshocton County, Ohio.

clay). The high resistance values displayed in hole CS-158 at approximately 955 to 960 feet above sea level correspond to the location of the number 6 coal. At the interval between 909 and 916 feet above sea level, a sandstone displays a higher resistance than the overlying dark-gray shale, but similar resistance to the underlying shale. The distinction between sandstone and shale, however, can be made through examination of the gamma log within the same interval. Between 909 and 916 feet above sea level, the gamma log shows a low response (in counts per minute) which corresponds to the sandstone, whereas the response within the over- and underlying shale intervals is relatively higher. The caliper log shows the enlarged bore hole adjacent to clay layers that were eroded from circulation of the drilling fluid (see, for example, Figure 79a at 955 feet). The caliper log also shows the depth at which a 3.5-inch bit was used to install a surface casing (975 feet).

The gamma logs from all three holes show a well-defined jump at the coal/underclay sequence (for example, hole CS-158 from 952.0 to 959.9 feet). The coal/underclay sequences were typically the most continuous features throughout the study area; therefore, the gamma logs were the most useful tool for correlating lithology between core holes. As seen with the combination of electric logs and gamma logs in the three core holes, however, the use of data from two or more types of logs can strengthen the interpretation of lithology. The combination of gamma logs and caliper logs reveals the underclay horizons (hole CS-158 from 952.0 to 955.5 feet above sea level) because the gamma log shows a characteristic high gamma count, and the caliper log shows a washout.

2.5.3.4.2 Vertical Hydraulic Conductivity of Underclay Samples

The results of permeability tests on underclay samples from cores at the study site are listed in Table 36. Vertical hydraulic conductivities ranged from about 1.53×10^{-5} to 4.56×10^{-4} feet per day; the average value was 1.9×10^{-4} feet per day. This range is similar to published hydraulic conductivities for other types of clays (Freeze and Cherry, 1979; Heath, 1983). Fractures noted in the underclays may have resulted from reduction of stresses induced by the erosion of overlying rocks, desiccation, movement of adjacent bedrock by tectonic forces, and (or) chemical leaching. Because the underclays may function as barriers to downward flow of water and solutes, fractures within the clays may be significant conduits for vertical ground-water flow.

2.5.3.4.3 Geochemistry of Composite Core Samples

Detailed results of the geochemical analyses of the composite core samples are listed in Table 37. A summary of the results is given in Haefner and Rowe (1992). Before being crushed and pulverized, duplicate samples (denoted with the subscript "dup") were submitted to evaluate the analytical precision of the laboratory. However, heterogeneities in the core samples caused difficulty in producing true split samples and resulted in large differences in reported concentrations for some constituents. Changes in core geochemistry with depth are related to vertical variations in lithology.

Table 36. Results of triaxial chamber tests on underclay samples, Coshocton County, Ohio. All analyses by The Ohio State University, Department of Civil Engineering; psi, pounds per square inch; cm/s, centimeters per second; ft/d, feet per day.

Hole number	Approximate elevation of top of clay	Clay thickness	Cell pressure	Gradient	Vertical hydraulic conductivity	
	-----feet-----		psi		cm s ⁻¹	ft d ⁻¹
Upper Freeport (number 7)						
CS-159	1128.0	2.0	37	63	8.95x 10 ⁻⁹	2.54x10 ⁻⁵
Lower Freeport (number 6a)						
CS-158	1046.0	2.0	40	53	5.40 x 10 ⁻⁹	1.53 x 10 ⁻⁵
Middle Kittanning (number 6)						
CS-158	955.0	3.5	-- ^a	-- ^a	-- ^a	-- ^a
CS-159	954.0	2.0	20	41	1.61 x 10 ⁻⁷	4.56 x 10 ⁻⁴
			35	41	1.61 x 10 ⁻⁷	4.56 x 10 ⁻⁴
			70	60	9.29 x 10 ⁻⁸	2.63 x 10 ⁻⁴
CS-160	973.5	3.0	10	24	3.05 x 10 ⁻⁸	8.65 x 10 ⁻⁵
			35	24	3.05 x 10 ⁻⁸	8.65 x 10 ⁻⁵
Lower Kittanning (number 5)						
CS-160	950.0	5.5 ^b	35	35	5.86 x 10 ⁻⁸	1.66 x 10 ⁻⁴

^a Sample was too fractured to test.

^b Poor recovery within this section of core; clay thickness was at least 5.5 ft.

Table 37. Results of inductively coupled plasma scans and atomic absorption hydride analyses for selected elements and analyses for forms of sulfur and carbon on composite core samples, Coshocton County, Ohio.

[Major- and minor-element data are calculated from the reported element concentration data; SiO₂ data were estimated by use of the oxide computation; --, below detection limit; the elements arsenic through zinc are in parts per million; the compounds silica through organic sulfur are in weight percent; dup, duplicate; ft, feet. Intervals are as follows: A, weathered strata in the unsaturated zone from hole CS-159 only; B, strata in the unsaturated zone for all three holes; C, strata in the transitional zone between the unsaturated and saturated zones; D, strata between the saturated zone and the top of the number 6 coal bed; E, number 6 underclay; F, strata between the number 6 and number 5 coal beds.]

Hole identi- fier	Inter- val	As	Ba	Ce	Co	Cr	Cu	Ga	La	Li	Nb	Nd	Ni	Pb
CS-158	B	4.3	340	75	9	47	11	15	42	34	6	32	18	15
	C	2.6	490	99	14	66	20	21	47	64	--	49	28	8
	D	5.3	420	85	13	58	16	18	45	59	--	41	27	8
	D _{dup}	14.0	250	91	17	63	18	18	48	66	5	42	31	15
	E	35.0	54	96	20	110	35	25	49	64	6	39	84	53
	F	3.1	540	84	18	87	19	22	38	120	6	44	36	13
CS-159	A	20.0	380	86	24	130	30	16	41	71	9	42	67	25
	B	7.9	230	53	12	35	8	8	27	20	4	24	22	15
	B _{dup}	15.0	270	60	13	39	7	9	29	21	4	30	24	19
	C	15.0	220	66	11	35	5	10	31	19	--	32	18	15
	D	8.6	430	69	15	62	22	17	36	55	9	32	26	18
	E	23.0	150	120	33	130	50	30	56	190	8	55	81	337
	F	3.4	380	93	13	74	10	19	50	120	10	42	31	16
CS-160	B	11.0	280	85	13	49	7	11	43	30	5	36	22	17
	C	5.1	440	98	20	83	29	21	51	68	12	47	38	23
	D	9.4	490	94	17	78	27	24	51	96	12	44	49	25
	E	5.7	330	140	4	96	20	26	77	170	15	52	26	31
	F	4.0	480	74	16	88	17	22	34	130	12	38	41	15
	F _{dup}	3.3	440	83	12	79	17	20	45	95	13	40	32	21
	RD ^a	1.9	350	96	12	53	16	15	54	35	7	43	17	17
	OX ^b	1.0	430	98	15	70	19	19	50	49	9	45	29	19

Table 37. continued.

Hole identifier	Interval ^a	Sc	Se	Sr	Th	V	Y	Zn	SiO ₂	Al ₂ O ₃	CaO	MgO	FeO	K ₂ O
CS-158	B	10	0.2	110	11	65	14	51	64.62	24.95	0.21	0.81	2.83	4.10
	C	22	0.5	90	17	84	21	88	43.28	41.58	0.69	1.82	4.50	5.54
	D	20	0.3	120	13	78	21	72	46.62	35.53	1.30	1.53	5.92	4.82
	D _{dup}	18	0.5	130	14	82	23	82	46.27	33.64	1.96	1.53	6.04	5.30
	E	31	1.5	290	19	120	21	83	37.78	45.36	0.17	0.66	5.53	3.37
	F	27	0.7	140	14	110	21	120	42.80	41.58	0.31	1.66	4.89	6.02
CS-159	A	15	7.7	100	11	170	32	140	52.83	26.84	1.40	1.48	6.17	4.82
	B	6	0.3	63	6	47	11	170	75.53	15.50	0.56	0.50	2.57	2.89
	B _{dup}	7	1.1	65	7	57	17	110	73.29	15.88	0.39	0.53	4.24	2.89
	C	6	0.1	68	7	50	17	68	77.00	14.36	0.32	0.41	3.22	2.65
	D	13	0.4	100	10	87	17	77	51.48	27.59	0.66	1.82	8.49	5.06
	E	38	1.7	290	20	150	22	200	34.21	52.92	0.38	0.83	3.47	3.61
CS-160	F	20	1.1	160	14	110	23	92	53.95	33.26	0.55	1.13	4.24	5.06
	B	8	0.3	65	9	59	14	56	71.88	18.52	0.27	0.61	3.22	3.13
	C	16	0.4	100	13	98	24	110	50.47	31.75	0.56	1.66	6.04	5.30
	D	18	0.5	130	15	120	24	100	42.39	37.42	0.59	1.82	6.56	6.99
	E	25	1.7	280	18	120	19	29	50.77	41.58	0.31	0.66	1.41	3.37
	F	26	0.8	160	15	130	17	96	45.71	41.58	0.25	1.59	3.22	6.02
	F _{dup}	15	0.8	130	13	120	20	79	54.74	32.51	0.56	1.09	3.99	5.30
	RD ^a	10	0.2	230	12	63	21	78	56.41	24.57	4.90	1.14	3.99	4.10
CS-160	OX ^b	12	0.2	82	14	83	19	81	55.78	29.86	0.36	1.33	4.12	4.82

Table 37. continued.

Hole identi- fier	Inter- val	Na ₂ O	P ₂ O ₅	TiO ₂	MnO	Forms of carbon			S, Organic total	Forms of sulfur		
						Organic C	CO ₂	CO ₃		Sulfide S	Sulfate S	S
CS-158	B	1.54	0.11	0.43	0.017	0.02	0.03	0.04	0.34	0.30	0.02	0.02
	C	1.48	0.18	0.58	0.034	0.13	0.08	0.10	0.10	0.08	0.01	0.01
	D	1.75	0.21	0.47	0.103	0.70	0.59	0.74	0.46	0.42	0.02	0.02
	D _{dup}	1.73	0.21	0.53	0.098	0.81	0.58	0.72	1.30	1.24	0.02	0.04
	E	0.24	0.14	1.00	0.007	0.76	0.00	--	4.97	4.59	0.16	0.22
	F	0.43	0.21	0.80	0.046	0.98	0.20	0.25	0.07	--	--	--
CS-159	A	1.21	0.60	0.53	0.067	2.74	0.22	0.27	1.09	0.97	0.06	0.06
	B	1.64	0.14	0.27	0.048	0.15	0.08	0.10	0.12	0.10	0.01	0.01
	B _{dup}	1.59	0.25	0.28	0.062	0.19	0.06	0.07	0.35	0.29	0.04	0.02
	C	1.40	0.28	0.28	0.070	0.00	0.00	--	0.00	--	--	--
	D	1.83	0.25	0.45	0.207	0.85	0.91	1.14	0.40	0.37	0.01	0.02
	E	0.32	0.14	1.22	0.006	0.79	0.00	--	2.10	1.88	0.10	0.12
	F	0.32	0.07	0.72	0.090	0.09	0.39	0.49	0.13	0.12	<0.01	0.01
CS-160	B	1.73	0.14	0.40	0.039	0.07	0.00	--	0.00	--	--	--
	C	1.97	0.25	0.68	0.081	0.74	0.27	0.34	0.22	0.19	0.01	0.02
	D	1.19	0.28	0.67	0.084	0.80	0.26	0.32	0.95	0.89	0.04	0.02
	E	0.30	0.14	1.10	0.004	0.00	0.00	--	0.35	0.33	--	0.02
	F	0.38	0.11	0.88	0.021	0.11	0.05	0.06	0.08	--	--	--
	F _{dup}	0.32	0.14	0.80	0.036	0.11	0.32	0.40	0.08	--	--	--
	RD ^a	2.97	0.18	0.50	0.107	0.10	0.98	1.23	0.06	--	--	--
	OX ^b	2.97	0.21	0.52	0.041	0.00	0.00	<0.01	0.00	--	--	--

^a Sample of gray-green shale collected at elevation of 1,024 feet above sea level.^b Sample of same shale as above, but reddish-brown in color, collected at 1,026 feet above sea level.

For example, an inverse correlation between aluminum and estimated silica concentrations is a function of the sandstone content in the sample: strata dominated by sandstone are silica rich whereas intervals dominated by finer grained siltstones and shales are aluminum rich because of higher clay content. A similar correlation is observed for potassium and sodium. Elevated potassium concentrations are found in intervals dominated by shales and siltstones whereas higher sodium concentrations are found in sandstone-dominated intervals where sodium feldspar (albite) is present. The depletion in sodium concentration in clay-rich rock could also reflect ion exchange, which favors exchange of monovalent sodium ions for divalent cations such as calcium and (or) magnesium.

Geochemical alteration of aquifer materials (dissolution of manganese-bearing calcite and oxidation of organic carbon and sulfide minerals) by slightly acidic, oxygenated ground water affected major-element concentrations in the core samples. In a typical interval, calcium, manganese, organic carbon, carbonate carbon, and total sulfur concentrations were depleted in a sample of reddish-brown shale (CS-160OX; Table 37) relative to overlying unweathered gray-green shale (CS-160RD; Table 37) collected at elevations of 1,026 and 1,024 feet above sea level (60 and 62 feet below land surface), respectively, at core hole CS-160.

The most notable variations in major- and trace-element chemistry in the composite core samples were variations associated with the number 6 underclay. In all three cores, depletions in the concentrations of calcium, manganese, sodium, and carbonate-carbon were observed in the underclay samples (interval E; Table 37; Fig. 80). The depletion of calcium and carbonate-carbon content reflects the lack of calcium carbonate in the underclay; the observed depletion in manganese may also reflect the lack of carbonate, because manganese is a common minor constituent in calcite. The lack of sodium may reflect an ion-exchange process, as mentioned previously. Minor and trace elements are markedly enriched in the underclay samples relative to the other core samples. These include As, Ce, Cr, Cu, Ga, La, Li, Nd, Ni, Pb, Sc, Se, Sr, Ti, and Th (interval E; Table 37; Fig. 81). Several chalcophile elements are positively correlated with sulfidic sulfur concentrations in the underclay samples, including Fe, Cu, Ni, Pb, and As. This correlation indicates that enrichment is due to scavenging of these trace metals during precipitation of pyrite (as evidenced by diminished sulfur and metal concentrations in CS-160, Table 37 and Fig. 81). Other elements, such as Sr, may be enriched as a result of ion-exchange processes or may be contained in resistate minerals (Ti, Sc). Alternatively, contamination of the underclay samples by the drill bit or the drilling fluid during the coring may have occurred, but contamination is unlikely because the underclays are soft and unabrasive. Furthermore, the fact that elements not commonly associated with drill-bit materials such as As, Ga, La, Li, Nd, Sc, Se, and Th also were enriched in the underclay samples indicates that drill-bit contamination was probably not the source. Water used in the drilling process may have also been a source of contamination, but the water was obtained from a local municipality and was deemed safe for human consumption, therefore, the chances of the water containing significant quantities of any of these elements is negligible. Additionally, if the bit or drilling water was a source of

Figure 80. Concentration profiles for calcium, sodium, manganese, and carbonate-carbon in composite core samples collected at the study site, Coshocton County, Ohio.

Figure 81. Concentration profiles for arsenic, copper, and lead in composite core samples collected at the study site, Coshocton County, Ohio.

contamination, it would be expected that other samples would have also been contaminated, not only underclay. Many of the trace elements enriched in the underclay would be mobile under the acidic, oxidizing conditions likely to characterize ground water in the postmining spoil pile. These data indicate that the underclays could be a significant source of trace metals to acid mine drainage.

2.5.3.4.4 Acid-Base Accounting

Acid-base accounting (ABA) techniques were developed in the early 1970's by soil scientists whose objective was to estimate lime requirement for revegetating acid-producing overburden. ABA techniques have also been used by regulatory authorities and researchers to predict if mining would produce acidic or alkaline water quality (diPretoro and Rauch, 1988; Erickson and Hedin, 1988; Brady et al., 1990). The basic principal involves measurement of the concentration of total sulfur and total carbonate concentration, or neutralization potential in overburden rock and calculation of the calcium-carbonate deficiency (also known as the net-neutralization potential or NNP) for each sampling interval (Sobek et al., 1978). Net neutralization potentials are calculated by use of the following formula:

$$\text{NNP} = \text{NP} - \text{MPA}, \quad (1)$$

where MPA is the maximum potential acidity and NP is the neutralization potential derived from the measured carbonate concentration. MPA, NP, and NNP can be volume- or mass-weighted and are expressed in units of tons CaCO_3 /1,000 tons. Maximum potential acidity is calculated by multiplying the total (or pyritic) sulfur concentration by a stoichiometric constant based on one of two reactions that describe the oxidation of pyrite (Cravotta et al., 1990):

The stoichiometry of the reaction in (2) indicates that acidity produced by the oxidation of 2 moles (64 grams) of sulfur in FeS_2 (pyrite) is neutralized by 2 moles (200 grams) of CaCO_3 (calcium carbonate), or 1 gram FeS_2 per 3.125 grams CaCO_3 . If reaction 3 stoichiometry is followed, four moles of CaCO_3 (400 grams) are required to complete neutralization, yielding a ratio of 1 gram FeS_2 per 6.25 grams CaCO_3 . MPA values are therefore calculated by multiplying total sulfur concentrations expressed as weight percent S by a factor of either 31.25 or 62.5. Results of a study by Brady and others (1990) indicated that use of 62.5 results in more accurate predictions of postmining water quality than use of 31.25.

At the study site, three sets of data were available for evaluating the acid-producing potential of the overburden by acid-base accounting techniques: (1)

total sulfur and NP data from core holes drilled by the mining company that were in the ODNR site permits; (2) forms-of-sulfur and -carbon data from the composite core samples collected by the USGS; and (3) detailed total sulfur and NP data from the three USGS cores sampled by use of techniques outlined by Sobek and others (1978). The first two data sets were combined, and the NNP of each core hole was determined on a thickness (column)-weighted basis (thickness of layer divided by the total thickness of the overburden, multiplied by the NNP). Maximum potential acidity was computed by multiplying the total sulfur concentration by a factor of 62.5. The data required for this evaluation and results of the NNP calculations are summarized in Table 38.

The thickness-weighted average NP for the nine complete core-data sets from the mining company is 111 tons CaCO_3 / 1,000 tons (range = 2 to 71.25; Table 38). An average NNP for the entire study area was calculated from an area-weighted NNP for each of the core holes. Area weighting was done by use of the Thiessen polygon method (Linsley et al., 1982; Smith and Brady, 1990). Areas to be mined within each of the Thiessen polygons were determined by use of the digitized map of the mine site and GIS software (Fig. 82).

The calculation is as follows: area of individual polygon divided by total area of all polygons multiplied by thickness-weighted NNP. Results of this calculation are shown in Table 39. For these calculations, the entire thickness of the coal seam was assumed to contribute acidity to postmining spoils. Therefore, the total area-weighted NNP given in Table 39 represents the worst-case NNP for the site (-11.6 tons CaCO_3 /1,000 tons). Of course, most of the coal seam is removed during mining and only 10 to 20% of the original coal seam remains in the spoil after mining. If one assumes that 20% of the seam is left in the spoil, the MPA contributed by sulfur in the coal refuse is reduced by a factor of 5. When this adjustment is calculated for the study site, the total area-weighted NNP drops to -8.1 tons CaCO_3 /1,000 tons, a value indicating that postmining acid mine drainage is still possible.

The first two data sets used in the ABA analysis described above are deficient in several ways. The most important deficiency involves the methods used to sample the cores. In the case of the mining company, ODNR regulations required only that a single grab sample be collected for each lithologic unit; thus thin, sulfur-rich horizons in thick shale or sandstone beds may have been missed during sampling. Deficiencies are similar for the multiple grab sampling technique used to collect the composite core samples (CS-158, CS-159, and CS-160). Standard ABA sampling protocols outlined by Sobek et al. (1978) for acid-base accounting involve compositing the entire core over intervals of 3 to 5 feet or less. By compositing the entire core in 3 feet sampling intervals, the possibility of missing a thin, sulfur- (or carbonate-) rich layer is avoided. The Pennsylvania Department of Natural Resources requires that all ABA data submitted with mining permits must be obtained by use of this sampling technique (Keith Brady, Pennsylvania Department of Natural Resources, written commun., 1992). Another deficiency in the approach used above is that thickness or column-based weighting of the NNP's requires the assumption of a constant density for each rock

Table 38. Thickness-weighted acid-base account for Ohio Department of Natural Resources permits and U.S. Geological Survey composite core samples, Coshocton County, Ohio.

[Polygons are shown in Figure 83; lithologic and analytical data for core holes CS-158, CS-159, and CS-160 are from U.S. Geological Survey analyses of composite core samples; data for other numbered core holes are from lithologic logs obtained from Ohio Department of Natural Resources mining permits; NP, neutralization potential; MPA, maximum potential acidity; tons CaCO_3 /1,000 tons, tons calcium carbonate per thousand tons of rock; NNP, net neutralization potential]

Core hole number	Lithologic description	Thickness	NP	Total sulfur	MPA ^a	NNP	Thickness weighted NNP
		feet	tons CaCO ₃ / 1,000 tons	%	—tons CaCO ₃ /1,000 tons —		
Polygon A							
9751	Brown shale	8	7.1	0.01	0.63	6.48	0.62
	Brown sandstone	15	31.05	0.01	0.63	30.43	5.5
	Gray shale	20	19.3	0.06	3.75	15.55	3.75
	Gray sandstone	10	28.8	0.2	12.5	16.3	1.96
	Gray shale	18	24.8	0.41	25.63	-0.82	-0.18
	#6 coal	2	5.58	3.03	189.38	-184	-4.43
	Gray shale	10	20.3	0.31	19.38	0.93	0.11
	Total thickness	83	Total thickness - weighted NNP =				7.34
Polygon B							
9017	Brown shale	14.5	4	0.01	0.63	3.38	0.33
	Brown sandstone	12.3	2	0.01	0.63	1.38	0.11
	Limestone	4.9	71	0.01	0.63	70.38	2.31
	Brown sandstone	4.5	32.75	0.01	0.63	32.13	0.97
	Gray shale	35.8	12	0.51	31.88	-19.9	-4.78
	Brown sandstone	5	40	0.01	0.63	39.38	1.32
	Gray sandstone	9	12.75	0.05	3.13	9.63	0.58
	Gray shale	44	15	0.26	16.25	-1.25	-0.37
	#6 coal	2.2	2.5	1.58	98.75	-96.3	-1.42
	Gray shale	11.8	14	0.2	12.5	1.5	0.12
	#5 coal	1	0.25	7.22	451.25	-451	-3.03
	Gray shale	4	13.5	0.73	45.63	-32.1	-0.86
	Total thickness	149	Total thickness - weighted NNP =				-4.71
Polygon C							
9752	Brown shale	33	7.03	0.01	0.63	6.41	2.24
	Gray sandstone	6	17.33	0.07	4.38	12.96	0.83
	Gray shale	17	26.88	0.08	5	21.88	3.95
	Brown sandstone	10	11.5	0.02	1.25	10.25	0.11
	Gray sandstone	16	21.43	0.43	26.88	-5.45	-0.93
	#6 coal	2.2	3	2	125	-122	-2.81
	Gray shale	10	36.28	1.45	90.63	-54.4	-5.77
	Total thickness	94.2	Total thickness - weighted NNP =				-1.40

Table 38. continued.

Core hole number	Lithologic description	Thickness	NP	Total sulfur	MPA ^a	NNP	Thickness weighted NNP
		feet	tons CaCO ₃ / 1,000 tons	%	—tons CaCO ₃ /1,000 tons —		
Polygon D							
CS-158	Gray sandstone; reddish-brown sandstone	45	0.73	0.34	21.25	-20.5	-6.28
	Gray shale; brown silt	35	1.82	0.1	6.25	-4.43	-1.05
	Gray shale, gray sandstone	35	13.18	0.88	55	-41.8	-9.96
	#6 coal	4	0.00 ^b	3.43 ^c	214.38	-214	-5.83
	#6 underclay	3	0	4.97	310.63	-311	-6.34
	Gray to black shale; gray silt	25	4.55	0.07	4.38	0.18	0.03
	Total thickness	147.0	Total thickness - weighted NNP =				-29.44
Polygon E							
CS-159	Reddish-brown sandstone, gray shale, brown clay	60	4.91	1.09	68.13	-63.2	-19.45
	Reddish-brown sandstone, gray sandstone	40	1.59	0.24	15	-13.4	-2.75
	Reddish-brown sandstone, gray sandstone	35	0	0	0	0	0
	Gray to black shale, gray sandstone, reddish-brown sandstone	40	20.73	0.4	25	-4.27	-0.88
	#6 coal	3	0.00 ^b	3.43 ^c	214.38	-214	-3.3
	#6 underclay	2	0	2.1	131.25	-131	-1.35
	Gray to black shale, gray sandstone	15	8.91	0.13	8.13	0.79	0.06
	Total thickness	195	Total thickness - weighted NNP =				-27.66
Polygon F							
5345	Brown sandstone	34	8.68	0.01	0.63	8.06	3
	Brown shale	20	17.43	0.21	13.13	4.31	0.94
	Gray shale	2.5	7.33	2.34	146.25	-139	-3.8
	Gray shale	14.8	6.4	0.56	35	-28.6	-4.64
	Gray sandstone	9	18.05	0.05	3.13	14.93	1.47
	Gray shale	3	43	0.51	31.88	11.13	0.37
	#6 coal	1.9	29.48	2.66	166.25	-137	-2.88
	Gray shale	6.1	15.73	0.46	28.75	-13	-0.86
	Total thickness	91.3	Total thickness - weighted NNP =				-6.40

Table 38. continued.

Core hole number	Lithologic description	Thickness	NP	Total sulfur	MPA ^a	NNP	Thickness weighted NNP
		feet	tons CaCO ₃ / 1,000 tons	%	—tons CaCO ₃ /1,000 tons —		
	Brown shale	4	5.8	0.02	1.25	4.55	0.08
	Brown sandstone	5	7.5	0.02	1.25	6.25	0.14
	Brown shale	31	5.48	0.04	2.5	2.98	0.42
	Gray shale	19	7.3	0.11	6.88	0.43	0.04
	Brown sandstone	7	4.83	0.05	3.13	1.71	0.05
	Gray shale	8	9.1	0.21	13.13	-4.03	-0.14
	Limestone	7	62.5	0.37	23.13	39.38	1.24
	Gray sandstone	3	20.8	0.11	6.88	13.93	0.19
	Gray shale	16	7.25	0.08	5	2.25	0.16
	Limestone	3	71.25	0.01	0.63	70.63	0.95
	Gray sandstone	13	44	0.11	6.88	37.13	2.17
	Gray shale	39	22.35	0.28	17.5	4.85	0.85
	#6 coal	3.2	8.58	2.85	178.13	-170	-2.44
	Gray shale	25.2	5.93	0.99	61.88	-56	-6.34
	#5 coal	1.5	7.78	3.44	215	-207	-1.4
	Gray shale	8.3	4.83	0.37	23.13	-18.3	-0.68
	Total thickness	222.2	Total thickness - weighted NNP =				-4.14
Polygon H							
CS-160	Reddish-brown sandstone	30	0	0	0	0	0
	Gray silt, gray shale	35	6.18	0.22	13.75	-7.57	-2.44
	Gray shale	22	5.82	0.95	59.38	-53.6	-10.86
	#6 coal	3	0.00b	3.43c	214.38	-214	-5.93
	#6 underclay	3.5	0	0.35	21.88	-21.9	-0.71
	Gray shale, gray silt	15	4.09	0.08	5	-0.91	-0.13
	Total thickness	108.5	Total thickness - weighted NNP =				-20.06

^aMPA = 62.5 x sulfur (g/100g)^bNo data obtained, therefore, NP values were set to zero^cValue is median total sulfur concentration of 32 Middle Kittanning (number 6) coal seam samples collected at various locations in eastern Ohio, reported in the U.S. Geological Survey National Coal Resources Data System (R.B. Finkelman, U.S. Geological Survey, written commun., 1992).

Figure 82. Map showing Theissen polygons used for area-weighting of acid-base accounting data obtained from the mining company (sampled using the single grab technique) and three U.S. Geological Survey cores (sampled using the multiple grab technique).

Table 39. Area-weighted net neutralization potential values for study site based on available core data, Coshocton County, Ohio.
[NNP, net neutralization potential]

Core hole number	Polygon	Area	NNP	
			Thickness-weighted	Area-weighted
		acres	-----tons CaCO ₃ /1,000 tons -----	
9751	A	83.17	7.34	1.09
9017	B	91.87	-4.71	-0.77
9752	C	66.33	-1.40	-0.17
CS-158	D	96.34	-29.44	-5.05
CS-159	E	81.00	-27.66	-3.99
5345	F	11.20	-6.40	-0.13
14285	G	76.78	-4.14	-0.57
CS-160	H	55.30	-20.66	-2.03
Total area =		561.99	Total area - weighted NNP =	
				-11.61

type. Although the variation in density between different rock types is small (+ 20%), the density difference between coal and rock exceeds a factor of 2. Thus, mass-based calculations are a more accurate means of calculating NNP than volume-weighted methods (thickness or column based). A more serious problem associated with volume-weighted calculations is that they are usually done on a unit basis. In other words, only the core hole itself is evaluated and actual spoil volumes (or masses) associated with each lithologic unit (which may vary considerably in areas of uneven topography) are not considered. Depending on the location of the dominant acid- or alkaline-producing lithologic units, significant errors could be made in estimating the spoil NNP. Deficiencies in volume-weighted ABA analysis and suggested techniques for overcoming these deficiencies are discussed by Smith and Brady (1990).

Given the above deficiencies, additional samples were collected from the three USGS cores, by use of techniques outlined by Sobek et al. (1978). Percent sulfur and NP data were obtained for each core (Table 40). Areas of influence for each of the three USGS core holes were evaluated by the Thiessen polygon technique (Fig. 83), and volumes associated with individual sampling intervals in each core hole were calculated by use of GIS software (Table 40). For the volume calculations, the strata were assumed to be horizontal and of equal thickness throughout the Thiessen polygon. These are obvious simplifications, and they introduce some uncertainty into the calculations; however, the same assumptions are included in the area-weighted estimates presented in Table 39. Mass-weighted MPA (using the 62.5 multiplication factor), NP, and NNP's were calculated for each interval by use of a computer spreadsheet technique by Smith and Brady (1990). Total mass-weighted and area-weighted NNP's for each core, evaluated by assuming that 100 or 20% of the coal-derived sulfur is incorporated into the resulting mine spoil, are given in Table 41.

Mass-weighted NNP's were calculated for the worst-case scenario that 100% of the coal-derived sulfur would be incorporated into the mine spoil. These NNP's are negative, indicating that alkaline amendments and (or) selective spoil handling may be required to prevent acid conditions from developing. The NNP's calculated for 20% incorporation of coal-derived sulfur into mine spoil are, with the exception of CS-159, near zero, indicating little if any alkaline amendments would be needed to prevent acid conditions from developing. Area-weighting of the mass-weighted NNP's yields overall NNP's for the study site of -18.8 tons CaCO_3 /1,000 tons for the worst-case scenario (100% coal-derived sulfur contribution) and -5.6 tons CaCO_3 /1,000 tons for the more realistic scenario (20% of coal-derived sulfur in the mine spoil). The negative values indicate that generation of acid mine drainage at the site is possible. Among all the techniques used, and despite the conservative assumptions used (MPA multiplication factor = 62.5; inclusion of coal-derived sulfur), none of the estimated NNP's were less than -35 tons CaCO_3 /1,000 tons. If less conservative assumptions are used (such as a MPA multiplication factor of 31.2; no coal-derived sulfur in spoil materials), the area-weighted NNP's for the site would be slightly positive (6 to 14 tons CaCO_3 /1,000 tons).

Table 40. "Fizz," total sulfur, and neutralization potential data for rock samples collected from U.S. Geological Survey core holes CS-158, CS-159, and CS-160, Coshocton County, Ohio
[tons CaCO₃/1,000 tons, tons calcium carbonate per thousand tons rock; ND, not determined; ox., oxidized; calc., calcareous]

Thickness of layer	Rock Type	"Fizz"	Total sulfur	Neutralization potential	Volume of layer within polygon
feet			wgt %	tons CaCO ₃ /1,000 tons	cubic feet
Core CS-158					
12	Soil	None	ND	ND	ND
3	Brown sandstone	None	0	1.89	3380000
3	Brown sandstone	None	0	0.97	3617000
3	Brown sandstone	None	0.01	0.86	3909000
3	Brown sandstone	None	0	1.08	4127000
3	Brown sandstone	None	0.01	-1.6	4448000
3	Brown sandstone	None	0.01	1.42	4683000
3	Brown sandstone	None	0.02	1.81	5068000
2	Brown sandstone	None	0	2.22	3575000
3	Gray siltstone	None	0.31	2.14	5594000
3	Gray siltstone	None	0.26	7.51	5983000
4	Gray siltstone	Moderate	0.29	54.36	8390000
2.5	Gray clay	Slight	2.62	28.22	5659000
3.5	Gray shale	Moderate	0.47	49.8	8211100
2.5	Gray siltstone	Moderate	0.28	42.24	6721000
1.8	Gray shale	None	0.66	8.09	5096000
1.3	Gray shale	Moderate	0.04	43.28	3876000
3.4	Gray shale	None	0.22	8.73	10480000
3	Gray shale	Slight	0.15	18.08	9647000
3	Gray shale	None	0.06	4.42	10205000
3	Gray shale	None	0.01	5.45	10539000
3	Gray shale	None	0.03	9.01	10981000
3	Gray shale	None	0.03	8.9	11790000
1	Ox. gray shale	Slight	0	11.19	3932000
3	Gray shale	None	0.02	9.45	12324000
3	Gray shale	None	0.03	7.76	12580000
3	Gray shale	weak	0.01	17.71	13106000
3	Gray shale	weak	0.11	20.76	13343000
3	Gray shale	None	0.1	10.9	13867000
3	Gray shale	None	0.14	10.65	14218000
3	Gray-brown shale	None	0.09	10.87	14645000
3	Gray shale	None	0.16	11.17	15166000
3	Gray shale	Slight	0.18	15.79	15349000
1.8	Gray shale	Moderate	0.09	78.82	9442000
1.1	Calc. gray sandstone	Strong	0.03	0	5898000

Table 40. continued.

Thickness of layer	Rock Type	"Fizz"	Total sulfur	Neutralization potential	Volume of layer within polygon
feet			wgt %	tons CaCO ₃ /1,000 tons	cubic feet
3	Silty sandstone	Moderate	0.14	95.53	17850000
2	Silty sandstone	Moderate	0.19	31.94	12063000
3	Gray shale	None	0.61	11.06	18143000
3	Gray shale	None	0.52	11.26	18521000
3	Gray shale	None	0.51	11.51	18499000
3	Gray shale	None	0.62	11.42	18885000
3	Gray shale	None	0.72	14.93	19070000
4	#6 coal	None	8.63	NDb	26010000
Core CS-159					
8	Soil	None	NDb	NDb	488000
3	Brown sandstone	None	0	3.39	401000
3	Brown sandstone	None	0	4.92	492000
3	Brown sandstone	None	0	1.75	595000
3	Brown sandstone	None	0.02	1.22	689000
3	Brown sandstone	None	0	1.11	805000
3	Brown sandstone	None	0	2.11	931000
2	Brown sandstone	None	0	1.36	676000
2.5	Brown sandstone	None	0	0.53	903000
3	Brown sandstone	None	0	1.78	1189000
2.1	Brown clay	None	0	6.17	893000
3.4	Gray clay	None	0.19	6.64	1555000
3	Gray shale	None	0.15	8.06	1520000
3	Gray shale	None	0.23	13.82	2079000
3	Black shale	Slight	0.56	24.86	2520000
3	Black shale	None	1.04	15.73	2724000
1.6	Black shale	None	0.95	13.09	1562000
2.4	Calc. black shale	Moderate	0.49	66.07	2454000
3	Black limestone	Strong	0.16	202.3	3309000
1.7	Black limestone	Strong	0.05	144.6	1939000
3.3	Calc. black shale	Strong	0.99	53.24	4065000
1.6	Calc. black shale	None	1.24	5.78	2121000
0.6	Calc. black shale	None	0.49	11.17	795000
2.8	Calc. black shale	Moderate	2.66	37.7	3876000
3	Calc. black shale	Moderate	1.89	83.68	4419000
3	Calc. black shale	Strong	2.13	115	4602000
3.6	Calc. black shale	Strong	2.17	106.2	5873000
1.8	Gray limestone	Strong	0.84	992.7	3064000
1.4	Calc. sandstone	Strong	0	635.3	2456000
1	Gray shale	None	2.63	0.14	1744000
2.2	Brown sandstone	Strong	1.68	553.7	4025000
2	Brown sandstone	None	0	5.28	3807000

Table 40. continued.

Thickness of layer	Rock Type	"Fizz"	Total sulfur	Neutralization potential	Volume of layer within polygon
feet			wgt %	tons CaCO ₃ /1,000 tons	cubic feet
2.8	Gray shale	None	0.55	9.84	5840000
1.1	Gray sandstone	None	0.26	3.86	2321000
1.4	Gray sandstone	None	0.21	7.76	3070000
1.7	Brown sandstone	Strong	0	529.2	3782000
1	Gray shale	None	0.16	11.12	2216000
2.8	Brown sandstone	None	0	2.5	6497000
3	Brown sandstone	None	0	2.22	7142000
3	Brown sandstone	None	0	1.7	7462000
3	Brown sandstone	None	0	2.67	7599000
3	Brown sandstone	None	0	3.06	7814000
3	Brown sandstone	None	0	3.11	8119000
3	Brown sandstone	None	0	3.34	8246000
3	Brown sandstone	None	0	3.7	8554000
3	Brown sandstone	None	0	2.89	8687000
2.4	Brown sandstone	None	0	4.7	7115000
1.7	Brown sandstone; clay	None	0	6.45	5148000
2.9	Brown sandstone	None	0	4.14	8961000
3	Brown sandstone	None	0	5	9358000
3	Brown sandstone	None	0	2.89	9554000
3	Brown sandstone	None	0	2.92	9854000
3	Brown sandstone	None	0	4.53	9947000
3	Brown sandstone	Moderate	0.14	8.73	10257000
3	Brown sandstone	None	0	4.23	10339000
3	Brown sandstone	None	0	1.81	10644000
3	Brown sandstone	None	0	2.5	10715000
3	Brown sandstone	None	0	1.25	11018000
2	Brown sandstone	None	0	1.58	7406000
2.2	Brown sandstone	None	0	0.95	8244000
2.8	Gray sandstone	None	1.42	1.42	10614000
2.2	Gray sandstone	None	0.47	2.36	8473000
1.4	Brown sandstone; clay	None	0.47	3.5	5486000
2.4	Gray sandstone	None	0.87	3.34	9402000
2.9	Gray sandstone	None	0.62	7.73	11477000
2.3	Gray shale	None	0.82	11.12	9240000
3.4	Gray sandstone	None	0.26	5.34	13909000
1.9	Gray sandstone	None	0.06	4.48	7862000
2.3	Gray sandstone; minor coal	None	0.54	5.45	9620000
3.2	Gray shale	None	0.59	12.4	13483000
3	Gray shale	None	0.95	13.12	12949000
3	Gray shale	None	0.48	11.95	12962000

Table 40. continued.

Thickness of layer	Rock Type	"Fizz"	Total sulfur	Neutralization potential	Volume of layer within polygon
feet			wgt %	tons CaCO ₃ /1,000 tons	cubic feet
2	Gray shale	None	1.79	6.84	23910000
3	#6 coal	None	5	0	8453000
Core CS-160					
10	Soil	None	ND	ND	14944000
0.8	Clay	None	0	5.84	504000
3.2	Brown sandstone	None	3.26	1.83	2141000
4	Brown sandstone	None	0	2.76	2934000
3	Brown sandstone	None	0	2.75	2398000
3	Brown sandstone	None	0.33	3.17	2534000
3	Brown sandstone	None	0	2.84	2736000
3	Brown sandstone	None	0	2.61	2872000
3	Brown sandstone	None	0	2.2	3052000
3	Brown sandstone	None	0	2.5	3165000
4	Brown sandstone	None	0.88	3.67	4467000
3	Brown sandstone	None	0.67	1.64	3496000
3	Brown-gray shale	None	0.1	7.76	3675000
3	Gray shale	None	0.07	7.26	3771000
3	Gray shale	None	0.2	8.26	3947000
3	Gray shale	None	0.1	6.81	4039000
2	Brown sandstone	None	0	6.2	2782000
3	Brown sandstone	None	0	16.87	4311000
3	Brown sandstone	Slight	0	63.75	4401000
3	Brown sandstone	Slight	0.83	17.94	4537000
3	Brown sandstone	Moderate	0.02	32.46	4785000
2	Brown sandstone	Moderate	0.31	43.41	3251000
0.2	Coal	None	2.77	5.53	339000
2.8	Gray shale	None	0.35	12.95	4967000
3	Gray shale	None	0.59	12.4	5165000
3	Gray shale	None	0.29	15.84	5251000
3	Gray shale	None	0.33	16.54	5453000
3	Gray shale	None	0.42	11.65	5538000
3	Gray shale	None	0.16	10.65	5730000
3	Gray shale	None	0.24	12.98	5963000
3	Gray shale	None	0.18	11.9	6066000
3	Gray shale	None	0.14	13.34	6304000
3	Gray shale	None	0.01	9.65	6404000
3	Gray shale	None	0.02	12.4	6646000
3	Gray shale	None	0.2	9.31	6764000
3	Gray shale	None	0.17	10.87	7084000
3	Gray shale	None	1.47	5.14	7221000
1.4	#6 coal	None	19.4	-4.69	3486000

Table 40. continued.

Thickness of layer	Rock Type	"Fizz"	Total sulfur	Neutralization potential	Volume of layer within polygon
feet			wgt %	tons CaCO ₃ /1,000 tons	cubic feet
2.7	Gray shale	None	0.09	4.03	7487000
2.3	Calc. brown sandstone	Strong	0	119.5	6468000
3	Gray shale	None	0	6.42	8628000
2	Gray shale	None	0	11.01	5929000
1	Gray limestone with pyrite	Strong	1.47	815.9	3053000
1.7	#5 coal	None	3.48	214.02	5039000

Table 41.

Mass-weighted maximum potential acidity, neutralization potential, and net neutralization values, and NP/MPA values for U.S. Geological Survey cores CS-158, CS-159, and CS-160, Coshocton County, Ohio.

[MPA, maximum potential acidity; NP, neutralization potential; NNP, net neutralization potential; tons CaCO₃, tons calcium carbonate per thousand tons rock]

Core hole	Area	Mass-weighted			Area-weighted, mass-weighted NNP	NP/MPA
		MPA	NP	NNP		
	acres	—————tons CaCO ₃ /1,000 tons —————				
100% of coal-derived sulfur incorporated into in mine spoil						
CS-158 (100) ^a	145.5	59.6	40.5	-15.1	6.4	0.75
CS-159 (100) ^a	117.4	61.3	29.8	-31.5	10.7	0.49
CS-160 (100) ^a	81.7	35.0	27.7	-7.3	1.7	0.78
Area-weighted, mass-weighted NNP value for all three cores = -18.8						
20% of coal-derived sulfur incorporated into in mine spoil						
CS-158 (20) ^a	145.5	35.8	40.5	4.2	3.7	1.13
CS-159 (20) ^a	117.4	57.4	30.0	-27.4	-9.4	0.52
CS-160 (20) ^a	81.7	25.6	26.2	0.6	0.1	1.02
Area-weighted, mass-weighted NNP value for all three cores = -5.6						

^a Number in parentheses denotes percentage of coal-derived sulfur included in MPA calculation.

Figure 83. Map showing Theissen polygons used for area-weighting of acid-base accounting data obtained from three U.S. Geological Survey cores sampled using the comprehensive compositing technique.

Because so much in the acid-base accounting calculations must be assumed, relating absolute NNP's to postmining water quality has proven difficult. Statistically significant correlations between volume-weighted NNP's and post mining water quality have not been demonstrated (Erickson and Hedin, 1988). Instead, only general guidelines have been suggested by the following investigators on the basis of a factor of 31.25 to compute MPA. Sobek et al. (1978) defined as potentially toxic those materials whose NNP's are less than -5 tons CaCO_3 /1,000 tons. Skousen et al. (1987) suggested that strata containing NNP's of less than 5 tons CaCO_3 /1,000 tons produce acid drainage, whereas strata with NNP's greater than 20 tons CaCO_3 /1,000 tons usually produce alkaline drainage. Studies by diPretoro and Rauch (1988) and Erickson and Hedin (1988) indicate that all mines whose NNP's were greater than 34 tons CaCO_3 /1,000 tons produce alkaline drainage. Acid mine drainage was produced by 41% of the mines whose volume-weighted NNP's in the overburden were between 7 to 27 tons CaCO_3 /1,000 tons, whereas acid mine drainage was produced by 89% mines whose NNP's in overburden were less than 7 tons CaCO_3 /1,000 tons. Work by Brady and Cravotta (1992) reevaluated the data of above-mentioned investigators and found that use of the 62.5 factor to compute MPA's improved prediction.

The general trend in NNP's based on available core-sample data in the study area indicates that acid-mine drainage could be produced. However, by varying the MPA multiplication factor and the percentage of coal-derived sulfur considered in the ABA analysis, slightly positive NNP's can be calculated. Such values are in the range where strong correlations between NNP's and postmining water quality do not exist. Therefore, it is important to look at water-quality data obtained from seeps, springs, and streams exiting reclaimed mines adjacent to the current active-mine site to verify acid-base accounting predictions. Data available for surface water exiting reclaimed areas southwest of the study site show that the adjacent reclaimed sites are producing alkaline water; however, sulfate concentrations are elevated in some of these samples, an indication that acid-generating reactions are occurring in the mine spoil (Table 33). The mass-weighted NNP's calculated from MPA and NP data obtained for USGS comprehensive core sampling are qualitatively similar to the NNP's calculated by the mine company and for USGS composite core samples. Thus, at least at this study site, the USGS comprehensive sampling strategy did not provide a more definitive determination of the acid-producing potential of the mine spoil than the grab-sampling techniques discussed previously. The USGS comprehensive sampling technique did, however, provide a more accurate determination of where potentially acid-producing strata are located in the core column. Figures 84 through 86 show the logarithm of the NNP as a function of depth and lithology and clearly distinguish between alkaline- and acid-producing strata. Specifically, the primary zones of carbonate deficiency are associated with shale beds most commonly located immediately above or below the coal seams. Sandstones and zones of calcareous shales, such as those between 1,040 and 975 feet in core CS-158, generally contain excess carbonate. Hence, a mining plan that includes selective handling of the carbonate-deficient shale zones would best ensure against the potential for acid-mine-drainage generation at this site.

Figure 84. Profile of the logarithm of the net neutralization potential of composite core samples for U.S. Geological Survey core CS-158, Coshocton County, Ohio.

Figure 85. Profile of the logarithm of the net neutralization potential of composite core samples for U.S. Geological Survey core CS-159, Coshocton County, Ohio.

Figure 86. Profile of the logarithm of the net neutralization potential of composite core samples for U.S. Geological Survey core CS-160, Coshocton County, Ohio.

2.5.3.5 Conceptual Model of Ground-water Flow

Preliminary work was done to conceptualize the ground-water flow system beneath the mine site. A cross-sectional, two-dimensional, steady-state model was necessary to help in the calculation of advective transport rates of ground water before mining and to compare these with post mining flow within spoils that may contain FGD-derived leachate. As with any numerical simulation of a hydrologic environment, many simplifying assumptions must be made to reduce the number of variables in the model. These assumptions included details about model boundary conditions, sources and sinks of water, and water-transmitting properties and extent of aquifers and aquitards.

Examples of previously published ground-water flow models that specifically deal with coal hydrology include those by Helgesen et al. (1982) and Saad and Cravotta (1991). These reports were used as a basis for the simplifying assumptions given below. Core data collected as part of this study also provided useful information for the conceptual model. The extent and thickness of geologic units in the study area were estimated from USGS core logs and are shown in Figure 87. Many of the units were too thin to be represented in the model and tended to vary in thickness over short horizontal distances. Therefore, an average lithology was assumed to exist across a defined interval. The triaxial chamber tests also yielded important data concerning water-transmitting properties of the underclays for inclusion in the model. Hydraulic conductivities for the lithologies and estimated values for use in the model are listed in Table 42.

The source of water in the conceptual model is solely precipitation. In Coshocton County, precipitation averages from 38 to 40 inches (960 - 1,020 millimeters) per year. Before mining, most of this water would leave the study site by way of surface runoff, evaporation, and transpiration by plants; only about 7 inches of water per year would recharge the uppermost saturated zone (Helgesen et al., 1982). After mining, sediment ponds decrease surface runoff and promote infiltration, and loss of topsoil creates poor conditions for plant growth. The model geologic section (A-A; Fig. 80) was placed perpendicular to the trend of a ridge. This placement allowed for the assumption that most of the ground-water flow would be towards the flanks of the hillside (to the east and west) and flow in or out of the plane created by the cross section would be negligible.

Ground-water flow within coal sequences is dominated by fractures, bedding planes, coal layers, and permeable sandstones, whereas underclay layers prohibit the downward flow of water under the influence of gravity. The upper limit of the ground-water system is the water table. However, a series of perched aquifers may overlie the principal saturated zone (Cunningham and Jones, 1990). Downward flow of water due to gravity is retarded by the presence of the poorly permeable shale and underclay. Ground water will therefore flow preferentially horizontally or downdip along the plane of the impermeable boundary. If the ground-water-flow path intersects a fracture, bedding plane, a relatively permeable sandstone, or an unplugged core hole,

Table 42. Estimates of hydraulic conductivity of lithologies in the conceptual ground-water-flow model in the study area.

[All hydraulic conductivities are in feet per day]

Lithology	Published hydraulic conductivities ^a		Hydraulic conductivities used in the model		
	Horizontal	Vertical	Horizontal	Vertical	Data source
Sandstone	$10^{-3} - 10^{-2}$	$10^{-6} - 10^{-1}$	10^{-2}	10^{-4}	Estimate from core data ^b
Shale	$10^{-8} - 10^{-2}$	$10^{-8} - 10^{-4}$	10^{-4}	10^{-6}	Estimate from core data
Coal	$10^{-1} - 10^2$	$10^{-1} - 10^2$	10^0	10^0	Estimate from core data
Underclay	$10^{-8} - 10^{-3}$	$10^{-8} - 10^{-5}$	10^{-5}	10^{-5}	Triaxial chamber tests, table 5
Alluvium	$10^{-1} - 10^2$	$10^{-3} - 10^0$	10^0	10^{-1}	Estimate from core data
Spoil	$10^{-2} - 10^2$	$10^{-4} - 10^0$	10^0	10^{-1}	Estimate from core data

^a Data sources: Anderson and Woessner, 1992; Saad and Cravotta, 1991; and Heath, 1983.

^b Sandstone hydraulic conductivity estimated for "silty sandstone" shown in figure 11 is lower than that for typical sandstone.

Figure 87. Schematic geologic section along A-A' illustrating the main principles of the conceptual model of ground-water flow in the study area (only strata considered in conceptual model are labeled).

the flow would take the path of least resistance and follow these features, eventually to exit the ground-water system through a surface-water body, spring, or seep, or perhaps a deeper aquifer. The deeper aquifer may have a regional flow component (far beneath the surface-mined interval) and therefore receives inputs from a much larger area. Inclusion of the deeper aquifer in the model was beyond the scope of the study and the current understanding of the ground-water system. If the results of the final model had revealed significant inputs of water from the spoils through the underlying aquifers and to the deeper aquifer, the model would have been revised to include the deeper (regional) aquifer. However, no data-collection strategies were planned for the deep aquifer.

The water-transmitting properties of the aquifers in the study area (sandstones, sandy shales, coal seams, and fracture zones) were to be further evaluated after the installation of the monitoring wells. Aquifer tests in each interval would have supplied information on how water levels within an aquifer or fracture zone respond to a stress (either an addition or withdrawal of water), thus allowing for the calculation of water-transmitting properties. Without aquifer test data, published estimates of water-transmitting properties of aquifers from previous studies could be used as a starting point to develop a conceptual ground-water-flow model (Table 42). The preceding information was used to develop a conceptual model only. Because ground-water levels and flow rates of springs and streams were not measured for this study, calibration of a numerical model to observed conditions would have been impossible. The information collected was useful, however, in designing an approach strategy for similar future investigations involved with the use of FGD by-products in reclamation of surface coal mines.

2.4 Economic Analyses

2.4.1 Overview

Under Title IV of the 1990 Clean Air Act, electric power generating plants will be required to reduce SO₂ and NO_x emissions by about 40 percent no later than the year 2000. These emission standards are most significant for power plants which burn coal, particularly high sulfur coal, as an energy source. In order to achieve compliance with these mandates, power plants will have to reduce demand for electricity thus reducing the quantity of coal burned, use fuels lower in sulfur, purchase emission allowances, retrofit existing power plants with clean air technology, or a combination of the foregoing. Currently, the only clean air technology available to existing power plants is flue gas desulfurization (FGD) technology. EPA has estimated that this technology can reduce SO₂ by as much as 95%. However, production of FGD by-products requires a strategy to dispose of or recycle the material produced. Based upon current coal consumption estimates, Ohio could potentially produce nearly 4 million tons of dry FGD by-product annually from six Ohio power plants. The objectives of this research were to estimate a least cost disposal model for the movement of this by-product to various geographic locations throughout Ohio and for use as a soil amendment for agricultural land, coal surface mine reclamation, and landfilling. In doing so, disposal costs and quantities were derived as well as shadow prices for each county (demand node) identified.

2.4.2 Analysis of FGD By-Product Disposal Alternatives

The objectives of this task were (1) to develop a model to identify the least cost disposal methods of FGD by-product among the three stated alternative end uses from the producer or power plants perspective, (2) estimate the quantity of by-product used in each alternative, and (3) estimate the shadow price associated with each alternative end use. This particular component of the research does not address the social amenities/disamenities associated with FGD by-product disposal. For example, deterioration of roads and bridges from increased traffic, property value gain/loss from landfill activities, strip mine reclamation, abandoned mine land reclamation, increased/decreased ground- and/or surface-water quality from surface mine reclamation and abandoned mine land reclamation, or landfilling activities, etc. However, work is in progress to quantify these amenities/disamenities. Once quantified, the least cost disposal model can be re-estimated with the appropriate increase/decrease in cost associated with the given end use alternative. That is, the model can be modified to consider the social gain/loss associated with each end use alternative.

Not all power plants are expected to convert to dry FGD technology. Of the 23 power plants identified in Ohio, potential convertors to dry FGD technology by the year 2000 were estimated. In this analysis, a least cost transportation model was developed depicting the least cost distribution of dry FGD by-product material from these potential electric power generating sites to numerous end use alternatives. The scenario assumes six power plants will retrofit with this dry injection technology. This analysis is

only preliminary since options regarding SO₂ emissions reduction at the plants are still being weighed.

Alternative Disposal Options. FGD by-product has many characteristics similar to agricultural lime, and it is expected that dry FGD by-product materials could be a close substitute for agricultural lime. For this reason it is appropriate to use agricultural lime sales as a proxy for estimating the quantity of FGD by-product demanded in each Ohio county. Maximum demand for the FGD by-product in agriculture reflects the usage of agricultural lime. Ohio Agricultural Statistics Service estimates of annual sales of agricultural lime by Ohio county (1985-90) were used to estimate the potential demand. Since FGD by product has 60 percent the neutralizing potential of agricultural lime, agricultural lime sales were adjusted to estimate FGD by-product demand. That is, FGD by-product has a lower total neutralizing potential (TNP), thus higher application rates are necessary to achieve the same neutralizing results as agricultural lime. The estimated quantity of FGD by-product demanded reflects these adjustments in TNP.

Transportation of the FGD by-product is expected to be similar to that of agricultural lime. Only slight modification of existing equipment is necessary to transport the by-product from the source (power plant) to the destination (farm, coal mine or landfill), and to apply the by-product to agricultural land. Under these assumptions, FGD by-product is expected to be back hauled from the power plant to various locations throughout the state by trucks. Once the by-product has reached the farm it is expected that conventional lime spreaders will apply this by-product to agricultural lands.

Coal surface mine operations are required to reclaim lands which have been mined. During the reclamation phase lime is often used to return the mined spoils back to a pH level conducive to plant growth. It is expected that FGD by-product materials can also be used for this same purpose. The estimated quantity of FGD by-product material demanded to meet surface mine reclamation work was derived from data reporting tons of surface mine coal sold in each Ohio coal mining county. Using these data, the number of acres displaced by surface coal mining in a given year was estimated. Based on conservative FGD by-product application rates of 20 tons per acre, an estimate for the quantity of FGD by-product demanded was determined.

The final end use alternative identified is that of landfilling. It is expected that FGD by-product will be landfilled in the event that the available quantity of the product is larger than its economical use in agricultural and/or coal surface mine reclamation. It is further assumed that landfilling is not constrained in the quantity of the by-product that can be accepted, and that landfills are in close proximity to the power plants or sources of FGD by-products.

Model Development. The least cost transportation model used for this analysis is based upon six source nodes (power plants) associated with the production of dry FGD by-product materials and its use as a substitute for agricultural lime in 88 Ohio counties, as a soil amendment at 21 Ohio coal surface mine reclamation sites, and as a

disposed material at six landfill sites located in proximity of the power plant.

To formulate this model mathematically, the following terms are defined.

a_i = the number of tons of FGD by-product material available at the power plant or source i , $i=1,2,\dots,m$;

b_j = maximum number of tons of by-product required at each destination or alternative use (e.g. county for agricultural lime, reclamation site, or landfill), $j = 1,2,\dots,n$;

c_{ij} = unit transportation and application cost from each source i to each destination j , ($i = 1,2,\dots,m$; $j = 1,2,\dots,n$).

The problem then becomes determining the amount of FGD by-product material shipped to each of these alternative end uses, given that the cost of distribution and application of the by-product is known or can be estimated. Thus, the decision variable, x_{ij} , equals the number of tons of by-product material shipped from each source i to each destination j annually given some cost per unit shipped.

The transportation model estimated is:

$$\text{Minimize cost} = \sum \sum c_{ij}x_{ij} \quad (1)$$

$$\sum x_{ij} = a_i \quad (i = 1,2,\dots,m); \quad (2)$$

$$\sum x_{ij} \leq b_j \quad (j = 1,2,\dots,n); \quad (3)$$

$$x_{ij} \geq 0 \quad (i = 1,2,\dots,m; j = 1,2,\dots,n). \quad (4)$$

Equation (1) represents the minimization of total distribution costs, assuming a linear cost structure for shipping, processing, and application of the dry FGD by-product material. Equation (2) shows that the quantity of by-product shipped from each source i to each alternative end use destination j must be equal to the quantity of by-product material available at source a_i . Equation (3) states that the quantity of by-product shipped from each source i to each destination j , must be less than or equal to the maximum quantity of by-product demanded at that destination. Finally, equation (4) indicates that the quantity of by-product shipped from each source i to each destination j can not be less than zero tons.

Supply, Demand, and Cost Estimates. All estimates pertaining to the quantity of dry FGD by-product demanded at various demand nodes for the two end use alternatives (agricultural and surface coal mine reclamation uses) have been adjusted for a 25 percent rate of adoption. It is expected that not all individuals using agricultural lime or reclaiming surface coal mines will completely adopt this new technology. Thus, the

model uses a conservative or lower rate of adoption. However, it is important to note that the model can be re-run at various adoption levels.

Linear distances from the power plant or source of dry FGD by-product to the center of each county were estimated. Once these distances were determined, costs associated with moving the specified distances were estimated. In the case of agricultural land application, cost estimates were derived from the agricultural lime industry. It is expected that the dry FGD by-product will be transported in much the same manner as current agricultural lime. Thus, an estimate of \$0.10 per ton per mile was used. In addition to moving the product from the source to the destination, an application expense is incurred. Again, the application of the dry FGD by-product is expected to be similar to agricultural lime which has an estimated application charge of \$3.50 per ton. This expense includes the spreader and mechanism to load the product from a pile onto the spreader. Therefore, transportation costs were calculated at \$0.10 per ton per mile, then an additional \$3.50 per ton was added to each for expected application costs.

Transportation of the dry FGD by-product to surface coal mine reclamation sites is also expected to cost \$0.10 per ton per mile. Application of the dry FGD by-product is expected to be at significantly higher levels, at least 20 tons per acre, than application rates associated with agricultural use. Thus, different equipment is expected to be used in distributing the by-product (e.g. a bulldozer or equivalent type of reclamation machinery), but application costs are assumed not to vary significantly and were estimated at \$3.50 per ton.

Cost estimates for landfilling were obtained from interviews with representatives of electric utilities. All landfilling activities are regulated by EPA. Because dry FGD by-product material has a relatively high pH, EPA may require that more stringent landfilling requirements be met. Electric power plants have estimated that it would cost about \$27.50 per ton of material to landfill dry FGD waste and meet current EPA guidelines.

Results. A summary of shipments by source and by disposal option are shown in Table 43. Of the total quantity of dry FGD by-product produced, little is used in agriculture (nine percent) or surface coal mine reclamation (one percent). This suggests that the amount of FGD by-product greatly exceeds these anticipated demands and that most of the by-product will be landfilled.

Shadow prices for dry FGD by-product are calculated as the difference between landfilling (non-binding constraint) and agricultural land application or coal surface mine reclamation options (both binding constraints). It would be expected that as the distance from the power plant increases, the cost to move this by-product also increases, therefore the imputed value or shadow price for binding end use options farther from the source or power plant would be lower. That is, the difference between landfilling and shipping dry FGD by-product material greater distances would be smaller. Thus, counties located farther from dry FGD by-product sources would have

Table 43. Summary of Shipment Estimates.

Option	Power Plant						Total Shipped
	Mountaineer	Edgewater	Muskingum	Niles	Sammis	Burger	
	Tons						
Ag.	120,393	70,000	60,490	46,256	37,823	39,314	374,276
Reclaim	5,412	0	10,063	409	15,933	10,690	42,507
Landfill	757,524	0	869,447	0	1,679,544	386,656	3,693,171
Total	883,330	70,000	940,000	46,665	1,733,300	436,660	4,109,955
	Percent of Total						
Ag.	13.6	100	6.4	99.1	2.2	9.0	9.0
Reclaim	0.6	0.0	1.1	0.9	0.9	2.5	1.0
Landfill	85.8	0.0	92.5	0.0	96.9	88.5	90.0
Total	100	100	100	100	100	100	100

lower shadow prices or lower cost savings to the utility companies than would land application sites closer in proximity to the power plant.

These shadow prices can also be interpreted as the amount the power plant would be willing to pay for the disposal of an additional ton of by-product in each end use alternative. For example, the shadow price for FGD by-product delivered to Williams county (northwest corner of the state) is \$3.29 per ton, while the shadow price for counties near power plants (east and southeast regions of Ohio) are over \$20 per ton. More detail on these methods and results is provided in Forster and Rausch (1993).

Conclusion. This analysis uses one scenario for future dry FGD by-product distribution and uses. Others are plausible and easily analyzed by this model. The linear transportation model estimated suggests that the least cost disposal of dry FGD by-product would cost \$107 million to dispose of 4.1 million tons of dry FGD by-product material annually. This is equivalent to average cost of \$26.10 per ton of dry FGD produced. In addition to landfilling, two alternative end uses currently identified are dry FGD by-product use as soil amendments on agricultural land and coal surface mine reclamation. Of these three options, landfilling accounts for about 90 percent of the total by-product produced, while agricultural and surface coal mine reclamation account

for about 9 and 1 percent, respectively. Therefore, on average land application of dry FGD by-product represents a savings of \$1.40 (\$27.50 (landfilling) - \$26.10 (average cost of disposal per ton). However, at the margin (one additional ton) land application represents a cost savings of about \$14.81 per ton. That is, electric power plants can reduce total cost of dry FGD by-product disposal by \$14.81 per ton through land application rather than landfilling.

Agricultural use of dry FGD by-product is concentrated in the western two-thirds of Ohio, while surface coal mine reclamation use is important in the eastern one-third of the state. Yet, of the alternative disposal options identified, the vast majority of dry FGD by-product is landfilled. Given these end use alternatives, more FGD by-product will be buried in landfills than will be used in other alternatives combined. However, electric utilities have an enormous economic incentive to supply dry FGD by-product for land application uses. They could pay farmers substantial amounts and save landfilling costs.

2.4.3. Agricultural Lime Supply and Demand Relationships

The objective of this task was to estimate supply and demand relationships that can best explain the observed variation in the usage of agricultural limestone in Ohio. The purpose of the demand equation is to quantify the sensitivity of farmers' demand for agricultural lime with respect to alternative factors such as prices of farm products, agricultural lime, and other inputs. The supply equation is used to estimate the effect of agricultural lime price, prices of key inputs such as transportation and fuel, and prices of non agricultural lime products on agricultural lime supplies.

Industry Structure in the United States. The agricultural lime industry is a subsector of a larger lime industry and the crushed stone industry. The United States is the largest producer of crushed stone among the market economy countries. In 1989, total production of crushed stone reached 1.2 billion short tons of which 75 percent was limestone and dolomite. Agricultural lime accounted for 2 percent of total limestone production and 6 percent of total dolomite production. Of 1989 total U.S. crushed stone production, 74 percent was from two major geographic areas, the south and the middle west. In the United States, limestone is produced in all states except Delaware, Louisiana, New Hampshire, and North Dakota.

The 1989 crushed dolomite output was reported in 27 states by 92 companies at 136 quarries. Of the 49 million tons of crushed dolomite consumed, 6 percent was used as agricultural lime. The United States is self sufficient in crushed stone of which agricultural lime is a part. Production of limestone meets all domestic demand.

Market Structure of Agricultural Limestone. The agricultural lime industry is characterized by its low unit value. Agricultural limestone is a material of low price at the point of production; however, transportation costs substantially increase price at the point of delivery. Consequently, agricultural lime generally is produced near its

marketable area and points of use, and the industry is mostly concentrated in high demand areas.

The limestone industry is highly competitive and characteristically serves local or regional markets as a result of an abundance of mineral deposits. Factors affecting production are mainly labor, equipment, energy, water, and compliance with environmental and safety regulations. These factors affect the cost of production which can vary with the natural formation of deposits, their geographic locations, and the quality and kind of product. The closer the markets are to the supply areas, the more significant the advantage to producers. Therefore, local production, even though impure or of lower grade is more economical to farmers than purchasing higher quality material from distant markets at a higher cost. Most producers of agricultural lime are relatively small and face competition from large companies producing agricultural lime as a joint product of limestone or industrial lime production.

Limestone and dolomite were produced or sold by 80 companies at 122 operations in 54 Ohio counties during 1990. Total sales of limestone and dolomite were 52.7 million short tons from which 37.2 percent was the contribution of 5 counties: Erie, Franklin, Wyandot, Ottawa, and Sandusky. The 1990 reported sales of agricultural crushed and broken limestone accounted for 3 percent of total limestone sales in the state (Ohio Department of Natural Resources, 1990).

Method of Estimation. A simultaneous system of equations, estimated by a two-stage regression technique, is used to estimate factors influencing demand and supply of agricultural lime. Because demand and supply are believed to be determined simultaneously, Two Stage Least Squares (TSLS) estimation technique is applied. This is to avoid biased and inconsistent estimates associated with Ordinary Least Squares (OLS) procedures between two or more dependent variables. It is assumed that usage and price of agricultural lime are simultaneously determined. That is, quantity used and produced and prices paid and received for agricultural lime depend on factors affecting demand and supply, and variations in usage of agricultural lime are related to fundamental economic forces.

Results. Two functional forms were used to estimate the model: one with linear equations and one composed of equations with logarithms of the same variables of the first equations, called a double-log linear regression model. The model that offers the better explanation of the demand and supply of agricultural lime is the double logarithmic model and only these results are presented. The equations from the TSLS estimation process are the following:

Demand and Supply Estimates.

Demand Equation.

$$\begin{aligned}\log(\text{QAL}) = & -1.9417 - 0.101 \log(\text{PAL}) + 1.176 \log(\text{IPP}) \\ & (-1.639) \quad (-0.271) \quad (4.764) \\ & + 0.118 \log(\text{NFI}) - 0.677 \log(\text{IIP}) + 0.539 \text{D} \\ & (1.992) \quad (-3.730) \quad (3.251) \\ R^2 = & 0.77 \quad \text{and} \quad \text{Adjusted } R^2 = 0.73\end{aligned}\tag{5}$$

Supply Equation.

$$\begin{aligned}\log(\text{QAL}) = & 0.1468 + 0.646 \log(\text{PAL}) - 0.014 \log(\text{PIL}) \\ & (0.063) \quad (1.557) \quad (-0.025) \\ & - 2.234 \log(\text{ITR}) - 0.349 \log(\text{IFU}) + 2.264 \log(\text{IME}) \\ & (-2.932) \quad (-1.179) \quad (2.712) \\ R^2 = & 0.58 \quad \text{and} \quad \text{Adjusted } R^2 = 0.48\end{aligned}\tag{6}$$

where

- QAL = quantity of agricultural limestone
- PAL = price of agricultural limestone
- IPP = index of price received by farmers for crops
- NFI = real Ohio net farm income
- IIP = index of price paid for farm inputs
- D = government program variable; 0 = no government program,
1 = government program.
- PIL = real price of industrial lime
- ITR = price index for transportation equipment
- IFU = price index for fuel
- IME = price index for material used in manufacturing

Coefficients preceding the variables identify the estimated percent change in the quantity of agricultural lime demanded or supplied associated with a one percent change in the variables. Numbers in parentheses below these coefficients are standard errors of these estimates.

Considering the demand side, in terms of explanatory power, the squared coefficient of determination (R^2) is very high (0.77) indicating that 77 percent of the variation in agricultural lime usage is explained by the model. Also R^2 in the supply estimation indicates that the model has an appreciable explanatory power.

The estimated demand and supply relationships identify the contribution of individual factors (for example price of agricultural limestone, PAL) in explaining the variations in the quantity of agricultural limestone demanded or supplied, holding all other variables constant.

The demand price of agricultural limestone is inelastic (-0.101); in this case, inelasticity implies that demand is relatively insensitive to price changes. There is a negative association between the price of agricultural limestone and its purchase by farmers. As agricultural limestone price increases by 1 percent, quantity demanded by farmers decreases by 0.101 percent. Lime usage proves to be income inelastic (0.118) but highly elastic with respect to crop prices paid to farmers (1.176). That is, a 1 percent increase in net farm income increases quantity demanded by 0.118 percent, while a 1 percent increase in crop prices increases quantity demanded of agricultural limestone by 1.176 percent. On the other hand, limestone application in agriculture is inversely related to prices of other farm production inputs, and a 1 percent increase in the price of items used for production (IIP) decreases the demand for agricultural limestone by 0.68 percent. Last, on the demand side, cost share conservation programs have had a significant positive effect on agricultural lime usage.

The supply price elasticity of agricultural limestone is considered inelastic (0.646). That is, agricultural limestone producers respond to a 1 percent increase in agricultural limestone price by increasing agricultural lime production by 0.646 percent. An inverse relationship between prices of industrial lime and the supply of agricultural limestone is estimated. That is, increasing industrial lime prices result in slightly lower agricultural limestone supplies. One percent increase in prices of transportation equipment and fuel leads respectively to a drop of 2.23 and 0.34 percent in limestone supply. However, one percent increase in the price of material used in manufacturing increases agricultural limestone supply by 2.264 percent, reflecting the fact that increases in manufacturing material prices is associated with increased supply of crushed limestone and a corresponding increase in the agricultural limestone by-product.

From the results of this econometric model of agricultural limestone supply and demand relationships, the price elasticity of demand is 0.10, and a rise in the price of agricultural lime, holding everything constant, has only a small effect on its use. However, the use of agricultural limestone is affected by prices of other farm inputs, by net farm income, and by crop prices. An elasticity of supply of 0.64 roughly means that a 10 percent increase in price of agricultural limestone would lead producers to increase supply by 6 percent. The industry appears to be capable of readily increasing supply in response to demand increases. Forster and Ennich (199) provide more detail on method and results of this analysis.

Summary and Implications. Agricultural limestone is a product of both the crushed stone industry and lime industry. It is a material of low price per ton at point of production. However, due to its bulk, transportation costs constitute a considerable part of its price at points of delivery. Because of high transportation costs and the

abundance of limestone deposits in the United States, agricultural limestone is marketed within small geographic areas and not transported large distances.

Agricultural limestone has experienced a declining market in agriculture even though the benefits from its application are known. A two-stage model is used to analyze the demand and supply forces of this input market. The double-log linear model is used to estimate these relationships. This model shows that the demand for limestone is inelastic with respect to its price. Lower agricultural limestone prices will have only slight impact on increasing its usage. Its use also is affected by the prices of other agricultural inputs used in production. Most importantly, its demand is affected by crop prices. Substantial increases in agricultural lime demand might occur if crop prices were to increase. The quantity of agricultural limestone supplied is also inelastic with respect to its price. It is affected by various factors of production such as transportation and fuel costs, which have an inverse relationship with respect to limestone supply. Results of this study indicate that the supply of agricultural limestone is enhanced by increased prices for crushed limestone, but increases in the price of industrial lime dampen the supply of agricultural limestone.

If FGD scrubbers are used by electric utility companies, FGD by-product promises to be an effective substitute for agricultural limestone, and characteristics of the agricultural limestone market will affect FGD by-product use. This new product will be entering a highly competitive market. If it is to be used in agriculture, it will have to replace agricultural lime. Agricultural limestone producers stand to lose market opportunities with the introduction of FGD by-products. However, limestone producers would probably gain an even bigger market if coal burners are retrofitted with scrubbers. The amount of limestone products used in the scrubbing process would far exceed the amount of the agricultural lime market replaced by the FGD by-product.

This analysis has implications for agricultural limestone producers under the scenario of expanded market opportunities with the adoption of FGD technology. This analysis estimates the price elasticity of supply to be inelastic (.646). Thus increases in demand would result in proportionately large increases in price. If electric utilities adopt FGD technology, the agricultural limestone industry can expect to produce a larger volume of output and receive higher prices.

2.4.4 Analysis of FGD By-Products Use in Beef Feedlots

One potential market for the FGD by-product is as surface paving material. Recently, interest has been expressed in its use in Ohio beef cattle feedlots. The purpose of this analysis is to investigate some economic issues related to the use of FGD by-product as a cattle feedlot surface material. Cattle feeding has been in a persistent decline in Ohio, as evidenced by cattle marketings declining from 858,000 in 1975 to 395,000 in 1992. Would supplying the cattle feeding industry with a low, or zero cost surface material improve the profitability of Ohio cattle feeding? Would it affect the future economic viability of Ohio cattle feeding?

Fundamental Economic Issue # 1: Consumer Demand. The cattle feeding industry can be characterized as facing stagnant consumer demand, and specialization and concentration of production. Until the mid-1970s cattle feeding was an expanding industry due to rising population and increasing per capita beef consumption. Since that time consumer preferences have changed. While demand for meat has increased, health conscious consumers have shifted away from beef to other meats. Per capita consumption of red meat, poultry and fish was 194 pounds in 1991, a 13 percent increase over 1975 consumption. Since 1975, per capital consumption of pork has remained steady (46.9 pounds in 1991), poultry has nearly doubled to 66.7 pounds, and fish has increased slightly to 15.4 pounds. However, per capita consumption of beef has declined from 83.0 to 63.9 pounds, and total U.S. beef production has reached a plateau.

Fundamental Economic Issue # 2: Location of Production. Cattle feeding is concentrated in the Plains States. Five states - Texas, Nebraska, Kansas, Colorado, and Oklahoma - accounted for 50 percent of U.S. cattle marketings in 1990. Their share was 30 percent in 1960. On the other hand, cow-calf herds are dispersed widely. Cow-calf operations tend to be located on marginal land and provide a method to market forage from pasture land that has little or no alternative use. Once calves reach 450 to 650 pounds, they are shipped to feedlots for fattening with concentrated feeds.

One of the reasons for the increased concentration of cattle feeding in the Plains States was the development of irrigated grain sorghum production in those states to supply needed concentrate feeds. This development was stimulated partly by increased feed grain prices in the early-1970s resulting from increased export opportunities. Until that time, the U.S. was essentially self-sufficient in feed grains. After that, it produced about one-fourth of its feed grains for an export market. Much of the feed grain exports came from the east North Central region, which in essence, increased feed prices for this region's beef feeders. Another factor contributing to cattle feeding concentration in the Plains States was an efficient transportation system that allowed feeder calves to be shipped to feedlots and beef products to be shipped to distant population centers at low cost. Also, fed beef housing and manure disposal costs were substantially less in the Plains States than in the North Central region.

Gains in cattle feeding in the Plains States came, in large part, at the expense of farmers in the eastern portion of the North Central region, which includes Ohio. The share of U.S. cattle marketings coming from the east North Central region declined from 14.2 percent in 1960 to 6.4 percent in 1990. In addition to east North Central states, Iowa, Minnesota, and Missouri lost substantial market share. The market share of these eight states (Ohio, Indiana, Illinois, Michigan, Wisconsin, Iowa, Minnesota, and Missouri) declined from 33 percent in 1960 to 16 percent in 1990. So, the location of beef feeding has changed dramatically. Thirty years ago, the eight-state Corn Belt region fed the same amount of beef as the five-state Plains States region; today, it feeds one-third as many.

Fundamental Economic Issue # 3: Industry Structure. These regional shifts resulted in a more geographically concentrated fed beef industry, and also produced a more concentrated industry structure with fewer, larger firms. The industry's production continues to shift to feedlots with more than 1,000 head capacity, and only 15 percent of fed beef are produced in feedlots with less than 1,000 capacity. Most feedlots in the east North Central region are farmer feedlots with less than 1,000 head capacity. On these farms, beef cattle feeding is typically one enterprise among several on the farm.

Costs and Returns. The U.S. Department of Agriculture (USDA) annually estimates average costs of production and returns for fed cattle production. Returns and costs are estimated for the two types of cattle feedlots: commercial feedlots and farmer feedlots. The estimates are based on financial and technical data obtained from producer surveys in Illinois, Iowa, Minnesota, Nebraska, and Kansas. Existing Ohio feedlots would most closely resemble the "farmer feedlots" depicted by USDA. Even large Ohio feedlots, those approaching 1,000 capacity, would resemble farmer feedlots because commercial feedlots represented in USDA estimates are Plains States feedlots which have relatively low building and lot investments.

Economic costs include all cash operating expenses, depreciation of facilities, interest charge for all capital (operating capital and investment in facilities), and labor charge for all hired and unpaid family labor. Over the 1972-90 period, costs averaged more than receipts from cattle marketings. Average residual returns (returns minus economic costs) were -\$1.89 per hundred pounds. However, residual returns were higher for commercial lots. The average commercial feedlot received a rate of return on capital investment that was comparable to off-farm investments with a similar degree of financial risk. Losses and low returns to capital investment were incurred primarily by farmer feeders.

On average, about 90 percent of the economic cost for fed cattle consists of expenses for purchasing the feeder calf and feed. Farmer feedlots typically purchase feeder calves at lighter weights and feed them longer than commercial feedlots. Thus, farmer feedlots have higher feed costs and lower feeder calf costs per hundred pounds of beef produced than commercial feedlots. There are substantial differences between commercial and farmer feedlots in labor and capital costs. Examples of labor inefficiency on farmer feedlots are observed in feeding, bedding, manure removal, and health care activities. Lower per unit capital costs on commercial feedlots reflect cost advantages due to scale economies and lower per unit investments in housing and feeding systems.

The cost estimates for 1990 highlight the cost disadvantages faced by farmer feedlots. In 1990, economic costs were about \$8 per hundred pounds higher for farmer feedlots than for commercial ones. Furthermore, Ohio farmer feedlots may face slightly higher feed costs than the average farmer feedlot pictured by these data. Feed grain prices at the farm level usually are higher in Ohio than in the western Corn Belt states (Iowa, Minnesota, and Nebraska) where USDA's farmer feedlot cost estimates

originate. Given these cost comparisons, it is little wonder that abandoned cattle feeding facilities are a common sight in rural Ohio.

Is FGD By-Product a Remedy? If FGD by-products were furnished at low, or zero cost to the Ohio cattle feeding industry, capital costs would be reduced. Concrete may account for as much as one-fourth of the investment in housing and lot facilities. However, the impact of excluding concrete floor investment on USDA's cost of production estimates is minimal. Capital costs decline by \$0.20 per hundred pounds - about the same magnitude as bedding costs - for the farmer feedlot if concrete costs are excluded.

Another potential effect of using the FGD by-product to pave existing feedlot surfaces is that feeding efficiency and weight gain could be enhanced. The reason housing investments are relatively high in Ohio and other Corn Belt states is that cattle need to be sheltered from wet weather that turn open lots into mud during some periods of the year. Muddy feedlots cause animals to expend more energy, reducing feed efficiency and weight gain. Typically, Ohio farmer feedlots are constructed with housing and a partially paved lot. During wet periods, the cattle have only partial reprieve from muddy surfaces. Computer simulation models were used to estimate the effect of muddy feedlot surfaces on production costs. Feed costs averaged \$1.02 per hundred pounds higher for feedlots with muddy lot surfaces than for those with paved lot surfaces. That is, average cost per hundred pounds would decrease from \$84.89 to \$83.87 if farmers pave portions of their feedlots that are muddy.

A third advantage of using FGD by-product on feedlot surfaces is that it would increase the market price of the livestock. Buyers pay less for beef that are coated with mud because it is apparent that there will be a lower percentage of dressed beef. However, producers should receive the same gross revenue for each animal whether it is clean or not. The per pound price discount for mud coated animals is offset by the added weight of the mud. The buyer is paying for the quantity of dressed beef in the animal, and the purchase price is adjusted to compensate for wastage.

Implications. The use of this by-product on farmer feedlots offers farm level benefits, but the distinct cost disadvantage faced by the Ohio farmer feeder would be affected only slightly. Potential benefits to Ohio farmer feedlots of supplying free feedlot surfacing material are estimated to be: (a) reduced capital costs totaling \$0.23 per hundred pounds of beef and (b) reduced feed costs of \$1.02 per hundred pounds. While the profitability of Ohio feedlots would be improved, their competitive position in the U.S. beef feeding industry would remain about the same. Currently, costs on the average U.S. farmer feedlot are estimated to be \$8.19 higher than costs on the average commercial feedlot, and there is reason to suspect that this cost difference is even greater for the average Ohio feedlot. Like the effect a thimble full of water has in relieving thirst, low cost FGD by-products would be beneficial, but they would have relatively little effect on the parched economic conditions in Ohio beef feedlots.

While Ohio beef feedlots would receive benefits from low, or zero cost surface material, suppliers might also benefit. Electric utility companies using the FGD technology generate a by-product that must be marketed in a beneficial use or disposed. Disposal by landfilling is likely to be costly; therefore, marketing the FGD by-product as feedlot surface material or other beneficial products need might be profitable for the company, even though cash receipts from the products are negligible.

Finally, a beneficial use for the FGD by-product, like using it as a feedlot surface material, could have substantial indirect benefits for communities. If beneficial uses are not utilized, landfills would impose "external" costs on communities, which are reflected in the depressed market value of real estate neighboring landfills.

2.4.5 Property Value Impacts from Landfilling: Preliminary Results from Gahanna Landfill

Using a small subsample of data from the Gahanna, OH area, the effect of a sanitary landfill on transaction prices in the market for new and existing residential single family housing was studied. In the present analysis, the underlying purpose was to study the social welfare impact of alternative uses of FGD by-product. There are two particularly interesting possible end uses for FGD by-product, one creating a potential environmental disamenity, i.e., landfilling, and the other a potential environmental amenity, i.e., surface coal mine reclamation.

After controlling for the confounding factors affecting housing prices, we would expect to observe lower real estate values close to a landfill site. A landfill's *life expectancy* may also affect real estate values, and after a landfill has been reclaimed, one would expect to see an increase in real estate values in the properties adjacent to the landfill site. However, the depressive price effects of a landfill may linger even after the landfill has been closed, if the landfill was either built in an undesirable area (near a freeway, for example), or attracted industrial development. In the present study, the life expectancy of the landfill would be expected to be manifest in differential price effects in the land and existing housing market. The negative (or positive) impact of proximity to a landfill would indicate social costs (or benefits) in terms of willingness to pay.

The ultimate goal of the economic analysis of landfills will be twofold. First, it will be necessary to estimate the social welfare impact of a landfill on homeowners living nearby by investigating the effects of a landfill's life expectancy and the proximity of property to the landfill. Second, we will try to determine any positive effect to be realized from landfill reclamation. Once we have estimated the social welfare impacts of the landfills which we are studying in Franklin County, we will attempt to extend the analysis by developing a predictive model to represent other landfill sites around the state of Ohio.

The preliminary analysis using the Gahanna, OH data subset is presented after a brief review of the hedonic pricing model which comprises the theoretical and empirical underpinnings of our analysis. In order to investigate one aspect of our

analysis, i.e. the impact of proximity to the landfill, 87 real estate transactions within a three mile radius of the Gahanna (Bedford) landfill were used in a statistical regression model.

The Hedonic Pricing Model

Theoretical Framework. To measure the effects of landfilling on real estate values, economic theory would suggest estimation of an implicit price function, or hedonic price function (HPF), to measure marginal willingness to pay (MWTP) for housing characteristics. MWTP can be interpreted as a consumer's demand for a particular characteristic of a housing unit from which welfare benefits can be quantified. In this case, the characteristic of interest is individuals' willingness to pay for distance from the landfill, both in terms of time and space. Estimates of the time aspect of the model will be derived from differing life expectancies of the landfills, while spatial aspects will be determined from distance of a house to the landfill.

The hedonic price gradient (HPG) derived from the HPF is the locus of equilibria of buyers' willingness to pay curves which arise as a result of consumers' utility (or satisfaction) maximization. Rosen (1974) presented a general framework for using the hedonic pricing model in the analysis of demand and supply for characteristics of a differentiated product (in our case, a house). Briefly, a consumer purchases one unit of a good with a vector of attributes z subject to the following utility maximization problem:

$$\text{Max } U = U(z, x; a) \text{ s.t. } y = p(z) + x \quad (7)$$

where z is a vector of characteristics of the product, which in this case consists of such factors as number of rooms in a house, lot size, distance from amenities and disamenities and neighborhood characteristics; x represents all purchases other than housing which a consumer can make, and a is a parameter which characterizes individual consumers. $p = p(z)$ is the hedonic (or implicit) price function. $y = p(z) + x$ is the budget constraint, or the maximum amount that an individual can spend on housing plus all other goods. This function may be nonlinear, in theory, since an additional unit of an attribute depends on the house being purchased, and there may also exist interaction among attributes. Each individual is assumed to take prices as fixed, and the price gradient is determined by forces over which the consumer has little control. The HPG results from the maximization problem in equation (7):

$$\text{HPG} = h(z, p-p(z); a) \quad (8)$$

Equation (8) is interpreted to be the implied demand for a good with characteristic z_i . An individual's utility is maximized at the point where his marginal willingness to pay for an additional unit of a characteristic is equal to the marginal implicit price. Thus, (2) represents the increase in expenditure needed for an increase in the consumption of characteristic z_i .

In our analysis, the characteristic, z_1 , of interest is the distance from the landfill. Theoretically, it would be expected that individuals' willingness to pay would be inversely related to this variable.

The Empirical Model. Figure 88 shows the extent of the preliminary study area; areas to the east and north of Gahanna are progressively more rural, with Whitehall more urban and Reynoldsburg more suburban. The area is centered on the Bedford sanitary landfill, a facility which in 1990 had an expected life of just two years.

The Housing Market Model. First, the hedonic price function (i.e. the integral of HPG) for the housing market is estimated. The general functional form for the estimated hedonic price function for new and existing housing is

$$\ln(\text{pricellotsize}) = a_0 + a_1 Z_1 + a_2 Z_1^2 + a_3 Z_2 + a_4 Z_2^2 + \sum_{i=5}^{20} a_i Z_{i-2} \quad (9)$$

where the z 's are as follows:

- z_1 = inverse distance from landfill in miles (1/distance);
- z_2 = direct distance from central business district (Columbus, OH);
- z_3 = square footage of residence/lot size;
- z_4 = fireplace (0,1 indicator variable);
- z_5 = age of building in years;
- z_6 = number of baths and half baths;
- z_7 = number of bedrooms;
- z_8 = condominium (0,1 indicator variable);
- z_9 = swimming pool (0,1 indicator variable);
- z_{10} = community crime rate/1000 persons;
- z_{11} = adjacent to country club (0,1 indicator variable);
- z_{12} = adjacent to railroad (0,1 indicator variable);
- z_{13} = adjacent to freeway (0,1 indicator variable);
- z_{14} = adjacent to airport (0,1 indicator variable);
- z_{15} = adjacent to park (0,1 indicator variable);
- z_{16} = Gahanna school district (0,1 indicator variable);
- z_{17} = median income of Census block group;
- z_{18} = average age of individuals in Census block group.

Except for the crime statistics, which came from *Columbus Monthly* (May, 1992), data for this study were provided by the City of Columbus Development Department Planning Division. Three data sets were used, two Census block group level data sets containing demographic and land use data, and one real estate transaction data set which contained all the housing characteristics. Variables for distance from the landfill and distance to the central business district were created using a straight line measure, and indicator variables were created from a city map of Columbus (See Appendix 6.6 for a description of the full data set to be used in Phase 3).

Figure 88. Gahanna landfill.

Although many more variables were available, the number of variables included in the initial analysis were limited to reduce statistical difficulties. Because of the extreme heterogeneity of the study area, which includes both rural and urban areas, both the transaction price and the square footage of the house were divided by lot size to mitigate heteroscedasticity associated with different settlement patterns. Table 44 contains simple descriptive statistics of the data used in the housing transactions model. The airport variable includes all properties in a 1.5 mile radius from the outer edge of the airport property, and the freeway and rail road variables include properties within one mile of a freeway or rail road. The park and country club variables include only properties within 0.5 miles.

Table 45 shows the results of a semi-log regression estimate of the model. The signs of the inverse distance and squared inverse distance to landfill terms suggest, as expected, that *the price of houses decreases at an increasing rate at closer distances to the landfill, i.e.,*

$$MWTP = (-8.962 + 10.4987z_1)\exp(a'z) \quad (10)$$

Table 44. Descriptive statistics of the data used in the housing model transactions.

N Obs 87	Variable	Minimum	Maximum	Mean	Std Dev
	price	9274.62	1980716.37	104554.04	214908.14
	price/lot	0.0003050	69.97	10.46	14.60
	1/distance	0.3367003	1.42	0.50	0.17
	1/distance ²	0.1133671	2.03	0.28	0.26
	distance to Bedford LF	0.7012500	2.97	2.17	0.51
	distance to Central Business District (CBD)	7.7500000	16.50	12.11	2.50
	CBD ²	60.0625000	272.25	152.78	61.79
	sq ft house	165.0000000	2940.00	1410.44	523.07
	bldg/lot	4.8330192E-6	1.00	0.32	0.43
	fireplace	0	1.00	0.37	0.48
	age of house	0	91.00	35.30	17.96
	lot size	686.00	457380000	11779081.76	55421703.5
	# baths	1.00	3.00	1.42	0.52
	# bedrooms	1.00	7.00	2.99	0.82
	condo	0	1.00	0.29	0.45
	pool	0	1.00	0.03	0.18
	crime	29.00	111.00	64.44	32.69
	country club	0	1.00	0.14	0.34
	railroad	0	1.00	0.18	0.38
	freeway	0	1.00	0.13	0.33
	airport	0	1.00	0.41	0.49
	park	0	1.00	0.20	0.39
	Gahanna School District	0	1.00	0.37	0.48
	Columbus School District	0	1.00	0.09	0.29
	Whitehall School District	0	1.00	0.36	0.48
	median census block group income	18203.00	76453.00	32779.07	11235.30
	% of block group white/Asian	45.55	98.84	90.88	10.48
	% of block group vacant homes	1.17	21.37	6.02	4.49
	Average Age Census block group	25.86	43.77	35.07	3.06
	sales date	1.00	12.00	7.28	3.24

Table 45. Semi-log regressions estimates of model.

Semi-Log Model					
Dependent Variable: log(price/lot)					
Analysis of Variance					
Source	DF	Sum of Squares	Mean Square	F-Value	Prob > F
Model	20	991.23579	49.56179	25.109	0.0001
Error	66	130.27253	1.97383		
C Total	86	1121.50832			
Root MSE	1.40493	R-square	0.8838		
Dep Mean	- 0.45404	Adj R-sq	0.8486		
C.V.	-309.42515				

Semi-Log Model				
Dependent Variable: log(price/lot)				
	Parameter Estimate	Standard Error	T for H0; Parameter = 0	Prob > T
INTERCEPT	-1.531305	6.87978966	-0.223	0.8245
1/DISTANCE	-8.961720	4.96766672	-1.804	0.0758
1/DINVGHA ²	5.249424	2.97740483	1.763	0.0825
CBD	0.248324	1.16588676	0.213	0.8320
CBD ²	-0.019665	0.04642606	-0.424	0.6733
SQFTBLD	27.025830	3.13464720	8.622	0.0001
FIREPLACE	0.403491	0.45130547	0.894	0.3745
AGE OF HOUSE	0.003394	0.01340806	0.253	0.8010
# BATH	0.677424	0.44206654	1.532	0.1302
# BEDRM	-0.198415	0.26098321	-0.760	0.4498
CONDO	-21.145170	2.71274121	-7.795	0.0001
POOL	0.929598	0.93059734	0.999	0.3215
CRIME RATE	-0.003927	0.00953113	-0.412	0.6816
COUNTRY CLUB	0.795694	0.66131427	1.203	0.2332
RAILROAD	-1.104336	0.61182149	-1.805	0.0756
FREEWAY	-0.627741	0.60391257	-1.039	0.3024
AIRPORT	0.174512	0.56153466	0.311	0.7569
PARK	0.381951	0.63839058	0.600	0.5506
GAHANNA SCHOOL DISTRICT	-1.769468	0.63006189	-2.808	0.0065
CENSUS BLOCK INCOME	0.00000879	0.00002157	0.407	0.6851
AVERAGE AGE OF CENSUS BLOCK	0.038331	0.06690969	0.573	0.5687

The negative effect of the landfill on marginal WTP occurs for a distance of about 1.17 miles from the landfill. The square foot of building/lot size term is significantly positive, most likely reflecting the fact that the more urban the area the higher the price of the property, and that in less urban areas larger houses are more valuable. The Gahanna school district variable is significantly negative, possibly reflecting the fact that Gahanna is the most rural part of the study area, as well as the part of the study area closest to the landfill.

2.4.6 Model Formulation for Muskingum Watershed Study Area

Reclamation of land used for mining is required by both state law and the federal Surface Mining Control and Reclamation Act. Since 1974, over two million acres have been reclaimed by U.S. coal producers, an area larger than the state of Delaware. Today operators must have a reclamation plan before mining begins, and mining companies are required to post a bond that is not returned until the land is restored to an approved level.

Mine reclamation is an important issue for Ohio, as mining occurs in 24 of the state's 87 counties, and four additional counties have abandoned mines. Until reclamation began, the 200 years of mining in the state left some 457,000 acres of land scarred by strip mining. By 1988, the state and federal government had reclaimed some 7,500 acres at a cost of \$52.4 million. It is estimated that total reclamation of all mine sites in the state will cost \$4 billion. Ohio ranks second in the U.S. with land needing to be reclaimed.

The Ohio Department of Natural Resources ranks reclamation sites by their potential threats--Priority 1 and 2 mines pose immediate health and human safety dangers, while Priority 3 mines present an environmental risk. ODNR lists 756 mines as Priority 1 and 2, and 418 as Priority 3. Almost 43% of Priority 1 and 2 sites have been reclaimed since 1982.

The study area chosen was the Muskingum Watershed Conservancy District (MWCD) which comprises approximately 1/5th of the land area of eastern Ohio (Fig. 89). The watershed is particularly appropriate for our study because it is located in an area of the state where there exist a number of both ongoing and abandoned strip and underground mines. Furthermore, a number of coal burning electric power plants are located in this part of the state. There are 10 lakes in the watershed originally created by the Army Corps of Engineers for flood control, but which are now widely used for recreational purposes. Deng et al. (1993) provide detail on the study area, as well as the methods and data to be utilized.

The social benefit-cost modeling strategy in the Muskingum study area will focus on possible cost savings and social benefits which may be realized by use of FGD by-product to reclaim both active and abandoned coal mines. In order to achieve our goal

Figure 89. Muskingum watershed conservancy district.

of producing estimates of these costs and benefits, we will employ two major techniques. First, we will use market-based techniques, such as the hedonic pricing and the travel cost methods, and second, we will use a survey-based method commonly known as contingent valuation (CVM). The hedonic price model is based on the notion that by controlling for property, neighborhood and demographic characteristics, the implicit "price" of an environmental amenity or disamenity can be estimated. The travel cost method uses survey data on expenditures made by individuals to travel to a recreation site as well as on-site expenditures to infer a willingness to pay for use of the recreational facilities. Finally, the contingent valuation method also employs surveys but directly questions respondents on how much additional tax, for instance, they would be willing to pay in order to improve environmental quality. All three of the foregoing estimation techniques are well known in the economics profession, are widely utilized as reliable measures of social costs and benefits and are summarized by Hitzhusen (1992).

The first objective is to obtain estimates of direct accounting cost savings to the power plants of back hauling FGD by-product for use in strip and underground mines to neutralize acidity during the reclamation process. In addition, of the 10 watershed lakes, four are directly impacted by coal mining in the area. One can thus compare these four lakes to the non-impacted lakes through use of the hedonic pricing, travel cost and contingent valuation methods. Using these combined methodologies, one can infer the social benefits from reclamation of coal mines from impacts on real estate values on the lakes, on the recreational value of the lakes, and on the aesthetic value of the area.

The study area has been defined, and initial data on the Ohio coal industry, abandoned coal mines and MWCD lake water quality were obtained. In addition, a mailing list of all individuals who own vacation homes on the lakes, has been purchased and a survey of these homeowners is under development. Plat maps of the cottage areas on the lakes have also been obtained; data on real estate transactions on the lakes will be obtained in Phase 3 both through the local auditors' offices and by directly surveying the homeowners. Through the direct survey method, it is possible to obtain demographic information on homeowners which would otherwise be impossible to obtain, and it will also be possible to directly question respondents on their attitudes towards mining activity in the watershed, as well as elicit willingness to pay estimates for reclamation activity. Surveying day users at the lakes makes it possible to gather comparative data on travel and on-site expenditures at impacted and non-impacted sites. In Phase 3, a joint estimation technique will be developed including the hedonic, travel cost and contingent data to get an overall estimate of willingness to pay for strip mine reclamation.

3 SUMMARY AND CONCLUSIONS OF PHASE 2 TASKS

3.1 Laboratory Studies

3.1.1 Chemistry and Mineralogy of Leachates From Spoil-FGD Mixtures

The study was designed to evaluate the potential impact of dry FGD by-products on water quality in mined land reclamation by following time-dependent changes in solution chemistry and solid phase composition resulting from the forced leaching of mine spoil-FGD mixes. Water soluble components and reaction products from mixtures of two eastern Ohio acid mine spoils and three dry FGD by-products from Ohio sources were examined. The highly alkaline FGD by-products selected for study were from three different scrubber processes. They were produced using lime injection multi-stage burner (LIMB), pressurized fluidized bed combustion (PFBC), and spray dryer (SPD) technologies. The mixtures (0, 10, 20, and 40 wt.% by-product) were sequentially extracted with deionized water for six sampling events over a 4-month period. The concentrations of elemental constituents released to solution were measured at each sampling event. Following the final extraction, the solid residues of spoils mixed with 40% FGD by-product were collected for subsequent X-ray diffraction analysis.

Leachate composition was primarily a function of FGD by-product type, application rate, and reaction time. Spoil type, on the other hand, had little influence on pore water chemistry except at the lowest FGD application rate. All rates of FGD by-products increased the solution pH significantly. The LIMB and SPD amended spoils attained initial pH levels above 12, whereas the less reactive PFBC amended spoils generally reached a pH of 10 or less. There was a tendency in most mixtures for the pH to decrease with time, and this effect was usually most pronounced at the lowest application rate.

Leachates from the LIMB and SPD amended spoils were dominated by Ca, S and Cl, whereas the PFBC mixes yielded significant amounts of dissolved Mg in addition to the aforementioned ions. Addition of FGD by-products resulted in initial increases in leachate Ca, Cl and S followed by decreases in solution concentrations with time. Sulfur concentrations were greater in leachates from the LIMB and PFBC amended spoils as compared to the SPD mixes due to the presence of more soluble forms of sulfate-S in the LIMB and PFBC materials. Significant quantities of dissolved Mg were measured only in leachate samples from the PFBC-spoil mixtures because the by-product contained both dolomite and periclase.

Leachate concentrations of Fe and Mn decreased to near or at detection limits with the addition of 10 wt.% by-product, whereas Al and Zn concentrations decreased with increasing rates of by-product. Reductions in soluble Al are particularly significant because this element is commonly present in phytotoxic levels in acid spoils.

In the amended mine spoils, As, Cd, Cr, Cu and Ni concentrations were below the maximum contaminant levels (MCL) defined for drinking water for the duration of the study. Leachate Pb concentrations were below the detection limit in all samples except in two amended spoil samples. Selenium concentrations were variable ranging from below detection to levels exceeding drinking water standards. Seventy-eight percent of the leachates exceeded the MCL for Se at day one. By comparison, only 20 to 30 % of the leachates exceeded the MCL for Se after 26 days of reaction. The reported concentrations were observed in mixtures with FGD amendment rates as high as 40 wt. %. A more realistic field application rate of 10-12.5 wt. % (100-125 tons/acre) by-product would result in lower solution concentrations for many elements.

Geochemical modeling of the leachate chemistry suggest that ettringite would form as the final reaction product in the LIMB and SPD amended materials while gypsum would be the solid phase controlling Ca and S solubilities in the PFBC-amended materials. Observed changes in solid phase composition supported the weathering scenario predicted by modeling. In the LIMB mixes, CaSO_4 and CaO hydrated rapidly to form $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ and $\text{Ca}(\text{OH})_2$. With longer reaction time the conversion of $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ to ettringite $[\text{C}_6\text{H}_6\text{Al}_2(\text{SO}_4)_3(\text{OH})_{12} \cdot 26\text{H}_2\text{O}]$ was observed. The precipitation of ettringite was also observed in the SPD-mixes but without $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ appearing as an intermediate phase. In the PFBC mixes, Ca weathering reactions resulted in the precipitation of $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ as a major Ca phase with no conversion of the $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$ to ettringite detected at 132 days. The significantly lower solution pH in the PFBC amended spoils most likely inhibited ettringite formation. In the 40 wt. % LIMB and SPD amended mixtures, pH's were significantly greater than in the PFBC-mixes and ettringite was present at 132 days; however, the long-term stability of ettringite as a final reaction product in these weathering environments is questionable.

3.1.2 Physical Properties of FGD/Spoil Mixtures

There is clearly a relationship between the increasingly finer texture and improved structure with increased FGD incorporation into these soils. This interrelationship, however, varies for the different FGD sources. Where addition of the finer textured FGD material to either soil would be expected to increase the mix compressibility, the opposite occurred. This is likely due to the increased aggregate strength afforded by FGD incorporation. LIMB by-product was generally the most effective and PFBC by-product the least effective as evidenced by higher rates of PFBC by-product needed to decrease compressibility. While the decreased bulk density with increasing FGD was highly and significantly correlated with the compression index ($r = 0.85$), a reduced bulk density is expected with increased incorporation of fine textured materials. Perhaps the most revealing information regarding the interrelationship between texture and structure is given in the hydraulic conductivity and water retention data. Low (i.e., 10%) rates of LIMB by-product yielded an improved structure for both soils with the greatest effect noted for the poorly structured R&F material. Increasing rates of LIMB by-product produced effects characteristic of an increasingly finer textured mix. Spray dryer by-product had little

structure improving effects with the Fleming spoil but did so at 20% addition with the R&F spoil. Again, higher rates of spray dryer by-product yielded a fine soil texture behavior. The PFBC material did not apparently improve the structure of the Fleming spoil and required 20 to 40% addition to the R&F spoil to yield improved structure.

3.1.3 Engineering Properties of FGD/Soil Mixtures

An estimate of the strength of the FGD material after cycles of freeze and thaw was needed since one application suggested for this material is as a road base or subbase course. The results obtained in the laboratory indicated that the dry FGD has some potential as a replacement for the select material currently specified by Departments of Transportation. The relevant engineering property measured (Resilient Modulus) was slightly lower for the FGD than for the materials currently specified, but the FGD samples tested showed only slight degradation of properties resulting from freeze thaw cycling.

Stabilizing on-site soil with dry FGD has been proposed for a number of applications. For those situations when hauling soil off-site is not practical, mixing weak natural soil with dry FGD may prove to be a cost effective use. Laboratory tests conducted on soil/dry FGD mixes showed a clear increase in the strength of the mixes over the strengths measured for the soil alone. Permeability was seen to be dependent on the percentages of soil to FGD in the mix. The highest permeabilities were obtained for samples made from approximately equal proportions of soil and FGD.

Laboratory tests on wet FGD samples showed strengths were observed to be dependent upon the source of the FGD, the amount of added fly ash and bottom ash, and whether or not lime was added to the material before compaction. The highest strengths were recorded for those samples compacted with additional lime. The results of these tests indicate that lime stabilized wet FGD ash could be used in a number of engineering applications where strengths of 200 to 400 psi are required. The durability of the wet FGD by-product was also determined. Strengths were affected by freeze-thaw cycling but samples that were allowed to cure for a period of 60 days exhibited good strengths and smaller amounts of strength degradation than those samples tested after shorter cure times.

On-going swelling tests have shown that some of the FGD samples continue to swell over time. Swelling rates have been decreasing however. The observed swelling is clearly dependent on the ash type.

The results of the laboratory tests on both wet and dry FGD samples give a clear indication that these materials should be considered suitable replacements for natural soil in many large volume earthworks. The dry FGD has high strength, typically low permeability and low susceptibility to degradation of strength due to freeze thaw. Wet FGD, when supplemented by small amounts of lime and properly cured, should also be considered as a soil replacement or enhancement in many engineered soil systems.

3.2 Greenhouse Studies

The FGD by-products studied were all found to be highly effective in neutralizing spoil and soil acidity and ameliorating phytotoxic conditions in spoils and soils. When applied in the amounts needed to neutralize active and potential acidity, plant growth was improved, and concentrations of Al, Fe, Mn, Ni, and Zn in leachates and in plant tissues were all reduced relative to unamended soil and spoil. Leachate concentrations of Ca, Mg, and S were all increased indicating some potential for downward leaching and amelioration of phytotoxic conditions in soil or spoil below the depth of application. With application of FGD by-products in amounts needed to neutralize acidity leachate concentrations of trace elements of environmental and regulatory concern remained very small. Most, in fact, were less than primary drinking water standards. Likewise, plant tissue concentrations of most trace elements were not increased by FGD amendments. It appears, therefore, that at these application rates, there is little potential for adverse effects on soil, water, or plant quality due to trace element contamination.

When applied in excess of the amount needed to neutralize spoil or soil acidity, FGD by-products containing oxides and hydroxides of Ca caused initially high pH (in the range 8.5 to 11) and in some cases cementing due to ettringite formation. FGD by-products containing Mg caused high levels of soluble salts. These conditions had a suppressive effect on early plant growth, and in the case of ettringite formation resulted in restricted rooting volume. The negative effects on plant growth could be avoided by applying FGD enough in advance of planting to allow for pH decrease by carbonation and for salts to decrease by leaching. Such over-application may be warranted if it would lead to increased leaching of Ca, Mg, and S and amelioration of phytotoxic conditions in deeper, unamended soil/spoil layers.

3.3 Economics

The economic analysis of recycling or disposing of FGD by-products on cropland, for feedlot, surfaces, for stripmine reclamation for highway construction, etc. is of two types—financial costs and returns and social costs and benefits. Both types of analysis have been utilized in Phase 2 in developing conceptual models and deriving empirical estimates useful for suppliers and users of FGD as well as for public policy formulation.

The specific conceptual and analytical focus in Phase 2 has been on (1) a linear programming analysis of least cost disposal, land application and stripmine reclamation options from a financial or power plant accounting stance perspective, (2) an econometric supply and demand analysis of the agricultural lime market to determine the potential for FGD by-product as an ag-lime substitute, (3) analysis of FGD by-product use in beef feedlots, (4) a preliminary analysis of property value impacts from the Gahanna landfill, and (5) formulation of hedonic pricing, travel cost and contingent valuation models and identification of data sources for the Muskingum Watershed reclamation study area.

Key preliminary economic findings and conclusions from Phase 2 include the following:

1. Under current regulations, assumptions and technologies regarding FGD end use or disposal, the average financial cost per ton is estimated at \$26.10 with 90 percent landfilled, nine percent applied to cropland and one percent used for stripmine reclamation.
2. The demand for ag lime is inelastic with respect to its price and its use is also affected by the prices of other agricultural inputs used in production and by crop prices. The quantity of ag lime supplied is also inelastic relative to its price. These characteristics of the ag lime market have important implications for any substitution of FGD by-products.
3. Use of FGD by-product on Ohio farmer feedlots offers some potential for reduction in costs. However, the magnitude is likely to be quite small relative to the distinct cost disadvantage faced by producers in Ohio and the Midwest generally.
4. In the Gahanna subset of the Columbus area hedonic price analysis of landfills and housing values, price of houses decreases at an increasing rate at closer distances to the landfill suggesting a potential negative social cost from FGD disposal.

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5 GLOSSARY

Abandoned mined land (AML). An area of land that was mined for the extraction of a specific mineral resource prior to the passage of the Surface Mining Control and Reclamation Act of 1977 and not returned to an environmentally acceptable state.

Acid insoluble residue. A solid residue composed primarily of fly ash that is produced by the dissolution of dry FGD by-product with a weakly acidic buffer solution (pH 5.0) prepared by mixing sodium acetate, acetic acid, and water.

Agricultural lime. A soil amendment containing calcium carbonate, magnesium carbonate, and other material, used to neutralize soil acidity and furnish calcium and magnesium for plant growth. Classification including calcium carbonate equivalent and limits in lime particle size is usually prescribed by law or regulation.

Alkaline amendment. Any material applied to a soil or spoil to raise the pH.

Anhydrite. The mineral name for anhydrous calcium sulfate, CaSO_4 .

Approach to saturation. A measurement of the moisture content of flue gasses, in terms of the number of degrees the gas temperature is above the temperature at which the gas would be saturated (100% relative humidity) with water at its current moisture content.

ASTM leachate. The aqueous extract produced by the American Society for Testing and Materials standard test method for shake extraction of solid waste with water (ASTM D 3987-85).

Available lime index. A method for measuring the amount of calcium oxide (CaO) in a material (ASTM C 25).

Baghouse. A particulate emission control device which utilizes fabric filters in the shape of elongated bags to remove particulate matter from flue gasses.

Base saturation. The extent to which the adsorption complex of a soil is saturated with alkali or alkaline earth cations expressed as a percentage of the cation exchange capacity measured at pH 7.0, which may include acidic cations such as H^+ and Al^{3+} .

Bed (or bottom) ash. Non-combusted particulate material which accumulates on furnace beds during combustion of coal.

Beneficial reuse. Use of an industrial by-product in a manner which provides an economic and/or environmental benefit as an alternative to disposal.

Benefit-cost analysis. A procedure for evaluating decisions, or projects, in terms of their full economic consequences (i.e., social benefits and costs).

Bulk density. The ratio of the dry mass of a material to its bulk volume.

Calcine. To remove all volatile components through heating. In this report the term is used to refer more specifically to the conversion of a carbonate mineral to its corresponding oxide by the removal of CO₂.

Calcite. The mineral name for calcium carbonate, CaCO₃.

Calcium carbonate equivalency (CCE). The quantity of acid-neutralizing substances in a material expressed relative to the acid-neutralizing capacity of CaCO₃.

Cation exchange capacity (CEC). The sum of exchangeable cations that a soil, soil constituent, or other material can adsorb at a specific pH. It is usually expressed in centimoles of charge per kilogram of exchanger (cmol_c kg⁻¹).

Coefficient of determination. The ratio of explained variation to total variation in multiple regression statistical analysis.

Compression index (strengths). A ratio of stress to strain used to characterize a soil subjected to one-dimensional strain.

Darcy's Law. A law describing the rate of flow of water through porous media. As formulated by Darcy the law is

$$Q = -K(dh/dl)A$$

where Q is the volume of water passed in unit time, A is the cross-sectional area of the porous media, dh/dl is the hydraulic gradient, and K is a coefficient dependent on the nature of the porous media (the hydraulic conductivity of the media).

Detection limit. The limit of an analytical technique to detect the presence of an analyte. Specifically the concentration of an analyte which produces a signal equal to twice the standard deviation of the background signal fluctuations.

Dolomite. The mineral name for calcium magnesium carbonate, CaMg(CO₃)₂.

Dry scrubber. Any of several technologies for removing SO₂ from flue gasses in which the reaction product is collected in a dry state in the particulate emission control devices (c.f. wet scrubber).

Duct injection. A dry scrubber technology in which the sorbent is injected into the ductwork downstream from the boiler air pre-heaters in the lower flue gas temperature regions.

Economic analysis. In contrast to financial analysis, economic analysis takes into account net economic returns to the whole society which frequently involves shadow prices to adjust for market or administered price imperfections.

Elasticity (economic).

a) price elasticity of supply (demand): the ratio of the percentage change in the quantity of a commodity supplied (demanded) per unit of time to the percentage change in the price of that commodity. If the elasticity is denoted e then:

$e > 1$	elastic demand/supply,
$e < 1$	inelastic demand/supply, and
$e = 1$	unitary elastic demand/supply.

b) income elasticity (e_m): The ratio of the percentage change in the amount of a commodity purchased per unit time to the percentage change in the buyer's income. If

$e_m > 0$	the commodity is normal,
$e_m < 0$	the commodity is inferior,
$e_m = 1$	the commodity is a luxury, and
$0 < e_m < 1$	the commodity is a necessity.

Electrostatic precipitator (ESP). A particulate emission control device in which flue gasses are passed through a charged field and particulate matter is removed due to electrostatic attraction to charged plates.

Environmental disamenity. Any environmental intrusion, such as a landfill or air pollution which is unpleasant or even dangerous.

Ettringite. The mineral name for hydrated calcium-aluminum hydroxysulfate,
 $\text{Ca}_6\text{Al}_2(\text{SO}_4)_3(\text{OH})_{12}\cdot 26\text{H}_2\text{O}$.

Externalities. Incidental, third party or external effects from economic producers or consumption activity which is not accounted for by market prices.

Financial analysis. Refers to net returns to private equity capital based on market or administered prices.

Flue gas desulfurization (FGD). The removal of sulfur dioxide from the combustion gasses of fossil fuel- (coal-) fired boilers.

Flue gas. Gasses resulting from the combustion of fossil fuels (coal).

Fluidized bed combustion (FBC). A coal combustion process in which a bed of mixed pulverized coal and sorbent (generally limestone) is fluidized by jets of air blowing up from the furnace floor.

Fly ash. The glassy inorganic particulate residue suspended in the flue gasses resulting from combustion of fossil fuel (coal).

Geochemical speciation. The equilibrium distribution of cationic, anionic, and neutral dissolved species in a solution expressed as the activity of the species.

Gypsum. The mineral name for hydrated calcium sulfate, $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$.

Hedonic prices. The implicit prices of attributes revealed to economic agents from the observed prices of differentiated goods and the specific amounts of characteristics associated with them.

Hedonic impacts. Impacts on the implicit prices of characteristics of differentiated goods.

Hematite. The mineral name for an iron oxide, Fe_2O_3 .

Hydrated lime. Portlandite, $\text{Ca}(\text{OH})_2$, formed by the reaction of lime (CaO) with water or steam.

Hydraulic conductivity. The rate of flow of pure water through a cross sectional area under a unit hydraulic gradient at the prevailing temperature. Synonymous with permeability coefficient, K .

Hydraulic gradient. The change in water level over a given distance, be it horizontal, vertical or somewhere in between. Defined as $\Delta h / \Delta l$ where Δh is the change in head or water level, and Δl is the change in distance or elevation.

ICP. Inductively Coupled Plasma atomic emission spectrophotometer.

Inclinometer. An instrument capable, when placed in an embankment, of measuring movement of a soil mass.

Instrument calibration and standardization. The optimization of an analytical instrument so as to provide specific output from analytical standards of known composition.

LIMB process. Lime Injection Multistage Burners. A dry scrubber technology in which the sorbent is injected into the furnace above the combustion zone

Lime test index (LTI). An index value generated by reaction of a soil with an alkaline buffer and used to determine the amount of agricultural lime required to raise the pH of the soil to a specified level.

Lime. The mineral name for calcium oxide, CaO .

Limestone. A rock of marine origin that is typically composed of calcite (CaCO_3).

Magnetite. The mineral name for a magnetic iron oxide, Fe_3O_4 .

NIST. National Institute of Standards and Technology.

Optimum density. The maximum density achievable with a standard compaction effort.

Optimum moisture content. The moisture content at which the optimum density is achieved.

Overburden. In surface mining practices, there is usually some quantity of rock or soil that overlies the mineral resource. This material is known as overburden and must be removed during surface mining to expose the mineral.

Oxidation-reduction potential (Eh). A measurement (expressed in mV) of the tendency of a system to oxidize (accept electrons from) or reduce (donate electrons to) chemical species introduced into the system.

Particle size distribution. The fractions of the various soil separates in a soil sample, often expressed as mass percentages.

Periclase. The mineral name for magnesium oxide, MgO .

Permeability. A measure of the relative ease of fluid flow under unequal pressure.

Piezometer. A device for measuring the water pressure at a specific elevation in a soil mass.

Portlandite. The mineral name for calcium hydroxide, $\text{Ca}(\text{OH})_2$.

Potential pareto improvement. An improvement in social welfare in which those who gain from the improvement could potentially compensate the perceived losses of the losers and still be no worse off.

Pressurized fluidized bed combustion (PFBC). A fluidized bed combustion technology modified such that the boiler is pressurized and the pressurized flue gas is used to turn a gas turbine for production of electricity in addition to steam generation to turn steam turbines.

Quality assurance (QA). Refers to a program of activities organized to insure that a laboratory produces data of adequate quality for its intended end use. To meet this data quality goal, an overall quality assurance program comprises numerous quality control activities.

Quality control (QC). Activities which address day-to-day analyses, as well as laboratory management, and are designed to insure that the data produced by a specific analytical procedure is of adequate quality for its intended end use.

Residuals. The amount of a resource which is left over after a material has been transformed from one state to another in a economic production or consumption process, e.g., fly ash is a residual of the process of transforming coal to electric energy.

Saturation index (SI). A measurement of the degree to which a chemical system is under- or oversaturated with respect to a particular solid phase mineral. When SI is greater than 0, the system is oversaturated and the mineral will tend to precipitate; when SI is less than 0, the system is undersaturated and the mineral will tend to dissolve.

Shift variable. In a supply and demand system, a shift variable is one which is exogenous, or outside, of the system. It effectively causes exogenous changes in supply and/or demand.

Soil stabilization. The application of physical or chemical additives to a soil mass for the purpose of improving soil strength.

Specific conductance. The ability of a solution to allow the passage of an electrical current, generally expressed in units of $\mu\text{mho cm}^{-1}$.

Specific surface area. The surface area per unit weight of a material, generally expressed in units of $\text{m}^2 \text{g}^{-1}$.

Spoil. Typically a mixture of overburden and uneconomic mineral deposits that are the result of mining operations.

Spray dryer. A dry scrubber technology in which a sorbent is injected into a dry scrubber vessel located downstream from the boiler air preheater in the lower flue gas temperature region. The scrubber vessel allows for increased flue gas residence time and therefore increased time for the sorbent to react with SO_2 .

Stiffness (soil). The ratio of applied stress to induced strain i.e. σ/ϵ .

Strength (soil). The maximum imposed stress a soil can withstand without failing.

TCLP leachate. The extract produced by the US EPA Method 1311 Toxicity Characteristic Leaching Procedure which is designed to determine the mobility of both organic and inorganic analytes present in liquid, solid, and multiphasic wastes.

Tensiometer. A device for measuring soil water tension.

Thermal reactivity. The amount of heat generated by a material upon addition of water. Generally expressed as the increase in temperature in a specified time period from the addition of a specified amount of water to specified amount of the material.

Thermal gravimetric analysis (TGA). An analytical method for the quantification of mineral components in a sample by measuring the weight loss induced by heating over a specified temperature range.

Tipping fee. The amount charged by a facility accepting waste materials for the disposal of a given weight or volume of a specified waste.

Total neutralizing potential (TNP). The total amount of alkalinity in a material, determined by causing a complete reaction of a specified amount of the material with a specified amount of a standard acid.

Wet scrubber. A process for removing SO_2 from flue gasses in which the sorbent is injected downstream from the particulate emission control system, and the reaction product is collected in the form of a slurry consisting primarily of gypsum (c.f. dry scrubber).

Willingness to accept. This is the supply curve of an economic agent; it is the price at which the agent is willing to engage in trade with a demander of a good.

Willingness to pay (WTP). The demand curve of an economic agent. The price at which the agent is willing to purchase a good from a supplier.

X-ray diffraction (XRD). An analytical technique for the qualitative identification of crystalline mineral components by measuring the scattering of X-rays by the atoms of a crystal in definite directions away from the crystal (diffraction pattern).

6 APPENDICES

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6.4 Supplementary Data from Laboratory Studies

Table 46. pH, E_H, specific conductance, and solution chemistry of leachates from time-dependent reaction products study.

SAMPLE	Time	pH	Eh	SC*	Cl	SO3**	SO4	Ag	Al	B	Ba	Ca	Fe
	days		mV					----- mg L-1 -----					
Fleming Spoil													
A	1	3.1	766	0.5	3.6	ND	93	<0.024	1.55	<0.015	0.037	2.9	0.51
A	7	3.4	761	0.3	1.1	ND	56	<0.024	1.48	0.09	0.027	3.1	0.13
A	14	3.6	837	0.1	2.1	ND	21	<0.024	0.29	<0.015	0.018	1.1	0.05
A	26	3.8	844	0.1	1.0	ND	19	<0.024	0.15	<0.015	0.027	0.5	0.12
A	62	3.6	863	0.2	<0.615	ND	27	<0.024	0.41	0.06	0.023	0.4	0.05
A	132	3.8	855	0.1	<0.615	ND	27	<0.024	0.61	0.07	0.033	0.8	0.06
B	1	3.1	740	0.5	2.5	ND	87	<0.024	1.25	<0.015	0.040	2.4	0.47
B	7	3.6	740	0.2	2.1	ND	36	<0.024	0.92	0.07	0.019	2.2	0.08
B	14	3.7	812	0.1	2.6	ND	19	<0.024	0.16	<0.015	0.013	0.6	0.03
B	26	3.7	834	0.1	1.8	ND	19	<0.024	0.12	<0.015	0.017	0.5	0.04
B	62	3.6	841	0.2	<0.615	ND	30	<0.024	0.46	0.04	0.028	0.5	0.05
B	132	3.7	804	0.1	<0.615	ND	29	<0.024	0.59	0.04	0.034	0.8	0.06
Flem-LIMB(10%)													
A	1	12.1	237	5.0	1714	ND	1755	<0.024	<0.050	0.07	0.528	1083	<0.029
A	7	11.3	202	1.4	3.0	ND	921	<0.024	0.41	0.18	0.093	408	<0.029
A	14	10.9	407	0.8	<0.615	ND	447	<0.024	1.62	0.17	0.073	214	<0.029
A	26	9.7	538	0.9	1.0	ND	540	<0.024	0.99	0.14	0.049	241	<0.029
A	62	NA	757	1.1	<0.615	ND	881	<0.024	0.73	0.33	0.005	354	<0.029
A	132	8.6	785	0.8	<0.615	ND	711	<0.024	0.04	0.29	0.024	270	0.15
B	1	11.9	271	4.0	34.8	ND	1773	<0.024	<0.050	0.03	0.079	898	<0.029
B	7	11.0	322	1.7	4.8	ND	1243	<0.024	0.23	0.21	0.093	572	<0.029
B	14	10.9	411	1.0	1.2	ND	577	<0.024	1.23	0.24	0.083	271	<0.029
B	26	10.1	522	1.0	<0.615	ND	620	<0.024	1.43	0.38	0.050	270	<0.029
B	62	7.4	494	0.9	<0.615	ND	764	<0.024	0.21	0.41	0.003	284	0.04
B	132	8.4	789	0.4	<0.615	ND	262	<0.024	0.07	0.31	0.030	120	0.56

Table 46. continued.

SAMPLE	Time days	pH	Eh mV	SC*	Cl	SO3**	SO4	Ag	Al	B	Ba	Ca	Fe
----- mg L-1 -----													
Flem-LIMB(20%)													
A	1	12.4	209	8.3	75.0	ND	2408	<0.024	<0.050	0.68	0.244	1544	<0.029
A	7	12.6	240	6.3	14.2	ND	1879	0.026	0.11	0.11	0.152	1518	<0.029
A	14	12.1	338	2.3	69.4	ND	1004	<0.024	0.11	<0.015	0.090	624	<0.029
A	26	11.7	418	1.3	<0.615	ND	768	<0.024	<0.050	<0.015	0.054	371	<0.029
A	62	11.1	371	1.2	<0.615	ND	725	<0.024	0.25	0.03	0.013	352	<0.029
A	132	10.3	496	1.4	<0.615	ND	1163	<0.024	0.58	0.14	0.031	513	<0.029
B	1	12.4	213	7.7	62.2	ND	1939	<0.024	<0.050	1.16	0.263	1698	<0.029
B	7	12.1	282	2.0	5.9	ND	582	<0.024	0.19	0.21	0.164	265	0.04
B	14	11.5	370	0.8	<0.615	ND	348	<0.024	0.22	<0.015	0.046	159	<0.029
B	26	11.3	442	0.8	<0.615	ND	393	<0.024	0.36	<0.015	0.038	195	<0.029
B	62	10.7	406	0.5	<0.615	ND	241	<0.024	0.37	0.04	<0.002	117	<0.029
B	132	10.4	475	0.6	<0.615	ND	336	<0.024	0.50	0.05	0.018	137	<0.029
A	1	12.5	207	7.9	284	ND	1393	0.058	<0.050	8.39	0.473	2005	<0.029
A	7	12.7	235	8.0	36.5	ND	1933	<0.024	0.07	1.17	0.219	1688	<0.029
A	14	12.7	316	6.8	<0.615	ND	1500	<0.024	0.16	0.03	0.096	1347	<0.029
A	26	12.8	370	6.3	<0.615	ND	1230	<0.024	<0.050	<0.015	0.101	1267	<0.029
A	62	12.5	246	5.2	<0.615	ND	875	<0.024	<0.050	0.02	0.075	888	<0.029
A	132	12.4	354	3.4	<0.615	ND	482	<0.024	0.10	<0.015	0.093	532	<0.029
B	1	12.5	205	8.8	235	ND	1911	0.064	<0.050	6.04	0.092	1079	<0.029
B	7	12.7	233	7.7	20.0	ND	1909	0.056	0.12	0.94	0.183	1705	<0.029
B	14	12.5	327	3.9	<0.615	ND	746	<0.024	<0.050	<0.015	0.160	674	<0.029
B	26	12.7	374	5.6	12.9	ND	1007	<0.024	<0.050	<0.015	0.079	1084	<0.029
B	62	12.4	248	5.2	<0.615	ND	911	<0.024	<0.050	0.02	0.058	953	0.03
B	132	12.2	360	1.9	1.2	ND	138	<0.024	0.23	<0.015	0.116	243	<0.029

Table 46. continued.

SAMPLE	Time days	pH	Eh mV	SC*	Cl	SO3**	SO4	Ag	Al	B	Ba	Ca	Fe
----- mg L-1 -----													
Flem-SPD(10%)													
A	1	11.9	247	2.6	20.2	#	686	0.063	0.10	0.38	0.055	411	<0.029
A	7	11.2	311	1.0	1.4	#	494	0.052	0.52	0.30	0.092	292	<0.029
A	14	10.5	394	0.6	1.3	4	249	<0.024	0.94	0.25	0.010	159	<0.029
A	26	9.8	467	0.7	<0.615	#	287	<0.024	0.71	0.15	<0.002	160	<0.029
A	62	9.2	378	1.0	<0.615	#	830	<0.024	0.33	0.33	<0.002	355	<0.029
A	132	8.4	477	1.5	<0.615	#	1385	<0.024	<0.050	0.42	<0.002	530	<0.029
B	1	11.5	245	3.0	19.6	#	966	0.061	0.47	0.35	0.067	600	<0.029
B	7	10.9	322	1.1	6.2	#	639	0.054	0.44	0.44	0.025	338	<0.029
B	14	11.0	387	0.8	<0.615	2	348	<0.024	0.44	0.38	0.015	195	<0.029
B	26	10.6	451	0.7	<0.615	2	293	<0.024	0.17	0.22	0.005	185	<0.029
B	62	9.6	368	0.8	<0.615	#	698	<0.024	0.28	0.40	<0.002	276	<0.029
B	132	8.3	477	1.5	<0.615	#	1278	<0.024	<0.050	0.40	0.004	500	0.06
A	1	12.1	216	6.6	53.1	#	774	0.060	0.23	0.61	0.534	1101	<0.029
A	7	12.5	255	3.0	7.0	#	261	0.060	0.19	0.34	0.211	238	<0.029
A	14	12.3	337	1.9	1.1	3	205	<0.024	<0.050	0.05	0.261	331	<0.029
A	26	11.9	408	1.1	<0.615	#	267	<0.024	<0.050	<0.015	0.153	237	<0.029
A	62	11.5	302	0.9	<0.615	1	484	<0.024	0.22	0.08	0.120	222	<0.029
A	132	11.1	392	1.0	1.1	3	481	<0.024	0.21	0.04	0.130	276	<0.029
B	1	12.4	212	6.0	41.5	#	1060	0.073	<0.050	0.69	0.464	1272	<0.029
B	7	12.3	260	2.3	15.1	#	330	0.025	<0.050	0.38	0.316	369	<0.029
B	14	12.1	345	1.3	9.4	#	381	<0.024	<0.050	0.13	0.157	202	0.04
B	26	11.8	410	1.3	0.9	#	50	<0.024	<0.050	<0.015	0.244	337	<0.029
B	62	11.1	312	1.5	<0.615	#	1079	<0.024	0.20	0.17	0.192	461	<0.029
B	132	10.1	428	1.5	<0.615	#	1277	<0.024	0.54	0.45	0.073	536	<0.029

Table 46. continued.

SAMPLE	Time days	pH	Eh mV	SC*	Cl	SO3**	SO4	Ag	Al	B	Ba	Ca	Fe
----- mg L-1 -----													
Flem-SPD(40%)													
A	1	12.4	284	8.0	119	#	1228	0.055	<0.050	1.46	0.983	1620	<0.029
A	7	12.7	203	6.6	11.7	#	263	0.039	0.22	0.67	1.352	1113	0.34
A	14	12.7	238	5.8	9.1	3	222	<0.024	<0.050	0.37	1.191	844	<0.029
A	26	12.8	251	5.3	<0.615	#	173	<0.024	<0.050	0.15	1.064	866	<0.029
A	62	12.5	237	5.1	1.6	#	103	<0.024	<0.050	0.27	0.973	699	<0.029
A	132	12.5	343	4.2	<0.615	4	75	<0.024	<0.050	0.09	1.123	674	<0.029
B	1	12.4	244	6.6	150	#	1238	0.079	<0.050	0.91	0.718	1251	<0.029
B	7	12.7	213	6.6	18.5	#	506	0.041	0.07	0.68	1.185	853	<0.029
B	14	12.7	273	5.6	13.6	#	197	<0.024	<0.050	0.34	1.126	757	<0.029
B	26	12.8	337	5.5	<0.615	#	173	<0.024	<0.050	0.12	1.055	865	<0.029
B	62	12.6	237	5.6	0.9	#	109	<0.024	0.06	0.23	1.042	780	<0.029
B	132	11.5	359	4.1	<0.615	4	102	<0.024	<0.050	0.07	0.911	644	<0.029
Flem-PFBC(10%)													
A	1	7.5	478	2.6	9.3	ND	2607	0.096	0.38	0.46	0.020	548	<0.029
A	7	7.8	535	1.3	2.7	ND	839	0.052	0.07	0.06	0.002	185	<0.029
A	14	8.5	541	0.6	10.6	ND	263	<0.024	<0.050	<0.015	0.002	74	<0.029
A	26	8.6	572	0.6	<0.615	ND	323	<0.024	<0.050	<0.015	<0.002	84	<0.029
A	62	8.6	528	0.4	<0.615	ND	154	<0.024	<0.050	0.14	<0.002	47	<0.029
A	132	8.1	535	0.2	1.3	ND	53	<0.024	0.16	0.03	0.025	23	1.00
B	1	7.1	508	2.6	7.8	ND	5046	0.101	0.41	0.27	0.022	399	0.03
B	7	7.7	539	1.5	2.1	ND	1420	0.056	0.07	0.12	0.004	255	<0.029
B	14	8.1	565	0.6	8.3	ND	295	<0.024	<0.050	0.03	0.003	81	<0.029
B	26	8.5	576	0.4	<0.615	ND	331	<0.024	<0.050	<0.015	<0.002	53	0.19
B	62	8.6	542	0.3	<0.615	ND	74	<0.024	<0.050	0.14	<0.002	29	<0.029
B	132	8.4	561	0.2	<0.615	ND	28	<0.024	0.05	0.05	0.009	21	0.27

Table 46. continued.

SAMPLE	Time days	pH	Eh mV	SC*	Cl	SO3**	SO4	Ag	Al	B	Ba	Ca	Fe
----- mg L-1 -----													
Flem-PFBC(20%)													
A	1	8.0	466	3.2	19.0	ND	3032	0.133	0.35	0.58	0.013	700	<0.029
A	7	9.3	443	2.3	2.6	ND	1928	0.038	0.05	0.06	0.010	585	<0.029
A	14	8.8	532	1.5	2.3	ND	1136	<0.024	<0.050	0.01	0.009	391	<0.029
A	26	8.3	604	1.2	<0.615	ND	776	<0.024	<0.050	<0.015	<0.002	273	<0.029
A	62	8.3	564	0.7	<0.615	ND	589	<0.024	<0.050	0.10	<0.002	146	<0.029
A	132	8.2	579	0.8	1.1	ND	615	<0.024	<0.050	0.02	0.003	141	<0.029
B	1	7.9	466	3.4	17.7	ND	3139	0.083	0.28	0.61	0.017	635	<0.029
B	7	8.8	480	1.3	2.9	ND	1062	<0.024	<0.050	0.07	0.012	258	<0.029
B	14	8.4	559	0.6	<0.615	ND	306	<0.024	0.06	<0.015	0.002	86	<0.029
B	26	8.4	605	0.7	<0.615	ND	421	<0.024	<0.050	0.06	0.061	109	2.17
B	62	8.2	550	1.5	<0.615	ND	1003	<0.024	<0.050	0.12	<0.002	285	<0.029
B	132	8.0	562	1.5	<0.615	ND	1022	<0.024	<0.050	0.14	0.006	276	<0.029
Flem-PFBC(40%)													
A	1	9.1	411	3.8	47.3	ND	3709	0.127	<0.050	0.79	0.024	957	<0.029
A	7	10.4	376	2.6	12.5	ND	2047	<0.024	<0.050	0.11	0.051	757	<0.029
A	14	9.9	442	2.4	10.5	ND	1865	<0.024	<0.050	0.18	0.059	765	<0.029
A	26	9.1	540	2.1	<0.615	ND	1864	0.030	<0.050	0.06	0.031	685	<0.029
A	62	7.9	476	2.4	<0.615	ND	1718	<0.024	<0.050	0.15	<0.002	632	<0.029
A	132	7.6	560	2.1	<0.615	ND	1808	<0.024	<0.050	0.09	0.014	536	<0.029
B	1	9.2	405	3.7	48.2	ND	3418	0.085	<0.050	0.44	0.022	739	<0.029
B	7	10.1	385	1.1	5.9	ND	740	<0.024	<0.050	0.04	0.027	246	<0.029
B	14	9.9	445	1.2	14.6	ND	702	<0.024	<0.050	0.12	0.032	263	<0.029
B	26	9.1	542	1.1	<0.615	ND	691	0.029	<0.050	<0.015	0.008	251	<0.029
B	62	8.0	521	1.5	<0.615	ND	967	<0.024	<0.050	0.09	<0.002	308	<0.029
B	132	7.9	563	1.6	<0.615	ND	1110	0.030	0.11	0.04	0.011	339	<0.029

Table 46. continued.

SAMPLE	Time days	pH	Eh mV	SC*	Cl	SO3**	SO4	Ag	Al	B	Ba	Ca	Fe
----- mg L-1 -----													
R&F Spoil													
A	1	5.1	549	1.5	2.7	ND	1005	0.062	0.19	<0.015	0.020	126	2.76
A	7	5.6	583	0.2	1.5	ND	81	<0.024	<0.050	<0.015	0.028	18	0.03
A	14	6.3	653	0.1	1.4	ND	41	<0.024	<0.050	0.06	0.022	12	<0.029
A	26	5.7	742	0.1	1.2	ND	49	0.035	<0.050	<0.015	0.027	12	<0.029
A	62	6.3	647	0.2	<0.615	ND	93	<0.024	<0.050	0.07	0.018	24	<0.029
A	132	6.9	636	0.3	<0.615	ND	122	<0.024	0.22	<0.015	0.031	37	<0.029
B	1	5.0	549	1.3	2.7	ND	1023	0.070	0.30	<0.015	0.024	139	2.31
B	7	6.0	571	0.1	2.1	ND	52	<0.024	<0.050	<0.015	0.023	13	<0.029
B	14	6.6	665	0.1	1.7	ND	50	<0.024	<0.050	0.06	0.030	14	<0.029
B	26	6.2	730	0.1	<0.615	ND	36	0.034	<0.050	<0.015	0.019	9	0.04
B	62	6.7	677	0.2	<0.615	ND	84	<0.024	<0.050	0.05	0.075	22	<0.029
B	132	6.8	559	0.2	<0.615	ND	120	<0.024	<0.050	0.14	0.043	33	<0.029
R&F-LIMB(10%)													
A	1	12.5	211	7.9	39.1	ND	1971	0.059	0.12	1.34	0.048	1606	<0.029
A	7	12.7	249	6.6	6.2	ND	1044	<0.024	<0.050	0.08	0.087	1218	<0.029
A	14	12.7	326	4.7	4.3	ND	306	<0.024	<0.050	0.05	0.103	659	<0.029
A	26	12.3	396	1.4	<0.615	ND	107	<0.024	0.25	<0.015	0.077	219	0.03
A	62	11.2	353	0.5	<0.615	ND	158	<0.024	0.59	0.06	0.010	98	<0.029
A	132	10.7	423	0.4	<0.615	ND	221	<0.024	0.48	0.13	0.030	114	<0.029
B	1	12.5	207	7.6	37.7	ND	1980	<0.024	<0.050	1.35	0.052	1631	<0.029
B	7	12.7	249	6.3	4.8	ND	939	<0.024	<0.050	0.08	0.086	1155	<0.029
B	14	12.6	324	4.1	<0.615	ND	281	<0.024	<0.050	0.03	0.100	611	<0.029
B	26	12.2	399	1.1	<0.615	ND	128	<0.024	0.33	<0.015	0.054	200	<0.029
B	62	11.0	363	0.5	<0.615	ND	244	<0.024	0.52	0.09	0.005	123	<0.029
B	132	10.0	449	0.6	<0.615	ND	537	<0.024	0.51	0.37	0.023	203	<0.029

Table 46. continued.

SAMPLE	Time days	pH	Eh mV	SC*	Cl	SO3**	SO4	Ag	Al	B	Ba	Ca	Fe
----- mg L-1 -----													
R&F-LIMB(20%)													
A	1	12.5	508	8.7	73.2	ND	1746	<0.024	<0.050	3.29	0.062	1651	<0.029
A	7	12.7	234	7.4	6.2	ND	1334	<0.024	0.08	0.11	0.069	1447	<0.029
A	14	12.7	299	7.1	8.8	ND	1021	0.033	<0.050	0.02	0.092	1397	<0.029
A	26	12.8	275	6.2	2.1	ND	842	0.037	0.05	<0.015	0.095	1107	0.25
A	62	12.5	152	4.6	<0.615	ND	34	<0.024	0.07	0.04	0.849	596	<0.029
A	132	11.6	380	0.8	<0.615	ND	82	<0.024	0.86	0.03	0.113	102	<0.029
B	1	12.5	492	7.9	89.1	ND	1848	0.053	0.08	2.99	0.043	1135	<0.029
B	7	12.7	237	7.3	8.0	ND	1384	<0.024	0.06	0.17	0.070	1405	<0.029
B	14	12.7	300	7.2	7.9	ND	1072	0.052	<0.050	0.02	0.075	1379	<0.029
B	26	12.8	285	6.6	<0.615	ND	914	<0.024	<0.050	<0.015	0.089	1197	<0.029
B	62	12.6	280	5.2	<0.615	ND	85	<0.024	<0.050	0.04	1.153	800	<0.029
B	132	11.5	398	0.7	<0.615	ND	64	<0.024	0.98	<0.015	0.101	88	<0.029
R&F-LIMB(40%)													
A	1	12.5	363	9.6	231	ND	2050	0.044	0.06	9.58	0.099	2087	<0.029
A	7	12.8	244	7.6	19.2	ND	1488	<0.024	<0.050	0.60	0.122	1535	<0.029
A	14	12.7	298	7.2	<0.615	ND	1154	0.051	0.09	0.02	0.117	1574	<0.029
A	26	12.8	311	6.8	<0.615	ND	1177	0.036	<0.050	<0.015	0.079	1275	0.73
A	62	12.6	286	7.4	2.8	ND	1141	<0.024	<0.050	0.04	0.075	1394	<0.029
A	132	12.6	354	6.2	<0.615	ND	610	<0.024	<0.050	<0.015	0.154	955	<0.029
B	1	12.5	343	9.5	239	ND	1852	0.090	0.19	8.73	0.086	1840	<0.029
B	7	12.8	238	7.5	17.4	ND	1486	<0.024	0.06	0.57	0.119	1536	<0.029
B	14	12.7	306	7.3	<0.615	ND	1140	0.072	0.11	0.02	0.106	1509	<0.029
B	26	12.8	321	6.7	<0.615	ND	1200	0.034	<0.050	<0.015	0.071	1300	<0.029
B	62	12.5	291	6.5	<0.615	ND	1139	<0.024	<0.050	0.04	0.063	1248	<0.029
B	132	12.6	358	5.5	<0.615	ND	654	<0.024	<0.050	0.04	0.137	970	<0.029

Table 46. continued.

SAMPLE	Time	pH	Eh	SC*	Cl	SO ₃ **	SO ₄	Ag	Al	B	Ba	Ca	Fe
	days		mV										
----- mg L-1 -----													
R&F-SPD(10%)													
A	1	12.4	369	6.4	27.0	#	936	0.032	<0.050	1.31	0.035	838	<0.029
A	7	12.3	273	1.6	2.5	3	86	<0.024	0.18	0.19	0.051	213	<0.029
A	14	12.0	329	1.0	1.4	6	17	0.042	0.36	0.05	0.059	137	<0.029
A	26	12.1	353	0.9	1.9	1	48	<0.024	0.21	<0.015	0.048	146	<0.029
A	62	11.4	370	0.4	<0.615	<1	20	<0.024	0.25	0.02	<0.002	39	<0.029
A	132	9.0	482	0.4	<0.615	#	99	<0.024	<0.050	<0.015	0.029	47	<0.029
B	1	12.4	373	6.6	23.7	#	1109	0.025	<0.050	1.20	0.030	893	<0.029
B	7	12.3	270	1.7	3.3	2	129	<0.024	0.09	0.39	0.039	276	<0.029
B	14	12.1	329	1.0	2.0	4	16	0.043	0.26	0.03	0.055	146	<0.029
B	26	12.4	348	2.0	<0.615	4	37	<0.024	0.20	<0.015	0.099	296	<0.029
B	62	11.8	350	0.6	<0.615	4	24	<0.024	0.34	0.04	0.031	110	<0.029
B	132	11.3	390	0.6	<0.615	3	135	<0.024	0.06	0.05	0.056	133	<0.029
R&F-SPD(20%)													
A	1	12.4	367	6.7	53.5	#	1263	0.050	0.06	2.76	0.046	1160	<0.029
A	7	12.7	245	6.1	4.1	4	279	<0.024	<0.050	0.85	0.097	878	<0.029
A	14	12.7	312	5.0	1.9	<1	77	0.046	0.08	0.27	0.130	587	<0.029
A	26	12.7	342	4.6	<0.615	3	59	<0.024	<0.050	0.06	0.198	660	0.06
A	62	12.6	302	5.7	<0.615	<1	32	<0.024	<0.050	0.12	0.409	758	<0.029
A	132	12.0	366	1.7	<0.615	5	86	<0.024	0.14	0.05	0.144	240	<0.029
B	1	12.4	363	6.8	57.2	#	1612	0.066	0.07	4.48	0.052	1394	<0.029
B	7	12.8	237	6.6	4.6	2	289	<0.024	0.06	1.14	0.111	949	<0.029
B	14	12.7	367	4.8	3.0	<1	53	0.044	<0.050	0.21	0.138	597	<0.029
B	26	12.8	345	4.8	<0.615	3	53	<0.024	<0.050	0.05	0.200	681	0.93
B	62	12.6	302	5.3	<0.615	<1	24	<0.024	<0.050	0.10	0.393	699	<0.029
B	132	12.5	357	4.9	<0.615	2	9	<0.024	<0.050	<0.015	0.657	633	<0.029

Table 46. continued.

SAMPLE	Time days	pH	Eh mV	SC*	Cl	SO3**	SO4	Ag	Al	B	Ba	Ca	Fe
----- mg L-1 -----													
R&F-SPD(40%)													
A	1	12.5	364	7.2	222	#	1390	0.082	0.29	5.22	0.096	1410	<0.029
A	7	12.8	234	7.2	31.6	#	756	<0.024	<0.050	3.25	0.139	1239	<0.029
A	14	12.8	308	6.1	3.1	#	193	0.044	<0.050	0.73	0.145	894	<0.029
A	26	12.7	344	4.8	<0.615	4	102	0.026	<0.050	0.14	0.158	737	<0.029
A	62	12.5	306	4.4	<0.615	<1	25	<0.024	<0.050	0.16	0.204	586	<0.029
A	132	12.4	357	3.3	<0.615	5	30	<0.024	0.41	0.08	0.237	329	<0.029
B	1	12.4	374	7.3	134	#	1606	0.034	<0.050	6.17	0.068	1323	<0.029
B	7	12.8	244	7.1	21.1	#	607	<0.024	<0.050	2.41	0.094	1166	<0.029
B	14	12.8	305	6.6	<0.615	#	227	0.041	<0.050	0.82	0.139	982	<0.029
B	26	12.8	344	6.4	<0.615	4	151	0.033	<0.050	0.31	0.199	904	<0.029
B	62	12.7	296	6.5	0.9	1	67	<0.024	<0.050	0.25	0.405	995	<0.029
B	132	12.6	360	6.0	<0.615	3	16	<0.024	<0.050	0.11	0.802	811	<0.029
R&F-PFBC(10%)													
A	1	9.5	380	2.6	11.5	ND	2048	<0.024	<0.050	<0.015	0.026	511	<0.029
A	7	10.3	367	1.6	13.4	ND	1004	<0.024	<0.050	0.20	0.039	465	<0.029
A	14	9.8	431	0.8	2.3	ND	382	0.035	<0.050	0.06	0.040	205	<0.029
A	26	7.8	738	0.6	<0.615	ND	338	0.033	<0.050	<0.015	0.029	102	<0.029
A	62	7.2	525	0.6	<0.615	ND	237	<0.024	<0.050	0.07	0.005	55	<0.029
A	132	8.6	564	0.2	<0.615	ND	48	<0.024	<0.050	0.04	0.051	11	<0.029
B	1	9.3	382	2.9	11.4	ND	936	0.050	<0.050	<0.015	0.024	526	<0.029
B	7	9.9	392	1.2	15.0	ND	752	<0.024	<0.050	0.08	0.032	343	<0.029
B	14	9.5	447	0.8	<0.615	ND	406	<0.024	<0.050	0.03	0.040	198	<0.029
B	26	8.7	596	0.6	<0.615	ND	352	0.036	<0.050	<0.015	0.016	106	<0.029
B	62	8.5	583	0.5	<0.615	ND	280	<0.024	<0.050	0.05	<0.002	74	<0.029
B	132	8.0	615	0.4	<0.615	ND	80	<0.024	<0.050	<0.015	0.015	17	<0.029

Table 46. continued.

SAMPLE	Time days	pH	Eh mV	SC*	Cl	SO ₃ **	SO ₄	Ag	Al	B	Ba	Ca	Fe
----- mg L-1 -----													
R&F-PFBC(20%)													
A	1	10.3	321	NA	19.8	ND	2078	<0.024	<0.050	0.10	0.031	579	<0.029
A	7	11.3	323	1.9	17.2	ND	1183	<0.024	<0.050	0.43	0.049	570	<0.029
A	14	10.8	394	1.4	17.0	ND	839	0.033	<0.050	0.28	0.048	426	<0.029
A	26	9.6	515	1.1	<0.615	ND	807	0.041	<0.050	<0.015	0.037	329	<0.029
A	62	8.4	578	0.8	<0.615	ND	514	<0.024	<0.050	0.08	0.007	147	<0.029
A	132	8.1	616	1.2	<0.615	ND	775	<0.024	<0.050	0.06	0.030	206	<0.029
B	1	10.3	312	3.0	20.0	ND	2386	<0.024	<0.050	0.15	0.031	685	<0.029
B	7	11.4	309	2.2	14.0	ND	1495	<0.024	<0.050	0.55	0.046	659	<0.029
B	14	11.0	399	1.6	<0.615	ND	1068	0.047	0.10	0.30	0.041	487	<0.029
B	26	9.9	503	1.2	<0.615	ND	850	0.033	<0.050	0.02	0.024	355	0.18
B	62	8.6	567	0.9	<0.615	ND	703	<0.024	<0.050	0.11	0.002	232	<0.029
B	132	8.2	618	0.9	<0.615	ND	544	<0.024	<0.050	0.05	0.025	128	<0.029
R&F-PFBC(40%)													
A	1	10.5	370	3.4	40.5	ND	2471	0.052	<0.050	1.16	0.054	849	<0.029
A	7	12.0	271	3.3	22.3	ND	1672	<0.024	<0.050	1.62	0.071	753	<0.029
A	14	11.8	341	2.7	16.5	ND	1605	0.054	0.05	1.03	0.050	727	<0.029
A	26	11.4	405	2.2	1.9	ND	1454	0.041	<0.050	0.60	0.038	668	<0.029
A	62	10.7	418	2.1	<0.615	ND	1259	<0.024	0.10	0.55	0.019	600	<0.029
A	132	9.7	523	1.8	<0.615	ND	1182	0.028	<0.050	0.32	0.042	511	<0.029
B	1	10.7	363	3.1	40.7	ND	2588	0.078	0.15	1.78	0.060	887	<0.029
B	7	12.1	267	3.3	19.7	ND	1570	<0.024	<0.050	1.54	0.076	770	<0.029
B	14	11.7	342	2.7	17.4	ND	1588	0.070	0.08	1.03	0.052	753	<0.029
B	26	11.3	405	2.3	<0.615	ND	1498	0.049	0.15	0.53	0.046	696	<0.029
B	62	10.0	448	2.2	<0.615	ND	1561	<0.024	0.10	0.31	0.021	691	<0.029
B	132	8.1	580	2.3	<0.615	ND	1773	0.038	<0.050	0.19	0.037	609	<0.029

Table 46. continued.

SAMPLE	Time	K	Li	Mg	Mn	Mo	Na	P	S	Si	Sr	Zn
	days	----- mg L-1 -----										
Fleming Spoil												
A	1	4.5	0.04	1.23	0.122	<0.018	0.4	2.77	<1.243	5.4	0.06	0.74
A	7	1.9	<0.008	1.24	0.093	0.02	0.3	0.16	16.	5.8	0.07	0.41
A	14	<0.248	<0.008	0.34	0.034	<0.018	<0.057	<0.109	6.	3.1	0.02	0.39
A	26	0.7	<0.008	0.19	0.017	0.02	<0.057	<0.109	<1.243	3.0	0.01	0.27
A	62	1.4	<0.008	0.29	0.026	<0.018	0.3	<0.109	8.	6.0	0.02	0.21
A	132	1.2	<0.008	0.47	0.033	<0.018	0.1	<0.109	9.	7.6	0.03	0.27
B	1	3.8	0.03	0.91	0.096	<0.018	0.3	3.05	<1.243	5.2	0.05	0.87
B	7	0.6	<0.008	0.70	0.043	<0.018	0.3	<0.109	10.	4.8	0.04	0.33
B	14	<0.248	<0.008	0.25	0.031	<0.018	0.1	<0.109	5.	2.6	0.02	0.57
B	26	<0.248	<0.008	0.19	0.014	<0.018	0.0	<0.109	<1.243	2.8	0.01	0.18
B	62	1.6	<0.008	0.34	0.023	<0.018	<0.057	<0.109	8.	5.4	0.02	0.26
B	132	0.8	<0.008	0.40	0.036	<0.018	0.0	<0.109	9.	7.1	0.03	0.40
Flem-LIMB(10%)												
A	1	37.2	<0.008	<0.043	<0.001	0.38	5.6	3.81	492	7.0	1.10	0.61
A	7	16.2	<0.008	<0.043	<0.001	0.13	1.6	0.12	307	3.6	0.64	0.04
A	14	6.2	<0.008	0.11	0.003	0.06	0.6	<0.109	155	2.3	0.31	0.03
A	26	6.6	<0.008	0.36	<0.001	0.04	0.5	<0.109	123	1.7	0.31	0.04
A	62	8.8	<0.008	0.97	<0.001	0.02	1.3	<0.109	283	1.8	0.40	0.14
A	132	5.9	<0.008	1.40	<0.001	<0.018	1.0	<0.109	234	2.4	0.26	0.27
B	1	34.5	<0.008	<0.043	<0.001	0.38	5.8	4.16	562	9.2	0.95	0.05
B	7	18.4	<0.008	0.18	<0.001	0.12	2.1	0.26	442	4.3	0.85	<0.006
B	14	6.7	<0.008	0.15	<0.001	0.04	0.6	0.21	193	2.4	0.39	0.09
B	26	7.2	<0.008	0.53	<0.001	0.03	0.6	<0.109	149	1.3	0.31	0.07
B	62	7.5	<0.008	2.62	<0.001	0.02	1.0	<0.109	224	2.6	0.30	0.21
B	132	4.4	<0.008	3.29	0.002	<0.018	0.6	<0.109	92	4.0	0.12	0.06

Table 46. continued.

SAMPLE	Time days	K	Li	Mg	Mn	Mo	Na	P	S	Si	Sr	Zn
----- mg L-1 -----												
Flem-LIMB(20%)												
A	1	84.8	<0.008	<0.043	<0.001	0.54	10.7	4.19	625	1.0	2.25	0.08
A	7	85.4	<0.008	<0.043	<0.001	0.41	5.4	0.35	653	1.3	3.77	0.08
A	14	30.4	<0.008	0.04	<0.001	0.13	3.1	<0.109	339	1.6	1.49	0.33
A	26	27.9	<0.008	<0.043	<0.001	0.08	3.2	<0.109	199	3.0	0.95	0.15
A	62	30.4	<0.008	<0.043	<0.001	0.04	4.4	<0.109	273	4.2	0.74	0.08
A	132	21.1	<0.008	0.20	<0.001	<0.018	4.0	<0.109	452	2.4	0.61	0.07
B	1	98.3	<0.008	<0.043	<0.001	0.63	12.2	3.07	692	8.2	2.27	0.14
B	7	38.3	<0.008	0.28	<0.001	0.26	14.3	0.31	<1.243	4.2	1.06	0.03
B	14	18.9	<0.008	0.06	0.003	0.13	2.3	0.15	116	1.9	0.62	0.05
B	26	21.2	<0.008	0.18	<0.001	0.14	2.7	<0.109	74	1.7	0.65	0.06
B	62	16.0	<0.008	<0.043	<0.001	0.06	2.0	<0.109	89	1.2	0.34	0.09
B	132	21.0	<0.008	<0.043	<0.001	0.05	3.5	<0.109	120	1.3	0.36	0.09
A	1	95.7	0.03	<0.043	<0.001	0.46	14.1	5.08	646	1.3	2.91	0.84
A	7	130.8	0.01	<0.043	<0.001	0.69	11.3	0.19	608	2.3	8.06	0.24
A	14	55.7	0.02	<0.043	<0.001	0.36	5.9	0.16	526	0.9	6.22	0.02
A	26	49.5	<0.008	<0.043	<0.001	0.16	6.8	<0.109	341	0.9	3.95	0.06
A	62	100.3	<0.008	<0.043	<0.001	0.16	17.1	<0.109	276	1.1	3.63	0.31
A	132	140.7	0.01	<0.043	<0.001	0.13	25.7	<0.109	194	1.6	3.11	0.19
B	1	91.7	0.02	<0.043	<0.001	0.48	13.0	<0.109	641	1.5	2.28	0.02
B	7	105.3	0.02	<0.043	<0.001	0.57	8.7	0.32	608	2.1	6.06	0.27
B	14	29.5	<0.008	<0.043	<0.001	0.15	3.3	<0.109	257	1.1	2.66	0.02
B	26	60.5	<0.008	<0.043	<0.001	0.21	6.6	0.13	277	0.9	4.53	0.06
B	62	115.1	0.01	<0.043	<0.001	0.15	17.4	<0.109	308	1.8	4.79	0.20
B	132	127.2	0.01	<0.043	<0.001	0.08	23.2	<0.109	65	2.2	2.03	0.28

Table 46. continued.

SAMPLE	Time days	K	Li	Mg	Mn	Mo	Na	P	S	Si	Sr	Zn
----- mg L-1 -----												
Flem-SPD(10%)												
A	1	12.6	0.10	<0.043	<0.001	0.20	5.5	3.31	276	8.4	0.45	0.05
A	7	10.2	0.03	0.23	<0.001	0.11	2.0	0.22	175	6.1	0.32	0.20
A	14	5.6	<0.008	0.59	<0.001	0.05	0.7	<0.109	293	4.0	0.18	0.05
A	26	4.6	<0.008	1.11	<0.001	0.02	0.4	<0.109	166	2.7	0.15	0.02
A	62	7.6	<0.008	4.16	<0.001	0.02	1.1	<0.109	300	2.7	0.29	0.26
A	132	6.6	<0.008	6.94	<0.001	<0.018	1.1	<0.109	471	4.3	0.35	0.13
B	1	11.7	0.02	<0.043	<0.001	0.24	3.0	4.11	534	7.1	0.47	0.06
B	7	11.6	0.03	0.56	<0.001	0.15	2.2	0.15	215	5.6	0.32	0.21
B	14	7.2	<0.008	0.38	<0.001	0.08	0.9	0.15	231	4.5	0.21	0.05
B	26	5.8	<0.008	0.36	<0.001	0.05	0.7	<0.109	204	5.1	0.18	0.05
B	62	5.7	<0.008	1.29	<0.001	0.02	0.9	<0.109	261	2.7	0.25	0.00
B	132	4.3	<0.008	11.39	<0.001	<0.018	1.0	<0.109	452	4.7	0.30	0.23
A	1	38.8	0.02	<0.043	<0.001	0.26	5.4	4.59	225	1.0	1.25	0.13
A	7	64.6	0.02	<0.043	<0.001	0.23	6.7	0.11	103	1.5	0.77	0.01
A	14	41.6	<0.008	<0.043	<0.001	0.19	3.7	<0.109	262	1.7	0.74	0.23
A	26	29.8	<0.008	<0.043	<0.001	0.16	2.6	<0.109	220	2.3	0.62	0.05
A	62	26.0	<0.008	0.15	<0.001	0.11	2.7	<0.109	270	3.7	0.59	0.13
A	132	19.5	<0.008	0.05	<0.001	0.07	2.3	<0.109	345	3.9	0.58	0.14
B	1	45.4	0.02	<0.043	<0.001	0.39	7.8	3.28	353	0.9	1.32	0.23
B	7	78.3	0.02	0.13	0.004	0.27	7.3	0.31	147	1.8	1.12	0.02
B	14	42.2	<0.008	0.05	0.036	0.17	3.5	0.12	245	2.7	0.80	0.01
B	26	29.1	<0.008	0.20	<0.001	0.13	2.2	<0.109	293	4.9	0.92	0.12
B	62	29.8	<0.008	0.37	<0.001	0.25	3.1	<0.109	433	5.0	0.94	0.06
B	132	19.1	<0.008	1.16	<0.001	0.07	2.6	<0.109	543	3.4	0.58	0.15

Table 46. continued.

SAMPLE	Time days	K	Li	Mg	Mn	Mo	Na	P	S	Si	Sr	Zn
----- mg L-1 -----												
Flem-SPD(40%)												
A	1	97.4	0.01	<0.043	<0.001	0.59	15.0	4.84	427	10.0	3.11	0.21
A	7	196.1	0.03	0.39	0.056	0.45	15.8	0.44	177	9.8	4.84	0.37
A	14	58.2	<0.008	<0.043	0.001	0.23	4.7	0.18	262	3.3	4.20	0.21
A	26	36.2	<0.008	<0.043	<0.001	0.12	3.3	<0.109	123	3.6	3.52	0.06
A	62	44.7	0.01	<0.043	<0.001	0.07	5.1	0.16	131	6.3	2.99	0.46
A	132	65.6	<0.008	<0.043	<0.001	0.04	7.5	<0.109	135	6.8	3.43	0.11
B	1	80.7	0.02	<0.043	<0.001	0.58	13.9	3.90	406	6.6	2.60	0.04
B	7	195.3	0.02	<0.043	<0.001	0.44	16.6	0.28	185	7.3	4.41	0.08
B	14	64.5	<0.008	<0.043	<0.001	0.23	4.4	0.13	236	2.7	3.90	0.03
B	26	35.1	<0.008	<0.043	<0.001	0.15	3.1	<0.109	121	2.5	3.10	0.08
B	62	43.9	<0.008	<0.043	<0.001	0.08	5.0	<0.109	127	4.2	2.74	0.22
B	132	55.9	<0.008	<0.043	<0.001	0.06	6.2	<0.109	154	5.2	2.36	0.15
Flem-PFBC(10%)												
A	1	3.1	0.06	447	0.060	<0.018	1.7	4.58	815	3.0	0.35	0.61
A	7	<0.248	0.03	126	<0.001	<0.018	0.6	0.21	268	2.9	0.06	0.44
A	14	<0.248	<0.008	43	<0.001	<0.018	0.2	<0.109	93	3.2	0.02	0.10
A	26	<0.248	<0.008	49	<0.001	<0.018	0.2	<0.109	41	3.7	0.02	0.07
A	62	<0.248	<0.008	31	<0.001	0.02	0.2	<0.109	59	4.7	0.02	0.04
A	132	<0.248	<0.008	17	<0.001	<0.018	0.1	0.12	19	5.7	0.01	0.21
B	1	1.6	0.06	336	0.074	<0.018	1.5	6.19	583	2.4	0.32	0.85
B	7	<0.248	0.03	229	0.001	<0.018	0.7	<0.109	431	3.8	0.08	0.37
B	14	<0.248	<0.008	54	<0.001	<0.018	0.3	<0.109	107	4.0	0.02	0.19
B	26	<0.248	<0.008	35	<0.001	<0.018	<0.057	<0.109	<1.243	4.3	<0.003	0.01
B	62	0.3	<0.008	21	<0.001	0.03	0.3	<0.109	25	5.4	0.01	0.02
B	132	<0.248	<0.008	16	<0.001	0.02	<0.057	0.19	12	6.8	0.01	0.15

Table 46. continued.

SAMPLE	Time days	K	Li	Mg	Mn	Mo	Na	P	S	Si	Sr	Zn
----- mg L-1 -----												
Flem-PFBC(20%)												
A	1	2.6	0.06	487	0.005	<0.018	3.6	3.96	964	3.5	0.40	0.76
A	7	9.1	0.03	166	<0.001	0.03	3.5	0.14	628	2.0	0.42	0.24
A	14	8.4	0.01	57	<0.001	0.05	2.8	<0.109	375	2.2	0.30	0.68
A	26	6.9	<0.008	46	<0.001	0.03	2.0	<0.109	206	2.2	0.20	0.13
A	62	5.1	<0.008	50	<0.001	0.01	1.7	0.11	176	2.5	0.10	0.19
A	132	5.0	<0.008	84	<0.001	0.02	2.0	0.13	210	3.4	0.09	0.83
B	1	3.8	0.06	452	0.025	<0.018	3.3	4.70	852	8.5	0.39	0.52
B	7	1.4	<0.008	106	0.004	<0.018	1.7	<0.109	328	3.3	0.14	0.37
B	14	<0.248	<0.008	42	0.004	<0.018	0.7	<0.109	107	1.9	0.04	0.30
B	26	1.4	<0.008	54	0.014	0.03	1.0	<0.109	84	2.2	0.05	0.14
B	62	4.2	<0.008	146	<0.001	<0.018	1.9	<0.109	376	4.2	0.16	0.33
B	132	3.3	0.01	149	0.002	0.03	1.5	<0.109	390	5.3	0.15	0.63
Flem-PFBC(40%)												
A	1	11.5	0.10	523	<0.001	0.04	12.4	4.89	1163	4.4	0.60	0.60
A	7	86.0	<0.008	45	<0.001	<0.018	15.2	<0.109	666	0.7	1.34	0.36
A	14	68.5	<0.008	31	0.001	<0.018	10.4	<0.109	678	1.0	1.37	0.44
A	26	44.7	<0.008	35	<0.001	0.02	6.7	<0.109	569	1.2	1.01	0.03
A	62	41.0	<0.008	106	<0.001	<0.018	7.6	<0.109	656	2.3	0.87	0.27
A	132	28.1	<0.008	145	<0.001	<0.018	5.5	0.14	624	3.3	0.59	0.33
B	1	9.9	0.08	429	<0.001	0.02	10.3	5.03	907	3.2	0.50	0.51
B	7	34.6	<0.008	16	0.002	<0.018	11.2	<0.109	218	0.8	0.41	0.08
B	14	62.7	<0.008	12	<0.001	<0.018	13.8	<0.109	258	1.0	0.58	0.11
B	26	49.3	<0.008	16	<0.001	0.02	8.2	<0.109	176	1.0	0.53	0.05
B	62	53.7	<0.008	51	<0.001	0.02	9.7	<0.109	327	1.9	0.65	0.27
B	132	45.1	<0.008	106	<0.001	<0.018	7.3	<0.109	423	2.5	0.63	0.33

Table 46. continued.

SAMPLE	Time days	K	Li	Mg	Mn	Mo	Na	P	S	Si	Sr	Zn
----- mg L-1 -----												
R&F Spoil												
A	1	14.2	0.04	63.06	5.094	<0.018	145.5	3.92	207	2.2	1.52	1.03
A	7	2.0	<0.008	5.17	0.622	<0.018	6.2	0.16	25	0.6	0.25	0.60
A	14	2.6	0.01	3.42	0.479	<0.018	2.1	<0.109	16	0.4	0.17	0.30
A	26	1.8	<0.008	3.87	0.529	<0.018	1.3	<0.109	<1.243	0.3	0.16	0.13
A	62	3.1	<0.008	7.93	0.097	<0.018	1.5	<0.109	32	0.7	0.31	0.28
A	132	2.4	<0.008	15.79	0.059	<0.018	1.5	<0.109	50	0.8	0.41	0.37
B	1	17.5	0.04	62.46	5.152	<0.018	129.7	3.71	201	2.6	1.63	0.84
B	7	1.6	<0.008	3.42	0.482	<0.018	3.4	<0.109	16	0.7	0.21	0.73
B	14	2.8	0.02	3.81	0.569	<0.018	1.6	<0.109	18	0.5	0.20	0.52
B	26	0.8	<0.008	2.48	0.373	<0.018	0.5	<0.109	<1.243	0.3	0.12	0.08
B	62	2.3	<0.008	6.59	0.972	<0.018	1.0	<0.109	28	0.6	0.29	0.37
B	132	1.3	<0.008	12.19	1.451	<0.018	1.1	<0.109	49	0.5	0.40	0.21
R&F-LIMB(10%)												
A	1	35.8	0.02	<0.043	<0.001	0.13	165.2	2.81	751	1.0	4.74	0.28
A	7	18.5	<0.008	0.11	0.001	0.07	9.3	0.43	329	2.2	3.81	0.48
A	14	10.1	0.01	0.09	<0.001	<0.018	3.5	<0.109	152	1.0	1.44	0.02
A	26	6.9	<0.008	0.06	<0.001	<0.018	3.0	<0.109	<1.243	2.0	0.49	0.07
A	62	7.3	<0.008	<0.043	<0.001	<0.018	4.0	<0.109	63	5.4	0.28	0.10
A	132	6.8	<0.008	<0.043	0.001	<0.018	2.9	<0.109	94	5.5	0.27	0.09
B	1	35.9	<0.008	<0.043	<0.001	0.12	179.9	3.13	778	0.7	4.95	0.05
B	7	18.6	<0.008	<0.043	<0.001	0.04	8.7	0.44	300	1.9	3.67	0.33
B	14	9.9	0.02	<0.043	<0.001	<0.018	3.6	<0.109	150	1.1	1.34	0.02
B	26	5.2	<0.008	<0.043	<0.001	<0.018	2.8	<0.109	<1.243	1.9	0.43	0.06
B	62	7.3	<0.008	<0.043	<0.001	0.02	3.7	<0.109	90	5.6	0.33	0.07
B	132	4.7	<0.008	0.12	0.005	<0.018	2.3	<0.109	179	4.2	0.37	0.06

Table 46. continued.

SAMPLE	Time days	K	Li	Mg	Mn	Mo	Na	P	S	Si	Sr	Zn
----- mg L-1 -----												
R&F-LIMB(20%)												
A	1	39.7	<0.008	<0.043	<0.001	0.18	171.8	3.66	901	5.7	5.61	0.19
A	7	21.7	<0.008	<0.043	<0.001	0.08	7.6	0.34	439	8.8	5.13	0.22
A	14	17.9	0.01	<0.043	<0.001	0.04	5.0	0.12	420	4.3	3.11	0.25
A	26	20.6	<0.008	<0.043	<0.001	0.02	4.7	<0.109	234	3.4	2.15	0.05
A	62	47.3	0.02	<0.043	<0.001	0.02	11.3	<0.109	16	5.0	2.16	0.04
A	132	33.8	<0.008	<0.043	<0.001	0.03	9.1	<0.109	31	5.9	0.36	0.18
B	1	38.8	0.01	<0.043	<0.001	0.18	165.0	3.24	755	0.5	4.94	0.07
B	7	23.8	<0.008	0.05	<0.001	0.09	9.3	0.50	423	0.7	5.36	0.17
B	14	17.4	0.02	<0.043	<0.001	<0.018	4.5	<0.109	442	0.3	3.05	0.05
B	26	19.3	<0.008	<0.043	<0.001	<0.018	4.8	<0.109	275	0.3	2.12	0.05
B	62	52.8	0.01	<0.043	<0.001	<0.018	13.0	<0.109	37	0.5	2.46	0.11
B	132	36.7	0.01	<0.043	0.001	<0.018	9.5	<0.109	23	3.0	0.34	0.12
R&F-LIMB(40%)												
A	1	46.9	0.01	<0.043	<0.001	0.25	230.8	5.41	1300	8.2	6.44	0.33
A	7	33.8	0.02	<0.043	<0.001	0.18	15.3	0.53	455	14.6	7.67	0.24
A	14	35.7	0.03	<0.043	<0.001	0.14	9.3	0.16	488	8.0	5.78	0.19
A	26	41.7	0.02	<0.043	<0.001	0.09	8.8	<0.109	359	4.2	3.92	0.04
A	62	116.7	0.06	<0.043	<0.001	0.06	24.2	<0.109	470	5.6	5.71	0.05
A	132	101.1	0.10	<0.043	<0.001	0.03	20.8	<0.109	212	5.4	3.15	0.40
B	1	46.1	0.03	<0.043	<0.001	0.25	213.9	4.30	1182	8.9	6.25	0.23
B	7	41.9	0.01	<0.043	<0.001	0.22	14.6	0.39	449	15.3	9.08	0.40
B	14	34.3	0.02	<0.043	<0.001	0.12	8.6	0.12	450	6.9	5.23	0.15
B	26	42.3	0.02	<0.043	<0.001	0.07	8.8	<0.109	371	4.1	3.93	0.13
B	62	123.7	0.06	<0.043	<0.001	0.05	25.4	0.13	485	5.6	5.66	0.10
B	132	128.0	0.12	<0.043	<0.001	<0.018	24.3	<0.109	231	7.7	4.02	0.18

Table 46. continued.

SAMPLE	Time days	K	Li	Mg	Mn	Mo	Na	P	S	Si	Sr	Zn
----- mg L-1 -----												
R&F-SPD(10%)												
A	1	27.8	<0.008	<0.043	<0.001	0.07	173.3	3.49	625	4.3	2.76	0.07
A	7	7.5	<0.008	0.10	0.003	<0.018	4.4	<0.109	36	4.5	0.81	0.06
A	14	6.3	0.02	0.09	<0.001	<0.018	2.8	<0.109	62	3.4	0.65	0.18
A	26	7.7	<0.008	<0.043	<0.001	<0.018	2.6	<0.109	104	4.3	0.78	0.13
A	62	9.5	<0.008	<0.043	<0.001	<0.018	2.4	<0.109	43	2.2	0.31	0.04
A	132	2.9	0.01	0.50	0.003	<0.018	2.1	<0.109	46	2.4	0.22	0.13
B	1	28.4	<0.008	<0.043	<0.001	0.05	175.5	3.94	660	0.5	2.95	0.05
B	7	9.3	<0.008	<0.043	0.002	<0.018	6.4	<0.109	58	1.3	1.24	0.12
B	14	5.7	0.02	0.07	<0.001	<0.018	2.4	<0.109	117	1.1	0.68	0.60
B	26	10.7	<0.008	<0.043	<0.001	0.03	2.8	<0.109	60	1.2	1.33	0.05
B	62	5.8	<0.008	<0.043	<0.001	0.04	1.9	<0.109	111	1.5	0.49	0.08
B	132	9.6	<0.008	<0.043	<0.001	<0.018	3.3	<0.109	208	5.5	0.51	0.15
R&F-SPD(20%)												
A	1	40.5	<0.008	<0.043	<0.001	0.11	177.6	2.84	554	5.4	3.88	0.06
A	7	27.4	<0.008	<0.043	<0.001	0.06	10.6	0.21	107	10.3	3.90	0.26
A	14	11.8	0.02	<0.043	<0.001	<0.018	2.9	<0.109	77	3.5	2.05	0.02
A	26	12.1	<0.008	<0.043	<0.001	0.02	2.0	<0.109	55	2.9	1.78	0.06
A	62	20.9	<0.008	<0.043	<0.001	0.02	3.4	<0.109	100	7.2	1.67	0.06
A	132	17.8	<0.008	<0.043	<0.001	<0.018	3.9	<0.109	133	6.2	0.58	0.19
B	1	54.3	0.01	<0.043	<0.001	0.17	175.9	3.76	626	6.2	5.44	0.07
B	7	28.0	0.01	<0.043	<0.001	0.07	9.5	0.22	114	8.0	4.53	0.39
B	14	8.3	0.02	<0.043	<0.001	<0.018	2.6	0.18	103	2.4	1.47	0.04
B	26	10.7	<0.008	<0.043	0.001	<0.018	2.4	<0.109	8	2.8	1.52	0.12
B	62	23.3	0.01	<0.043	<0.001	<0.018	4.0	<0.109	102	5.1	2.02	0.04
B	132	30.0	<0.008	<0.043	<0.001	<0.018	3.9	0.15	89	5.2	2.00	0.20

Table 46. continued.

SAMPLE	Time days	K	Li	Mg	Mn	Mo	Na	P	S	Si	Sr	Zn
----- mg L-1 -----												
R&F-SPD(40%)												
A	1	83.9	0.04	<0.043	<0.001	0.23	194.5	5.39	510	7.2	4.79	0.22
A	7	49.8	0.02	<0.043	<0.001	0.15	18.5	0.14	308	9.7	6.16	0.15
A	14	15.1	0.03	<0.043	<0.001	0.04	3.4	0.17	179	2.7	2.74	0.11
A	26	16.2	0.01	<0.043	<0.001	0.03	4.0	<0.109	53	3.1	2.10	0.02
A	62	40.6	0.02	<0.043	<0.001	<0.018	7.2	<0.109	98	3.6	2.64	0.04
A	132	39.4	0.01	<0.043	<0.001	<0.018	5.5	<0.109	115	4.1	1.99	0.06
B	1	79.5	0.01	<0.043	<0.001	0.26	187.0	3.06	529	9.3	5.87	0.06
B	7	42.1	0.01	<0.043	<0.001	0.13	11.5	0.38	241	11.6	6.64	0.17
B	14	20.1	0.03	<0.043	<0.001	0.04	4.2	0.23	178	3.9	4.16	0.18
B	26	20.1	<0.008	<0.043	<0.001	0.04	3.5	<0.109	85	3.2	3.22	0.06
B	62	45.2	0.02	<0.043	<0.001	0.02	7.1	<0.109	152	4.5	3.79	0.09
B	132	57.6	0.04	<0.043	<0.001	<0.018	6.5	<0.109	79	4.2	3.93	0.25
R&F-PFBC(10%)												
A	1	26.6	0.01	103.01	0.048	<0.018	134.9	2.84	591	2.0	2.76	0.48
A	7	23.6	0.02	6.11	0.001	<0.018	12.7	0.39	341	2.4	1.60	0.13
A	14	6.4	0.03	5.57	<0.001	<0.018	2.4	0.51	146	1.4	0.60	0.38
A	26	4.7	0.02	23.68	<0.001	<0.018	1.3	<0.109	35	1.4	0.38	0.05
A	62	6.9	0.02	40.84	0.001	<0.018	1.5	<0.109	84	2.7	0.33	0.13
A	132	1.5	0.02	18.19	0.001	<0.018	0.6	0.14	15	2.3	0.14	0.31
B	1	28.0	0.02	135.58	0.062	<0.018	151.8	4.85	634	2.7	2.97	0.55
B	7	15.6	0.01	11.29	0.002	<0.018	8.1	<0.109	283	1.8	1.28	0.27
B	14	8.2	0.03	7.82	<0.001	<0.018	2.4	<0.109	163	1.3	0.63	0.33
B	26	5.2	0.01	23.44	<0.001	<0.018	1.1	<0.109	37	1.3	0.35	0.11
B	62	4.7	0.02	34.33	0.003	<0.018	1.1	<0.109	96	1.9	0.33	0.09
B	132	1.2	<0.008	19.71	0.003	<0.018	0.6	<0.109	28	1.8	0.13	0.12

Table 46. continued.

SAMPLE	Time days	K	Li	Mg	Mn	Mo	Na	P	S	Si	Sr	Zn
----- mg L-1 -----												
R&F-PFBC(20%)												
A	1	30.3	0.03	85.35	0.004	0.04	145.0	3.19	597	2.0	2.34	0.42
A	7	49.4	0.03	1.58	<0.001	<0.018	17.1	0.18	434	2.5	1.76	0.11
A	14	31.0	0.03	1.19	<0.001	<0.018	6.3	0.13	304	3.3	1.15	0.18
A	26	20.0	0.03	4.96	<0.001	<0.018	3.7	<0.109	208	2.3	0.79	0.07
A	62	14.8	0.03	21.15	<0.001	<0.018	3.8	0.15	145	1.6	0.47	0.06
A	132	18.6	0.05	74.30	<0.001	<0.018	3.4	<0.109	277	2.5	0.70	0.28
B	1	32.6	0.04	94.45	0.002	0.03	157.9	2.32	693	1.9	2.87	0.34
B	7	52.7	0.03	2.26	<0.001	<0.018	19.4	0.39	456	3.0	2.02	0.11
B	14	25.1	0.04	0.98	<0.001	<0.018	5.7	0.11	356	4.1	1.11	0.08
B	26	16.8	0.02	4.36	<0.001	<0.018	3.7	0.18	229	2.8	0.78	0.19
B	62	17.4	0.04	26.38	<0.001	<0.018	4.1	<0.109	221	2.4	0.63	0.12
B	132	13.0	0.05	53.68	<0.001	<0.018	2.6	<0.109	180	2.6	0.45	0.36
R&F-PFBC(40%)												
A	1	47.5	0.08	19.71	<0.001	0.10	165.9	2.51	753	1.2	2.46	0.13
A	7	97.9	0.03	0.05	<0.001	<0.018	26.4	0.51	520	1.9	2.30	0.03
A	14	70.1	0.06	0.66	<0.001	<0.018	15.1	0.27	528	2.7	1.66	0.04
A	26	60.4	0.04	4.22	<0.001	<0.018	11.0	<0.109	477	4.8	1.31	0.07
A	62	69.3	0.05	3.58	<0.001	<0.018	13.5	<0.109	494	6.9	1.17	0.03
A	132	63.9	0.08	6.13	<0.001	<0.018	10.2	<0.109	474	5.3	1.06	0.12
B	1	58.3	0.11	70.23	<0.001	0.13	191.1	3.85	915	2.7	3.03	0.06
B	7	99.9	0.04	<0.043	<0.001	<0.018	25.0	0.12	513	5.2	2.48	0.02
B	14	67.6	0.08	0.21	<0.001	<0.018	13.8	<0.109	557	4.9	1.67	0.02
B	26	59.4	0.08	0.61	<0.001	<0.018	10.4	<0.109	501	6.6	1.24	0.06
B	62	81.0	0.15	9.68	<0.001	<0.018	13.9	<0.109	572	4.7	1.37	0.02
B	132	64.1	0.19	79.85	<0.001	<0.018	11.2	<0.109	655	2.4	1.27	0.25

Table 47. Leachate concentrations of elements regulated[†] by Ohio EPA.

SAMPLE	Time	As	Cd	Co	Cr	Cu	Ni	Pb	Se
	days	----- mg L-1 -----							
Fleming Spoil									
A	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
A	7	NA	<0.003	<0.005	<0.002	<0.002	0.011	0.059	<0.052
A	14	NA	<0.003	<0.005	0.006	0.004	0.006	<0.020	<0.052
A	26	NA	<0.003	<0.005	0.002	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	0.003	<0.002	<0.005	<0.020	<0.052
A	132	0.052	<0.003	<0.005	0.002	<0.002	0.014	<0.020	0.058
B	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	7	NA	<0.003	<0.005	<0.002	<0.002	0.014	<0.020	0.084
B	14	NA	<0.003	<0.005	0.002	<0.002	0.009	<0.020	<0.052
B	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	132	0.096	<0.003	<0.005	0.008	<0.002	0.011	<0.020	<0.052
Flem-LIMB(10%)									
A	1	NA	<0.003	<0.005	0.003	0.059	<0.005	<0.020	<0.052
A	7	NA	<0.003	<0.005	0.006	<0.002	0.006	0.039	0.277
A	14	NA	<0.003	<0.005	0.009	<0.002	<0.005	<0.020	0.155
A	26	NA	<0.003	<0.005	0.009	<0.002	0.007	<0.020	<0.052
A	62	NA	<0.003	<0.005	0.010	<0.002	<0.005	<0.020	<0.052
A	132	<0.027	<0.003	<0.005	0.003	<0.002	<0.005	<0.020	<0.052
B	1	NA	<0.003	<0.005	0.002	0.058	<0.005	<0.020	<0.052
B	7	NA	<0.003	<0.005	0.003	<0.002	0.012	<0.020	0.063
B	14	NA	<0.003	<0.005	0.003	<0.002	<0.005	<0.020	<0.052
B	26	NA	<0.003	<0.005	0.007	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	0.006	<0.002	<0.005	<0.020	0.138
B	132	0.030	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052

Table 47. continued.

SAMPLE	Time	As	Cd	Co	Cr	Cu	Ni	Pb	Se
	days								
-----mg L-1-----									
Flem-LIMB(20%)									
A	1	NA	<0.003	<0.005	<0.002	0.038	<0.005	<0.020	0.148
A	7	NA	<0.003	<0.005	0.002	0.018	0.034	<0.020	0.099
A	14	NA	<0.003	0.017	0.008	<0.002	<0.005	<0.020	0.108
A	26	NA	<0.003	<0.005	0.005	<0.002	<0.005	<0.020	0.090
A	62	NA	<0.003	<0.005	0.023	<0.002	<0.005	<0.020	<0.052
A	132	<0.027	<0.003	<0.005	0.031	<0.002	0.009	<0.020	<0.052
B	1	NA	<0.003	<0.005	0.003	0.045	<0.005	<0.020	0.113
B	7	NA	<0.003	<0.005	0.012	<0.002	0.010	0.216	<0.052
B	14	NA	<0.003	0.009	<0.002	<0.002	<0.005	<0.020	0.089
B	26	NA	<0.003	<0.005	0.004	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	0.005	0.002	<0.005	<0.020	0.096
B	132	<0.027	<0.003	<0.005	<0.002	<0.002	0.010	<0.020	<0.052
Flem-LIMB(40%)									
A	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.216
A	7	NA	<0.003	0.015	<0.002	<0.002	0.043	0.041	0.124
A	14	NA	<0.003	0.018	0.007	0.011	0.014	<0.020	0.409
A	26	NA	<0.003	<0.005	<0.002	<0.002	0.010	<0.020	<0.052
A	62	NA	<0.003	<0.005	0.002	<0.002	0.048	<0.020	<0.052
A	132	<0.027	<0.003	<0.005	<0.002	<0.002	0.050	<0.020	<0.052
B	1	NA	<0.003	<0.005	<0.002	0.005	<0.005	<0.020	0.175
B	7	NA	<0.003	0.019	<0.002	<0.002	0.041	<0.020	<0.052
B	14	NA	<0.003	0.012	0.005	<0.002	<0.005	<0.020	0.131
B	26	NA	<0.003	<0.005	0.003	<0.002	0.011	<0.020	<0.052
B	62	NA	<0.003	<0.005	0.010	0.004	0.064	<0.020	0.091
B	132	<0.027	<0.003	<0.005	<0.002	0.004	0.030	<0.020	<0.052

Table 47. continued.

SAMPLE	Time	As	Cd	Co	Cr	Cu	Ni	Pb	Se
	days	----- mg L-1 -----							
Flem-SPD(10%)									
A	1	NA	<0.003	<0.005	<0.002	0.079	<0.005	<0.020	0.192
A	7	NA	<0.003	0.007	<0.002	<0.002	0.012	<0.020	<0.052
A	14	NA	<0.003	0.023	<0.002	<0.002	<0.005	<0.020	<0.052
A	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	<0.002	0.047	<0.005	<0.020	<0.052
A	132	<0.027	<0.003	<0.005	<0.002	<0.002	0.014	<0.020	<0.052
B	1	NA	<0.003	<0.005	0.004	0.033	<0.005	<0.020	0.212
B	7	NA	<0.003	0.005	<0.002	<0.002	<0.005	<0.020	0.083
B	14	NA	<0.003	<0.005	0.007	0.002	<0.005	0.052	<0.052
B	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.066
B	132	<0.027	<0.003	<0.005	<0.002	<0.002	0.012	<0.020	0.243
Flem-SPD(20%)									
A	1	NA	<0.003	<0.005	<0.002	0.041	<0.005	<0.020	0.110
A	7	NA	<0.003	<0.005	<0.002	<0.002	0.018	0.023	0.067
A	14	NA	<0.003	<0.005	0.003	0.024	<0.005	<0.020	0.218
A	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	0.003	0.005	<0.005	<0.020	<0.052
A	132	<0.027	<0.003	<0.005	<0.002	0.002	<0.005	<0.020	0.052
B	1	NA	<0.003	<0.005	0.004	0.056	<0.005	<0.020	<0.052
B	7	NA	<0.003	0.017	0.007	0.025	<0.005	<0.020	0.065
B	14	NA	<0.003	0.026	0.010	0.015	0.062	0.026	<0.052
B	26	NA	<0.003	<0.005	0.003	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	0.010	<0.002	<0.005	<0.020	<0.052
B	132	<0.027	<0.003	<0.005	0.003	<0.002	0.035	<0.020	<0.052

Table 47. continued.

SAMPLE	Time	As	Cd	Co	Cr	Cu	Ni	Pb	Se
	days	----- mg L-1 -----							
Flem-SPD(40%)									
A	1	NA	<0.003	<0.005	0.007	0.099	<0.005	<0.020	0.189
A	7	NA	<0.003	0.026	0.005	0.040	0.036	0.050	0.159
A	14	NA	<0.003	0.008	0.003	0.016	0.012	<0.020	<0.052
A	26	NA	<0.003	<0.005	0.002	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	0.005	0.008	<0.005	<0.020	0.153
A	132	<0.027	<0.003	<0.005	<0.002	0.037	0.032	<0.020	<0.052
B	1	NA	<0.003	<0.005	0.008	0.121	<0.005	<0.020	0.168
B	7	NA	<0.003	0.019	0.003	0.030	0.023	<0.020	0.069
B	14	NA	<0.003	0.017	<0.002	0.016	0.011	<0.020	0.140
B	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	0.005	0.003	<0.005	<0.020	<0.052
B	132	<0.027	<0.003	<0.005	<0.002	0.004	0.034	<0.020	<0.052
Flem-PFBC(10%)									
A	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
A	7	NA	<0.003	0.019	0.003	<0.002	<0.005	<0.020	0.149
A	14	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
A	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
A	132	<0.027	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.189
B	7	NA	<0.003	0.035	<0.002	<0.002	<0.005	<0.020	<0.052
B	14	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.143
B	132	<0.027	<0.003	<0.005	<0.002	<0.002	0.006	<0.020	<0.052

Table 47. continued.

SAMPLE	Time	As	Cd	Co	Cr	Cu	Ni	Pb	Se
	days	----- mg L-1 -----							
Flem-PFBC(20%)									
A	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.061
A	7	NA	<0.003	0.018	0.010	0.002	<0.005	<0.020	<0.052
A	14	NA	<0.003	0.009	<0.002	<0.002	<0.005	<0.020	0.150
A	26	NA	<0.003	<0.005	0.002	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	0.003	<0.002	<0.005	<0.020	<0.052
A	132	<0.027	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	1	NA	<0.003	<0.005	0.003	<0.002	<0.005	<0.020	0.152
B	7	NA	0.003	0.013	0.002	0.006	<0.005	<0.020	0.076
B	14	NA	<0.003	0.008	0.002	<0.002	<0.005	<0.020	0.052
B	26	NA	<0.003	<0.005	0.004	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	0.002	<0.002	<0.005	<0.020	<0.052
B	132	0.042	0.005	<0.005	0.002	<0.002	0.007	<0.020	<0.052
Flem-PFBC(40%)									
A	1	NA	<0.003	<0.005	0.013	<0.002	<0.005	<0.020	0.109
A	7	NA	<0.003	0.005	0.005	0.003	<0.005	<0.020	0.087
A	14	NA	<0.003	<0.005	0.005	<0.002	<0.005	<0.020	0.209
A	26	NA	<0.003	<0.005	0.004	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	0.003	<0.002	<0.005	<0.020	0.184
A	132	<0.027	<0.003	<0.005	<0.002	<0.002	0.010	<0.020	<0.052
B	1	NA	<0.003	<0.005	0.009	<0.002	<0.005	<0.020	<0.052
B	7	NA	<0.003	<0.005	<0.002	0.008	0.007	<0.020	<0.052
B	14	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	0.005	<0.002	<0.005	<0.020	<0.052
B	132	<0.027	<0.003	<0.005	0.004	<0.002	<0.005	0.021	<0.052

Table 47. continued.

SAMPLE	Time	As	Cd	Co	Cr	Cu	Ni	Pb	Se
	days	----- mg L-1 -----							
R&F Spoil									
A	1	NA	<0.003	0.076	<0.002	<0.002	0.125	<0.020	<0.052
A	7	NA	<0.003	0.012	<0.002	<0.002	<0.005	<0.020	<0.052
A	14	NA	<0.003	<0.005	<0.002	<0.002	0.029	<0.020	0.085
A	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	0.005	<0.002	<0.005	0.020	<0.052
A	132	<0.027	<0.003	<0.005	0.010	<0.002	0.044	<0.020	0.064
B	1	NA	<0.003	0.065	<0.002	<0.002	0.111	<0.020	0.054
B	7	NA	<0.003	0.018	<0.002	<0.002	0.013	<0.020	<0.052
B	14	NA	<0.003	<0.005	<0.002	<0.002	0.035	<0.020	0.130
B	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	132	<0.027	<0.003	<0.005	<0.002	0.003	0.010	0.061	<0.052
R&F-LIMB(10%)									
A	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.146
A	7	NA	0.003	0.005	0.007	0.005	0.013	<0.020	0.186
A	14	NA	<0.003	0.010	0.007	<0.002	0.022	<0.020	0.140
A	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	0.002	<0.002	<0.005	<0.020	0.052
A	132	<0.027	<0.003	<0.005	0.002	<0.002	<0.005	0.026	0.068
B	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.283
B	7	NA	<0.003	<0.005	0.006	0.003	<0.005	<0.020	<0.052
B	14	NA	<0.003	0.005	0.002	<0.002	0.013	<0.020	<0.052
B	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.081
B	62	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	132	0.030	<0.003	<0.005	0.008	<0.002	0.009	<0.020	0.069

Table 47. continued.

SAMPLE	Time	As	Cd	Co	Cr	Cu	Ni	Pb	Se
	days	----- mg L-1 -----							
R&F-LIMB(20%)									
A	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.217
A	7	NA	<0.003	0.009	0.007	0.004	0.011	<0.020	0.183
A	14	NA	<0.003	<0.005	<0.002	<0.002	0.012	<0.020	<0.052
A	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	0.003	<0.002	<0.005	<0.020	<0.052
A	132	<0.027	<0.003	<0.005	<0.002	<0.002	0.012	<0.020	0.157
B	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.271
B	7	NA	<0.003	<0.005	0.006	<0.002	0.022	0.031	0.143
B	14	NA	<0.003	<0.005	0.002	<0.002	0.024	<0.020	0.069
B	26	NA	<0.003	<0.005	0.002	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	132	<0.027	<0.003	<0.005	<0.002	<0.002	0.005	<0.020	0.102
R&F-LIMB(40%)									
A	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.373
A	7	NA	<0.003	<0.005	0.016	<0.002	0.012	<0.020	0.289
A	14	NA	<0.003	<0.005	0.012	<0.002	0.017	<0.020	0.089
A	26	NA	<0.003	<0.005	0.007	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	0.004	<0.002	<0.005	<0.020	<0.052
A	132	<0.027	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.181
B	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.308
B	7	NA	<0.003	<0.005	0.019	0.002	0.021	<0.020	<0.052
B	14	NA	<0.003	0.021	0.004	<0.002	0.008	<0.020	<0.052
B	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.098
B	62	NA	<0.003	0.007	0.004	<0.002	<0.005	<0.020	<0.052
B	132	<0.027	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052

Table 47. continued.

SAMPLE	Time	As	Cd	Co	Cr	Cu	Ni	Pb	Se
	days	----- mg L-1 -----							
R&F-SPD(10%)									
A	1	NA	<0.003	<0.005	0.002	<0.002	<0.005	<0.020	0.421
A	7	NA	0.004	0.016	<0.002	0.003	<0.005	<0.020	0.101
A	14	NA	<0.003	0.006	0.002	<0.002	0.005	<0.020	<0.052
A	26	NA	<0.003	0.010	<0.002	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	<0.002	<0.002	<0.005	0.038	<0.052
A	132	<0.027	<0.003	<0.005	<0.002	<0.002	0.017	<0.020	<0.052
B	1	NA	<0.003	<0.005	0.002	<0.002	<0.005	<0.020	0.080
B	7	NA	<0.003	<0.005	<0.002	0.004	<0.005	<0.020	0.095
B	14	NA	<0.003	<0.005	<0.002	<0.002	0.019	<0.020	0.107
B	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	132	<0.027	<0.003	<0.005	<0.002	<0.002	0.008	<0.020	0.080
R&F-SPD(20%)									
A	1	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.053
A	7	NA	<0.003	<0.005	0.005	0.002	0.005	<0.020	<0.052
A	14	NA	<0.003	0.022	<0.002	<0.002	0.009	<0.020	<0.052
A	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.119
A	62	NA	<0.003	<0.005	0.007	<0.002	<0.005	<0.020	<0.052
A	132	<0.027	0.005	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	1	NA	<0.003	<0.005	0.005	<0.002	<0.005	<0.020	0.263
B	7	NA	<0.003	<0.005	0.005	0.005	0.006	<0.020	0.287
B	14	NA	<0.003	0.007	<0.002	<0.002	0.017	<0.020	<0.052
B	26	NA	<0.003	<0.005	0.003	<0.002	0.005	<0.020	0.117
B	62	NA	<0.003	0.010	0.004	<0.002	<0.005	<0.020	<0.052
B	132	<0.027	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052

Table 47. continued.

SAMPLE	Time	As	Cd	Co	Cr	Cu	Ni	Pb	Se
	days	----- mg L-1 -----							
R&F-SPD(40%)									
A	1	NA	<0.003	0.017	0.030	<0.002	<0.005	0.054	0.399
A	7	NA	<0.003	0.009	0.009	<0.002	<0.005	<0.020	0.152
A	14	NA	<0.003	0.013	<0.002	<0.002	0.013	<0.020	<0.052
A	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.130
A	132	<0.027	<0.003	<0.005	<0.002	<0.002	<0.005	0.026	<0.052
B	1	NA	<0.003	<0.005	0.009	<0.002	<0.005	<0.020	<0.052
B	7	NA	<0.003	<0.005	0.005	<0.002	0.012	<0.020	<0.052
B	14	NA	<0.003	<0.005	0.003	<0.002	0.030	<0.020	0.113
B	26	NA	<0.003	0.005	0.004	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	0.003	<0.002	<0.005	<0.020	0.070
B	132	<0.027	<0.003	<0.005	0.002	<0.002	<0.005	<0.020	<0.052
R&F-PFBC(10%)									
A	1	NA	<0.003	<0.005	0.004	<0.002	<0.005	<0.020	0.239
A	7	NA	0.003	0.006	0.005	<0.002	0.009	<0.020	0.061
A	14	NA	<0.003	0.013	<0.002	<0.002	0.011	<0.020	0.122
A	26	NA	<0.003	<0.005	0.003	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
A	132	<0.027	<0.003	0.013	<0.002	<0.002	<0.005	<0.020	<0.052
B	1	NA	<0.003	<0.005	0.007	<0.002	<0.005	<0.020	0.226
B	7	NA	<0.003	<0.005	0.004	<0.002	<0.005	<0.020	<0.052
B	14	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	26	NA	<0.003	<0.005	0.003	<0.002	<0.005	<0.020	<0.052
B	62	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052
B	132	<0.027	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.133

Table 47. continued.

SAMPLE	Time	As	Cd	Co	Cr	Cu	Ni	Pb	Se
	days								
-----mg L-1-----									
R&F-PFBC(20%)									
A	1	NA	<0.003	<0.005	0.009	<0.002	<0.005	<0.020	0.082
A	7	NA	<0.003	<0.005	0.005	<0.002	0.019	<0.020	0.075
A	14	NA	<0.003	<0.005	<0.002	<0.002	0.009	<0.020	0.140
A	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.058
A	62	NA	<0.003	<0.005	0.005	<0.002	<0.005	<0.020	<0.052
A	132	<0.027	<0.003	<0.005	<0.002	<0.002	<0.005	0.058	0.124
B	1	NA	<0.003	<0.005	0.014	<0.002	<0.005	<0.020	0.096
B	7	NA	<0.003	<0.005	0.007	<0.002	<0.005	<0.020	<0.052
B	14	NA	<0.003	0.017	0.002	<0.002	<0.005	<0.020	0.109
B	26	NA	<0.003	<0.005	<0.002	<0.002	<0.005	<0.020	0.096
B	62	NA	<0.003	<0.005	0.005	<0.002	<0.005	<0.020	0.121
B	132	<0.027	<0.003	<0.005	0.003	<0.002	<0.005	0.027	<0.052
R&F-PFBC(40%)									
A	1	NA	<0.003	<0.005	0.027	<0.002	<0.005	<0.020	0.164
A	7	NA	<0.003	<0.005	0.004	<0.002	<0.005	<0.020	0.115
A	14	NA	<0.003	<0.005	0.009	<0.002	<0.005	<0.020	<0.052
A	26	NA	<0.003	0.008	0.006	<0.002	<0.005	<0.020	<0.052
A	62	NA	<0.003	<0.005	0.003	<0.002	<0.005	<0.020	<0.052
A	132	<0.027	<0.003	<0.005	<0.002	<0.002	<0.005	0.023	<0.052
B	1	NA	<0.003	<0.005	0.034	<0.002	<0.005	<0.020	0.269
B	7	NA	<0.003	0.006	0.007	<0.002	<0.005	<0.020	0.265
B	14	NA	<0.003	0.014	0.002	<0.002	<0.005	<0.020	0.143
B	26	NA	<0.003	<0.005	0.005	<0.002	<0.005	<0.020	0.312
B	62	NA	<0.003	<0.005	0.005	<0.002	<0.005	<0.020	0.112
B	132	<0.027	0.003	<0.005	<0.002	<0.002	<0.005	<0.020	<0.052

Figure 90. X-ray diffraction pattern of Fleming-SPD (40wt. %) after 132 days of reaction.

Figure 91. X-ray diffraction pattern of R&F-LIMB (40 wt.%) after 132 days of reaction.

Figure 92. X-ray diffraction pattern of R&F-PFBC (40 wt.%) after 132 days of reaction.

6.5 Supplementary Data from Greenhouse Studies

Table 48. Composition of the first leachates collected in the Fleming spoil greenhouse experiment.

Soil	FGD	Amount	pH	EC	Al	As	B	Ba	Ca	Cd	Cl	Co
		wgt %		mmho/cm								
									mg kg ⁻¹			
Clay	—	0	4.73	3.77	20.61	<0.03	0.53	3.60	397	<0.005	263	0.13
Clay	PFBC	3	7.41	6.80	1.08	<0.03	0.35	3.35	475	<0.005	168	0.02
Clay	PFBC	6	7.28	6.96	0.57	<0.03	0.33	<0.02	467	<0.005	184	0.01
Clay	PFBC	12	7.69	6.68	0.70	<0.03	0.36	2.63	396	<0.005	150	0.01
Clay	PFBC	24	9.04	9.50	1.66	<0.03	0.40	0.27	530	<0.005	923	0.06
Clay	LIMB	3	8.18	4.24	7.27	<0.03	1.05	0.02	781	<0.005	503	0.15
Clay	LIMB	6	10.72	6.24	1.33	<0.03	0.21	0.40	1696	<0.005	1116	0.43
Clay	LIMB	12	11.87	12.54	0.99	0.04	0.15	0.84	3472	<0.005	2545	0.68
Clay	LIMB	24	12.03	18.10	1.01	0.03	0.15	0.98	5160	<0.005	3896	0.86
Spoil	—	0	3.76	3.17	49.61	0.06	0.78	10.14	289	<0.005	340	0.03
Spoil	PFBC	3	7.34	6.00	0.50	<0.03	0.56	<0.02	505	<0.005	258	0.01
Spoil	PFBC	6	7.28	12.66	1.17	<0.03	0.61	0.04	568	<0.005	317	0.01
Spoil	PFBC	12	6.99	17.43	0.55	<0.03	0.60	0.53	533	<0.005	914	0.01
Spoil	PFBC	24	7.85	13.86	0.89	0.09	0.48	1.70	579	<0.005	905	0.01
Spoil	LIMB	3	6.97	4.87	1.19	<0.03	0.80	0.14	1080	<0.005	716	<0.01
Spoil	LIMB	6	8.20	6.22	0.90	<0.03	0.39	3.91	1624	<0.005	1134	0.02
Spoil	LIMB	12	9.81	9.75	1.52	0.29	0.17	0.50	2694	<0.005	2938	0.09
Spoil	LIMB	24	11.97	16.52	0.68	0.29	0.38	1.79	4424	<0.005	4162	0.16
LSD _{0.05}			1.11	2.52	14.23	0.26	0.51	4.86	641	—	1129	0.14
No Sewage Sludge Treatments												
Spoil	—	0	3.07	3.61	134.50	<0.03	0.27	0.14	105	<0.005	860	0.04
Spoil	PFBC	12	7.29	8.76	0.20	<0.03	0.24	<0.02	487	<0.005	626	<0.01
Spoil	LIMB	12	11.11	10.30	0.66	<0.03	0.09	0.34	2264	<0.005	3050	<0.01
LSD _{0.05}			0.74	2.49	23.18	—	0.15	6.59	445	—	1004	0.11

Table 48. continued.

Soil	FGD	Amount	Cr	Cu	F	Fe	K	Li	Mg	Mn	Mo
		wgt %	mg L ⁻¹								
Clay	--	0	0.01	0.17	2.3	0.63	66.7	0.12	46	3.63	0.05
Clay	PFBC	3	<0.01	<0.01	2.2	0.13	58.2	<0.03	787	1.19	0.08
Clay	PFBC	6	<0.01	<0.01	5.0	0.12	57.9	<0.03	768	0.55	0.20
Clay	PFBC	12	0.01	0.09	3.1	0.09	64.6	<0.03	754	0.15	0.31
Clay	PFBC	24	0.02	4.74	30.5	0.09	147.5	<0.03	1243	0.12	0.42
Clay	LIMB	3	0.02	7.06	20.1	0.31	77.9	<0.03	17.27	0.79	0.96
Clay	LIMB	6	0.02	36.07	51.5	0.04	143.0	<0.03	2.55	<0.05	1.94
Clay	LIMB	12	0.02	40.05	103.6	0.04	318.4	<0.03	1.64	0.05	1.72
Clay	LIMB	24	0.02	57.87	121.6	0.03	389.9	<0.03	1.31	<0.05	1.23
Spoil	--	0	0.02	0.09	0.5	1.63	38.9	0.23	56	4.88	<0.01
Spoil	PFBC	3	<0.01	0.11	1.9	0.16	57.4	<0.03	688	1.51	<0.01
Spoil	PFBC	6	<0.01	0.02	11.0	0.20	39.9	<0.03	2888	0.97	<0.01
Spoil	PFBC	12	<0.01	0.03	20.1	0.14	60.0	<0.03	4519	0.48	0.05
Spoil	PFBC	24	<0.01	0.11	23.0	0.08	78.8	<0.03	3139	0.12	0.19
Spoil	LIMB	3	<0.01	<0.01	1.0	0.09	48.8	<0.03	40.94	0.23	<0.01
Spoil	LIMB	6	0.01	1.93	30.4	0.02	65.2	<0.03	9.46	<0.05	0.35
Spoil	LIMB	12	0.02	10.27	111.0	0.06	145.5	<0.03	1.55	0.07	1.26
Spoil	LIMB	24	<0.01	14.03	187.7	0.03	362.3	<0.03	<0.05	<0.05	1.42
LSD _{0.05}			--	8.90	18.8	0.43	47.4	--	500	1.31	0.36
No Sewage Sludge Treatments											
Spoil		0	0.02	<0.01	<0.1	4.30	112.2	0.43	28.78	2.12	<0.01
Spoil	PFBC	12	<0.01	<0.01	0.9	0.01	56.5	<0.03	1539	<0.05	0.01
Spoil	LIMB	12	<0.01	0.13	97.0	0.01	180.4	<0.03	2.17	<0.05	0.94
LSD _{0.05}			--	2.75	22.5	0.50	36.3	--	397	1.11	0.15

Table 48 continued.

Soil	FGD	Amount	Na	Ni	P	Pb	S	Se	Si	Sr	Zn
		wgt %	mg L ⁻¹								
Clay	—	0	54.58	0.91	5.49	<0.09	666	<0.16	17.80	2.58	3.64
Clay	PFBC	3	57.15	0.17	2.84	<0.09	1545	0.23	6.94	5.29	0.09
Clay	PFBC	6	69.59	0.25	3.16	<0.09	1601	0.18	5.18	2.50	0.13
Clay	PFBC	12	70.87	0.37	2.29	<0.09	1543	0.17	5.60	2.10	0.10
Clay	PFBC	24	210.24	1.77	2.34	<0.09	2090	0.41	9.75	2.39	0.33
Clay	LIMB	3	50.71	1.94	1.57	<0.09	708	0.26	21.03	2.89	0.59
Clay	LIMB	6	79.23	5.88	3.97	<0.09	691	0.50	13.96	5.87	0.34
Clay	LIMB	12	117.94	9.26	4.81	<0.09	729	0.69	16.93	14.28	0.28
Clay	LIMB	24	139.07	8.96	2.53	<0.09	786	0.92	13.21	21.87	0.14
Spoil	—	0	53.71	0.65	2.32	<0.09	561	<0.16	40.54	2.46	6.31
Spoil	PFBC	3	44.46	0.32	2.78	<0.09	1428	<0.16	6.95	1.79	0.19
Spoil	PFBC	6	54.01	0.15	1.84	<0.09	4047	0.21	7.12	1.48	0.13
Spoil	PFBC	12	73.86	0.32	2.32	<0.09	5942	0.38	7.57	1.41	0.09
Spoil	PFBC	24	94.61	0.47	1.99	<0.09	4128	0.35	8.00	1.45	0.08
Spoil	LIMB	3	37.92	0.09	0.97	<0.09	847	<0.16	7.84	1.60	0.08
Spoil	LIMB	6	38.81	0.73	1.12	<0.09	997	<0.16	7.07	2.21	0.09
Spoil	LIMB	12	54.39	2.91	2.63	<0.09	711	0.17	16.73	5.78	0.10
Spoil	LIMB	24	75.90	3.77	1.61	<0.09	686	0.19	13.84	12.07	0.10
LSD _{0.05}			33.3	1.42	2.06	—	582	0.19	9.45	2.63	1.58
No Sewage Sludge Treatments											
Spoil		0	14.95	0.41	1.53	<0.09	77	<0.16	26.72	1.46	2.01
Spoil	PFBC	12	13.24	0.03	0.97	<0.09	2025	<0.16	1.00	0.39	0.09
Spoil	LIMB	12	33.40	0.08	0.13	<0.09	515	<0.16	1.24	4.30	0.04
LSD _{0.05}			12.8	0.25	1.43	—	364	—	9.65	1.09	1.92

Table 49. Composition of the second leachates collected in the Fleming greenhouse experiment.

Soil	FGD	Amount	pH	EC	Al	As	B	Ba	Ca	Cd	Co
		wgt %		dS m ⁻¹	mg L ⁻¹						
Clay	—	0	4.25	1.16	2.78	<0.08	0.56	0.02	141	0.008	0.03
Clay	PFBC	3	7.52	5.33	0.12	<0.08	0.59	0.01	446	<0.003	0.02
Clay	PFBC	6	7.56	8.50	0.23	<0.08	0.95	0.02	480	0.004	0.04
Clay	PFBC	12	7.79	12.98	0.25	<0.08	1.36	0.01	485	<0.003	0.04
Clay	PFBC	24	7.74	23.82	0.36	0.20	1.21	0.04	551	<0.003	0.09
Clay	LIMB	3	7.36	4.22	0.17	<0.08	2.01	0.01	939	<0.003	0.03
Clay	LIMB	6	7.31	8.39	0.21	<0.08	4.63	0.08	1723	<0.003	0.09
Clay	LIMB	12	9.65	7.06	0.39	<0.08	0.54	0.33	1844	<0.003	0.58
Clay	LIMB	24	11.64	10.02	0.24	<0.08	0.30	0.33	2581	<0.003	0.51
Spoil	—	0	3.95	1.10	3.48	<0.08	0.46	0.01	89	<0.003	0.01
Spoil	PFBC	3	7.24	3.53	0.14	<0.08	0.44	0.01	340	<0.003	0.01
Spoil	PFBC	6	7.33	5.18	0.13	<0.08	0.49	0.01	404	<0.003	0.02
Spoil	PFBC	12	7.78	8.50	0.20	<0.08	0.85	0.01	462	<0.003	0.03
Spoil	PFBC	24	7.74	28.28	0.40	<0.08	0.89	0.01	464	<0.003	0.07
Spoil	LIMB	3	7.14	2.17	0.10	<0.08	0.74	0.00	578	<0.003	<0.01
Spoil	LIMB	6	7.41	2.67	0.08	<0.08	1.49	0.01	723	<0.003	<0.01
Spoil	LIMB	12	7.35	4.69	0.13	<0.08	4.16	0.04	1177	<0.003	0.03
Spoil	LIMB	24	9.28	6.93	0.43	<0.08	0.27	0.07	1814	<0.003	0.23
LSD _{0.05}			0.32	2.66	0.46	—	0.79	0.06	415	—	0.09
No Sewage Sludge Treatments											
Spoil	—	0	2.92	1.95	45.13	<0.08	0.18	0.03	63	<0.003	0.02
Spoil	PFBC	12	7.91	6.94	0.51	<0.08	0.67	0.01	450	0.004	0.02
Spoil	LIMB	12	8.00	3.72	0.52	<0.08	0.83	0.06	1012	0.017	0.03
LSD _{0.05}			0.28	1.71	9.19	—	0.93	0.02	170	—	0.03

Table 49. continued.

Soil	FGD	Amount	Cr	Cu	Fe	Hg	K	Li	Mg	Mn	Mo
		wgt %	mg L ⁻¹								
Clay	--	0	0.010	0.41	0.58	<0.04	5.0	0.093	143	4.59	0.02
Clay	PFBC	3	0.016	0.20	0.04	<0.04	10.7	0.016	685	0.68	0.12
Clay	PFBC	6	0.025	0.21	0.04	<0.04	7.0	0.017	1422	0.79	0.34
Clay	PFBC	12	0.016	0.27	0.03	<0.04	16.6	0.029	3006	0.29	0.79
Clay	PFBC	24	0.041	0.56	0.07	<0.04	230.1	0.151	6491	0.50	1.09
Clay	LIMB	3	0.009	0.18	0.03	<0.04	9.5	0.010	87	0.25	0.04
Clay	LIMB	6	0.005	0.49	0.04	<0.04	84.5	<0.006	114	0.35	0.05
Clay	LIMB	12	0.069	31.4	0.08	<0.04	372.8	0.015	11	0.04	1.94
Clay	LIMB	24	0.022	42.0	0.02	<0.04	749.0	0.008	6	0.02	1.21
Spoil	--	0	0.009	0.20	0.11	<0.04	3.7	0.048	148	2.69	0.01
Spoil	PFBC	3	0.014	0.27	0.04	<0.04	3.8	0.008	446	0.17	0.01
Spoil	PFBC	6	0.011	0.10	0.04	<0.04	1.2	<0.006	728	0.06	0.00
Spoil	PFBC	12	0.021	0.17	0.03	<0.04	3.5	<0.006	1556	0.08	0.09
Spoil	PFBC	24	0.043	0.35	0.10	<0.04	9.0	<0.006	8906	0.18	0.39
Spoil	LIMB	3	0.005	0.04	0.02	<0.04	0.9	0.009	67	0.05	0.00
Spoil	LIMB	6	<0.004	0.05	0.02	<0.04	1.5	0.006	65	0.03	0.00
Spoil	LIMB	12	0.008	0.24	0.03	<0.04	30.3	0.007	86	0.06	0.02
Spoil	LIMB	24	0.146	8.01	0.02	<0.04	495.9	0.009	13	0.02	3.42
LSD _{0.05}			0.039	4.85	0.35	--	53.2		687	0.98	0.16
No Sewage Sludge Treatments											
Spoil	--	0	0.008	0.08	1.10	<0.04	80.8	0.221	26	0.94	0.01
Spoil	PFBC	12	0.013	0.11	0.02	<0.04	9.9	<0.006	1617	0.01	0.07
Spoil	LIMB	12	0.110	0.06	0.04	<0.04	217.4	0.016	16	0.03	0.81
LSD _{0.05}			0.036	0.17	0.28	--	23.1		371		0.18

Table 49. continued.

Soil	FGD	Amount	Na	Ni	S	Se	P	Pb	Si	Sr	Zn
		wgt %	mg L ⁻¹								
Clay	—	0	9.0	0.56	363	<0.27	0.83	<0.04	13.16	1.87	5.68
Clay	PFBC	3	16.4	0.11	1113	0.33	1.31	<0.04	8.90	1.95	0.20
Clay	PFBC	6	21.2	0.13	2006	0.51	1.58	<0.04	8.70	2.21	0.24
Clay	PFBC	12	35.0	0.12	3901	0.58	3.80	<0.04	6.07	2.50	0.12
Clay	PFBC	24	143.0	0.20	7191	0.76	14.97	<0.04	6.89	3.64	0.08
Clay	LIMB	3	27.3	0.09	488	<0.27	0.53	<0.04	7.05	2.85	0.18
Clay	LIMB	6	48.3	0.11	520	0.30	0.94	<0.04	9.83	4.53	0.14
Clay	LIMB	12	137.1	4.04	521	0.31	3.88	<0.04	16.39	12.29	0.14
Clay	LIMB	24	178.1	5.84	435	<0.27	1.49	<0.04	19.31	26.56	0.08
Spoil	—	0	4.5	0.29	346	<0.27	0.40	<0.04	13.81	1.14	3.72
Spoil	PFBC	3	3.7	0.08	889	<0.27	0.25	<0.04	7.18	1.55	0.18
Spoil	PFBC	6	2.8	0.07	1296	<0.27	0.26	<0.04	6.06	1.34	0.17
Spoil	PFBC	12	4.8	0.09	2296	<0.27	1.05	<0.04	6.91	2.16	0.09
Spoil	PFBC	24	11.5	0.21	10475	0.86	5.49	<0.04	5.80	2.05	0.11
Spoil	LIMB	3	3.0	0.04	518	<0.27	0.18	<0.04	7.54	0.94	0.13
Spoil	LIMB	6	4.2	0.04	574	<0.27	0.29	<0.04	6.66	1.25	0.10
Spoil	LIMB	12	36.1	0.07	697	0.38	0.58	<0.04	7.83	2.33	0.07
Spoil	LIMB	24	127.8	1.38	941	0.35	1.15	<0.04	8.19	6.15	0.05
LSD _{0.05}			26.4	0.66	697	0.27	2.15	—	2.92	2.38	1.14
No Sewage Sludge Treatments											
Spoil	—	0	6.4	0.13	72	<0.27	<0.09	<0.04	22.03	0.65	0.92
Spoil	PFBC	12	5.8	0.01	2202	0.37	0.32	0.04	5.42	0.27	0.09
Spoil	LIMB	12	35.0	0.02	795	<0.27	<0.09	<0.04	5.91	1.96	0.08
LSD _{0.05}			13.2		545	0.22	0.22	—	3.89		0.87

Table 50. Composition of the third leachates collected from the greenhouse Fleming experiment.

Soil	FGD	Amount	pH	EC	TDC	DOC	Al	As	B	Ba	Ca	Cd	Cu	Fe
		wgt %		dS m ⁻¹	mg L ⁻¹									
Clay		0	3.65	1.56	52	39	2.89	<0.08	0.53	0.012	278	0.004	0.25	20.3
Clay	PFBC	3	7.33	4.81	138	84	<0.05	<0.08	0.74	0.015	486	0.009	0.13	0.01
Clay	PFBC	6	7.60	4.99	159	94	<0.05	<0.08	1.03	0.005	454	<0.003	0.15	0.01
Clay	PFBC	12	7.79	9.20	272	132	<0.05	<0.08	1.60	0.005	448	<0.003	0.19	0.01
Clay	PFBC	24	7.78	21.50	497	323	<0.05	0.21	1.30	0.020	465	<0.003	0.46	0.04
Clay	LIMB	3	7.50	2.37	92	47	<0.05	<0.08	1.41	<0.001	656	<0.003	0.06	0.07
Clay	LIMB	6	7.63	2.06	129	77	<0.05	<0.08	2.68	0.035	553	0.020	0.19	0.04
Clay	LIMB	12	8.54	6.39	3255	2245	0.17	0.22	2.63	0.068	1599	<0.003	11.1	0.11
Clay	LIMB	24	10.6	6.23	5773	4493	<0.05	<0.08	0.63	0.462	1157	<0.003	26.1	<0.01
Spoil		0	3.72	0.96	21	14	3.04	<0.08	0.41	0.008	83	<0.003	0.04	2.48
Spoil	PFBC	3	6.87	3.83	99	74	<0.05	<0.08	0.55	0.003	461	<0.003	0.16	0.08
Spoil	PFBC	6	7.47	4.21	154	96	<0.05	<0.08	0.57	<0.001	477	<0.003	0.08	<0.01
Spoil	PFBC	12	7.63	6.34	298	130	<0.05	<0.08	0.92	<0.001	452	<0.003	0.06	<0.01
Spoil	PFBC	24	7.73	21.97	992	632	0.09	<0.08	0.96	0.011	425	0.008	0.21	0.07
Spoil	LIMB	3	7.26	2.39	95	57	<0.05	<0.08	0.84	0.021	705	0.007	0.02	0.14
Spoil	LIMB	6	7.62	2.35	104	53	<0.05	<0.08	1.02	0.004	692	<0.003	0.01	<0.01
Spoil	LIMB	12	7.56	2.88	238	151	<0.05	<0.08	3.73	0.012	771	<0.003	0.03	<0.01
Spoil	LIMB	24	8.52	5.44	1125	1048	0.07	0.14	0.61	0.059	1278	0.004	5.55	0.02
LSD _{0.05}			0.52	1.77	329	353	1.1	0.13	1.43	0.095	202	—	2.58	6.44
No Sewage Sludge Treatments														
Spoil			2.81	2.24	7	7	50.2	<0.08	0.24	0.032	50	<0.003	0.06	1.14
Spoil	PFBC	12	7.98	6.36	113	55	<0.05	<0.08	0.69	<0.001	391	<0.003	<0.01	<0.01
Spoil	LIMB	12	7.86	2.92	67	52	<0.05	<0.08	1.93	0.022	696	<0.003	0.01	<0.01
LSD _{0.05}			0.29	1.16	63	62	15.7	—	0.71	0.01	49	—	0.03	2.83

Table 50. continued.

Soil	FGD	Amount	K	Mg	Mn	Na	Ni	P	Pb	S	Se	Si	Sr	Zn
		wgt %	mg L ⁻¹											
Clay	—	0	5.53	55	8.281	7.96	0.963	0.35	<0.04	373	<0.27	16.95	1.21	4.30
Clay	PFBC	3	6.94	787	0.297	22.40	0.070	1.09	<0.04	1173	<0.27	8.35	0.79	0.16
Clay	PFBC	6	0.85	1138	0.087	17.37	0.056	1.13	<0.04	1628	<0.27	7.36	0.80	0.20
Clay	PFBC	12	1.68	2227	0.117	29.27	0.064	3.07	<0.04	2759	0.32	6.01	0.83	0.09
Clay	PFBC	24	106	6421	0.318	80.59	0.141	13.06	<0.04	6760	0.62	5.97	1.45	0.15
Clay	LIMB	3	0.85	52	0.044	13.7	0.013	0.14	<0.04	540	<0.27	5.91	1.14	0.17
Clay	LIMB	6	7.40	41	0.042	10.5	0.050	0.19	<0.04	411	<0.27	7.36	1.08	0.14
Clay	LIMB	12	316	10	0.021	98.1	1.401	2.21	<0.04	988	0.38	6.93	5.81	0.08
Clay	LIMB	24	586	3	0.002	149.	3.957	0.73	<0.04	205	<0.27	13.30	17.84	0.06
Spoil	—	0	1.81	35	2.528	2.55	0.243	0.13	<0.04	169	<0.27	16.83	0.35	3.34
Spoil	PFBC	3	3.04	560	0.063	6.53	0.033	0.12	<0.04	1062	<0.27	7.44	0.25	0.20
Spoil	PFBC	6	0.60	674	0.038	7.85	0.017	0.11	<0.04	1170	<0.27	5.75	0.21	0.16
Spoil	PFBC	12	2.32	1299	0.056	5.74	0.012	0.45	<0.04	1761	<0.27	6.48	0.30	0.06
Spoil	PFBC	24	1.04	7088	0.098	11.5	0.110	4.11	<0.04	7486	0.54	6.03	0.53	0.12
Spoil	LIMB	3	0.85	33	0.103	5.65	0.015	<0.09	<0.04	557	<0.27	7.62	0.30	0.10
Spoil	LIMB	6	<0.23	23	0.008	2.97	<0.009	0.10	<0.04	524	<0.27	5.62	0.47	0.03
Spoil	LIMB	12	8.35	58	0.010	15.1	0.016	0.18	<0.04	631	<0.27	6.64	0.92	0.05
Spoil	LIMB	24	369	4	<0.001	103	0.853	1.15	<0.04	970	0.28	5.85	4.40	0.07
LSD _{0.05}			48	550	2.212	16.8	0.453	1.13		554	0.14	2.38	1.25	1.86
No Sewage Sludge Treatments														
Spoil	—		88.5	16	0.868	6.10	0.110	<0.09	<0.04	47	<0.27	22.14	0.50	0.74
Spoil	PFBC	12	4.42	1268	<0.001	4.94	<0.009	0.30	<0.04	1903	<0.27	5.26	0.14	0.15
Spoil	LIMB	12	148	11	0.005	27.6	<0.009	0.09	<0.04	648	<0.27	4.64	1.17	0.13
LSD _{0.05}			11	302	0.810	5.82	0.860	0.17		364		3.34	0.16	1.03

Table 51. Fescue tissue composition from the sixth harvest of Fleming experiment.

Soil	EGD	Amount	N	P	K	S	Ca	Mg	Al	As	B
		Wgt %	-----%-----						-----mg/kg-----		
Clay	--	0	3.52	0.565	0.754	0.617	0.640	0.289	92.9	3.31	36.8
Clay	PFBC	3	2.57	0.484	2.919	0.710	0.492	0.774	52.7	0.91	21.7
Clay	PFBC	6	3.27	0.397	3.320	0.666	0.394	0.805	42.1	0.64	31.3
Clay	PFBC	12	3.29	0.357	3.050	0.586	0.363	0.784	29.9	0.44	34.2
Clay	PFBC	24	3.52	0.260	3.018	0.593	0.321	0.808	19.8	0.51	31.7
Clay	LIMB	3	3.59	0.377	3.102	0.541	0.831	0.403	36.0	0.52	57.0
Clay	LIMB	6	3.23	0.280	3.550	0.555	0.875	0.430	43.3	0.66	114.3
Clay	LIMB	12	3.70	0.174	3.575	0.583	0.901	0.294	26.4	0.90	123.5
Clay	LIMB	24	2.55	0.141	3.350	0.385	0.921	0.287	17.7	0.46	136.9
Spoil	--	0	3.62	0.542	0.484	0.575	0.475	0.264	112.6	1.27	27.6
Spoil	PFBC	3	2.81	0.424	2.548	0.779	0.556	0.806	68.2	0.20	18.8
Spoil	PFBC	6	2.63	0.354	2.803	0.650	0.440	0.717	20.6	0.14	18.6
Spoil	PFBC	12	2.71	0.360	2.552	0.554	0.362	0.845	32.4	0.32	28.1
Spoil	PFBC	24	2.85	0.293	2.326	0.591	0.490	0.748	21.9	0.26	31.6
Spoil	LIMB	3	2.63	0.430	3.084	0.696	0.884	0.386	21.2	0.51	22.2
Spoil	LIMB	6	2.71	0.352	2.860	0.609	1.099	0.409	41.9	0.14	48.1
Spoil	LIMB	12	2.83	0.300	3.096	0.704	1.100	0.408	25.3	0.25	88.4
Spoil	LIMB	24	2.54	0.161	3.100	0.562	0.987	0.320	25.0	0.23	122.8
LSD _{0.05}				0.070	0.439	0.163	0.111	0.084	66.0	0.73	18.6
No Sewage Sludge Treatments											
Spoil	PFBC	12	1.84	0.239	3.390	0.349	0.294	0.633	26.9	0.61	15.3
Spoil	LIMB	12	2.89	0.140	3.818	0.274	0.718	0.297	34.8	0.50	42.8
LSD _{0.05}				0.044	0.292	0.120	0.100	0.071	47.3	0.73	8.5

Table 51. continued.

Soll	FGD	Amount	Ba	Be	Cd	Co	Cr	Cu	Fe	Mn	Mo
		Wgt %	mg/kg								
Clay	--	0	2.73	0.05	0.94	1.51	0.58	32.36	75.8	448	1.6
Clay	PFBC	3	1.88	0.00	0.91	0.20	0.39	9.23	57.9	363	12.9
Clay	PFBC	6	0.72	0.01	0.71	0.30	0.41	12.47	60.8	188	7.2
Clay	PFBC	12	0.97	0.00	0.64	0.17	0.36	10.79	53.6	189	9.3
Clay	PFBC	24	2.45	0.01	0.29	0.28	0.38	11.07	53.1	114	4.9
Clay	LIMB	3	1.96	0.03	0.47	0.33	0.34	9.46	54.9	267	9.0
Clay	LIMB	6	3.24	0.02	0.39	0.26	0.42	11.40	61.8	258	8.4
Clay	LIMB	12	3.69	0.00	0.20	0.23	0.38	12.54	66.2	171	12.3
Clay	LIMB	24	1.92	0.01	0.23	0.19	0.25	9.63	56.2	252	21.2
Spoil	--	0	1.36	0.01	0.53	0.61	0.57	18.45	97.7	298	0.6
Spoil	PFBC	3	5.38	0.00	0.62	0.12	0.43	7.44	76.4	435	9.4
Spoil	PFBC	6	2.30	0.00	0.45	0.09	0.28	5.83	48.8	97	12.0
Spoil	PFBC	12	1.01	0.01	0.46	0.07	0.38	8.65	62.3	108	7.4
Spoil	PFBC	24	12.06	0.00	0.29	0.13	0.37	6.77	53.2	178	7.3
Spoil	LIMB	3	4.57	0.00	0.55	0.05	0.35	6.11	55.6	295	12.0
Spoil	LIMB	6	2.57	0.01	0.37	0.08	0.39	7.15	63.2	144	11.2
Spoil	LIMB	12	7.78	0.00	0.24	0.07	0.41	7.66	55.3	180	10.9
Spoil	LIMB	24	8.68	0.01	0.14	0.01	0.45	7.74	67.3	161	16.6
LSD _{0.05}			3.10		0.34	0.22	0.12	3.42	29.1	116	22.2
No Sewage Sludge Treatments											
Spoil	PFBC	12	1.22	0.01	0.10	0.17	0.43	4.12	55.8	26	7.5
Spoil	LIMB	12	2.99	0.01	0.12	0.11	0.37	5.78	72.2	54	12.7
LSD _{0.05}			2.95		0.17	0.17	0.11	2.05	24.3	58	2.9

Table 51. continued.

Soil	FGD	Amount	Na	Ni	Pb	Sb	Se	Sr	Zn
		Wgt %	mg/kg						
Clay	--	0	2139	59.60	1.14	1.32	3.18	38.75	383.1
Clay	PFBC	3	161	3.07	0.63	0.24	3.51	11.35	93.2
Clay	PFBC	6	185	2.89	1.07	0.57	2.50	7.46	93.9
Clay	PFBC	12	74	1.92	1.00	0.17	1.93	8.80	54.8
Clay	PFBC	24	128	1.45	0.50	0.85	1.42	7.17	23.5
Clay	LIMB	3	328	1.82	0.38	0.79	5.37	17.66	50.6
Clay	LIMB	6	435	1.65	0.66	0.57	3.48	20.00	47.2
Clay	LIMB	12	204	1.45	1.19	0.74	6.26	18.28	30.9
Clay	LIMB	24	661	2.01	0.47	0.49	1.24	20.42	33.7
Spill	--	0	3036	30.43	0.92	0.21	1.23	21.85	264.1
Spill	PFBC	3	197	2.77	0.68	0.12	1.22	8.58	55.9
Spill	PFBC	6	133	1.60	0.47	0.00	0.61	4.55	41.4
Spill	PFBC	12	208	1.50	0.84	0.25	0.49	3.76	49.7
Spill	PFBC	24	110	1.83	0.28	0.24	0.00	6.02	31.0
Spill	LIMB	3	114	2.16	0.49	0.20	0.97	8.44	47.0
Spill	LIMB	6	241	3.01	0.61	0.23	0.40	9.51	34.5
Spill	LIMB	12	231	1.54	0.70	0.08	2.02	13.81	30.0
Spill	LIMB	24	164	1.75	0.20	0.07	2.78	15.60	27.7
LSD _{0.05}			541	4.83	ns	0.75	1.95	4.62	44.5
No Sewage Sludge Treatments									
Spill	PFBC	12	86	0.56	0.07	0.42	0.54	1.64	17.6
Spill	LIMB	12	146	0.70	0.36	0.33	1.57	9.52	16.0
LSD _{0.05}			ns	1.72	ns	0.50	1.48	3.20	31.6

Table 52. Leachate composition of first leachates from R&F spoil amended with FGD by-products.

FGD	Rate	pH	EC	Al	B	Ba	Be	Ca	Cd	Co	Cr	Cu	Fe	K	Li
	wgt %	mg/L													
LIMB	0	6.51	15.18	0.44	1.84	1.22	<0.002	464	0.006	1.36	<0.01	0.02	0.92	99	0.36
LIMB	4	11.94	15.35	0.47	0.67	0.05	<0.002	653	<0.005	<0.01	0.01	<0.01	<0.01	60	<0.03
LIMB	8	12.00	18.33	0.22	1.24	0.79	<0.002	867	<0.005	<0.01	<0.01	<0.01	<0.01	80	<0.03
LIMB	16	11.98	20.05	0.16	0.45	0.08	<0.002	785	<0.005	<0.01	<0.01	<0.01	<0.01	108	<0.03
LIMB	32	12.14	17.65	0.15	0.21	0.10	<0.002	641	<0.005	<0.01	<0.01	0.03	<0.01	143	<0.03
PFBC	0	6.68	12.60	0.66	1.58	0.24	<0.002	426	0.007	1.07	<0.01	<0.01	0.21	77	0.28
PFBC	4	8.51	11.45	0.91	0.30	0.16	<0.002	498	<0.005	<0.01	0.01	<0.01	0.03	72	0.04
PFBC	8	8.70	12.50	0.38	0.31	0.03	<0.002	498	<0.005	<0.01	0.04	<0.01	0.02	83	0.09
PFBC	16	8.64	13.10	0.52	0.43	0.04	<0.002	542	<0.005	<0.01	0.02	<0.01	0.03	112	0.11
PFBC	32	9.51	13.18	0.54	1.08	0.72	<0.002	546	<0.005	<0.01	<0.01	<0.01	0.02	151	0.11
LSD _{0.05}		0.71	1.90	0.68	0.65	1.09	ns	203	ns	0.13	0.02	ns	0.46	19	0.06

FGD	Rate	Mg	Mn	Mo	Na	Ni	P	Pb	S	Sb	Se	Si	Sr	V	Zn
	wgt %	mg/L													
LIMB	0	663	58.6	<0.01	3790	2.05	<0.1	<0.09	3119	<0.06	3.16	9.63	13.48	<0.03	2.07
LIMB	4	0.98	0.09	1.18	2835	0.03	<0.1	<0.09	1137	<0.06	1.86	1.23	6.57	<0.03	0.03
LIMB	8	<0.05	<0.05	1.16	3508	0.01	<0.1	<0.09	1135	<0.06	1.67	0.26	9.38	<0.03	0.03
LIMB	16	<0.05	<0.05	1.26	4130	0.05	<0.1	<0.09	1145	<0.06	1.49	0.39	12.64	<0.03	0.03
LIMB	32	<0.05	<0.05	1.39	3901	0.02	<0.1	<0.09	1135	<0.06	1.20	0.38	13.67	<0.03	0.04
PFBC	0	488	45.2	<0.01	2935	1.70	<0.1	<0.09	2632	<0.06	2.25	8.96	11.35	<0.03	1.66
PFBC	4	473	0.11	0.06	2550	<0.01	<0.1	<0.09	2458	<0.06	1.87	2.06	6.52	<0.03	0.05
PFBC	8	334	<0.05	0.07	2892	0.01	<0.1	<0.09	2352	<0.06	2.17	1.49	4.50	<0.03	0.06
PFBC	16	168	<0.05	0.06	3294	<0.01	<0.1	<0.09	2142	<0.06	1.88	2.44	3.42	<0.03	0.03
PFBC	32	26	<0.05	0.10	3444	<0.01	<0.1	<0.09	2045	<0.06	1.86	4.47	2.66	<0.03	0.03
LSD _{0.05}		107	5.47	0.18	607	0.21	ns	ns	377	ns	0.50	2.28	1.43	ns	0.17

Table 53. Composition of leachates from R&F spoil 6 months after amendment with FGD by-products.

FGD	Rate	pH	EC	Organic C	Al	As	B	Ba	Ca	Cd	Cr	Cu	Fe
	wgt %		mmho/cm	mg/L									
LIMB	0	5.93	9.56	8.33	2.57	<0.08	1.40	0.02	404	0.046	0.02	0.04	1.37
LIMB	4	7.57	6.29	85.7	<0.05	<0.08	1.34	0.02	674	0.013	0.02	0.04	<0.01
LIMB	8	9.46	5.91	169	0.57	<0.08	0.55	0.02	225	0.013	0.04	0.06	<0.01
LIMB	16	9.67	5.92	278	5.87	<0.08	4.16	0.01	2.37	0.010	0.02	0.03	<0.01
LIMB	32	9.76	15.9	255	1.04	0.09	2.67	0.01	2.93	0.010	0.01	0.23	0.02
PFBC	0	6.28	10.6	11.8	<0.05	<0.08	1.53	0.02	419	0.032	0.02	0.02	0.06
PFBC	4	7.92	20.2	73.5	<0.05	<0.08	1.00	0.01	438	0.007	0.02	0.01	<0.01
PFBC	8	7.94	24.7	96.7	<0.05	<0.08	0.91	0.02	443	0.007	0.03	0.01	<0.01
PFBC	16	8.07	29.5	80.5	<0.05	<0.08	0.79	0.02	447	0.005	0.04	<0.01	<0.01
PFBC	32	7.77	8.82	36.4	<0.05	<0.08	1.35	0.03	442	<0.003	0.01	<0.01	<0.01
LSD _{0.05}		0.77	3.23	24.6	2.51	ns	1.63	0.01	64.5	0.02	0.02	0.1	1.21

FGD	Rate	K	Mg	Mn	Mo	Na	Ni	P	Pb	S	Se	Si	Sr	Zn
	wgt %	mg/L												
LIMB	0	52	1180	145	<0.02	1612	2.45	<0.09	0.12	2985	0.94	4.68	8.56	2.67
LIMB	4	66	7	0.10	0.58	1306	<0.01	<0.09	<0.04	1318	1.17	6.63	4.96	0.02
LIMB	8	127	3	0.03	1.50	1342	0.01	<0.09	<0.04	719	0.65	12.1	2.81	<0.01
LIMB	16	272	3	0.01	0.88	1456	<0.01	0.10	<0.04	335	<0.266	67.3	0.09	0.02
LIMB	32	1631	3	0.00	0.96	4572	0.03	1.82	0.06	2028	0.55	33.0	0.06	0.01
PFBC	0	56	1220	135	<0.02	1816	1.77	<0.09	<0.04	3260	1.33	4.28	8.76	1.33
PFBC	4	145	4738	1.92	0.09	2642	0.02	0.10	<0.04	7546	2.17	3.65	8.18	0.04
PFBC	8	326	6288	0.30	0.20	3254	0.01	0.11	<0.04	9530	3.03	4.18	6.08	0.03
PFBC	16	782	7632	0.12	0.34	3801	<0.01	0.10	<0.04	11259	2.83	4.12	2.43	<0.01
PFBC	32	458	40	0.03	0.25	1970	<0.01	<0.09	<0.04	2059	0.55	6.16	2.85	0.01
LSD _{0.05}		677	1007	74.8	0.12	2132	1.26	1.07	ns	1890	0.87	22.1	2.14	1.77

Table 54. Plant tissue composition from the sixth harvest of fescue grown on FGD amended R&F spoil.

FGD	Rate	N	P	K	Ca	Mg	S	Al	As	B	Ba	Cd	Co	Cr
	wgt %	%						mg/kg						
LIMB	0	1.94	0.33	3.42	0.50	0.30	0.31	45.07	0.42	25.17	16.24	0.34	0.49	0.51
LIMB	4	2.05	0.33	3.49	0.59	0.30	0.25	31.19	0.28	46.40	16.57	0.47	0.39	0.49
LIMB	8	1.87	0.33	3.14	0.70	0.27	0.25	32.19	1.19	40.72	18.03	0.36	0.76	0.56
LIMB	16	1.97	0.36	3.38	0.66	0.29	0.25	25.49	0.13	48.26	15.59	0.22	0.41	0.48
LIMB	32	2.19	0.30	3.40	0.62	0.25	0.22	39.47	0.34	32.86	18.41	0.21	0.54	0.55
PFBC	0	1.96	0.36	3.38	0.44	0.53	0.35	40.34	0.44	21.67	17.25	0.33	0.69	0.51
PFBC	4	1.78	0.33	3.28	0.45	0.60	0.31	32.89	0.46	20.66	16.18	0.38	0.58	0.50
PFBC	8	1.82	0.36	3.45	0.41	0.59	0.31	32.60	0.22	20.40	15.22	0.32	0.38	0.44
PFBC	16	2.01	0.34	3.25	0.44	0.60	0.29	47.86	0.34	21.46	20.01	0.27	0.43	0.48
PFBC	32	1.87	0.34	3.36	0.39	0.63	0.33	32.28	0.51	21.88	21.56	0.63	0.76	0.80
LSD _{0.05}		ns	ns	ns	0.13	0.14	ns	ns	0.58	12.01	ns	ns	ns	ns

FGD	Rate	Cu	Fe	Hg	Li	Mn	Mo	Na	Ni	Pb	Se	Si	Sr	Zn
	wgt %	mg/kg												
LIMB	0	8.59	67.4	0.08	0.62	266	1.65	1001	1.94	0.14	<0.27	29.7	15.1	38.8
LIMB	4	8.26	62.4	0.10	1.11	226	2.98	888	1.23	<0.04	<0.27	37.5	13.6	36.6
LIMB	8	7.15	74.8	0.15	0.54	247	3.24	555	1.27	<0.04	<0.27	28.7	14.3	24.9
LIMB	16	7.09	55.4	0.15	0.35	197	3.89	211	1.58	<0.04	<0.27	34.0	12.5	22.4
LIMB	32	7.24	110.1	0.14	0.44	161	3.32	295	0.98	0.11	<0.27	29.4	12.1	29.2
PFBC	0	7.21	68.7	0.12	0.58	272	2.06	1416	1.53	0.06	<0.27	32.1	14.1	31.6
PFBC	4	7.20	55.5	<0.01	0.50	281	2.87	446	0.91	<0.04	<0.27	30.5	10.8	29.3
PFBC	8	7.64	56.5	0.03	0.48	254	2.51	352	1.07	0.08	0.30	26.3	9.41	43.1
PFBC	16	7.15	65.1	0.14	0.57	208	2.63	553	1.16	0.46	<0.27	30.3	8.94	33.1
PFBC	32	7.06	53.9	0.30	0.87	206	2.74	223	1.26	0.38	<0.27	27.0	7.86	34.1
LSD _{0.05}		ns	ns	ns	ns	104	ns	266	ns	ns	ns	ns	2.94	ns

Table 55. Leachate composition of the first leachates from Wooster silt loam soil amended with FGD by-products.

FGD ¹	Rate	Crop ²	pH	EC ³	Al	As	B	Ba	Ca	Cd	Cr
	wgt %				mg/L						
PFBCc	0	F	5.17	1.41	1.51	<0.03	0.06	2.05	127	<0.005	<0.01
PFBCc	0.35	F	6.89	2.74	0.23	<0.03	0.22	0.20	326	<0.005	<0.01
PFBCc	0.7	F	7.17	3.51	0.28	<0.03	0.29	0.27	441	<0.005	<0.01
PFBCc	1.4	F	7.54	4.66	0.35	0.12	0.47	0.27	500	<0.005	<0.01
PFBCc	2.8	F	7.97	6.25	0.40	<0.03	0.58	0.17	514	<0.005	<0.01
PFBCc	0	A	5.56	0.85	1.06	0.05	0.07	1.12	82	<0.005	<0.01
PFBCc	0.35	A	6.56	1.93	0.26	0.05	0.16	0.22	231	<0.005	<0.01
PFBCc	0.7	A	6.90	3.23	0.27	0.06	0.30	0.21	401	<0.005	<0.01
PFBCc	1.4	A	7.24	4.21	0.34	0.09	0.39	0.19	466	<0.005	<0.01
PFBCc	2.8	A	7.57	4.32	0.57	<0.03	0.40	0.16	400	<0.005	<0.01
PFBCb	0	F	5.17	1.41	1.51	<0.03	0.06	2.05	127	<0.005	<0.01
PFBCb	0.35	F	5.74	2.28	1.22	0.13	0.19	0.56	254	<0.005	<0.01
PFBCb	0.7	F	5.77	2.22	0.97	0.06	0.25	0.40	253	<0.005	<0.01
PFBCb	1.4	F	5.89	2.91	0.96	<0.03	0.32	0.28	405	<0.005	<0.01
PFBCb	2.8	F	6.50	2.86	0.89	<0.03	0.33	0.22	380	<0.005	<0.01
PFBCb	0	A	5.56	0.85	1.06	0.05	0.07	1.12	82	<0.005	<0.01
PFBCb	0.35	A	5.37	1.25	0.78	<0.03	0.10	0.44	151	<0.005	<0.01
PFBCb	0.7	A	5.92	1.56	0.82	<0.03	0.21	0.27	204	<0.005	<0.01
PFBCb	1.4	A	6.16	2.45	0.93	<0.03	0.42	0.30	373	0.020	<0.01
PFBCb	2.8	A	6.19	2.42	0.48	<0.03	0.28	0.21	353	<0.005	<0.01
LIMB	0	F	5.17	1.41	1.51	<0.03	0.06	2.05	127	<0.005	<0.01
LIMB	0.35	F	7.06	2.61	0.24	<0.03	0.39	0.27	542	<0.005	<0.01
LIMB	0.7	F	7.41	3.91	0.23	<0.03	0.69	0.45	931	<0.005	<0.01
LIMB	1.4	F	7.41	3.58	0.29	<0.03	0.42	0.52	1100	<0.005	<0.01
LIMB	2.8	F	7.24	3.46	0.73	0.22	0.28	0.30	953	<0.005	<0.01
LIMB	0	A	5.56	0.85	1.06	0.05	0.07	1.12	82	<0.005	<0.01
LIMB	0.35	A	6.85	2.03	0.25	0.12	0.28	0.30	414	<0.005	<0.01
LIMB	0.7	A	7.19	3.30	0.30	0.17	0.68	0.53	814	<0.005	<0.01
LIMB	1.4	A	7.20	2.89	0.25	0.19	0.38	0.44	767	<0.005	<0.01
LIMB	2.8	A	7.28	2.08	0.57	0.09	0.31	0.20	556	<0.005	<0.01
LSD _{0.05}			0.55	1.58	0.59	s	0.23	0.24	388	ns	ns

¹PFBCc=PFBCcyclone, PFBCb=PFBCbed²A=alfalfa, F=fescue³EC=electrical conductivity, mmho/cm

Table 55. continued.

FGD ¹	Rate	Crop ²	Cu	Fe	Hg	K	Mg	Mn	Mo	Na
	wgt %		mg/L							
PFBCc	0	F	<0.01	0.05	<0.24	46.6	30	18.9	<0.01	9.80
PFBCc	0.35	F	<0.01	0.04	<0.24	47.8	177	17.2	<0.01	8.56
PFBCc	0.7	F	<0.01	0.10	<0.24	28.1	316	18.6	<0.01	11.48
PFBCc	1.4	F	0.07	0.56	<0.24	54.4	613	16.4	<0.01	11.68
PFBCc	2.8	F	<0.01	0.31	<0.24	48.4	1045	5.31	0.03	13.83
PFBCc	0	A	<0.01	0.05	<0.24	26.4	19	14.4	<0.01	7.94
PFBCc	0.35	A	<0.01	0.33	<0.24	38.4	118	21.9	<0.01	8.33
PFBCc	0.7	A	<0.01	0.63	<0.24	66.5	271	29.7	<0.01	13.10
PFBCc	1.4	A	0.02	0.21	<0.24	35.4	499	21.8	<0.01	10.15
PFBCc	2.8	A	<0.01	0.38	<0.24	23.7	645	9.58	0.02	8.19
PFBCb	0	F	<0.01	0.05	<0.24	46.6	30	18.9	<0.01	9.80
PFBCb	0.35	F	<0.01	0.06	<0.24	63.3	89	28.5	<0.01	13.16
PFBCb	0.7	F	<0.01	0.05	<0.24	43.8	93	23.3	<0.01	9.56
PFBCb	1.4	F	<0.01	0.06	<0.24	81.8	137	26.1	<0.01	13.09
PFBCb	2.8	F	0.08	0.04	<0.24	58.8	167	15.8	<0.01	8.73
PFBCb	0	A	<0.01	0.05	<0.24	26.4	19	14.4	<0.01	7.94
PFBCb	0.35	A	<0.01	0.03	<0.24	33.8	46	17.5	<0.01	7.68
PFBCb	0.7	A	<0.01	0.05	<0.24	35.2	71	17.4	<0.01	7.06
PFBCb	1.4	A	0.08	0.22	<0.24	37.5	154	27.6	<0.01	9.17
PFBCb	2.8	A	0.03	0.03	<0.24	38.0	149	17.1	<0.01	8.20
LIMB	0	F	<0.01	0.05	<0.24	46.6	30	18.9	<0.01	9.80
LIMB	0.35	F	<0.01	0.38	<0.24	34.2	47	15.3	<0.01	7.62
LIMB	0.7	F	<0.01	0.38	<0.24	33.1	72	13.3	<0.01	7.57
LIMB	1.4	F	<0.01	0.55	<0.24	40.6	19	4.25	0.03	4.11
LIMB	2.8	F	0.12	0.16	<0.24	15.3	6	0.84	0.12	3.38
LIMB	0	A	<0.01	0.05	<0.24	26.4	19	14.4	<0.01	7.94
LIMB	0.35	A	<0.01	0.16	<0.24	31.5	41	16.0	<0.01	7.54
LIMB	0.7	A	<0.01	1.09	<0.24	45.2	56	16.5	<0.01	7.42
LIMB	1.4	A	<0.01	0.22	<0.24	21.2	25	4.32	<0.01	4.17
LIMB	2.8	A	<0.01	0.06	<0.24	9.2	10	0.70	0.03	2.43
LSD _{0.05}			ns	ns	ns	33.5	165	11.1	ns	4.19

¹PFBCc=PFBCcyclone, PFBCb=PFBCbed²A=alfalfa, F=fescue

Table 55. continued.

FGD ¹	Rate	Crop ²	Ni	P	Pb	S	Se	Si	Sr	V	Zn
	wgt %		mg/L								
PFBCc	0	F	0.04	3.34	0.17	18	<0.16	6.23	0.70	0.07	0.35
PFBCc	0.35	F	0.04	1.48	<0.09	357	<0.16	2.01	0.70	0.08	0.06
PFBCc	0.7	F	0.02	1.60	<0.09	603	<0.16	1.67	0.72	0.07	0.04
PFBCc	1.4	F	0.02	2.19	<0.09	947	<0.16	1.60	0.74	0.13	0.06
PFBCc	2.8	F	0.03	3.86	<0.09	1328	<0.16	0.80	0.73	0.11	0.03
PFBCc	0	A	0.03	1.45	<0.09	19	<0.16	6.15	0.50	0.05	0.25
PFBCc	0.35	A	0.03	1.25	<0.09	272	<0.16	2.48	0.64	0.07	0.08
PFBCc	0.7	A	0.02	1.65	<0.09	570	<0.16	2.16	0.80	0.08	0.08
PFBCc	1.4	A	0.04	1.89	<0.09	858	<0.16	1.28	0.85	0.06	0.04
PFBCc	2.8	A	0.03	2.31	<0.09	913	<0.16	1.33	0.57	0.05	0.06
PFBCb	0	F	0.04	3.34	0.17	18	<0.16	6.23	0.70	0.07	0.35
PFBCb	0.35	F	0.06	2.49	0.14	185	<0.16	7.56	1.09	0.08	0.30
PFBCb	0.7	F	0.04	2.19	<0.09	215	<0.16	7.18	0.85	0.08	0.28
PFBCb	1.4	F	0.05	2.20	<0.09	352	<0.16	7.20	1.12	0.04	0.22
PFBCb	2.8	F	0.04	1.06	<0.09	416	<0.16	3.67	0.99	<0.03	0.14
PFBCb	0	A	0.03	1.45	<0.09	19	<0.16	6.15	0.50	0.05	0.25
PFBCb	0.35	A	0.04	5.95	<0.09	128	<0.16	6.31	0.70	0.10	0.25
PFBCb	0.7	A	0.03	0.83	<0.09	213	<0.16	6.62	0.65	0.04	0.18
PFBCb	1.4	A	0.05	1.54	0.18	432	<0.16	6.52	0.97	<0.03	0.23
PFBCb	2.8	A	0.04	4.38	<0.09	421	<0.16	4.69	0.91	0.07	0.15
LIMB	0	F	0.04	3.34	0.17	18	<0.16	6.23	0.70	0.07	0.35
LIMB	0.35	F	0.02	1.68	0.21	323	<0.16	1.45	0.84	0.07	0.05
LIMB	0.7	F	0.02	1.61	<0.09	511	<0.16	1.14	1.25	<0.03	0.03
LIMB	1.4	F	0.03	2.37	<0.09	634	<0.16	0.37	1.12	0.08	0.02
LIMB	2.8	F	0.13	4.84	<0.09	450	<0.16	0.72	0.96	<0.03	0.05
LIMB	0	A	0.03	1.45	<0.09	19	<0.16	6.15	0.50	0.05	0.25
LIMB	0.35	A	0.02	1.45	<0.09	279	<0.16	1.95	0.90	0.07	0.10
LIMB	0.7	A	0.03	1.54	0.18	461	<0.16	1.17	1.08	0.03	0.07
LIMB	1.4	A	0.02	1.60	0.20	434	<0.16	0.50	0.92	0.05	0.04
LIMB	2.8	A	0.03	1.58	0.13	354	<0.16	0.40	0.63	0.03	0.03
LSD _{0.05}			0.04	2.61	ns	310	ns	1.86	ns	ns	0.08

¹PFBCc=PFBCcyclone, PFBCb=PFBCbed²A=alfalfa, F=fescue

Table 56. Composition of leachate collected from Wooster silt loam soil 8 months after amendment with FGD by-products.

FGD	Rate	Crop	pH	EC	Al	As	B	Ba	Ca	Cd
	wgt %				mg/L					
PFBCc	0	F	5.17	0.00	1.80	<0.08	0.24	0.06	7.86	0.009
PFBCc	0.35	F	6.61	1.94	<0.05	<0.08	0.48	0.06	201	0.008
PFBCc	0.7	F	6.87	4.08	<0.05	<0.08	0.76	0.05	388	0.010
PFBCc	1.4	F	7.52	6.20	<0.05	<0.08	1.09	0.07	558	0.010
PFBCc	2.8	F	7.51	7.19	<0.05	<0.08	1.32	0.06	529	0.014
PFBCc	0	A	4.52	0.00	1.86	<0.08	0.13	1.04	167	0.021
PFBCc	0.35	A	5.86	3.39	<0.05	<0.08	0.27	0.03	316	0.008
PFBCc	0.7	A	6.65	6.64	<0.05	<0.08	0.52	0.04	461	0.008
PFBCc	1.4	A	7.25	9.42	<0.05	<0.08	0.80	0.05	474	0.019
PFBCc	2.8	A	7.76	11.05	<0.05	<0.08	1.54	0.08	593	0.012
PFBCb	0	F	5.17	0.00	1.80	<0.08	0.24	0.06	7.86	0.009
PFBCb	0.35	F	6.35	3.32	0.13	<0.08	0.59	0.03	288	0.010
PFBCb	0.7	F	7.12	2.42	<0.05	<0.08	0.44	0.03	244	0.007
PFBCb	1.4	F	7.39	4.92	0.48	<0.08	0.55	0.04	339	0.008
PFBCb	2.8	F	7.89	4.97	<0.05	<0.08	0.34	0.02	205	0.012
PFBCb	0	A	4.52	0.00	1.86	<0.08	0.13	1.04	167	0.021
PFBCb	0.35	A	6.04	5.41	<0.05	<0.08	0.25	0.04	390	0.014
PFBCb	0.7	A	6.93	8.65	<0.05	<0.08	0.38	0.04	446	0.010
PFBCb	1.4	A	7.18	11.74	0.08	<0.08	0.54	0.06	507	0.008
PFBCb	2.8	A	7.53	17.26	0.10	<0.08	0.60	0.08	568	0.009
LIMB	0	F	5.17	0.00	1.80	<0.08	0.24	0.06	7.86	0.009
LIMB	0.35	F	6.65	0.61	0.14	<0.08	0.50	0.04	109	0.025
LIMB	0.7	F	7.43	2.19	<0.05	<0.08	1.11	0.07	613	0.005
LIMB	1.4	F	7.60	2.73	<0.05	<0.08	1.90	0.06	733	0.012
LIMB	2.8	F	7.61	3.83	<0.05	<0.08	2.89	0.07	950	0.013
LIMB	0	A	4.52	0.00	1.86	<0.08	0.13	1.04	167	0.021
LIMB	0.35	A	6.11	3.22	<0.05	<0.08	0.91	0.06	594	0.015
LIMB	0.7	A	6.92	4.77	<0.05	<0.08	2.02	0.13	1184	0.010
LIMB	1.4	A	7.26	6.00	<0.05	<0.08	3.02	0.22	1597	0.010
LIMB	2.8	A	7.47	4.91	<0.05	<0.08	3.00	0.14	1208	0.010
LSD _{0.05}			0.45	2.28	0.84	ns	0.55	0.23	212	ns

Table 56. continued.

FGD	Rate	Crop	Cu	Fe	K	Mg	Mn	Na	Ni
	wgt %		mg/L						
PFBCc	0	F	<0.01	1.14	0.52	6.77	0.34	2.07	<0.01
PFBCc	0.35	F	<0.01	2.28	1.78	150	4.90	2.02	<0.01
PFBCc	0.7	F	<0.01	0.43	3.40	492	4.25	1.75	<0.01
PFBCc	1.4	F	<0.01	0.66	8.18	1093	6.02	4.55	0.01
PFBCc	2.8	F	0.01	0.32	3.05	1314	3.47	3.83	0.03
PFBCc	0	A	<0.01	0.30	4.04	44	24.46	19.18	0.06
PFBCc	0.35	A	<0.01	0.48	3.81	266	6.52	2.37	<0.01
PFBCc	0.7	A	<0.01	0.14	6.85	830	1.31	3.97	<0.01
PFBCc	1.4	A	0.07	0.22	11.79	1460	0.30	5.75	0.04
PFBCc	2.8	A	0.06	1.05	25.25	2116	4.40	12.10	0.03
PFBCb	0	F	<0.01	1.14	0.52	6.77	0.34	2.07	<0.01
PFBCb	0.35	F	<0.01	0.29	4.24	335	3.85	4.25	<0.01
PFBCb	0.7	F	<0.01	0.23	1.11	253	0.83	1.09	<0.01
PFBCb	1.4	F	0.01	0.18	1.11	698	1.00	1.06	<0.01
PFBCb	2.8	F	<0.01	0.07	1.60	716	0.49	0.67	<0.01
PFBCb	0	A	<0.01	0.30	4.04	44	24.46	19.18	0.06
PFBCb	0.35	A	<0.01	0.11	8.23	689	3.74	4.54	<0.01
PFBCb	0.7	A	<0.01	0.13	19.11	1370	1.64	4.22	0.01
PFBCb	1.4	A	0.07	0.37	28.15	2536	2.96	4.96	0.02
PFBCb	2.8	A	0.11	0.49	31.16	3781	1.55	5.58	0.05
LIMB	0	F	<0.01	1.14	0.52	6.77	0.34	2.07	<0.01
LIMB	0.35	F	<0.01	0.66	0.43	13.2	1.51	1.48	<0.01
LIMB	0.7	F	<0.01	0.36	1.27	16.4	2.98	1.41	<0.01
LIMB	1.4	F	<0.01	0.33	1.03	18.2	1.00	1.23	<0.01
LIMB	2.8	F	<0.01	0.43	3.93	24.4	0.47	2.52	<0.01
LIMB	0	A	<0.01	0.30	4.04	43.9	24.46	19.18	0.06
LIMB	0.35	A	<0.01	0.39	2.98	78.7	7.40	2.65	0.02
LIMB	0.7	A	0.01	0.25	5.90	65.0	1.04	2.87	<0.01
LIMB	1.4	A	0.04	0.22	6.26	48.3	1.67	3.15	0.02
LIMB	2.8	A	<0.01	0.08	6.14	32.1	0.44	4.28	<0.01
LSD _{0.05}			0.03	ns	8.69	568	5.79	4.31	0.03

Table 56. continued.

FGD ¹	Rate	Crop ²	P	Pb	S	Se	Si	Sr	Zn
	wgt %		mg/L						
PFBCc	0	F	0.09	<0.04	<0.9	<0.27	3.25	<0.01	0.05
PFBCc	0.35	F	<0.09	<0.04	295	<0.27	<0.02	0.30	0.10
PFBCc	0.7	F	0.38	<0.04	796	<0.27	1.13	0.54	0.03
PFBCc	1.4	F	1.10	<0.04	1434	<0.27	2.07	0.74	0.02
PFBCc	2.8	F	1.27	<0.04	1577	<0.27	2.02	0.72	0.05
PFBCc	0	A	<0.09	<0.04	<0.9	<0.27	19.48	0.88	0.64
PFBCc	0.35	A	<0.09	<0.04	518	<0.27	3.66	0.56	0.12
PFBCc	0.7	A	0.83	<0.04	1161	<0.27	2.18	0.72	0.05
PFBCc	1.4	A	1.46	<0.04	1913	<0.27	3.26	0.67	0.01
PFBCc	2.8	A	2.27	<0.04	2700	<0.27	3.36	0.90	0.03
PFBCb	0	F	0.09	<0.04	<0.9	<0.27	3.25	<0.01	0.05
PFBCb	0.35	F	0.22	<0.04	618	<0.27	<0.02	0.52	0.09
PFBCb	0.7	F	0.22	<0.04	457	<0.27	<0.02	0.31	0.05
PFBCb	1.4	F	0.36	<0.04	1162	<0.27	0.08	0.42	0.04
PFBCb	2.8	F	0.16	<0.04	1025	<0.27	1.32	0.21	0.09
PFBCb	0	A	<0.09	<0.04	<0.9	<0.27	19.48	0.88	0.64
PFBCb	0.35	A	0.29	<0.04	879	<0.27	3.08	0.72	0.06
PFBCb	0.7	A	1.06	<0.04	1864	<0.27	1.18	0.74	0.04
PFBCb	1.4	A	2.29	<0.04	2783	<0.27	2.67	0.79	0.05
PFBCb	2.8	A	3.23	<0.04	5298	<0.27	2.40	0.81	0.09
LIMB	0	F	0.09	<0.04	<0.9	<0.27	3.25	<0.01	0.05
LIMB	0.35	F	0.12	<0.04	45	<0.27	2.55	0.09	0.38
LIMB	0.7	F	0.36	<0.04	359	<0.27	1.38	0.70	0.23
LIMB	1.4	F	0.53	<0.04	414	<0.27	1.27	0.78	0.12
LIMB	2.8	F	0.43	<0.04	510	<0.27	1.54	1.07	0.16
LIMB	0	A	<0.09	<0.04	<0.9	<0.27	19.48	0.88	0.64
LIMB	0.35	A	<0.09	<0.04	400	<0.27	5.23	0.94	0.11
LIMB	0.7	A	0.85	<0.04	559	<0.27	2.67	1.30	0.04
LIMB	1.4	A	1.29	0.04	591	<0.27	3.12	1.82	0.04
LIMB	2.8	A	0.35	<0.04	581	<0.27	2.66	1.33	0.02
LSD _{0.05}			0.57	ns	611	ns	1.87	0.26	0.18

Table 57. Tissue composition of fescue grown in pots of Wooster silt loam soil amended with FGD by-products.

FGD	Crop ¹	Rate	N	P	K	Ca	Mg	S	Al	As
		wgt %	%						mg/kg	
PFBCc	F	0	2.06	0.28	2.82	0.46	0.29	0.25	33.88	<0.08
PFBCc	F	0.35	1.86	0.32	2.66	0.48	0.55	0.34	18.77	<0.08
PFBCc	F	0.7	1.88	0.32	2.45	0.53	0.68	0.40	17.29	<0.08
PFBCc	F	1.4	2.39	0.28	2.51	0.56	0.73	0.41	15.47	<0.08
PFBCc	F	2.8	2.08	0.28	2.48	0.46	0.81	0.49	12.48	<0.08
PFBCc	A	0	2.01	0.42	3.43	1.31	0.41	0.55	132.0	<0.08
PFBCc	A	0.35	3.03	0.37	2.09	1.09	0.46	0.24	27.73	<0.08
PFBCc	A	0.7	3.24	0.37	1.84	1.14	0.49	0.25	15.77	<0.08
PFBCc	A	1.4	3.26	0.35	2.18	1.24	0.52	0.26	15.06	<0.08
PFBCc	A	2.8	3.02	0.31	2.15	1.37	0.48	0.30	13.23	<0.08
PFBCb	F	0	2.06	0.28	2.82	0.46	0.29	0.25	33.88	<0.08
PFBCb	F	0.35	2.07	0.31	2.91	0.46	0.65	0.42	15.95	<0.08
PFBCb	F	0.7	2.67	0.31	2.53	0.43	0.72	0.49	13.73	<0.08
PFBCb	F	1.4	3.17	0.29	2.61	0.38	0.83	0.54	12.42	<0.08
PFBCb	F	2.8	3.83	0.25	2.99	0.28	0.98	0.43	11.90	<0.08
PFBCb	A	0	2.01	0.42	3.43	1.31	0.41	0.55	132.0	<0.08
PFBCb	A	0.35	2.89	0.36	2.34	0.97	0.39	0.25	22.95	0.33
PFBCb	A	0.7	3.13	0.37	2.26	1.06	0.48	0.28	18.22	<0.08
PFBCb	A	1.4	3.13	0.32	2.09	1.19	0.49	0.31	18.57	0.10
PFBCb	A	2.8	2.57	0.29	2.11	1.29	0.66	0.38	13.27	<0.08
LIMB	F	0	2.06	0.28	2.82	0.46	0.29	0.25	33.88	<0.08
LIMB	F	0.35	1.90	0.31	2.89	0.67	0.31	0.33	17.24	<0.08
LIMB	F	0.7	1.82	0.33	2.52	0.90	0.30	0.38	14.47	<0.08
LIMB	F	1.4	1.93	0.20	2.10	0.90	0.29	0.35	9.58	<0.08
LIMB	F	2.8	2.45	0.17	2.40	1.00	0.31	0.39	11.25	0.41
LIMB	A	0	2.01	0.42	3.43	1.31	0.41	0.55	132.0	<0.08
LIMB	A	0.35	3.30	0.33	1.90	1.60	0.22	0.22	24.65	<0.08
LIMB	A	0.7	3.26	0.32	2.05	1.79	0.18	0.24	17.51	<0.08
LIMB	A	1.4	3.03	0.29	2.11	2.04	0.18	0.27	15.38	0.35
LIMB	A	2.8	2.72	0.23	2.23	2.60	0.18	0.42	14.69	<0.08
LSD _{0.05}			0.67	0.05	0.47	0.18	0.08	0.06	20.21	ns

Table 57. continued.

FGD	Crop	Rate ¹	B	Ba	Cd	Co	Cr	Cu	Fe
mg/kg									
PFBCc	F	0	21.84	54.94	0.36	0.79	0.28	6.63	56.01
PFBCc	F	0.35	26.06	26.47	0.30	0.22	0.23	8.38	51.70
PFBCc	F	0.7	21.34	17.41	0.50	0.44	0.38	6.80	51.37
PFBCc	F	1.4	19.67	11.46	0.30	0.20	0.27	6.63	58.60
PFBCc	F	2.8	21.22	9.47	0.26	0.30	0.23	5.03	48.97
PFBCc	A	0	65.12	116.92	0.81	0.90	0.28	8.96	44.09
PFBCc	A	0.35	58.31	57.49	0.18	0.30	0.32	10.01	58.48
PFBCc	A	0.7	54.59	28.34	0.11	0.26	0.27	10.90	56.60
PFBCc	A	1.4	54.18	24.37	0.10	0.22	0.28	9.41	59.90
PFBCc	A	2.8	52.53	11.41	0.10	0.14	0.24	7.80	55.46
PFBCb	F	0	21.84	54.94	0.36	0.79	0.28	6.63	56.01
PFBCb	F	0.35	25.06	16.42	0.39	0.32	0.24	6.79	48.73
PFBCb	F	0.7	22.78	9.15	0.18	0.26	0.22	7.58	49.39
PFBCb	F	1.4	19.20	6.57	0.23	0.18	0.22	6.80	47.93
PFBCb	F	2.8	13.53	3.90	0.39	0.36	0.23	6.80	57.14
PFBCb	A	0	65.12	116.92	0.81	0.90	0.28	8.96	44.09
PFBCb	A	0.35	50.34	66.07	0.20	0.09	0.23	9.80	56.55
PFBCb	A	0.7	53.46	52.05	0.12	0.23	0.31	10.14	57.77
PFBCb	A	1.4	51.49	21.53	0.11	0.24	0.28	8.57	56.20
PFBCb	A	2.8	56.14	15.34	0.08	0.16	0.23	7.70	58.21
LIMB	F	0	21.84	54.94	0.36	0.79	0.28	6.63	56.01
LIMB	F	0.35	29.35	22.19	0.26	0.25	0.20	5.99	49.76
LIMB	F	0.7	27.80	17.04	0.28	0.20	0.19	6.91	50.64
LIMB	F	1.4	25.02	13.00	0.38	0.48	0.24	4.59	49.00
LIMB	F	2.8	39.93	11.48	0.33	0.31	0.22	4.90	52.11
LIMB	A	0	65.12	116.92	0.81	0.90	0.28	8.96	44.09
LIMB	A	0.35	58.12	57.18	0.22	0.30	0.22	10.62	59.84
LIMB	A	0.7	57.81	41.54	0.12	0.22	0.25	9.76	53.90
LIMB	A	1.4	58.32	26.11	0.25	0.38	0.31	7.46	53.97
LIMB	A	2.8	76.75	8.16	0.10	0.28	0.20	6.06	61.75
LSD _{0.05}			7.90	17.77	0.23	0.24	ns	0.16	10.11

Table 57. continued.

FGD	Crop	Rate	Mn	Mo	Na	Ni	Se	Si	Sr	Zn
		wgt %	mg/kg							
PFBCc	F	0	608.9	0.24	446.2	3.73	<0.27	18.31	21.08	30.09
PFBCc	F	0.35	246.2	0.86	226.0	1.24	2.11	21.39	11.58	20.40
PFBCc	F	0.7	225.5	1.78	196.4	1.00	1.47	17.35	8.73	22.20
PFBCc	F	1.4	176.6	2.89	166.2	0.75	<0.27	19.88	6.95	15.42
PFBCc	F	2.8	237.7	5.08	126.8	0.82	0.33	11.84	5.58	13.57
PFBCc	A	0	520.6	<0.02	77.32	7.21	2.32	4.42	58.87	88.27
PFBCc	A	0.35	131.5	0.35	53.56	1.20	0.83	1.78	27.27	31.23
PFBCc	A	0.7	85.27	1.46	84.49	0.68	2.42	2.15	18.39	22.80
PFBCc	A	1.4	86.23	2.07	133.3	0.89	1.36	1.04	16.34	24.05
PFBCc	A	2.8	68.58	3.61	64.39	0.34	1.46	0.26	13.40	16.88
PFBCb	F	0	608.9	0.24	446.2	3.73	<0.27	18.31	21.08	30.09
PFBCb	F	0.35	293.7	0.63	174.2	1.25	0.61	22.30	9.92	21.70
PFBCb	F	0.7	172.6	1.62	250.1	0.88	1.60	15.90	7.09	16.73
PFBCb	F	1.4	192.8	3.69	182.4	0.85	1.33	14.62	5.13	15.35
PFBCb	F	2.8	122.7	4.90	241.8	1.07	1.97	16.42	3.22	16.80
PFBCb	A	0	520.6	<0.02	77.32	7.21	2.32	4.42	58.87	88.27
PFBCb	A	0.35	120.9	0.11	84.48	1.29	<0.27	1.66	24.33	32.23
PFBCb	A	0.7	99.32	0.89	72.00	0.83	1.02	1.92	20.60	28.11
PFBCb	A	1.4	75.28	2.21	66.97	0.95	<0.27	<0.02	13.92	19.50
PFBCb	A	2.8	70.97	2.55	70.53	0.56	1.10	0.92	11.80	15.78
LIMB	F	0	608.9	0.24	446.2	3.73	<0.27	18.31	21.08	30.09
LIMB	F	0.35	339.3	0.61	198.1	1.55	1.97	18.29	12.18	18.21
LIMB	F	0.7	289.2	1.92	198.8	1.27	1.24	18.05	10.67	15.28
LIMB	F	1.4	495.2	4.27	201.4	0.89	1.87	8.88	8.85	9.65
LIMB	F	2.8	396.2	6.66	112.1	0.92	0.83	12.48	9.62	11.14
LIMB	A	0	520.6	<0.02	77.32	7.21	2.32	4.42	58.87	88.27
LIMB	A	0.35	147.0	0.17	125.2	1.50	<0.27	2.57	28.96	27.43
LIMB	A	0.7	136.3	1.20	117.2	1.38	0.29	2.00	23.94	26.96
LIMB	A	1.4	137.7	3.10	81.66	0.88	<0.27	<0.02	18.82	17.73
LIMB	A	2.8	124.1	3.98	42.34	0.55	2.51	1.61	19.43	11.96
LSD _{0.05}			175.4	0.90	137.4	0.94	ns	14.05	4.70	7.59

6.6 Economics Data Set Description

The following describes the contents of the data set which will be finalized and analyzed during Phase 3. All analysis performed during Phase 2 was preliminary and comprised a small subset of the final data.

All 1990 single family home real estate individual transactions, as well as any residential land transactions, within a 3-mile radius of four landfills in Franklin County.

Approximately 3000 expected observations.

- Gahanna (Sanitary Landfill), life expectancy in 1990 is 2 years
- Grove City (Sanitary Landfill), life expectancy 20 years
- Alum Creek (Demolition Landfill), closed 14 years
- Obetz (Demolition Landfill), closed 6 years

Contents:

1. Transaction price and month (will be converted to yearly rental value using prevailing index of mortgage interest rates and deflated monthly by CPI)
2. Number of rooms, bedrooms, baths, half baths and porches
3. Square footage of structure, garage and lot
4. Age of structure
5. Dummy variables for central air conditioning, fireplace, full and finished basement
6. Previous sale price of sales occurring between 1980-1990
7. Dummy variable for intra-family sales
8. Dummy variables for presence of airport, railroad, freeway, park country club
9. Distance of each property to all four landfills, and distance to CBD
10. Dummy variables for school district
11. Census block group data for racial characteristics, income, age distribution, family size
12. Neighborhood crime rate
13. National indexes of mortgage interest rates and CPI for real estate

From Survey:

1. Knowledge of landfill's presence prior to home purchase.
2. Detailed demographic data
3. Detailed employment data
4. Improvements made on property
5. Previous place of residence

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