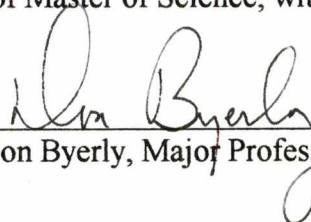
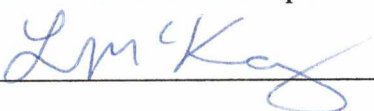


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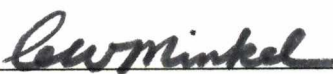

Don Byerly, Major Professor

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Associate Vice Chancellor and
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**The Geologic and Hydrologic Factors Influencing Acid Mine
Drainage at a Surface Coal Mine in East Tennessee:
An Environmental Case History**

**A Thesis
Presented for the
Master of Science
Degree
The University of Tennessee, Knoxville**

**Charles Grant Blackburn
December 1996**

DEDICATION

**This thesis is dedicated to my wife, Margaret, for her steadfast support,
to my son, Charles Grant, Jr., for giving me inspiration,
and to my grandparents, Mr. and Mrs. C. C. Wright, for their support and
for instilling in me the value of an education.**

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ABSTRACT

The oxidation and hydrolysis of sulfide minerals exposed during coal mining operations can result in the production of high levels of acidity, high metal concentrations, and high sulfate concentrations in groundwater and receiving streams, a condition commonly referred to as acid mine drainage (AMD), or acid drainage (AD). The geochemical and microbiological factors influencing the production of AD are well understood, but the extent to which these problems can occur at a given mine site depends on a variety of conditions including: 1) mineralogy of the deposits, 2) surface/groundwater hydrology, 3) mining/backfill handling methods, and 4) reclamation practices.

This study documented and evaluated the geology, hydrology, water quality, and reclamation activities at a large-scale (i.e., 500,000 tons/yr) surface coal mine in the coalfields of the southern Cumberland Plateau of Tennessee from the perspective of environmental impacts related to AD. Geochemical analyses, including electron microprobe (EMP), x-ray fluorescence (XRF), and x-ray diffraction (XRD) of representative overburden lithotypes found within the study area were performed to determine potential sources of AD within mine overburden, in particular sources of elevated manganese (Mn) concentrations detected in backfill groundwater. The hydrology of the study area was documented using groundwater elevations from observation wells emplaced in backfill, stream elevations, and hydrogeologic cross-sections of the mine site to determine general groundwater flow paths and points of discharge from the backfill. Groundwater chemical characteristics of the backfill during mining and following final pit closure were evaluated using graphs of specific groundwater chemical parameters. Modern mining techniques implemented in the study area, in particular a plan referred to as a Toxic Material Handling Plan (TMHP), was evaluated to determine the effectiveness of the plan in abating AD.

Geochemical analyses of representative mine overburden of the study area identified two potential sources of AD: 1) a siderite lithology containing nearly 1 weight percent Mn, and 2) a sandy shale unit containing an average of 1.35 weight percent sulfur. The siderite lithology could be considered as a potential source of mineral acidity by increasing Mn concentrations and the sandy shale unit capable of acid

production in terms of lowering pH in backfill groundwater.

Groundwater flow within the backfill is apparently to the east and southeastern portions of the study area with discharge from the backfill into Spring Branch and Glady Fork Creek most likely occurring on the east side of the study area. Discharge in the southeastern corner of the study area is not apparent at the present time due to pumping from dewatering wells in this area. It is expected that shutdown of the dewatering wells in this area would cause the previously observed seeps to reappear, possibly with negative impacts on stream water quality.

Graphs of chemical groundwater characteristics suggests that the TMHP is having a positive affect on groundwater quality within the backfill of the study area in terms of increasing alkalinity, lowering metal concentrations, and increasing pH. Groundwater quality in observation wells located closest to the area where the TMHP was implemented exhibit higher alkalinity concentrations, higher pH values, and lower metal concentrations than observation wells located farthest from the TMHP area. Increasing alkalinity and decreasing metal concentrations in observation wells located farthest from the TMHP area suggests that the TMHP area is influencing water quality beyond the area where it was implemented. Evaluation of groundwater chemical characteristics suggests that the TMHP is a reclamation technique that could be beneficial in mitigating many of the adverse environmental impacts associated with surface coal mining.

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Chapter 1.

Introduction

Fossil fuels, including coal, are the primary source of energy in the United States, with over 87 percent of the total energy coming from petroleum, natural gas, and coal (Foley et al, 1993). Utility companies are the primary consumers of coal, where it is used to provide about 57 percent of the electrical energy (Foley et al., 1993). Utility consumption of coal rose from 759 million tons in 1988 to 813 million tons in 1993 (COAL, July, 1995).

The United States has the world's largest known recoverable reserves of bituminous and anthracite coal (COAL, July, 1995). The recovery of coal reserves, however, has been a controversial issue between environmentalists and the coal-producing industry. Adverse environmental impacts associated with coal mining, especially mining conducted prior to the enactment of the Surface Mining Control and Reclamation Act (SMCRA) of 1977, has led to an environmental awareness concerning coal production. Coal production from Tennessee, particularly in the coalfields of the southern Cumberland Plateau, has continually diminished since the 1970's when the first regulations governing mining and reclamation were enacted. A major factor in the decrease in mining activity is the legacy of past adverse environmental impacts from pre-regulated mining, particularly in the form of high levels of acidity and dissolved metals in streams and groundwater, a condition commonly referred to as acid drainage (AD).

Acid drainage (AD), relative to coal mining, results from the oxidation of sulfide minerals in freshly exposed overburden and waste material from coal cleanings. The geochemical and microbiological factors influencing production of AD waters and related high levels of dissolved metals are well understood, but the extent to which these problems can occur at a given mine site depends on a variety of conditions including:

- 1) mineralogy of the deposits, 2) surface/groundwater hydrology, 3) mining/backfill handling methods, and
- 4) reclamation practices.

Predictions of the long-term environmental impacts of mining are complicated because important factors such as mining/backfill handling methods may change during the operational life of the mine. Also, the hydrology of the area after mine closure is usually very different than during the mine's operational life. For example, pumping from the mine pit or backfill and diversion of surface runoff that might have been performed during a mine's operation, might not be carried out following closure of the mine. Due to the complexity of these factors it is difficult to design cost-efficient reclamation methods to control AD or to develop the monitoring methods necessary to assess the effectiveness of such efforts. For example, if water quality in an adjacent stream improves or degrades after mine closure it may not be clear which of the many changing conditions (i.e. rising water table in the backfill, changes in surface runoff, vegetative growth in the reclaimed area, etc.) may be responsible for the improvement or degradation.

Chapter 2.

Purpose of Investigation

Many of the coal mines in east Tennessee, especially the southern coalfield, are located in similar geologic and hydrologic settings and, in many cases, experience similar types of environmental problems. Future mining and ongoing operations can often benefit by comparison with well-documented environmental case histories based on existing operations. The goal of this thesis is to document and evaluate the geology, hydrology, water quality and reclamation activities at a large-scale (i.e. 500,000 tons/yr) surface coal mine in east Tennessee from the perspective of environmental impacts related to acid mine drainage. In order to make this assessment, this study will include the following:

- 1) Analyze and document the mineralogy of mine overburden within the study area. In particular, the mineralogies of a lithologic unit recognized as siderite-bearing (FeCO_3) will be examined relative to the production of AD.
- 2) Study groundwater elevations and determine general groundwater flow patterns within the backfill of the study area.
- 3) Evaluate of backfill water quality by studying concentrations of selected chemical constituents measured within the backfill during mining and during water table recovery after closure of the mine.
- 4) Evaluate the mining techniques currently being implemented in the study area as to their effectiveness in minimizing potential adverse environmental impacts associated with coal mining in the southern coalfields of the Cumberland Plateau.

In essence, this study will provide a case history that studies physical and geochemical characteristics of mine overburden and water quality for a surface coal mine, during and shortly after closure, and will provide an assessment of modern mining practices used to recover coal resources in the southern coal fields of the Cumberland Plateau.

Chapter 3.

Acid Drainage Generation

Acid Drainage (AD), in a coal-mining environment, is normally characterized by elevated levels of heavy metals, low levels of dissolved oxygen, high sulfate content and low pH. The acidity of AD originates not from free acid but from the hydrolysis of certain iron sulfates (by products of pyrite oxidation) (Caruccio, 1988). The exposure of abnormal amounts of rock material during mining can result in elevated levels of acidity and metals due to accelerated weathering of freshly broken or exposed mineral surfaces to air and/or water. Acid drainage can also occur naturally in undisturbed deposits, but the role of acid production is extremely small compared to what occurs at many mine sites. The major factors influencing the generation of AD include the presence of sulfide minerals, forms of sulfur contained in the rock, the availability of oxygen, the pH of the aqueous environment, and the presence of iron-oxidizing bacteria.

Forms of Sulfur in Rock

Sulfur associated with coal and coal-bearing rocks can occur as organic sulfur, sulfate sulfur, and pyritic sulfur (Skousen, 1995). AD is a function of the amount and type of sulfur encountered during coal mining operations. Organic sulfur, believed to be complexed and combined with organic constituents of coal, is not chemically reactive and is thought to have little effect on acid generation (Skousen, 1995). Sulfate sulfur is commonly the result of weathering of sulfide minerals and is not considered as a primary acid producer. The third type of sulfur, sulfide or pyritic sulfur, is the dominant sulfur species in coal and associated rocks and of greatest concern in regards to AD generation.

Of all the sulfide sulfur species, iron disulfides predominate and are considered the major acid-producers. The polymorphs, pyrite and marcasite, common disulfide minerals associated with coal stratigraphy, are generally recognized as the primary cause of acidity due to their spontaneous oxidation

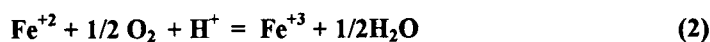
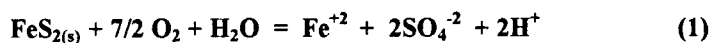
potential and subsequent hydrolysis of the oxidation products (Caruccio and Geidel, 1996). Table 1 lists some important sulfide minerals which, when exposed to oxidizing conditions, can generate acidity and increased metal concentrations in receiving waters.

Table 1: Common metal sulfides (Skousen, 1995)

FeS ₂ ----- Pyrite	CuS----- Covellite	MoS ₂ --- Molybdenite
FeS ₂ ----- Marcasite	Cu ₂ S ----- Chalcosite	NiS ----- Millerite
Fe _x S _x ----- Pyrrhotite	CuFeS ₂ --- Chalcopyrite	PbS ----- Galena

Oxidation of Sulfide Minerals

Oxidation of once deeply buried rocks is an unavoidable chemical reaction associated with surface mining operations. The oxidation of iron disulfides and subsequent conversion to acidity occurs through several reactions. Acidity, as used here, is defined as a measure of the capacity of a water to neutralize strong base (Snoeyink and Jenkins, 1980). The oxidation of pyrite by oxygen can occur as a purely chemical process or with catalysis by iron-oxidizing bacteria such as *Thiobacillus ferrooxidans*. The major reactions responsible for pyrite oxidation are outlined as follows (Kleinmann, et al, 1981):



The reactions in Equations 1 through 3 produce the initial acidity in micro environments surrounding the surfaces of the sulfide mineral particles (Byerly, 1996). Equation 1 summarizes the oxidation of pyrite by oxygen. In equation (1) iron disulfide is oxidized, releasing ferrous iron (Fe⁺²), sulfate, and acidity (in the form of H⁺). The ferrous iron is oxidized to form ferric iron (Fe⁺³) as illustrated in equation (2). The ferric iron (Fe⁺³) can then, in turn, either be hydrolyzed and form ferric hydroxide and acidity (equation 3), or it can directly react with sulfide minerals and act as a catalyst in generating much greater amounts of ferrous iron, sulfate, and acidity (equation 4) (Skousen, 1995). Equation 4 outlines the

oxidation of pyrite by ferric iron. Equation 4 is significant only under the acidic conditions necessary for ferric iron solubility (Erickson et al, 1977). The ferric hydroxide generated as a result of equation (3) is commonly referred to as "yellow boy" and is responsible for forming the gelatinous, yellowish orange floc in many streams receiving AD.

The stoichiometry of equations (1) to (4) gives the expected ratio of dissolved ions in AD effluent. The net acidity of equations (1) to (3) indicates that for every 1 mole of FeS_2 , 4 moles of acidity (in the form of H^+) are produced. The increasing number of H^+ ions released into solution results in lowering of pH. As the pH of water lowers, the water is considered to become more "acidic" because it is becoming increasingly able to neutralize a strong base. As the pH decreases conditions become favorable for many species of metal ions to remain in solution (Snoeyink and Jenkins, 1980).

Equation 2 is known to be the rate-limiting step because ferrous iron conversion to ferric iron is rather slow under abiotic (or strictly chemical) conditions (Skousen, 1995; Singer and Stumm, 1970). Oxidation of ferrous iron proceeds most rapidly at pH above 4.5, and quite slowly at pH's below about 3, where half-reaction times are slower. However, the presence of iron-oxidizing bacteria can greatly enhance and accelerate iron oxidation which greatly speeds acid generation (Skousen, 1995).

Oxidation reactions are believed to occur in a multistage sequence dependent upon the activity of iron-oxidizing bacteria, such as *Thiobacillus ferrooxidans*, and solution Eh and pH. According to Kleinmann et al. (1981) there are three stages involved during the oxidation of sulfide minerals:

Stage 1:

Chemical Environment: pH above 4.5; high sulfate; low iron; little or no acidity.

Mechanism: Equation 1: Proceeds both abiotically and by direct bacterial oxidation.

Equation 2: Proceeds abiotically, slows down as pH falls.

Stage 2:

Chemical Environment: Approximate pH range of 2.5-4.5; high sulfate; acidity, and total iron increasing; low $\text{Fe}^{+3}/\text{Fe}^{+2}$ ratio.

Mechanism: Equation 1: Proceeds both abiotically and by direct bacterial oxidation.

Equation 2: Proceeds at rate determined primarily by activity of *T. ferrooxidans*.

Stage 3:

Chemical Environment: pH below approximately 2.5; high sulfate, acidity, total iron and $\text{Fe}^{+3}/\text{Fe}^{+2}$.

Mechanism: Equation 3: Proceeds at rate totally determined by activity of *T. ferrooxidans*.

Equation 4: Proceeds at rate primarily determined by rate of Reaction 3.

Thiobacillus ferrooxidans increases initial acidification above pH 4.5 and below pH 4.5 it allows acidification to continue by oxidizing Fe^{+2} (Kleinmann et al, 1981). Below a pH of approximately 2.5, the activity of Fe^{+3} is significant and results in steady-state cycling between oxidation of pyrite by Fe^{+3} and bacterial oxidation of Fe^{+2} (Kleinmann et al, 1981). Under these biotic conditions, acid generation rates can be increased up to a million times (Leathen et al, 1953).

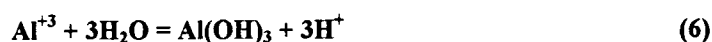
Several main factors controlling the oxidation and hydrolysis of sulfide minerals include solution pH, the presence of iron-oxidizing bacteria, the availability of water and the presence or absence of oxygen. Removal of air and/or water from the system, which are the two most critical factors, would stop pyrite from oxidizing (Skousen, 1995). When pyrite is found in geologic materials beneath the water table where oxidizing conditions are limited, the pyrite basically remains stable (Skousen, 1995).

The extent of AD generated from sulfide minerals is largely determined by three interrelated factors: 1) the oxidation rate of sulfide minerals, 2) the presence of bacterial catalysts, and 3) the neutralizing capacity of the geologic material (Gunderson, 1991). The rate of pyrite oxidation depends on variables such as reactive surface area of pyrite (Singer and Stumm 1968), form of pyritic sulfur (Caruccio et al. 1988), oxygen concentrations (Smith and Shumate 1970), catalytic agents (Caruccio et al. 1988), flushing frequencies (Caruccio et al. 1988), and the presence of *Thiobacillus* bacteria (U.S. Environmental Protection Agency 1971). The possibility of identifying and quantifying the effects of these and other controlling factors with all the various rock types in a field setting is unlikely; however, precise knowledge regarding the oxidation rates is not needed when the acid-producing potential has been accurately measured from weight percent sulfide sulfur (Skousen, 1995).

Oxidation of Non-Sulfide Minerals

Although sulfide mineral oxidation is the primary contributor to AD, other mineral species can undergo hydrolysis and oxidation reactions to produce AD. According to Hedin, et al (1994), acidity associated with coal mine drainage arises from proton acidity associated with pH (a measure of free H^+ ions)

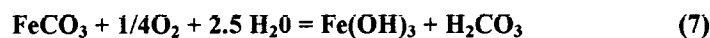
and mineral acidity from the hydrolysis of dissolved Fe, Mn, and Al. The following reactions (Equation 5 and Equation 6) illustrate the hydrolysis of manganese and aluminum-bearing rocks:



The dissolution of Mn is dependent on the presence or absence of an oxidizing environment and solution pH. In an aerobic environment, MnO_2 should precipitate at a pH above 4, however, below a pH of 8, the kinetics of precipitation are quite slow (Hedin et al, 1994). Mn^{+2} may be oxidized to either a $+3$ or a $+4$ valence, either one of which precipitates as MnOOH. If MnOOH precipitates, over time it likely oxidizes to the more stable MnO^{+2} (Hedin et al, 1994). In an anaerobic environment Mn^{+2} does not oxidize and oxyhydroxide precipitates do not form (Hedin et al, 1994).

In an alkaline environment (i.e. $\text{pH} > 8$), Mn^{+2} can precipitate as a carbonate (MnCO_3), which may also be oxidized by oxygen to MnO_2 (Hedin et al, 1994). Under acidic or circumneutral pH, hydroxides of reduced Mn ions, Mn(OH)_2 , do not form due to their high solubility (Hedin et al, 1994).

Siderite (FeCO_3), a non-sulfide mineral, is a common carbonate mineral associated with overburden of strip mines in the Appalachian coal fields (Geidel et al., 1986). Geidel, et al, (1986) state that siderite is 14 times more soluble than pyrite. Substitution of Fe^{+2} by other metallic metals, including Mn, is common in siderite and the mineral is rarely found as pure FeCO_3 (Deer et al., 1992). The dissolution of the non-sulfide, iron-bearing carbonate, siderite, is illustrated below:



Equation (7) indicates that the hydrolysis of siderite results in the formation of ferric hydroxide and carbonic acid (H_2CO_3), the dominant carbonate species at low pH (Snoeyink and Jenkins, 1980). The presence of siderite in overburden can influence the results in the acid-base accounting (ABA) procedure for AD prediction. In the customary ABA analysis, the Neutralization Potential (NP) laboratory procedure (Sobek et al., 1978) measures alkalinity by digestion of carbonates with HCL. When siderite is present during the test, it reacts quickly with the acid indicating that the rock is an alkaline contributor; however, in nature upon complete weathering, siderite is a neutral to slightly acid-producing mineral (Frisbee and

Hossner, 1989; Cravotta, 1991; Doolittle et al., 1992). The acidity associated with siderite weathering is a product of ferrous iron oxidizing to ferric iron.

To avoid the possibility of erroneous NP results due to the presence of siderite, Wiram (1992) suggests using a Modified Net Neutralization Potential (MNNP) which involves adding a small quantity of 30% hydrogen peroxide (H_2O_2) to the sample prior to back titration. The peroxide addition causes the ferrous iron to oxidize to ferric iron before back titrating, resulting in precipitation of $Fe(OH)_3$ upon titration, and yielding a more accurate NP value (Evans and Skousen, 1995).

Chapter 4.

Acid Drainage Prevention and Mitigation Strategies

In order to effectively control acid drainage generation, prevention of the oxidation of ferrous iron to ferric iron is a key element. Limiting the oxidation of ferrous to ferric iron effectively reduces the amount of acidity produced by almost half (Caruccio and Geidel, 1996).

Mitigation Strategies

Mitigation strategies devised to limit the oxidation of ferrous to ferric iron, include (Caruccio and Geidel, 1996):

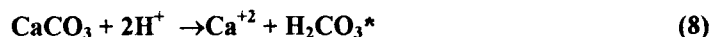
- 1) Caustic substances (i.e. sodium carbonate briquettes, calcium oxide) are added to neutralize acidity.
- 2) Ferric iron complexes are introduced (i.e. phosphate, in the form of pulverized apatite). For example, introducing a phosphate compound into an acid system serves to complex the free iron radical and inhibit the hydrolysis reaction, thereby reducing the amount of acidity produced (Flynn, 1969).
- 3) Plastic liners are used to reduce the flow of water through acid material which can effectively reduce the acid load of the system. To protect the plastic from ultraviolet ray exposure, which decomposes the plastic, a three foot layer of soil covers the plastic and is vegetated to afford mass stability.
- 4) Toxic material is selectively placed in order to isolate the material from oxidation reactions caused by infiltrating groundwater and oxygen diffusion through the soil/backfill cover.
- 5) Detergents and surfactant bactericides are used to inhibit growth of bacterial catalysts.
- 6) Wetlands are used to remove acidity, sulfate, and metals from mildly to moderately acidic mine drainages by filtering and cation exchange, oxidation and reduction reactions, and microbiological processes.
- 7) Submerge toxic material to limit oxygen supply.

- 8) Revegetate and place organic-rich soils to consume oxygen.
- 9) Use of alkaline treatments to neutralize acidity. Alkaline treatments include:
 - a) deep well injection of limestone slurry,
 - b) limestone blankets,
 - c) anoxic limestone trenches, and
 - d) induced alkaline recharge zones.

Alkaline Additions

Alkaline additions, (calcium carbonate, CaCO_3), are probably the most widely-used mitigation strategy, particularly in Appalachia where local sources of limestone (calcium carbonate) are readily available. They have been used in acid drainage treatment with variable success (Caruccio et al, 1984; Pearson and McDonnell, 1974; Nairn, Hedin and Watzlaf, 1992; Hedin, Nairn and Kleinmann, 1994). When limestone contacts acidic water, it dissolves and produces dissolved calcium and bicarbonate alkalinity. The principal source of alkalinity in mine water is dissolved carbonate, which can exist in a bicarbonate (HCO_3^-) or carbonate form (CO_3^{2-}) (Hedin et al., 1994). The alkalinity neutralizes mineral acidity and buffers against decreases in pH (Nairn, Hedin and Watzlaf, 1994).

The reaction sequence for carbonate dissolution in acid waters, as described by Cravotta et al. (1990) are:



When acid waters contact limestone, the limestone dissolves and produces dissolved calcium and dissolved carbon dioxide (8). Dissolved CO_2 is a weak acid and continues to react with limestone, producing dissolved calcium and bicarbonate alkalinity (9). The conventional notation for dissolved CO_2 is " H_2CO_3^* ", and includes both $\text{CO}_{2(\text{aq})}$ and carbonic acid (Stumm and Morgan 1981). This relationship is shown in reactions (8) and (10). Reaction (9) illustrates that for every mole of carbonic acid, two moles of

bicarbonate are produced, thus increasing the bicarbonate alkalinity of the system. The bicarbonate alkalinity produced in reaction (9) is available for acid neutralization reactions.

Caruccio and Geidel (1996) suggest that by restricting the acid reactions to the aqueous oxidation of pyrite by oxygen, the levels of acidity can be reduced to concentrations that can be accommodated by carbonate produced bicarbonate alkalinity. If, however, oxidation of pyrite by ferric iron occurs, the rate of acid production is greatly enhanced and overwhelms the low solubility limited calcareous-generated alkalinity (Caruccio and Geidel, 1996).

Alkaline production from calcareous material is constrained by the low solubility of the specific carbonate mineral in water (siderite excluded) (Geidel, 1980). As a result, alkalinity concentrations are fixed by the partial pressure of carbon dioxide, the duration of water contact, and the solubility of the specific mineral or rock, e.g., calcite, dolomite, limestone (Caruccio and Geidel, 1996).

The equations for pyrite oxidation (Equation 1) show that a material containing 1 weight % sulfur, all as pyrite, would yield upon complete reaction an amount of sulfuric acid that requires 31.25 tons of CaCO_3 to neutralize 1000 tons of the material (Skousen, 1995). The 31.25 tons of CaCO_3 required to neutralize 1000 tons of material is based on two moles of calcium carbonate needed to neutralize one mole of FeS_2 .

Calcareous minerals have the capability of producing alkalinity, and a highly buffered and potentially neutralizing leachate (Caruccio and Geidel, 1996). Perry and Brady (1995) determined that calcareous content as low as 1 to 3 weight % can produce alkaline drainage. In addition, the calcareous material renders the pH of the groundwater regime alkaline, which effectively suppresses the iron bacteria microbial activity and reduces the catalysis of the oxidation of ferrous iron (Caruccio and Geidel, 1996).

The bicarbonate alkalinity produced by the dissolution of limestone has proven beneficial in abating AD in many reclamation activities. By strategically placing limestone in the groundwater recharge areas of a mine and inducing surface water flow into these areas, alkaline loads within the mine backfill are increased to enhance neutralization and inhibit pyrite oxidation (Caruccio et al., 1984).

The alkalinity produced from limestone dissolution can be inhibited, however, by mineral coatings. In an aerobic environment minerals, including Fe^{+2} and Mn^{+2} , oxidize forming oxyhydroxide coatings which can effectively eliminate or decrease the dissolution of limestone by "armoring" the limestone. If limestone can remain in an anaerobic environment, however, where oxygen is absent, Fe^{+2} and Mn^{+2} do not oxidize and the oxyhydroxide precipitates do not form.

The diversion of runoff and infiltrating water from reclaimed areas into limestone ammendments is a key element to be considered while devising mitigation strategies. For example, if reclaimed mine areas are not properly graded to divert surface runoff and groundwater infiltration into areas that have been ammended with limestone additions, the maximum potential alkalinity generated from limestone dissolution may not be achieved.

Chapter 5.

Geologic and Hydrologic Setting of Study Area

General Geologic Setting

The mine selected for this study is located within the physiographic section of Tennessee known as the Cumberland Plateau. The Cumberland Plateau is part of the Appalachian Plateaus physiographic province of the eastern United States (Fenneman, 1938). The Cumberland Plateau, in Tennessee, is a long, narrow arch between coal basins of the central Appalachians and the Black Warrior Basin in Alabama. It was divided by Nelson (1925a) and Glenn (1925) into two coal fields--the southern and the northern using the Tennessee Central Railroad (Norfolk-Southern) as the dividing line (Milici, 1974). Now, Interstate 40 almost paralleling the railroad can be considered representative of the boundary between the northern and southern coal fields of Tennessee.

In general, the southern Cumberland Plateau is a broad, asymmetrical syncline, the axis of which is parallel to, and near, the western side of Sequatchie Valley (Luther, 1959). Along the western escarpment of Sequatchie Valley, the rocks dip steeply to the northwest, then gradually flatten out and begin to rise gently to the northwest in response to the syncline (Luther, 1959). The southern Cumberland Plateau is a broad and relatively flat upland, ranging in elevation from 1,700 to 2,000 feet, capped by resistant Pennsylvanian age sandstone. The tableland character of the Cumberland Plateau is due to the erosional character of the nearly flat-lying, alternating layers of sandstone and shale.

Stratigraphy

The study area is underlain by rocks of the Crab Orchard Mountains group of the Pottsville series of the Pennsylvanian system. The Crab Orchard Mountains group consists of, in stratigraphically descending

order, the Rockcastle conglomerate, Vandever formation, Newton sandstone, Whitwell shale, and the Sewanee conglomerate. These formations are comprised mainly of well-cemented, commonly conglomeratic sandstone, olive-gray shale and silty to sandy olive-gray shale. Coal seams of varying thicknesses are typically associated with the shale zones throughout the group.

A columnar section representing the general stratigraphy in the study area is depicted in Figure 1. Two units of the Crab Orchard Mountains group comprise the section within the study area — 50 to 100 feet of Newton sandstone overlying 50 to 70 feet of Whitwell shale. The Newton sandstone is a fine to medium-grained, massive, well indurated, micaceous, ortho-quartzitic sandstone. Sparry calcite cement in the Newton, which may contribute to alkalinity in the backfill, has been noted in drill cores; however, this particular cement type is rare and is not believed to comprise a significant percentage of the Newton. Sedimentary structural features include cross-bedding, especially festoon-type, and channel-fill deposits characterized by preferentially oriented siderite clasts of various sizes (on the average, 1 inch thick by two to five inches long) bounded by a matrix of sandstone. This type of deposit is referred to as a siderite conglomerate and overburden core drilling indicates that it is randomly distributed within the Newton.

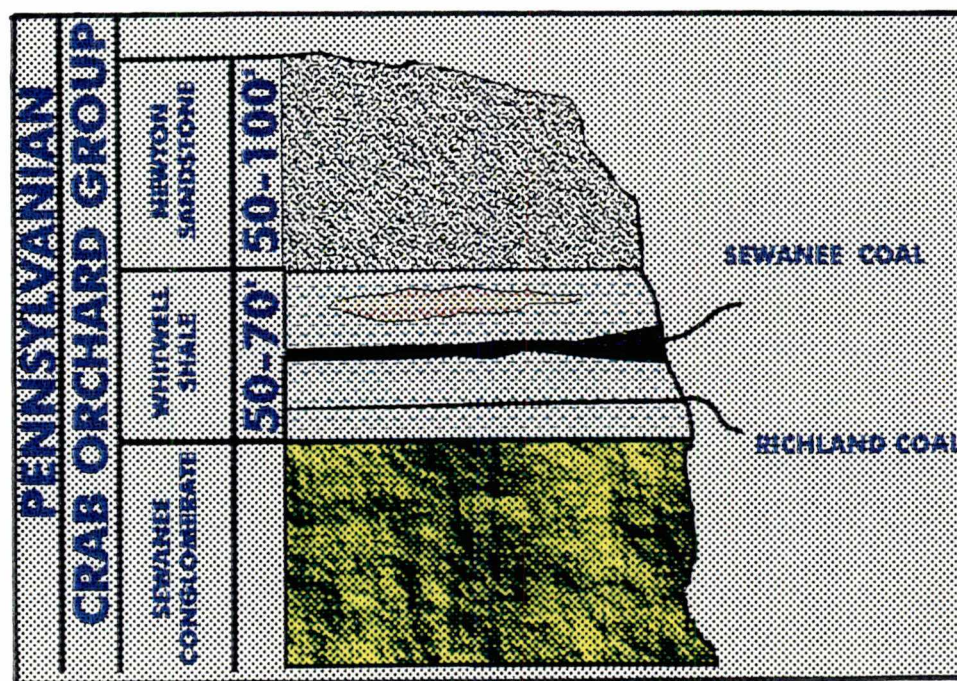


Figure 1: Representative stratigraphic section of study area prior to mining.

The underlying Whitwell shale is an olive-gray, thinly bedded shale that grades laterally and vertically to silty or sandy shale near the contact with the Newton sandstone. The sandy shale unit near the top of the Whitwell shale is generally pyritic and considered a potential source of acidic drainage. Common to the Whitwell are thin (1/2 inch) beds of siderite separated by 2 to 3 inches of shale. Randomly distributed carbonaceous shale in the Whitwell shale has been noted by mine operators; however, it does not comprise a major portion of the shale in the study area (Wiram and Naumann, 1995).

Coal units within the Whitwell include the Sewanee coal, the seam mined within the permit area, and the Richland coal seam, occurring below the Sewanee and near the base of the Whitwell. The thin and discontinuous nature of the Richland coal combined with its high sulfur content are primary reasons why it is not mined in the permit area. The Sewanee coal seam, near the middle of the Whitwell shale in the study area, is 15 to 60 inches thick, with an average thickness of 3 feet. On the average 20 to 30 feet of Whitwell shale lies between the base of the coal and the top of the underlying Sewanee conglomerate. The Richland coal, when present, lies approximately 15 feet below the Sewanee coal.

The Sewanee conglomerate, not exposed at the study mine, ranges in thickness from 60 to 200 feet throughout the Plateau and generally occurs as a medium, to coarse-grained, crossbedded, massive, conglomeritic sandstone (Luther, 1959). Neither the Sewanee Conglomerate nor the Richland coal are disturbed during mining in the study area.

Site Structure

The study area lies near the western limits of the decollement of the Alleghenian Orogeny. Although the rocks of the Cumberland Plateau can be characterized as being relatively flat-lying, tectonic deformation during the Alleghenian Orogeny resulted in fracturing of competent rock units like the Newton Sandstone, tectonic thinning and thickening of beds, and minor folding. Coal thickness variation in this area is typically due to tectonic movement along the beds. Cleavage and minor folding within the Sewanee coal provide evidence of tectonism. In general, bedrock in the study area strikes northeast between 10° and 25°

and dips to the southeast between 1° and 2°. No major structural features (i.e. faults or major folds) are observable within or immediately adjacent to the study area.

General Watershed Characteristics

The study area is contained within the Tennessee River and the Cumberland River drainage basins. The hydrologic setting of the study area, relative to drainage patterns, is depicted in Figure 2. The bold, red line in Figure 2 is a surface water drainage boundary representing the Tennessee Valley Divide. The Tennessee Valley Divide crosses the northwestern portion of the study area, trending in a northeast-southwest direction. All streams south of the divide generally flow southeasterly into Big Brush Creek, a tributary of the Sequatchie River which is a tributary of the Tennessee River. Streams north of the divide flow in a northerly direction into Rocky River, a tributary of Caney Fork which is a tributary of the Cumberland River.

The average annual precipitation for the area is about 54 inches, but ranges from 35 inches in dry years to about 70 inches in wet years (May et al., 1981). The climate of the study area is classified as humid continental. A humid continental climate is one that lacks a definitive "wet" and "dry" season. The variability in precipitation is illustrated in Figure 3, depicting the total monthly rainfall accumulations measured by the mine operators at the study area from January, 1990 to October, 1996.

In terms of precipitation amounts, Figure 3 indicates the considerable seasonal and monthly variability in terms of "wet" versus "dry" months within the study area. In order to characterize "wet" versus "dry" months in the study area, monthly precipitation totals were averaged over a six and a half year period (1990 to October, 1996). Table 2 indicates the precipitation days per month, total monthly precipitation, average monthly precipitation totals, yearly precipitation totals, and average precipitation/event from 1990 to October, 1996. The data presented in Table 2 indicates that, in terms of precipitation totals, February, March, December, and June can be considered "wet" months compared to July through November, which appear to be months that are relatively "dry".

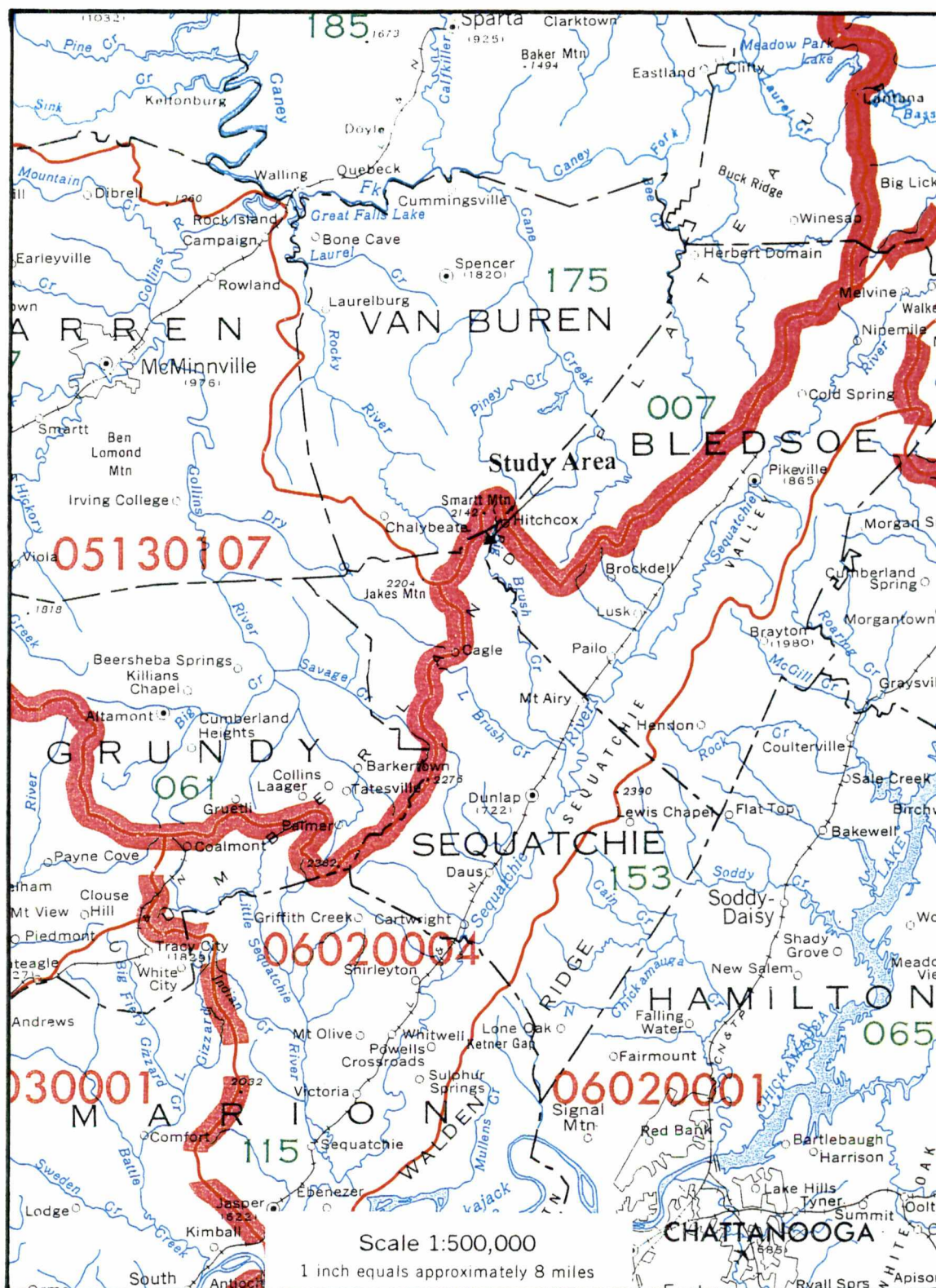


Figure 2: A portion of the Hydrologic Unit Map of the State of Tennessee, depicting drainage patterns in the study area.

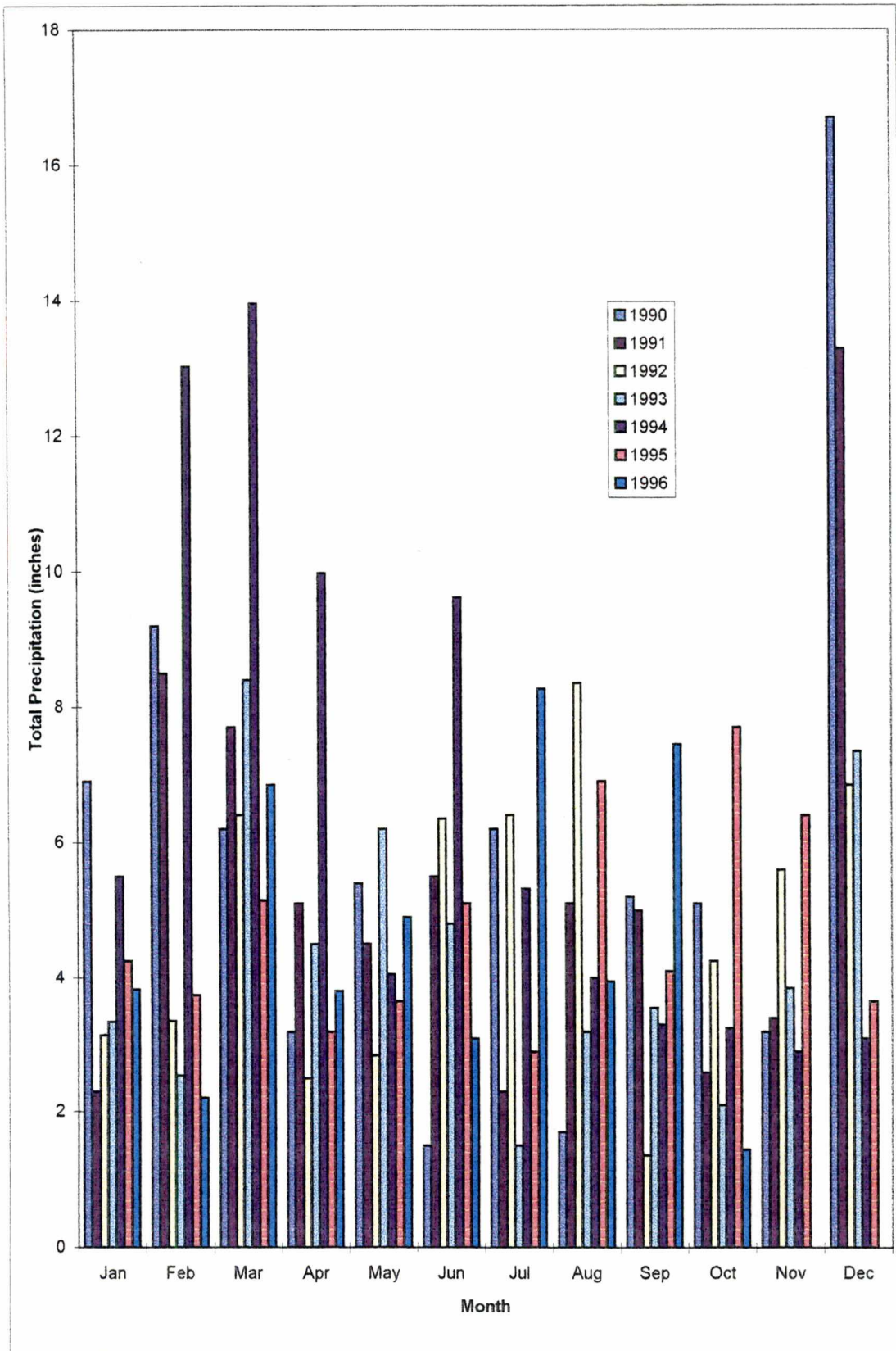


Figure 3: Monthly Precipitation Totals from 1990-October, 1996.

Table 2: Precipitation (Prp.) Data from 1990-October, 1996. Precipitation days/month, total precipitation/month and year, average precipitation/month and average amount of precipitation/event (Precipitation reported in inches).

	1990		1991		1992		1993		1994		1995		1996		Total Prp.	Avg. Prcp./ Month
	Prcp. Days	Total Prcp.	Prcp. Days	Total Prcp.	Prcp. Days	Total Prcp.	Prcp. Days	Total Prcp.	Prcp. Days	Total Prcp.	Prcp. Days	Total Prcp.	Prcp. Days	Total Prcp.		
Jan.	7	6.9	8	2.3	6	3.15	10	3.65	10	5.55	7	4.25	6	3.83	29.63	4.23
Feb.	7	9.2	7	8.5	6	3.36	5	2.55	7	13.03	6	3.74	4	2.21	42.59	6.08
Mar.	8	6.2	7	7.7	9	6.4	11	8.4	10	13.96	5	5.15	9	6.85	54.66	7.81
Apr.	6	3.2	13	5.1	7	2.5	10	4.5	9	9.98	6	3.2	6	3.8	32.28	4.61
May	7	5.4	7	4.5	8	2.85	8	6.2	5	4.05	8	3.65	5	4.9	31.55	4.51
June	2	1.5	10	5.5	9	6.35	8	4.8	17	9.62	10	5.1	7	3.1	35.97	5.14
July	8	6.2	8	2.3	11	6.4	4	1.5	14	5.32	5	2.9	11	8.27	32.89	4.70
Aug.	3	1.7	7	5.1	6	8.35	7	3.2	8	4	9	6.9	6	3.95	33.20	4.74
Sep.	5	5.2	4	5	3	1.35	8	3.55	6	3.3	7	4.1	7	7.45	29.95	4.28
Oct.	6	5.1	3	2.6	7	4.25	5	2.1	9	3.25	6	7.7	3	1.44	26.44	3.78
Nov.	5	3.2	6	3.4	6	5.6	4	3.85	4	2.9	6	6.4			25.35	4.23
Dec.	8	16.7	7	13.3	12	6.85	7	7.35	6	3.1	6	3.65			50.95	8.49
Total	72	70.5	87	65.3	90	57.41	87	51.65	105	78.06	81	56.74				
Avg. Prcp. /Day		0.98		0.75		0.64		0.59		0.74		0.70				

Surface Water Hydrology

Streamflow for the typical stream on the Cumberland Plateau is composed of direct surface runoff from precipitation events and from groundwater base flow (May et al., 1981). Most of the streams on the Plateau exhibit ephemeral flow characteristics, going dry or experiencing periods of very low flow during the summer and fall months. The peak flows within the watersheds are typically during wet months of winter and early spring.

Figure 4 is a topographic map depicting the drainage pattern of the study area prior to the mining activity documented in this study. As depicted in Figure 4, the study area is bounded on the east by an ephemeral stream, Spring Branch, and on the south by Glady Fork Creek. Spring Branch exhibits periods of surface flow during wet seasons, while Glady Fork Creek flows year-round with peak flow during wet seasons. The mottled purple areas indicate areas that have been mined prior to 1988. As indicated in Figure 4, Glady Fork Creek flows through areas mined prior to and unrelated to the current mining operations.

Groundwater Hydrology

The major aquifers of the Cumberland Plateau are considered to be the essentially flat-lying formations of sandstone and conglomerate, with interbedded shale and coal often acting as local confining beds (May et al., 1981). Although some of the sandstone and conglomerate contain significant primary porosity in the form of intergranular pores, most of these rocks are firmly cemented, and exhibit mainly secondary or fracture porosity (May et al., 1981). Groundwater flow in strata of the Cumberland and Appalachian Plateau is mainly controlled by the permeability created by stress-relief fractures (Hawkins et al., 1996). The fracture systems provide for storage and transmission of most of the groundwater and as a result, aquifer yields depend on degree of fracturing and are highly variable (May et al., 1981).

The Newton sandstone and Sewanee conglomerate are the main aquifers in the region of the Cumberland Plateau where the study area is located, with the Newton being a primary source of drinking water for residents outside of the study area. Infiltrating groundwater reaching the Newton sandstone-Whitwell shale interface, most likely migrates southeasterly, following the general regional direction of dip.

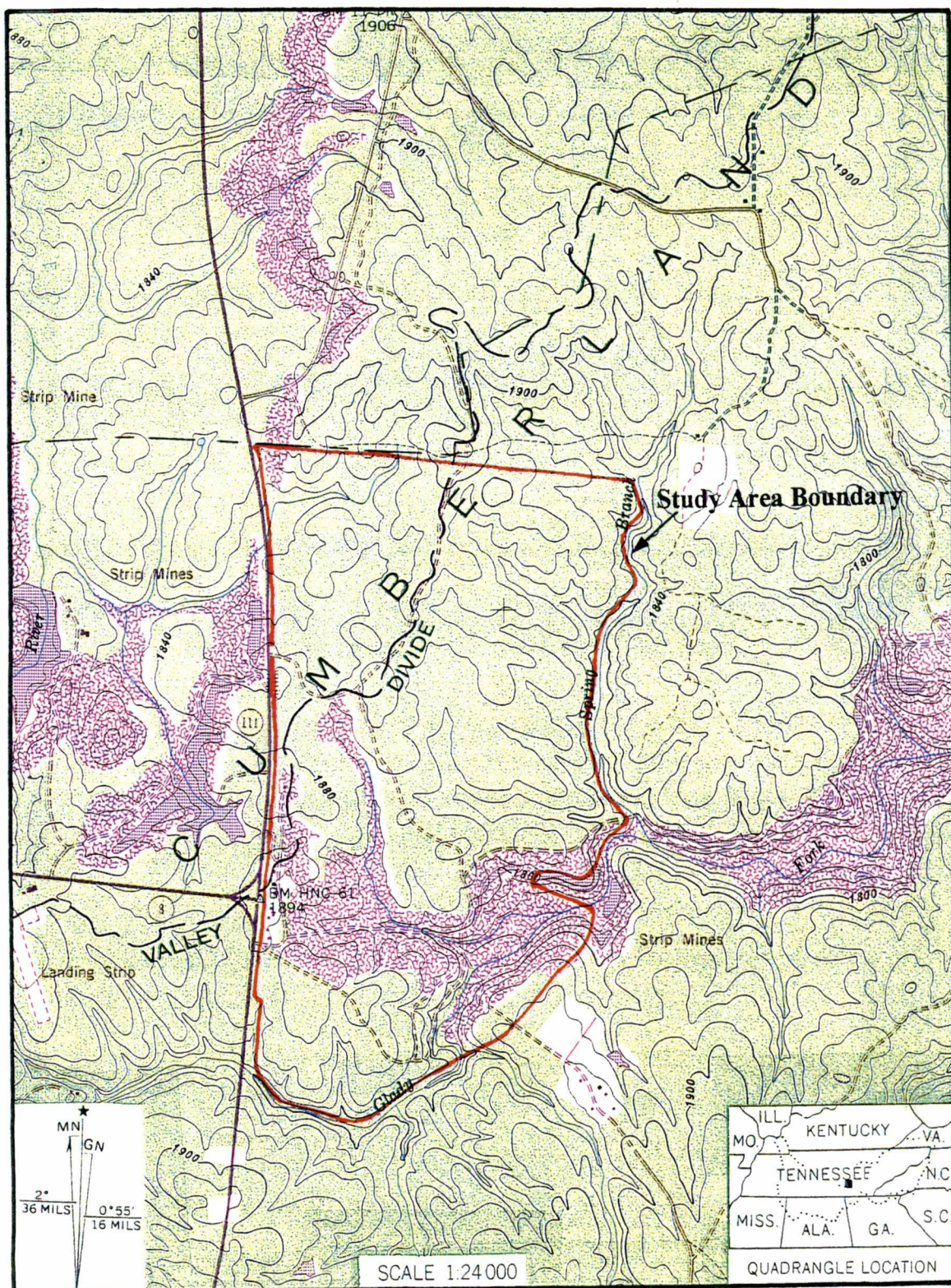


Figure 4: Topography of study area prior to mining indicating streams and drainage patterns (TVA Topographic Map 7.5 Minute Series 103-SW).

Chapter 6.

Mining Operations and Mitigation Procedures

History of Mining Operations

Coal mining had been conducted in the study area prior to the initiation of mining operations documented in this study (see Figure 4). The mining toward which this study is directed began in September, 1987 and was completed in May, 1993. Acid-base accounting (ABA) analyses of mine overburden conducted prior to mining by mine operators indicated that mining could be done with minimum concerns relative to the generation of AD and related adverse hydrologic impacts (Wiram and Naumann, 1995). In mid-1990, however, three, slightly acidic to alkaline, manganese-enriched, ferruginous seeps were observed emanating from the mine site into an adjacent stream. Seepage flows were determined by mine operators to be originating from the mine disturbance areas with movement to the stream via natural fracture systems existing in the stream buffer zone and stream bed itself (Wiram and Naumann, 1995). The mine operators determined that alkalinity associated with the dissolution of siderite within certain horizons in the overburden had interfered with results of the first ABA tests, causing inaccurate prediction of AD.

Primary water quality parameters in the seeps were as follows: pH = 3.4 to 7.5; alkalinity = 0 to 121 mg/l as CaCO_3 ; total iron (Fe) = 4.8 to 48.6 mg/l; total manganese (Mn) = 2.3 to 34 mg/l; and, sulfate (SO_4^{2-}) = 8 to 812 mg/l (Wiram and Naumann, 1995). The mining company was issued a Notice of Violation (NOV) defining the seepage discharge as a pollution source to waters of the State of Tennessee, thus requiring remedial action plans to eliminate or minimize water quality degradation.

In order to eliminate the seeps, mining operators devised a strategy that included:

- 1) Removing a sediment basin above the creek that was creating a hydraulic gradient allowing discharge to the creek (seep #1).
- 2) Installing two dewatering wells adjacent to the creek to lower the water table below the creek,

thereby eliminating discharge from mine backfill to the creek (seep #2).

3) The installation of a sand-bag dike system to be constructed upstream and downstream of seep #3 to channel seepage through natural wetlands, allowing iron to drop out prior to re-entering the creek.

Figure 5 indicates the location of the seeps, the dewatering wells, the sediment basin that was removed, and the area where the sand-bag dike system was installed. The sediment basin was removed in August, 1990 with an immediate reduction in flow from seep #1. Dewatering wells #1 and #2 were completed and began operating in October, 1990. Flow at seep #2 stopped as a result of pumping dewatering wells #1 and #2. The sand-bag dike system, near seep #3, was completed in October, 1990.

Toxic Material Handling Plan

As a course of action to mitigate future adverse environmental impacts, the mine operators developed a comprehensive mining and reclamation plan involving selective handling and placement of overburden during mining and reclamation. The plan, referred to as a Toxic Materials Handling Plan (TMHP), was implemented after roughly ninety percent of the mine acreage had been mined using normal stripping and backfilling methods.

Stratigraphic complexity due to the depositional paleoenvironment (see Ferm, 1974, and Milici, 1977), was responsible for considerable variability of the geochemistry of mine overburden, necessitating certain baseline information before the TMHP could be implemented. This information essentially concerns the geology of the mine, including the three salient elements:

- 1) Overburden stratigraphy.
- 2) Structural aspects of the rock.
- 3) Mineral chemistry of the coal and overburden.

The TMHP was designed to meet the following objectives:

- 1) Subdue the effects of bacteria on pyritic oxidation by controlling pH through the use of limestone additions and surficial surfactants.
- 2) Neutralize acidity created by reactions occurring within a sandy shale unit in mine overburden.

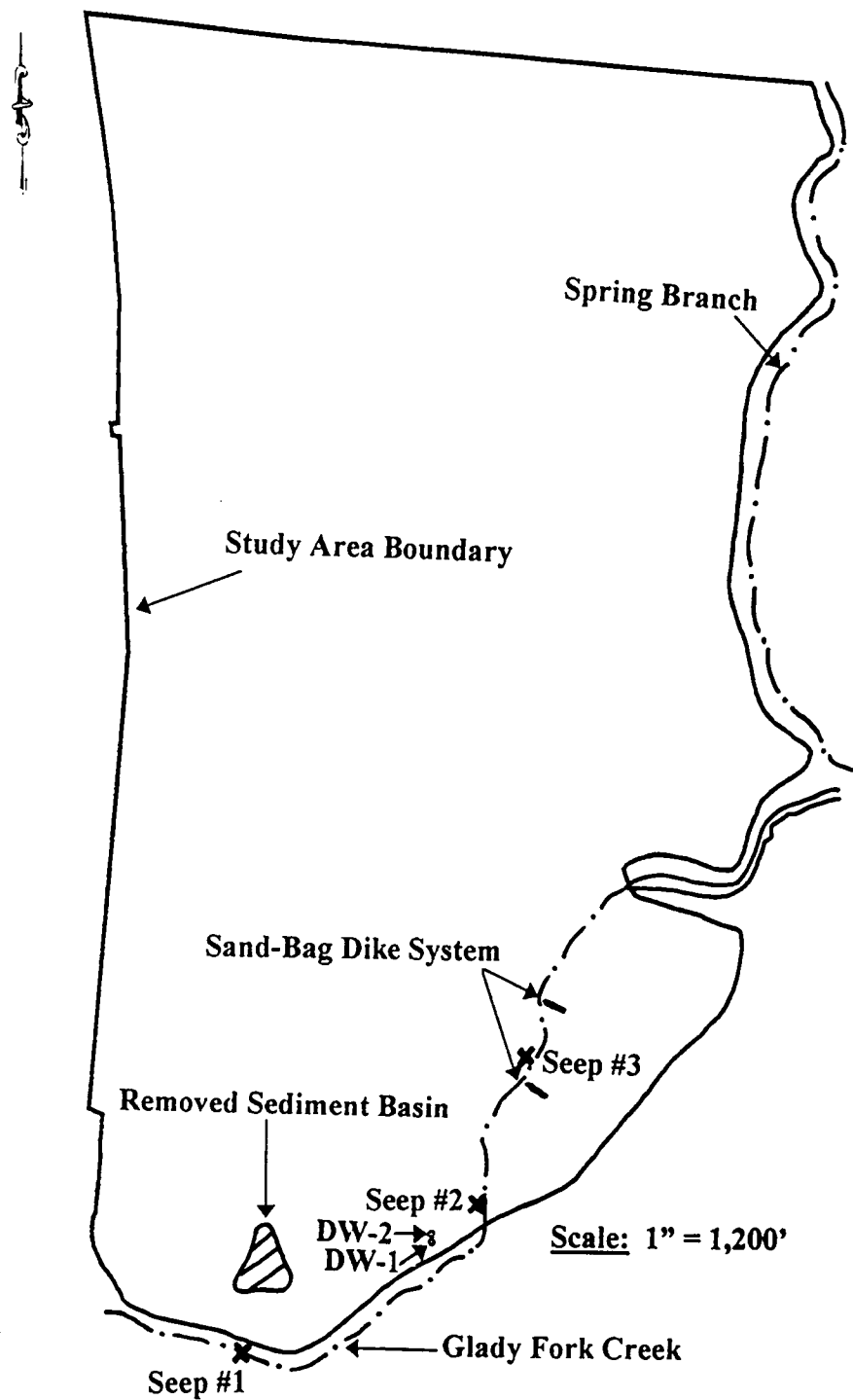


Figure 5: Map of study area indicating location of seeps, dewatering wells, removed sediment basin, and sand-bag dike system.

3) Provide alkaline recharge capabilities through construction of pre-defined hydrologic flow-paths to promote long-term alkaline loading in the backfill.

4) Enhance the stability field of siderite (a possible source of elevated Fe and Mn values in mine discharge) contained within the overburden through strategic addition of limestone to the reclaimed spoil.

5) Ensure that sufficient alkalinity levels are maintained in the backfill during "wetting" and "drying" cycles of a fluctuating water table associated with initial groundwater re-establishment.

Six key elements, or steps, were designed to meet the TMHP objectives. These elements include:

1) The use of interlayered limestone additions in designated positions in the backfill targeted to prevent and/or treat the immediate impacts of pyrite oxidation and siderite dissolution. Six areas were targeted for limestone and agricultural limestone amendments:

- Agricultural limestone on replaced topsoil

- Topsoil/graded spoil interface.

- Spoil-side dragline bench.

- Pit coal cleanings.

- Underclay/shale of the pit floor.

- Alkaline Recharge Structures.

Because of the substantial quantities of limestone to be applied to the backfill to account for the neutralization deficiencies identified for the involved strata and to compensate for possible unforeseen shortcomings in the mine's overall success at achieving total selective handling in the backfill, the mine operators split the limestone application into three equal portions (Wiram and Naumann, 1995).

The three sections of the backfill that were targeted for limestone amendments included: 1) topsoil/backfill interface, 2) backfill dragline bench, and 3) direct cast profile

The mine operators further split the lime targeted for the "direct cast" profile of the muck pile into two separate applications (Wiram and Naumann, 1995). Half of the required lime amount was applied to the pit floor and spread in a uniform lift while the remaining half of the lime planned for the direct cast profile was placed and spread over the tractor push area (Figure 6) of the muck pile (Wiram and Naumann, 1995).

2) Selective placement and handling of potentially acid forming materials by specially designed cast-blasting techniques and movement of overburden via a model 1300 dragline machine, bulldozers, and loading equipment. The selective handling and placement of problematic material, illustrated in Figure 6, was designed to isolate the material from air and rapid infiltration of groundwater.

3) Compaction of pyrite bearing overburden and coal-cleaning materials via equipment trafficking and dragline movement during normal stripping activities forming impermeable layers within the backfill to establish preferred routing of groundwater recharge around underlying potentially acid-producing material.

4) Routing of surface flow within the backfill area through Alkaline Recharge Structures (ARSs) to limit the contact of groundwater with potentially acid-producing material and to increase the bicarbonate alkalinity within the backfill.

5) Groundwater inundation of backfill as a long-term preventative measure to ensure future avoidance of pyrite oxidation and subsequent AD from the buried coal cleanings. At a minimum, the basal five feet of the pit floor is planned for permanent inundation following mining and reclamation.

6) Contemporaneous reclamation (contouring and vegetating the surface) enhances rapid build-up of organic matter (oxygen sink) in the replaced soil profile by routinely mowing and dropping vegetation cuttings in the reclaimed land.

Key Methods, Parameters, and Assumptions of the TMHP

The key methods used to implement the TMHP as outlined by Wiram and Naumann, 1995 are indicated below:

1) Overburden was sampled ahead of mining on a close-spaced grid (approximately 500 feet) using air-rotary or core recovery methods. Individual samples were collected at 5-foot vertical intervals in each exploratory hole.

2) All amendment rates associated with alkaline additions to the backfill were determined by utilizing the modified neutralization potential (MNP) procedure for the purpose of eliminating "siderite masking" (Wiram, 1992). MNP was expressed in terms of tons CaCO_3 /1000 tons of material.

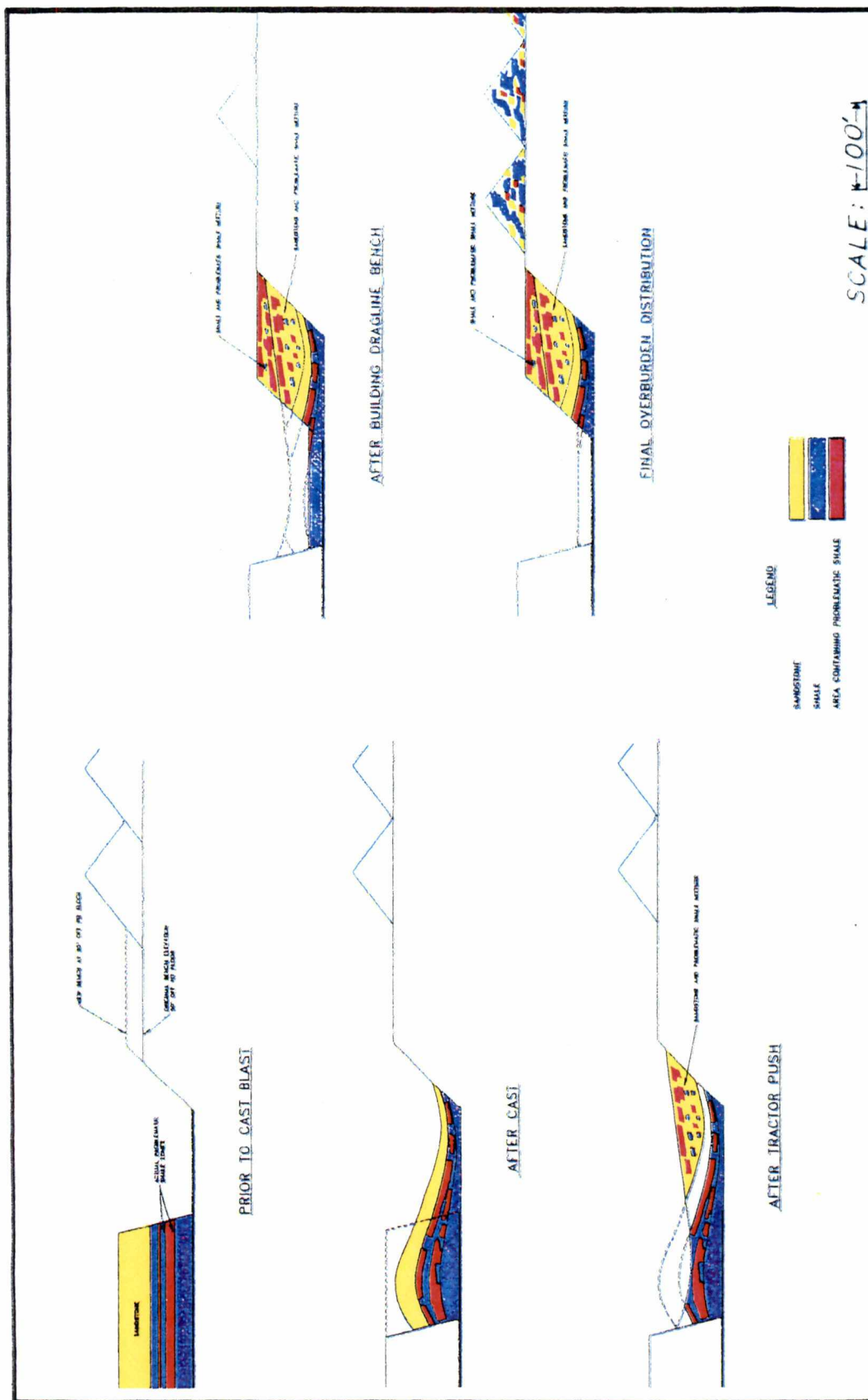


Figure 6: Selective Handling and Placement Schematic.

3) The standard multiplication factor of 31.25 times the pyritic sulfur content was used in calculating potential acidities (PA's) of the various overburden materials. PA's were expressed in terms of tons CaCO_3 /1000 tons of material.

4) Modified Net Neutralization Potential (MNNP) was defined as: MNP minus PA.

5) The acid-potential determined in overburden within a 250 foot radius around each borehole was used to estimate the total quantity of alkalinity required.

6) Limestone application rates were based on "negative MNNP" zones identified in each borehole. These were also denoted as "net deficient" zones. Individual net deficiencies (tons CaCO_3 /1000 tons of material) were deemed applicable to the 5-foot interval from which they were determined.

7) The alkalinity of "net neutral" zones in each exploratory hole was not factored into the limestone quantity estimation calculations.

8) Limestone application rates were based on a stone quality of 90% pure CaCO_3 .

As previously mentioned the TMHP was implemented after roughly 90% of the study area had been mined and as a result, only the final stages of the mining operation incorporated the TMHP. Figure 7 depicts the boundary where the TMHP was implemented, the location of groundwater observation wells, and the location of Alkaline Recharge Structures (ARSs).

In order to control AD in the area where the TMHP had not been implemented, mine operators designed Alkaline Recharge Structures (ARS's) and added 10 tons/acre of "crusher run" limestone to the graded spoil/topsoil interface to enhance alkalinity and control pyrite oxidation in the backfill. The addition of the "crusher run" limestone began in August, 1992 in the area where OW-8 is presently located.

The alkaline-enhanced groundwater migrating through the backfill is believed, by mine operators, to eventually abate adverse water quality within the entire backfill of the study area. The mine operators base their belief, in part, on the findings of Caruccio (1996) where studies have shown alkaline material (calcium carbonate) occurring hydrologically upgradient of the acid producing material, produces alkaline infiltrating waters which may effectively or partially neutralize the acidity produced by toxic materials.

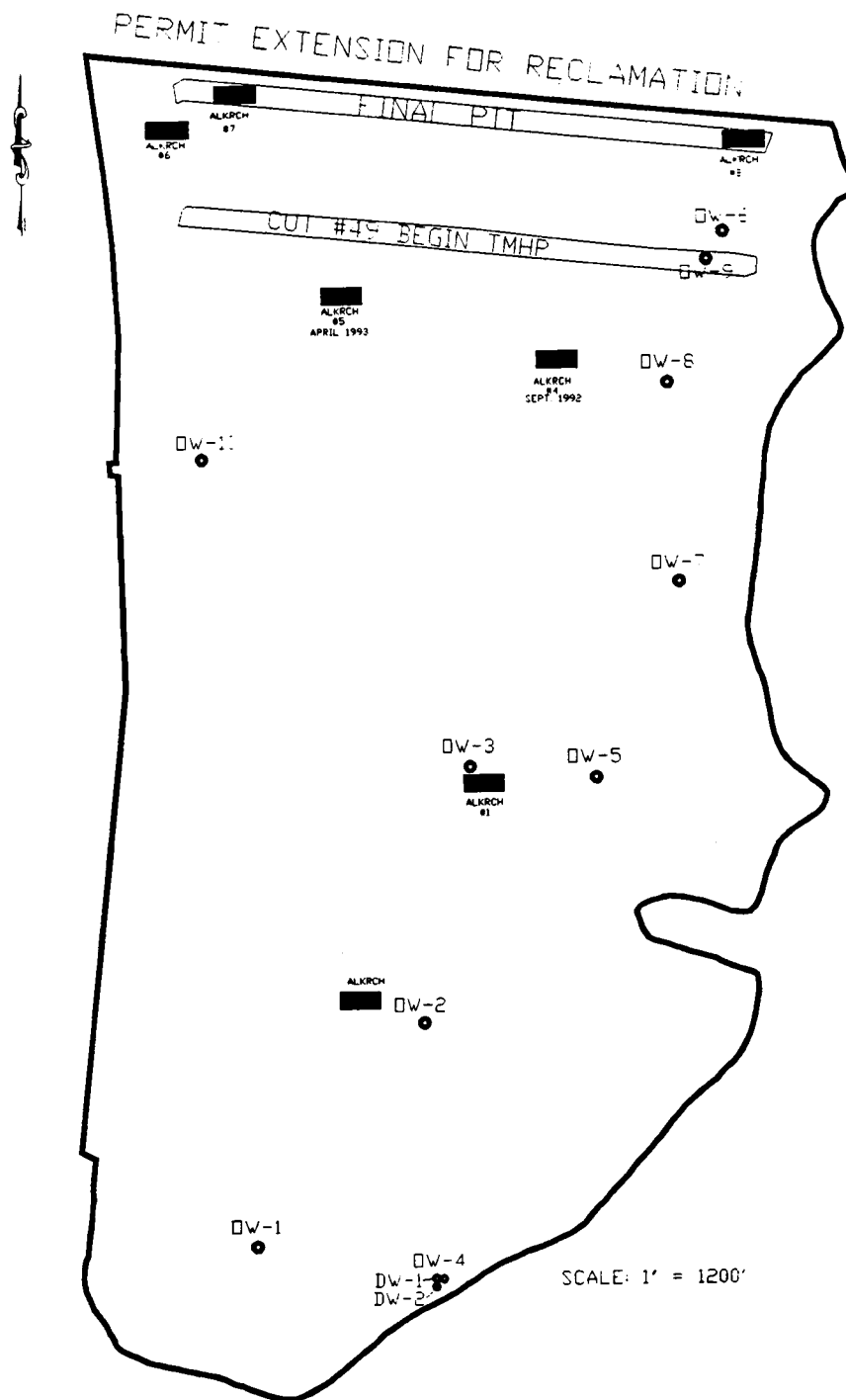


Figure 7: Diagram illustrating boundary where the TMHP was implemented, the location of alkaline recharge structures, and groundwater observation wells.

Chapter 7.

Geochemical Analysis of Mine Overburden

Introduction

In order to identify geologic horizons capable of generating AD in the study area, in particular, to determine the source, or potential sources, of elevated Mn concentrations in backfill groundwater, a geochemical analysis of representative overburden material was performed. Previous studies by Caruccio (1991) and Wiram (1995) have identified potential acid-producing material in the overburden of the study area; however, the studies did not concentrate on, or determine a source for Mn contained in the overburden. The representative samples that were collected for this study were based largely on findings of Caruccio (1991) and Wiram (1995) that identified geologic horizons as potential sources for AD.

Representative overburden samples were collected for geochemical analyses from the spoil pile of an active, surface coal mine located adjacent to the study area. The samples are considered to be representative of overburden in the study area prior to mining because the active mine is mining the same stratigraphic section that was mined in the study area.

A combination of analytical techniques including Electron Microprobe (EMP), X-ray Fluorescence (XRF), and X-ray Diffraction (XRD) analyses were used to determine the geochemistry of the representative samples. Representative highwall samples that were analyzed included:

- 1) Siderite clasts and siderite beds.
- 2) Whitwell shale unit.
- 3) Whitwell shale between siderite beds.
- 4) A sandy shale unit.
- 5) Newton sandstone.

- a) Gray Newton sandstone.

b) Tan Newton sandstone.

c) Pink Newton sandstone.

All representative samples were analyzed using XRF. XRD was used to determine phases present in samples of a siderite clast, the clay fraction of the siderite clast, and the Whitwell shale between siderite beds. A representative siderite clast was the only sample analyzed using EMP. EMP analyses are normally not used on sedimentary rocks due to the small degree of crystallinity relative to igneous rocks. This study, however, concentrated on identifying homogeneous phases in the siderite clast that would enable EMP analyses. In order to compare EMP results to traditional sedimentary rock analytical techniques, XRF was used to analyze the same siderite clast.

Sample selection and the methods that were used to prepare the siderite clast for geochemical analyses are included in Appendix A. The analytical techniques and methods used to determine the geochemistry of the siderite clast and other representative overburden samples are included in Appendix B.

EMP Results

The EMP analysis of the siderite clast was performed using two methods, one assuming the siderite was an oxide and the other assuming the siderite was a carbonate. Table 3 contains the EMP results produced by the broad-beam (15 μ m) traverse analysis of the siderite clast. The broad-beam traverse analysis analyzed the siderite clast as an oxide, assuming CO₂ by difference (i.e. adding CO₂ to total 100%). The values in parentheses represent the standard deviation (sigma values) among replicate analyses. The EMP analyses resulted in relatively high sigma values. The reasons for the high sigma values are discussed in Appendix C. The weight percent MnO determined from the broad-beam analysis averaged 1.16 weight % MnO, equivalent to 0.89 weight % Mn.

Results obtained from EMP analyses of the siderite clast, assuming the siderite as a carbonate, are contained Table 4. The Mn weight % determined from this analysis revealed an average of 1.80 weight % MnCO₃ which is equivalent to 0.86 weight % Mn. Table 5 contains an individual point analysis of the siderite clast and clay phase assuming the siderite as an oxide. This analysis revealed an average of 1.12

Table 3: EMP Broad-beam analysis. Values are elemental weight percent, values in parentheses are standard deviation.

		SAMPLE #																	
Ele.	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	24	25	Average	
FeO	35.8	33.8	41.4	41.8	26.3	45.1	45.2	45.9	46.9	45.5	46.4	46.8	43.3	45.5	44.0	40.8	45.8	42.4(6)	
MnO	0.96	0.99	1.17	1.14	0.87	1.25	1.31	1.24	1.25	1.23	1.23	1.22	1.18	1.17	1.10	1.09	1.26	1.16(.12)	
CaO	3.01	3.04	3.88	3.71	2.23	4.20	3.86	4.14	3.98	4.30	3.78	3.58	4.33	4.16	3.99	3.58	3.60	3.73(.54)	
MgO	3.03	3.44	4.05	3.74	3.09	3.86	3.23	3.49	3.15	3.26	3.17	3.29	3.45	3.28	3.65	3.81	4.61	3.51(.41)	
K2O	0.60	0.90	0.72	0.66	0.94	0.44	0.29	0.40	0.37	0.39	0.55	0.47	0.97	0.57	0.54	0.79	0.27	0.58(.22)	
SiO2	24.1	18.1	8.27	7.67	22.4	4.50	7.40	4.21	2.86	4.20	3.69	3.58	6.20	4.79	7.09	8.69	2.52	8.26(.67)	
Al2O3	4.01	4.84	3.84	5.22	11.3	1.99	1.29	1.84	1.54	1.78	2.06	1.75	3.63	2.61	2.71	4.01	1.56	3.29(.41)	
Na2O	0.05	0.07	0.09	0.08	0.07	0.08	0.07	0.09	0.07	0.08	0.09	0.08	0.11	0.09	0.10	0.07	0.06	0.08(.02)	
TiO2	0.07	0.03	0.16	0.10	0.19	0.16	0.06	0.06	0.09	0.08	0.06	<0.03	0.12	0.07	0.14	0.08	0.45	0.12(.1)	
P2O5	0.42	0.45	0.71	0.62	0.24	0.83	0.73	0.87	0.63	1.00	0.68	0.69	1.02	0.96	0.68	0.57	0.39	0.67(.22)	
SO3	0.05	0.14	<0.03	<0.03	0.17	<0.03	<0.03	<0.03	<0.03	0.04	<0.03	0.05	0.05	0.07	<0.03	0.03	0.05	0.07(.05)	
CO2	27.7	34.2	35.7	35.2	32.3	37.6	36.6	37.7	39.1	38.1	38.2	38.4	35.7	36.8	36.1	36.5	39.5	36.2(3)	
Total	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	

Table 4: EMP Analysis of Siderite Clast as a Carbonate. Reported values are elemental weigh percent. Standard deviation in parentheses.

Cmpd.	SAMPLE						
	1.2	1.21	1.22	1.23	M-1	M-2	Avg.
Mg(CO ₃)	6.66	6.57	6.84	7.51	6.89	6.01	6.75(0.5)
Ca(CO ₃)	7.56	6.16	7.45	8.06	7.98	8.31	7.59(0.8)
Mn(CO ₃)	1.77	1.95	1.86	1.76	1.38	2.07	1.80(0.3)
Fe(CO ₃)	74.4	79.3	81.2	76.5	79.4	76.9	77.9(2.5)
Total	90.4	93.9	97.4	93.9	95.6	93.3	
Cations based on 6 oxygens:							
Mg	0.195	0.185	0.186	0.211	0.19	0.174	
Ca	0.186	0.146	0.171	0.191	0.186	0.198	
Mn	0.038	0.04	0.037	0.036	0.028	0.043	
Fe	1.581	1.628	1.606	1.563	1.596	1.585	
Total	2	1.999	2	2.001	2	2	

Table 5: EMP Individual Point Analysis of Siderite Clast. Values are elemental weight percent. Standard deviation in parentheses.

Cmpd.	SAMPLE						
	Sid.G-1	Sid.G-2	Sid.G-3	Average	Siderite Clay G-1	Siderite Clay -2	Average
FeO	48.9	49.8	48.7	49.1(0.6)	2.82	6.61	4.72
MnO	1.04	1.11	1.22	1.12(0.1)	0.13	0.23	0.18
CaO	3.61	4.02	3.99	3.88(0.2)	0.17	0.61	0.39
MgO	4.97	3.06	2.99	3.67(1.1)	2.81	2.13	4.94
K ₂ O	<0.03	0.09	0.18	0.13(0.1)	8.83	7.38	8.10
SiO ₂	0.10	0.63	1.26	0.66(0.6)	47.6	40.4	44.0
Al ₂ O ₃	0.06	0.42	0.88	0.45(0.4)	30.4	26.9	28.6
Na ₂ O	0.04	0.07	0.09	0.07(0.02)	0.14	0.14	0.14
TiO ₂	<0.03	<0.03	<0.03	NA	0.09	0.61	0.35
P ₂ O ₅	0.33	0.54	0.50	0.46(0.1)	<0.03	<0.03	NA
SO ₃	<0.03	<0.03	<0.03	NA	<0.03	<0.03	NA
CO ₂	40.9	40.3	40.1	40.42	7.0	15.0	11.0
Total	100	100	100	99.93	100	100	NA

weight % MnO, which is equivalent to 0.86 weight % Mn. The results of all analyses obtained by EMP revealed an average of 0.87 weight % Mn contained in the siderite.

Figure 8 is a ternary diagram developed using the EMP results obtained by analyzing the siderite clast as an oxide and as a carbonate. The cluster of points indicates that this sample can be classified as siderite, however the Mn concentrations in the siderite clast determined by EMP analyses are significant and could contribute to Mn content in the AD waters.

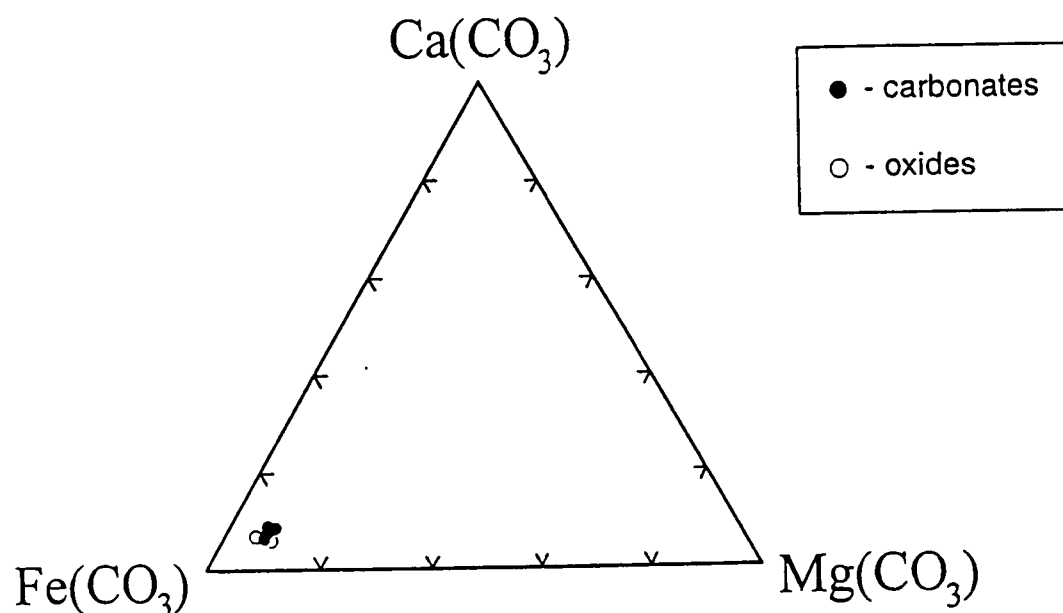


Figure 8: Ternary diagram representing EMP analytical results.

XRF Results

XRF analytical results of the siderite clast, siderite beds, Whitwell shale, a sandy shale unit within the Whitwell shale, Whitwell shale between siderite beds, and the Newton sandstone are included below. The list of compounds and elements that were analyzed for each representative sample are not consistent because different protocols are designed to detect specific compounds and elements.

Siderite Clast

Table 6 and Table 7 include XRF analytical results of the siderite clast (samples SCXRF1 and SCXRF2) using the LSDOL and FECLAY protocols. XRF analyses revealed Mn concentrations ranging from 1.4 weight % Mn, using the LSDOL protocol, to 0.28 weight % Mn, using the FECLAY protocol. The Mn concentration determined using the FECLAY protocol was out of the range of standards used in the protocol and the results were not the same as those determined using the LSDOL protocol. A discussion of the reasons for the variation is included in Appendix C.

Siderite Beds

The siderite bed samples were analyzed using the FECLAY protocol. Analyses of the siderite beds (sample numbers SB1 to SB6), included in Table 8, indicated that Mn concentrations were out of the range of intensity of the standards, a problem discussed in Appendix C. The average of 0.41 weight % MnO, or 0.32 weight % Mn, indicates that the siderite beds possess a minor amount of Mn that can contribute to AD. This value can be considered a “low end” value and concentrations may be larger. Iron concentrations in the siderite beds averaged 13.67 weight % Fe. The relative high Fe concentrations and Mn content indicate that the siderite beds can be considered potential acid-producers. The sulfur content of the siderite beds was below detection limits.

Table 6: XRF Analyses of Siderite
Clast Using LSDOL Protocol.

Cmpd.	SCXRF1	SCXRF2	Average
Fe2O3	33.9	33.9	33.9
Mn	1.38	1.38	1.38
CaO	0.66	0.66	0.66
MgO	2.98	3.04	3.01
K2O	0.37	0.39	0.38
SiO2	9.24	9.32	9.28
Al2O3	0.23	0.25	0.24
Na2O	0.02	0.02	0.02
P2O5	0.14	0.14	0.14
Total	48.9	49.0	49.0

Table 7: XRF Analyses of Siderite
Clast Using FECLAY Protocol.

Cmpd.	SCXRF1	SCXRF2	Average
Fe2O3	61.30	60.40	60.85
MnO	[0.35]	[0.36]	[0.36]
CaO	3.87	3.85	3.86
MgO	3.87	3.83	3.85
K2O	0.51	0.51	0.51
SiO2	9.87	9.79	9.83
Al2O3	3.64	3.72	3.68
Na2O	0.91	1.09	1.00
P2O5	0.31	0.31	0.31
S	0.03	0.03	0.03
TiO2	0.35	0.34	0.35
Cr	0.01	0.01	0.01
Ba	0.04	0.08	0.06
Rb	0.01	0.01	0.01
Sr	0.01	0.01	0.01
Zr	0.01	0.01	0.01
Total	85.10	84.40	84.37

Note: [] denotes intensity out of range

Table 8: Siderite Beds XRF Analyses.

Cmpd.	SAMPLE					
	SB1	SB2	SB3	SB4	SB5	SB6
Fe2O3	37.9	43.4	15.4	37.4	40.8	59.4
MnO	[0.43]	[0.40]	[0.33]	[0.44]	[0.41]	[0.45]
CaO	1.31	1.77	0.78	1.31	1.27	3.05
MgO	1.81	1.97	1.26	1.79	1.62	3.54
K2O	1.39	1.13	2.23	1.41	1.16	0.05
SiO2	38.7	34.2	60.5	39.5	36.1	5.72
Al2O3	10.1	8.68	13.9	10.3	9.54	2.92
Na2O	0.69	0.57	0.49	0.48	0.48	1.06
P2O5	0.17	0.17	0.18	0.17	0.19	[1.01]
S	[0.02]	[0.02]	0.001	[0.02]	[0.02]	0.13
TiO2	0.55	0.51	0.79	0.56	0.51	0.32
Cr	0.011	0.009	0.012	0.011	0.011	0.009
Ba	0.046	0.044	0.054	0.041	0.036	0.035
Rb	0.007	0.007	0.01	0.008	0.006	0.004
Sr	0.012	0.013	0.014	0.013	0.011	0.021
Zr	0.017	0.018	0.029	0.018	0.017	0.012
Total	92.7	92.5	95.7	93.0	91.8	76.3

Note: [] indicates intensity out of range.

Whitwell Shale

Analyses of the Whitwell shale (samples WSH1 to WSH4) using the FECLAY protocol, included in Table 9, indicate that the Whitwell shale does not pose a threat to the generation of AD. The low Fe concentrations (averaging 1.64 weight %) and low Mn concentrations (0.09 weight %) combined with low pyritic sulfur content (averaging 0.02 weight %), do not justify classifying the Whitwell shale as a potential AD contributor.

Sandy Shale Unit Within the Whitwell Shale

Analyses of the sandy shale (SSH1-SSH4) unit, using the FECLAY protocol, included in Table 10, indicate concentrations of FeO and MnO (1.99 weight % Fe and 0.09 weight % Mn) similar to the Whitwell shale, however the sulfur content of the sandy shale averages 1.35 weight %. Byerly (1996) suggests units containing pyritic sulfur in excess of 0.5 weight % and low neutralization potential are prone to produce AD.

Whitwell Shale Between Siderite Beds

Table 11 contains the results obtained from XRF analyses of the Whitwell shale between siderite beds using FECLAY protocol. MnO weight % averaged 0.068 weight % MnO, equivalent to 0.05 weight % Mn. Fe₂O₃ weight % in the shale averaged 3.8 weight % Fe₂O₃, or 1.33 weight % Fe. Sulfur concentrations, averaging 0.02 weight %, were similar to concentrations in the Whitwell shale analyses. The relative low Fe, Mn, and sulfur concentrations indicate that the Whitwell shale between siderite beds is similar in geochemistry to the Whitwell shale and does not pose a threat to the generation of AD.

Newton Sandstone

Table 12 contains the results obtained from the XRF analyses of the Newton sandstone using the SIAL protocol. Samples GSS2, TSS1, and PSS1 represent analyses of the gray, tan, and pink Newton sandstone lithotypes respectively.

Table 9: Whitwell Shale XRF Analyses.

Cmpd.	SAMPLE			
	WSH1	WSH2	WSH3	WSH4
Fe2O3	4.65	4.69	4.69	4.72
MnO	0.11	0.11	0.11	0.11
CaO	0.33	0.33	0.34	0.033
MgO	1.15	1.14	1.17	1.25
K2O	3.23	3.26	3.29	3.17
SiO2	69.2	69.9	69.6	69.6
Al2O3	17.9	18.1	18.2	18.3
Na2O	0.61	0.51	0.69	0.59
P2O5	0.16	0.16	0.16	0.21
S	0.02	0.02	0.02	0.02
TiO2	0.97	0.99	0.98	0.99
Cr	0.007	0.007	0.007	0.007
Ba	0.063	0.066	0.066	0.074
Rb	0.016	0.016	0.015	0.015
Sr	0.016	0.017	0.017	0.017
Zr	0.031	0.031	0.031	0.031
Total	98.5	99.3	99.4	99.1

Table 10: Sandy Shale XRF Analyses.

Cmpd.	SAMPLE			
	SSH1	SSH2	SSH3	SSH4
Fe2O3	5.68	5.68	5.69	5.64
MnO	0.11	0.11	0.10	0.11
CaO	0.77	0.77	0.78	0.77
MgO	1.35	1.41	1.47	1.54
K2O	2.86	2.87	2.86	2.84
SiO2	54.5	55.2	55.2	55.1
Al2O3	25.1	25.5	25.4	25.5
Na2O	0.93	1.12	0.98	1.05
P2O5	0.61	0.63	0.61	0.63
S	1.33	1.36	1.34	1.35
TiO2	1.61	1.62	1.64	1.62
Cr	0.011	0.011	0.011	0.011
Ba	0.137	0.143	0.149	0.154
Rb	0.025	0.027	0.025	0.026
Sr	0.089	0.088	0.092	0.091
Zr	0.034	0.035	0.033	0.033
Total	95.1	96.6	96.4	96.5

Table 11: Whitwell Shale Between Siderite Beds.

Cmpd.	SAMPLE							
	BSH1	BSH2	BSH3	BSH4	BSH5	BSH6	BSH7	BSH8
Fe2O3	3.81	3.79	3.74	3.75	3.67	3.67	4.01	3.98
MnO	0.06	0.06	0.06	0.06	0.07	0.07	0.08	0.08
CaO	0.32	0.32	0.32	0.35	0.26	0.24	0.27	0.29
MgO	1.17	1.15	1.19	1.34	0.98	1.29	1.04	0.95
K2O	3.26	3.26	3.23	3.23	2.78	2.58	2.79	2.76
SiO2	68.2	69.2	68.5	68.7	73.4	73.7	73.1	71.9
Al2O3	18.5	18.5	18.4	18.6	15.9	16.5	16.1	15.8
Na2O	0.61	0.51	0.61	0.65	0.45	0.71	0.64	0.61
P2O5	0.17	0.17	0.17	0.21	0.18	0.33	0.17	0.18
S	0.02	0.02	0.02	0.02	[0.007]	[0.009]	[0.009]	0.01
TiO2	1.04	1.05	1.04	1.04	0.92	0.92	0.92	0.89
Cr	0.007	0.007	0.007	0.006	0.007	0.007	0.007	0.008
Ba	0.065	0.057	0.056	0.058	0.056	0.057	0.063	0.063
Rb	0.016	0.016	0.016	0.017	0.013	0.013	0.013	0.013
Sr	0.017	0.017	0.017	0.017	0.016	0.016	0.015	0.015
Zr	0.038	0.038	0.037	0.037	0.039	0.038	0.037	0.037
Total	97.3	98.2	97.4	98.1	98.7	100.1	99.3	97.6

Note: [] indicates intensity out of range.

Table 12: Newton Sandstone XRF Analyses.

Cmpd.	Sample		
	GSS2	TSS1	PSS1
Fe ₂ O ₃	0.68	1.79	0.62
MnO	0.01	0.02	0.01
CaO	0.07	[-0.22]	[-0.21]
MgO	0.26	0.24	0.31
K ₂ O	0.19	0.41	0.42
SiO ₂	97.5	98.3	98.8
Al ₂ O ₃	4.69	4.48	6.68
Na ₂ O	0.14	0.12	0.23
P ₂ O ₅	[-0.05]	[-0.05]	0.08
S	[-0.01]	[-0.01]	[-0.01]
TiO ₂	0.24	0.22	0.18
Ba	0.03	0.23	0.02
Rb	0.0004	0.0013	0.002
Sr	0.002	0.003	0.002
Zn	0.01	0.01	0.01
Zr	0.05	0.02	0.01
Ni	0.001	0.001	0.001
Total	103.9	105.3	107.2

Note: [] indicates intensity out of range.

The XRF analyses of the Newton sandstone resulted in high totals (i.e. above 100%). The reasons for the high reported totals are discussed in Appendix C. The XRF analyses of the Newton sandstone did reveal geochemical differences among strata within the unit. TSS1 contains 0.02 weight % MnO, equivalent to 0.015 weight % Mn. The GSS1 and PSS1 samples analyses indicated 0.01 weight % MnO, or roughly half the Mn contained in the TSS1 sample. The TSS1 sample also contained nearly three times as much Fe₂O₃, 1.79 weight %, than the GSS1 and PSS1 samples. Sulfur concentrations in the three sandstone samples were below detection limits (i.e. intensity was out of range).

XRD Results

XRD was used to determine mineralogical phases present in the siderite clast and in the Whitwell shale between siderite beds.

Siderite Clast and Clay Fraction in Siderite Clast

The major phases that were distinguished during petrographic and EMP analyses of the siderite clast included a quartz, a framboidal sulfide, a siderite and a clay phase. XRD analyses identified siderite, quartz and clay phases in the siderite clast. The XRD combined with the particle-size separation method was successful in determining a mineralogy for the clay phase. The clay phase is composed of illite, kaolinite and pennantite (chlorite) clay minerals.

Whitwell Shale Between Siderite Beds

XRD analyses of the Whitwell shale between siderite beds identified the siderite, quartz and clay phases that were also identified in the clay fraction of the siderite clast. The similarity in the mineralogy of the clay phases, suggest that the source of the clay fraction in the siderite clast originated from the Whitwell shale.

Summary

Geochemical analyses of the representative overburden samples identified the siderite lithology as the unit most capable of contributing elevated levels of Mn to the hydrologic regime. Although the siderite comprises a minor percentage of the total overburden and is considered non-toxic, EMP and XRF analyses revealed Mn concentrations that can contribute to the mineral acidity of the strata. EMP and XRF analyses determined a range of Mn (0.28 to 1.38 weight %) in the siderite clast. This range in Mn concentrations is believed to be most likely attributable to the analytical accuracy inherent in the analytical methods used.

The Mn concentrations averaging 0.87 weight % determined by EMP analyses appear to rest between the low and high end values determined by XRF. The broad-beam traverse analysis, individual point analysis and carbonate analysis, reveal a consistent concentration of 0.9 weight % Mn contained in the siderite clast. The exacting nature of EMP analyses, particularly by wave-dispersion spectrometry (WDS),

and with the extensive array of standards coupled with the ZAF correction procedure, make the EMP analyses the preferred values to use.

EMP analyses were combined with modal percentages to calculate a Reconstructed Whole-Rock Analysis (RWRA) of the siderite clast. The RWRA (see Appendix C) was compared to total elemental percentages determined by XRF to determine an average composition for the siderite clast and to compare results determined using XRF and EMP analyses. Excluding Mn, elemental analyses determined by EMP and XRF are similar.

Trace element concentrations, including Ni, were found to be below detection limits for EMP analyses and can be considered to compose a relatively insignificant portion of the siderite clast.

The methods used to determine the geochemistry of the siderite are expensive and not typically used by mine operators; however, this study has shown that the methods used can characterize acid-producing and toxic strata in mine overburden. Proper analytical methods should be conducted on mine overburden prior to mining in order to provide an adequate knowledge of the acid-producing capabilities of geologic horizons contained in mine overburden. The accurate geochemical characterization of mine overburden can serve as an aid in predicting the impact of geologic horizons on the water quality within the backfill after reclamation. For example, this study has identified a Mn component in the siderite lithology that was not detected during traditional ABA methods. The Mn component is considered by the writer to be a potential source of acid-drainage. Further studies involving the dissolution rate of the siderite clast are recommended in order to determine the dissolution rate of the Mn contained in the siderite lithology.

Chapter 8.

Mine Site Hydrology

Introduction

The hydrology of a mine site is often very complex, mainly because of variability in the hydraulic properties of both the backfill and the surrounding bedrock. Backfill materials can be especially heterogeneous because of differences in source material, blasting techniques, and placement and compaction of backfill material. Furthermore, the mine site is commonly subjected to changing hydraulic conditions due to varying rates of pumping or artificial recharge and planting vegetation related to changes in mining or reclamation activities. This chapter will provide an overview of the mine site hydrology based on topography, mine layout, water level monitoring, records of pumping and estimated backfill material properties.

Physical Properties of Backfill Material and Bedrock

The hydraulic conductivity of the backfill in the study area was determined by mine operators using pump test data in three recently installed (11/96) monitoring wells and one existing observation well (OW-8). The hydraulic conductivity measured within the backfill was 1.6×10^{-3} ft/s, which is higher than the range of values (1.41×10^{-4} to 5.04×10^{-7} ft/sec) determined by Hawkins (1993) in similar backfill at other sites. The high value of hydraulic conductivity (1.6×10^{-3} ft/s) measured at this site indicates loose zones with large voids (pseudokarst), but it is not clear whether this is representative of all of the backfill. The overall hydraulic conductivity of the backfill is expected to consist of a combination of high hydraulic conductivity zones with some lower conductivity zones due to compaction by the dragline or due to compositional differences. Diodato et al. (1988) found that zones constructed from disaggregated shale material typically have a lower hydraulic conductivity than disaggregated blocky sandstone and both of these

materials are present in the backfill at this site. Also, Schubert (1980) found that horizontal permeabilities of siltstone and shale were consistently greater than vertical permeabilities, due in part to grain fabric and differential settling. This suggests that layers of shale backfill material formed by compaction of the dragline could preferentially direct flow horizontally within the backfill. The lower conductivity zones of the compacted layers could also slow down the rate of vertical infiltration sufficiently to cause the formation of perched water tables within the backfill, however, there are no monitoring wells emplaced within the backfill to substantiate this.

The hydraulic conductivity of the Newton sandstone has not been measured in the study area, but it can be estimated by comparison with measured transmissivity values ranging from 750 to 20 ft²/d for rocks of the Cumberland Plateau (Hollyday et al., 1982). Dividing the transmissivity by an average saturated thickness of 100 ft. (Hollyday, personal communication, 1996), yields a range of hydraulic conductivity of 8.7×10^{-5} to 2.3×10^{-6} ft/s. Details of the well construction and the lithology for the tests summarized in Hollyday et al. (1982) are not available, but the results are expected to be controlled by the higher hydraulic conductivity units, which were likely sandstones. Typical values of hydraulic conductivity for sandstones given by Freeze and Cherry (1979) range from 3×10^{-10} to 6×10^{-6} ft/s, which are lower than most of the values measured by Hollyday et al. (1982). The higher hydraulic conductivity values of Hollyday et al. (1982) most likely represent fracturing in the Newton sandstone. The seeps into Glady Fork Creek in the study area provide further evidence of fractures in the Newton sandstone.

The Whitwell shale is expected to represent the lowest hydraulic conductivity unit at the site. The hydraulic conductivity of lithologically similar unweathered shales in the Oak Ridge area of East Tennessee range from 5.6×10^{-9} to 1.5×10^{-10} ft/s (Solomon et al., 1992). Typical values of hydraulic conductivity for shales given by Freeze and Cherry (1979) range from 10^{-7} to 10^{-11} ft/s, and the values for the Whitwell shale are expected to fall within this range.

The hydraulic conductivity of the Sewanee coal seam has not been determined, however coal seams commonly act as aquifers, which suggests that the Sewanee may have a higher hydraulic conductivity than the surrounding Whitwell shale.

Hydrology Data and Observations

Groundwater elevations within the backfill have been monitored by mine operators beginning with the installation of observation wells (hereafter referred to as OW's) associated with the TMHP. The locations of the observation wells are depicted on Figure 9, which also shows surface water drainage patterns within the study area. There are no visible signs of erosion in the study area and there are no indications of large-scale surface runoff to either Spring Branch or Glady Fork Creek. The present topography of the study area was graded during reclamation in a manner designed to direct the majority of surface runoff into sediment ponds OO2-A, OO4-A and OO4-B (Figure 9). Alkaline recharge structures (ARS's), positioned in areas that receive surface runoff, were excavated in the backfill to a depth of twelve feet and filled with limestone gravel of a stone quality of 90% pure calcium carbonate. Riprap berms were constructed upgradient from the ARS's to prevent sediment from settling into the structures. The ARS's are designed to enhance alkaline loading in the backfill by saturating infiltration with dissolved calcium carbonate.

All observation wells were completed in mine spoil with the exception of OW-4 which was completed in bedrock adjacent to the backfill. OW's 1, 2, 3, and 4, were completed in September, 1990, with OW's 5, 7, 8, and 10 completed in the fall of 1992. Additional observation wells, OW-6 and OW-9, were completed in January, 1996. Depth and elevation of the top of the casing (casings typically extend three feet above ground surface) for all the observation wells are presented in Table 13..

Table 13: Observation well casing elevation and well depth.

OW#	Casing Elevation (ft.)	Observation Well Depth (ft.)
1	1839	68
2	1872	115
3	1865	81
4	1808	72
5	1909	165
6	1901	95
7	1883	131
8	1900	151
9	1904	148
10	1892	107

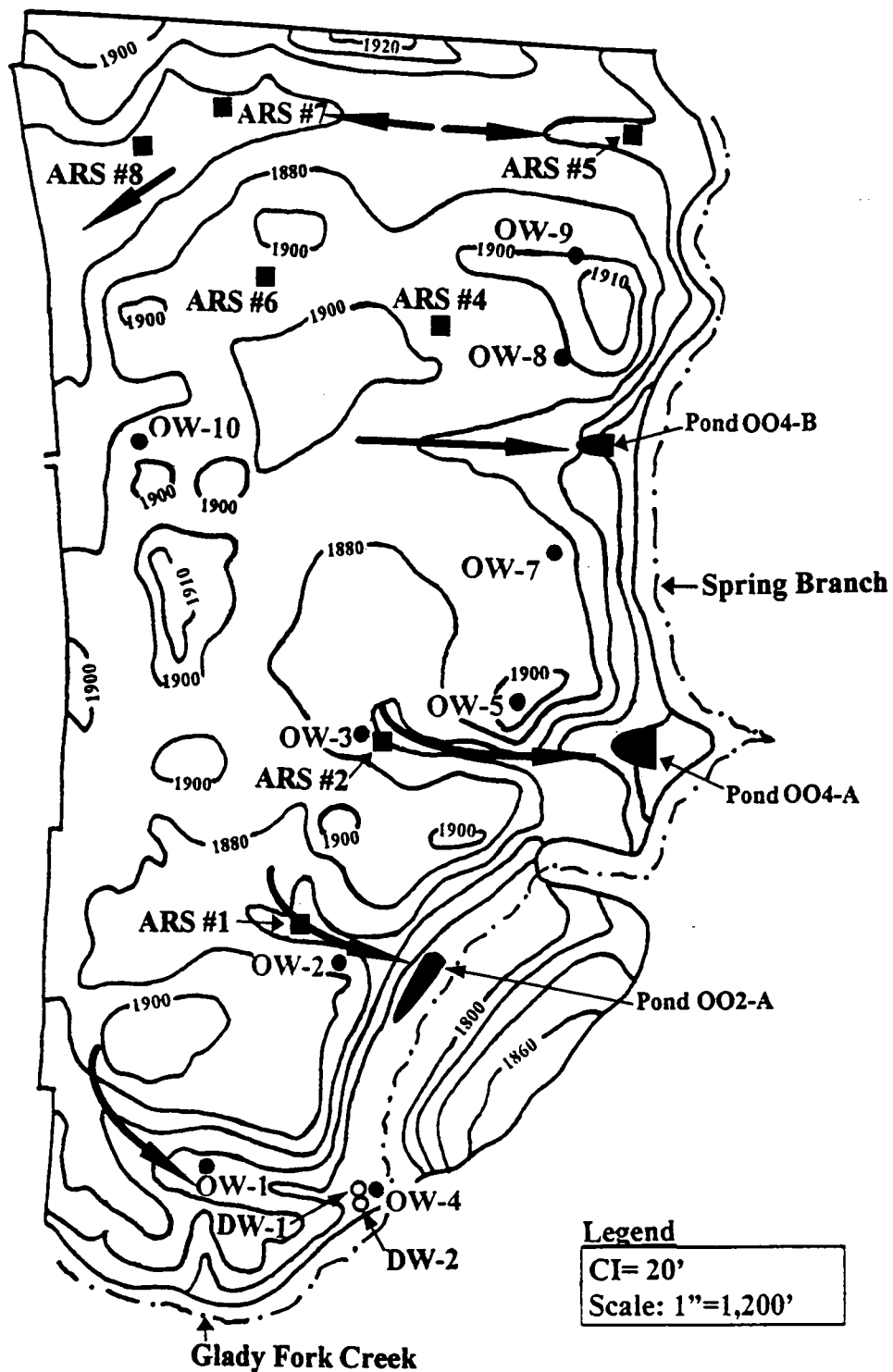


Figure 9: Surface water drainage patterns within study area and locations of alkaline recharge structures, observation wells, and sediment ponds.

A Fisher ® depth indicator was used to determine groundwater elevations in observation wells, usually on a monthly basis. The groundwater elevation data for the observation wells is included in Appendix D.

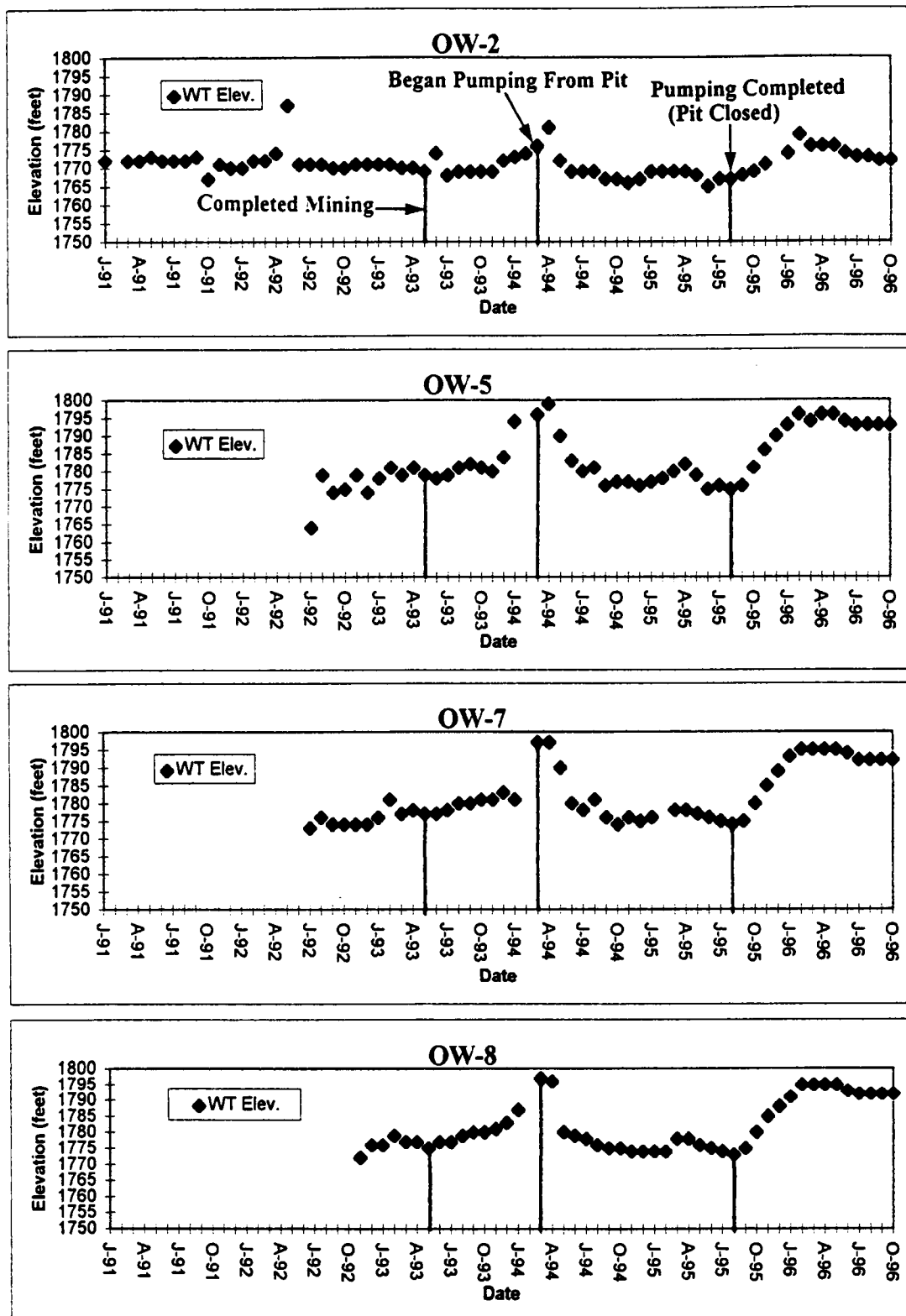
Groundwater elevations within the backfill have been manipulated during mining by two methods:

- 1) Pumping water from the active mine pit to avoid the accumulation of water in the open pit.
- 2) The dewatering of backfill by two dewatering wells emplaced in the southern section of the mine.

The dewatering wells (DW-1 and DW-2) were completed to depths of 65 and 66 feet for DW's 1 and 2 respectively. The bottom 20 feet of the dewatering wells were screened, with half of the screen completed in the underlying Whitwell shale. DW's 1 and 2 began operating at a 43 gpm maximum pumping capacity in October, 1990. The pumps in DW's 1 and 2 are activated when water levels reach the elevation of the pumps in the wells. If the water table should rise above the pump in DW-2 (positioned lower than the pump in DW-1) and reach the pump in DW-1, a switch is activated that starts the pump in DW-1. The water from DW's 1 and 2 is pumped to sediment pond OO2-A. The water in OO2-A is pumped to the alkaline recharge structure (ARS) next to OW-3. The other ARS's are recharged by surface run-off.

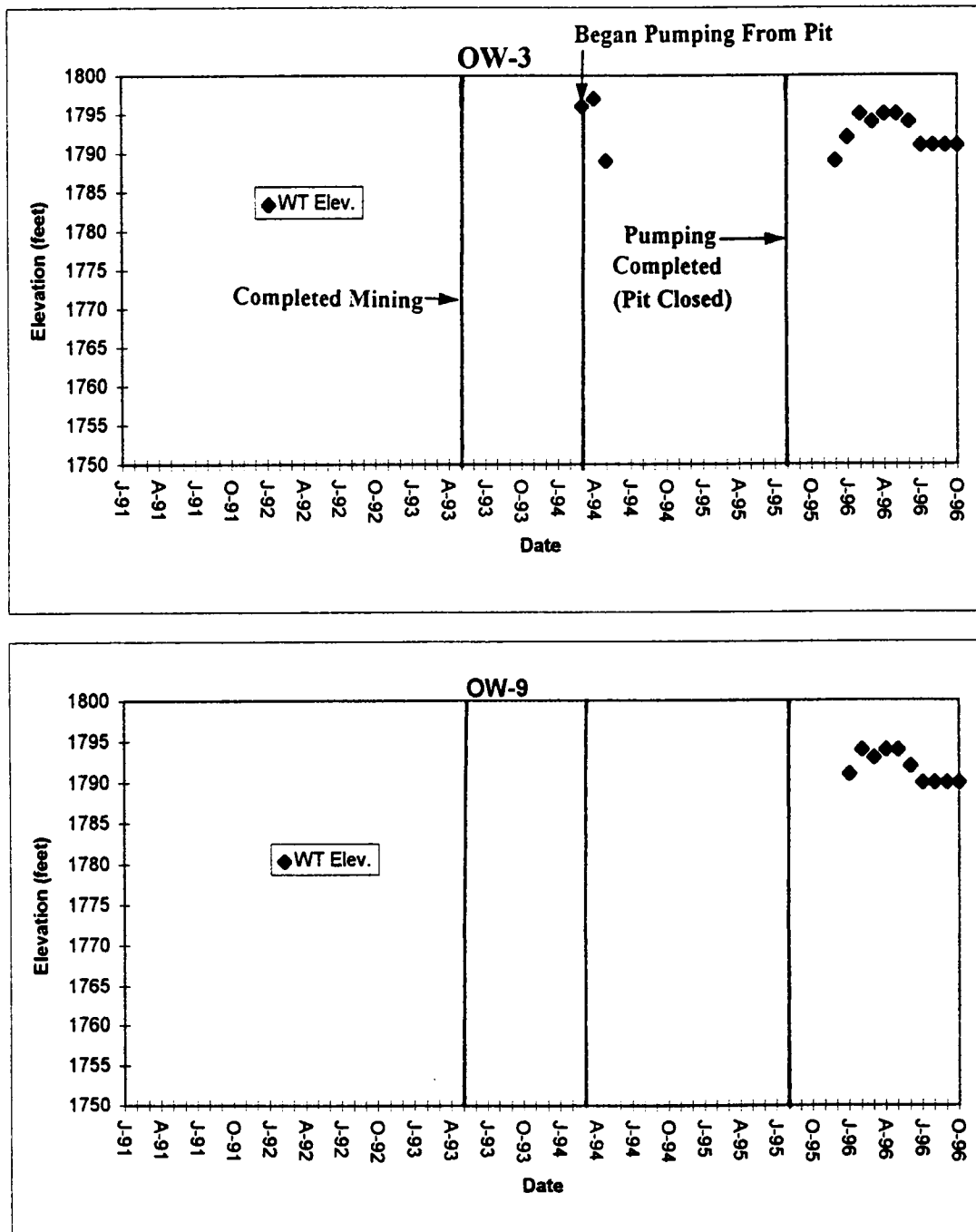
The impact on groundwater elevations due to pumping from the pit and from the dewatering wells is evident in Figure 10, depicting groundwater elevations in OW's 1, 2, 3, 4, 5, 7, 8, 9 and 10. The relative isolation of OW's 1 and 10 in regards to groundwater flow within the backfill is also evident in Figure 10. Two significant features are revealed in Figure 10:

- 1) After mining was completed in May, 1993 and pumping from the final pit to pond OO4-B ceased, groundwater elevations began to rise in the backfill and peaked in March and April, 1994. The peak in groundwater elevations corresponds with high precipitation totals during the "wet" months from January to April, 1994 (Table 2). Figure 10a reveals that after pumping resumed in March, 1994 to prepare for final reclamation, groundwater elevations in OW's 2, 5, 7, and 8, declined, apparently in response to the pumping. The hydrographs indicate that water levels decreased in wells closest to the final pit (OW's 7 and 8) before decreases in wells located furthest from the final pit (OW's 2 and 5), indicating the development of



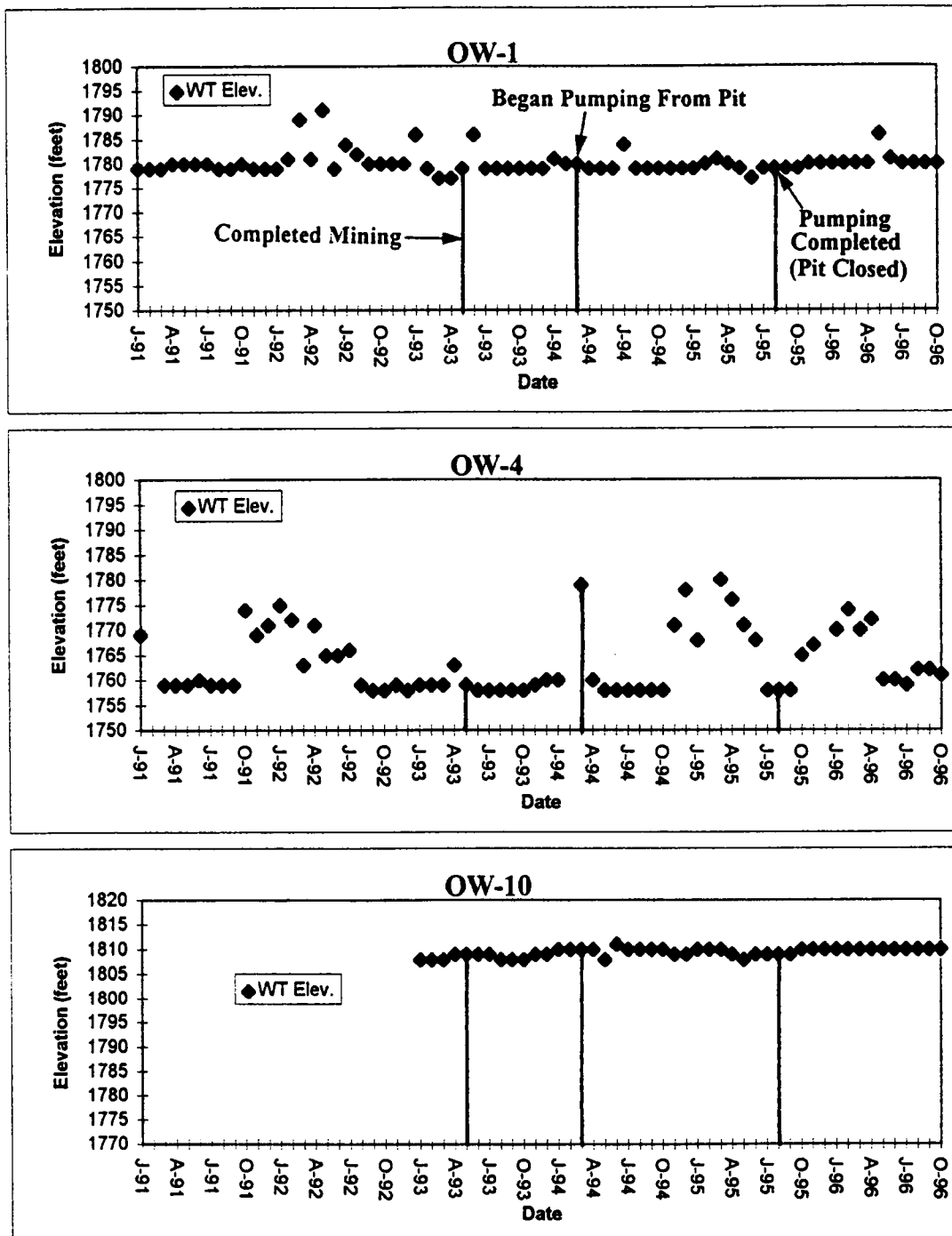
a. Water Table Elevations for OW's 2, 5, 7, and 8.

Figure 10: Water Table Elevations: a) OW's 2, 5, 7, 8; b) OW's 3, 9; c) OW's 1, 4, 10.



b. Water Table Elevations for OW's 3 and 9.

Figure 10: (continued)



c. Water Table Elevations for OW's 1, 4, and 10.

Figure 10: (continued)

a cone of depression generated from the final pit in the backfill and the hydraulic connection between the wells.

2) Following pit closure in August, 1995, groundwater levels began to rise in wells 2, 5, 7, and 8 (Figure 10a) in the backfill and apparently stabilized in February, 1996. Groundwater elevations decreased slightly from June to August, 1996, reflecting expected variations in recharge for this region. The similar responses in OW's 2, 5, 7, and 8 provide further evidence that the wells are hydraulically connected.

The influence of pumping from the pit on groundwater elevations in OW's 3 and 9 is evident in Figure 10b. OW-3 remained dry during mine operation, however, after pumping from the pit was completed and the final pit was filled with spoil in August, 1995, groundwater levels in OW-3 began to rise as the backfill became saturated. Groundwater elevations in OW-9, installed in January, 1996, began to rise immediately after installation, closely following the trend in OW-3. Water levels in both OW-3 and OW-9 indicate apparent water level stabilization in the backfill after February, 1996. The groundwater responses in OW-3 (and possibly OW-9) following pit closure are similar to those observed in OW's 2, 5, 7, and 8, indicating that OW-3 (and possibly OW-9) are hydraulically connected to OW's 2, 5, 7, and 8.

Groundwater elevations in OW-3 do not appear to have been influenced by pumping from pond OO2-A to ARS #2 located adjacent to OW-3. OW-3 remained dry during the course of mining and water levels did not rise in OW-3 after pumping from pond OO2-A was initiated in April, 1994, indicating that either water pumped from OO2-A to ARS #2 was not infiltrating the backfill or was being diverted in the backfill by a perched water table. Another possibility is that water was infiltrating the backfill, reaching the pit floor, and flowing downgradient to Gladly Fork Creek. Water levels in OW-5, located downgradient from ARS #2, did not indicate increases after pumping to ARS #2 was initiated, providing further evidence that pumping to ARS #2 was not influencing water elevations in this area of the backfill.

Groundwater elevations in OW's 1 and 4 (Figure 10c) varied significantly, but not at the same time as water levels observed in OW's 2, 3, 5, 7, 8, and 9. The variations in groundwater elevations in OW's 1 and 4 are believed to be related to pumping from DW-1 and DW-2. OW-1 is completed in backfill, roughly 1,080 feet west of DW-1 and DW-2 and at a higher elevation than the two dewatering wells. The cone of

depression generated by pumping from DW's 1 and 2 is believed to influence water levels in OW-1. OW-4 is completed in bedrock, roughly 50 feet east of DW's 1 and 2. Pumping from DW's 1 and 2 may be responsible for the variations in groundwater elevations in OW-4, which indicates that the shale bedrock is fractured at least near the mine excavation. The greater distance from the dewatering wells and higher elevation, relative to OW-4, could explain why water levels in OW-1 do not fluctuate as dramatically as water levels in OW-4.

OW-10, presented in Figure 10c, is not influenced by pumping from the mine pit or from the dewatering wells, however it is included in order to indicate the relative isolation of this well in regards to water table variations within the backfill. Groundwater elevations in OW-10 have remained relatively constant compared to other observation wells.

Groundwater Flow and Discharge

Recharge to the groundwater system within the study area occurs mainly by surface infiltration. The Newton Sandstone may contribute some groundwater to the study area, however, the off-site groundwater contribution is likely to be small because the mine site is located on a ridge and most of the natural flow would be away from the mine.

Groundwater flow within the study area was investigated using cross-sections and groundwater elevations from the observation wells. The cross sections were prepared using the topography of the study area after mining (Figure 11) and an isochore map (a line drawn on a map through points of equal drilled thickness of a specified subsurface unit) of the top of the Sewanee coal (Figure 12). Figure 11 was modified from a post-mining topographic map of the study area supplied by IRTEC, Engineering, a consulting firm employed by mine operators. Figure 12 is a map modified from a pre-mining map constructed by mine operators that was used to determine the thickness and areal extent of the Sewanee seam. The elevation of the pit floor, representing the base of the cross-sections, was determined by subtracting the average thickness of the Sewanee coal (3 feet) from the isochore map. Four geologic cross-sections showing water table locations were prepared to aid in the identification of groundwater flow and discharge from the study

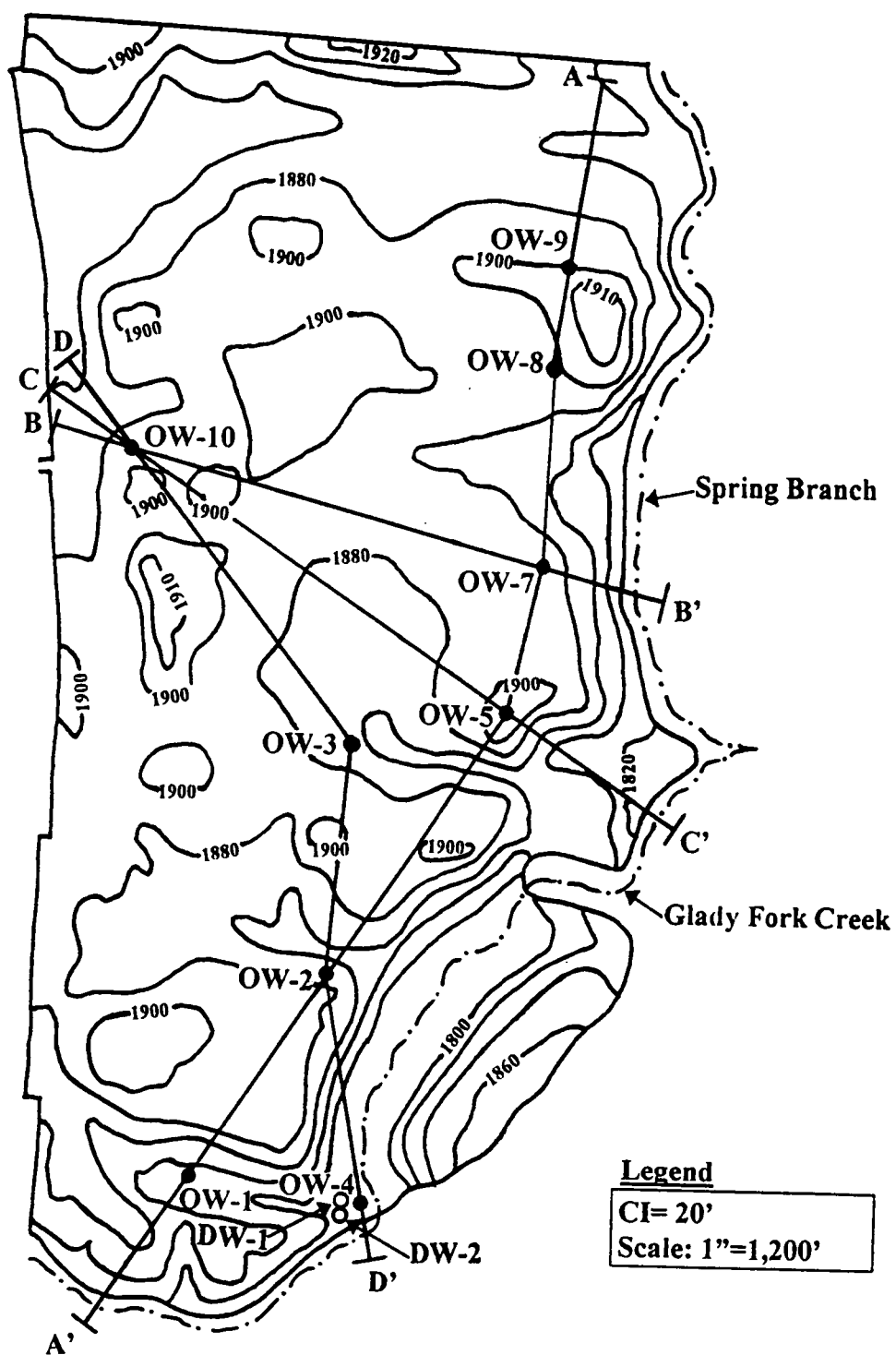


Figure 11: Present topography of study area and location of hydrogeologic cross-sections.

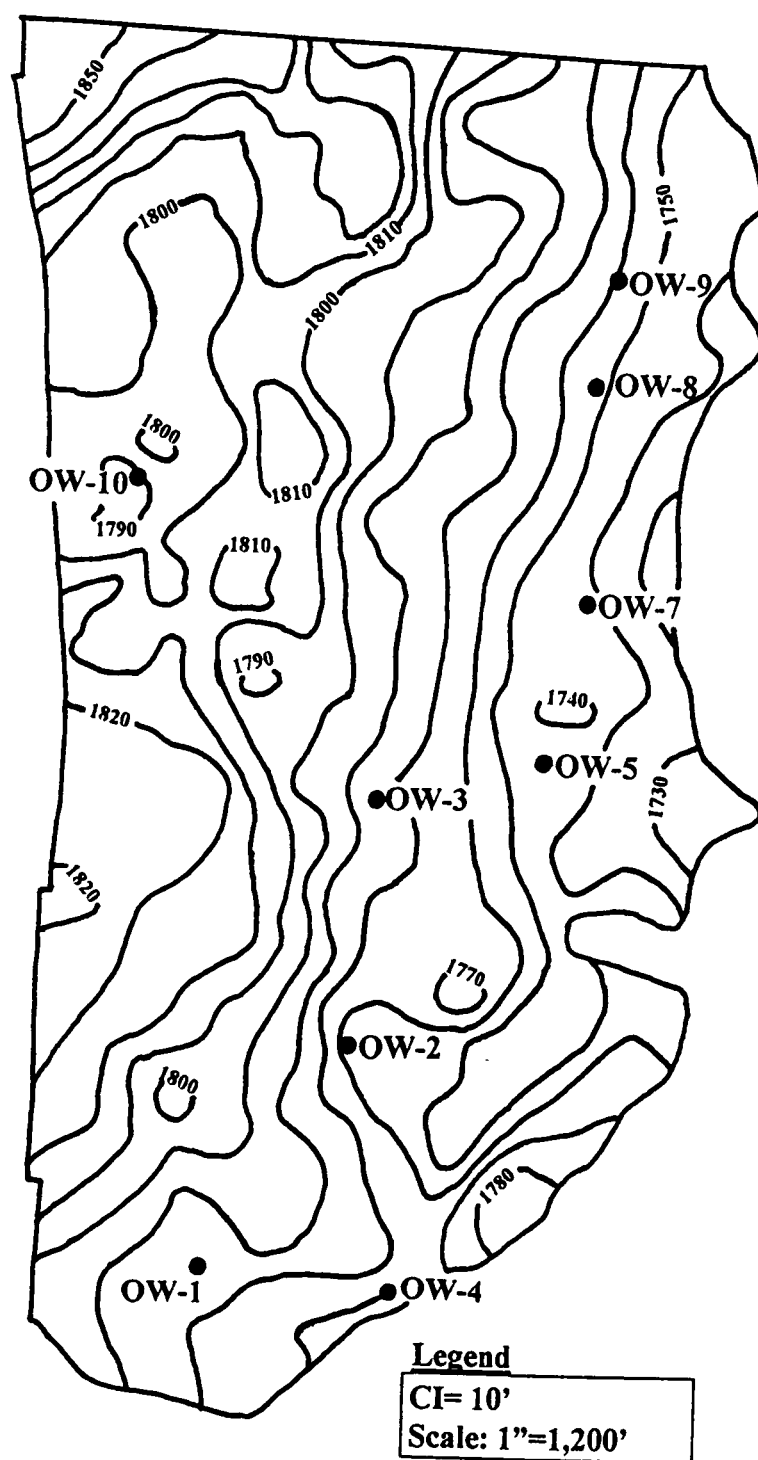


Figure 12: Isochore map of the top of the Sewanee coal.

area. The locations of the cross-sections are indicated on Figure 11.

Cross-section A-A', presented in Figure 13, extends from OW-9 to OW-1 and depicts groundwater elevations during mining (5/95), and after mining was completed (4/96 and 8/96). The after mining water tables represent a relatively "wet" month (April, 1996) and a relatively "dry" month (August, 1996). The groundwater elevations are approximately 3 feet higher during April, 1996 than in August, 1996, in all observation wells with the exception of OW-1 where groundwater elevations remained constant. As illustrated in Figure 13, an apparent groundwater divide exists between OW-5 and OW-8 (where groundwater elevations were one foot higher in OW's 5 and 7 than OW-8), resulting in groundwater flow to the North, in the direction of OW-8, and to the South, in the direction of OW-2. The isolation of OW-1 relative to groundwater flow from the direction of OW-2 is evident in Figure 13. The cross-section A-A' is mainly parallel to the steep slope going down to Spring Branch and Glady Fork Creek, so much of the groundwater flow is expected to be perpendicular to this cross-section.

Groundwater elevations cannot be extended beyond OW-8 in May, 1995 because OW-9 was installed after the final pit had been closed and mining was completed. Figure 13 illustrates that the direction of groundwater flow during mining is the same as groundwater flow after mining, however groundwater elevations are roughly 40 feet lower during mining. As evidenced in Figure 13, the groundwater elevation is below the elevation of Glady Fork Creek in the southern section of the study area, indicating that there was no possibility of direct discharge into Glady Fork Creek in this part of the study area, at the time of monitoring.

Figure 14 includes cross-sections B-B' and C-C', from west to east in the study area, depicting nearly identical groundwater elevations in April, 1996. The isolation of OW-10 relative to groundwater flow in the study area is also illustrated in Figure 14. The ridge depicted on the pit floor partly isolates OW-10 from the rest of the backfill, although flow from the area of OW-10 can spill over the ridge and flow down towards the lower portion of the backfill. Figure 14 indicates groundwater elevations are higher than the elevation of Spring Branch and Glady Fork Creek, indicating discharge from the backfill to the streams in this portion of the study area is possible through the Newton sandstone or along the bedding plane

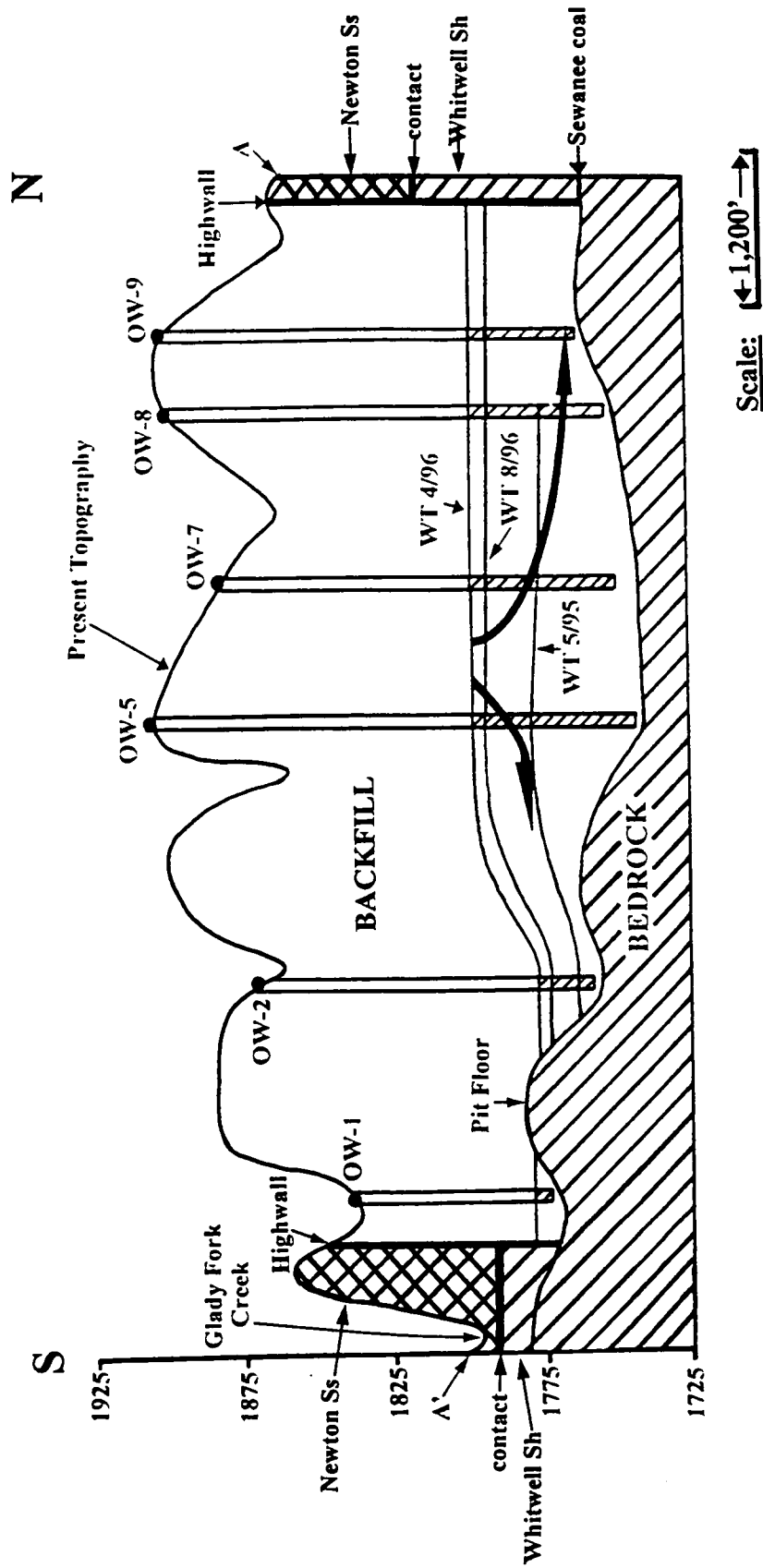


Figure 13: Cross-section A-A", indicating water table (WT) elevations in April and August, 1996 and May, 1995. Vertical exaggeration=24.

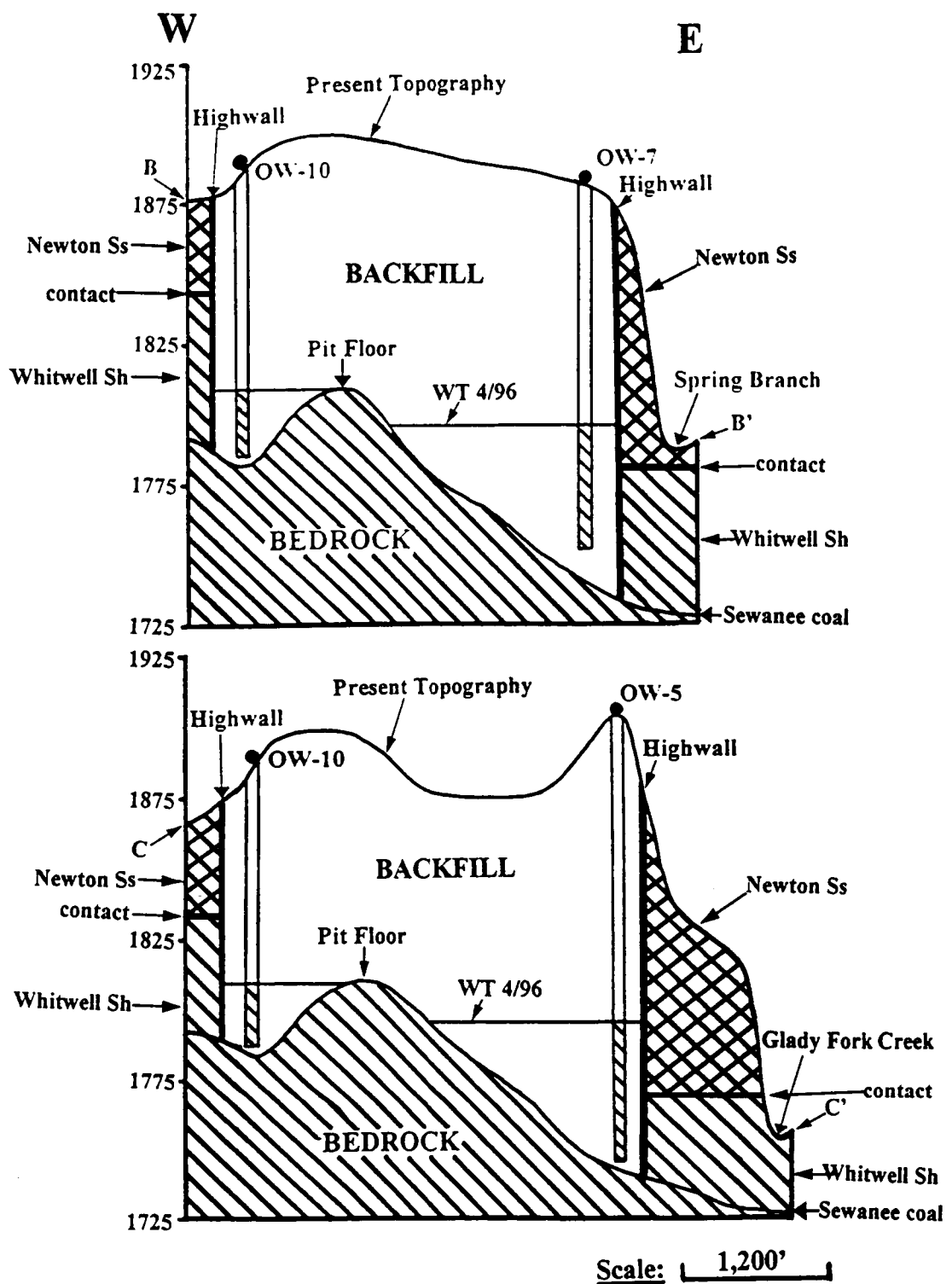


Figure 14: Cross-sections B-B' (top) and C-C' (bottom) indicating water table (WT) elevations in April, 1996. Vertical exaggeration=24.

(contact) between the Newton sandstone and Whitwell shale. Off-site discharge through the Sewanee coal seam is also possible in this area, however discharge would not likely be into either of the creeks because the Sewanee coal seam is overlain by 20-60 feet of shale near the creeks.

Cross-section D-D' (Figure 15) extends from the northwest to the southeast of the study area, depicting groundwater elevations in April and May, 1996. The low elevation of the water table at the southeast end of the section and the low levels in OW-4 indicate the influence of the nearby dewatering wells. During April, 1996 the dewatering wells were temporarily inoperative and groundwater elevations in OW-4 increased 12 feet, indicating a strong hydraulic connection between the dewatering wells and OW-4. In May, 1996 the dewatering wells were reactivated and groundwater levels decreased in OW-4, while elevations in OW's 2, 3, and 10 remained the same. The proximity of the dewatering wells to OW-4 accounts for the groundwater response in OW-4 that is not observed in OW's 2, 3, and 10.

The partial isolation of OW-10 relative to groundwater flow is again evident in Figure 15. Groundwater elevations remain constant at 1810 feet in OW-10 indicating that pumping from the dewatering wells and pumping from the final pit have not influenced groundwater elevations in OW-10. Groundwater elevations also remain constant in OW-10 during "wet" and "dry" seasons, indicating that during "wet" seasons recharging infiltration could be flowing over the ridge into the rest of the backfill.

Figure 15 indicates groundwater flow is in the direction of the dewatering wells, however the groundwater elevations in this area are slightly below elevations in Glady Fork Creek, thereby eliminating the possibility of direct discharge from the backfill into Glady Fork Creek in this area at the time the water levels were measured. Off-site discharge through the Sewanee coal seam is also possible in this area.

A potentiometric map for the backfill (Figure 16) was constructed using the cross-sections and groundwater elevations during a relatively "wet" month, April, 1996. The flow paths will remain the same during "dry" months because observation well data indicate consistent increases and decreases in groundwater elevations relative to one another. The flow path map indicates discharge from the backfill is most likely to the east, into Spring Branch, and to the southeast of the study area in the direction of Glady Fork Creek north of the pumping wells DW-1 and DW-2. The groundwater elevations in these portions of

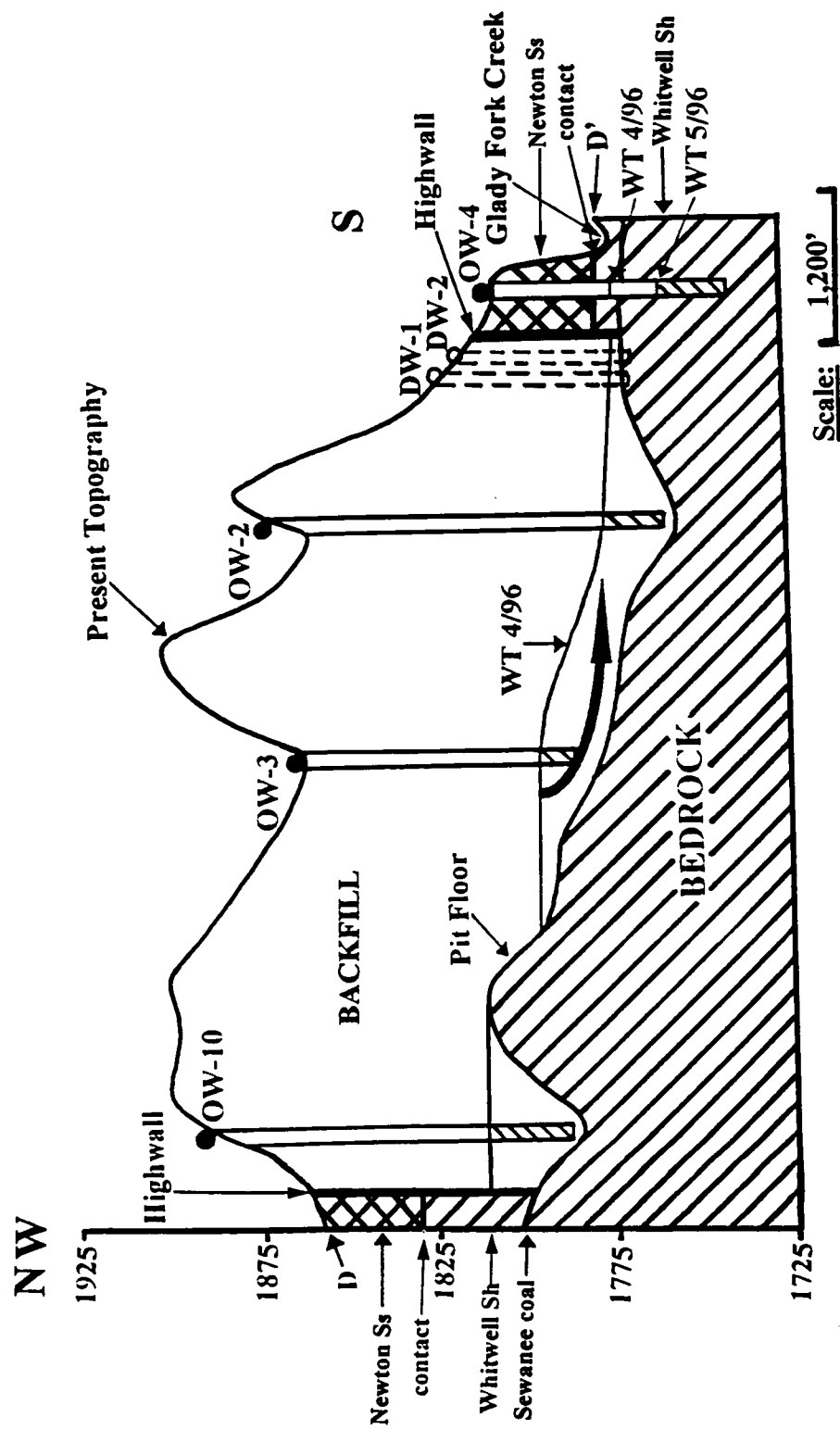


Figure 15: Cross-section D-D' indicating water table (WT) elevations in April, 1996. Vertical exaggeration=24.

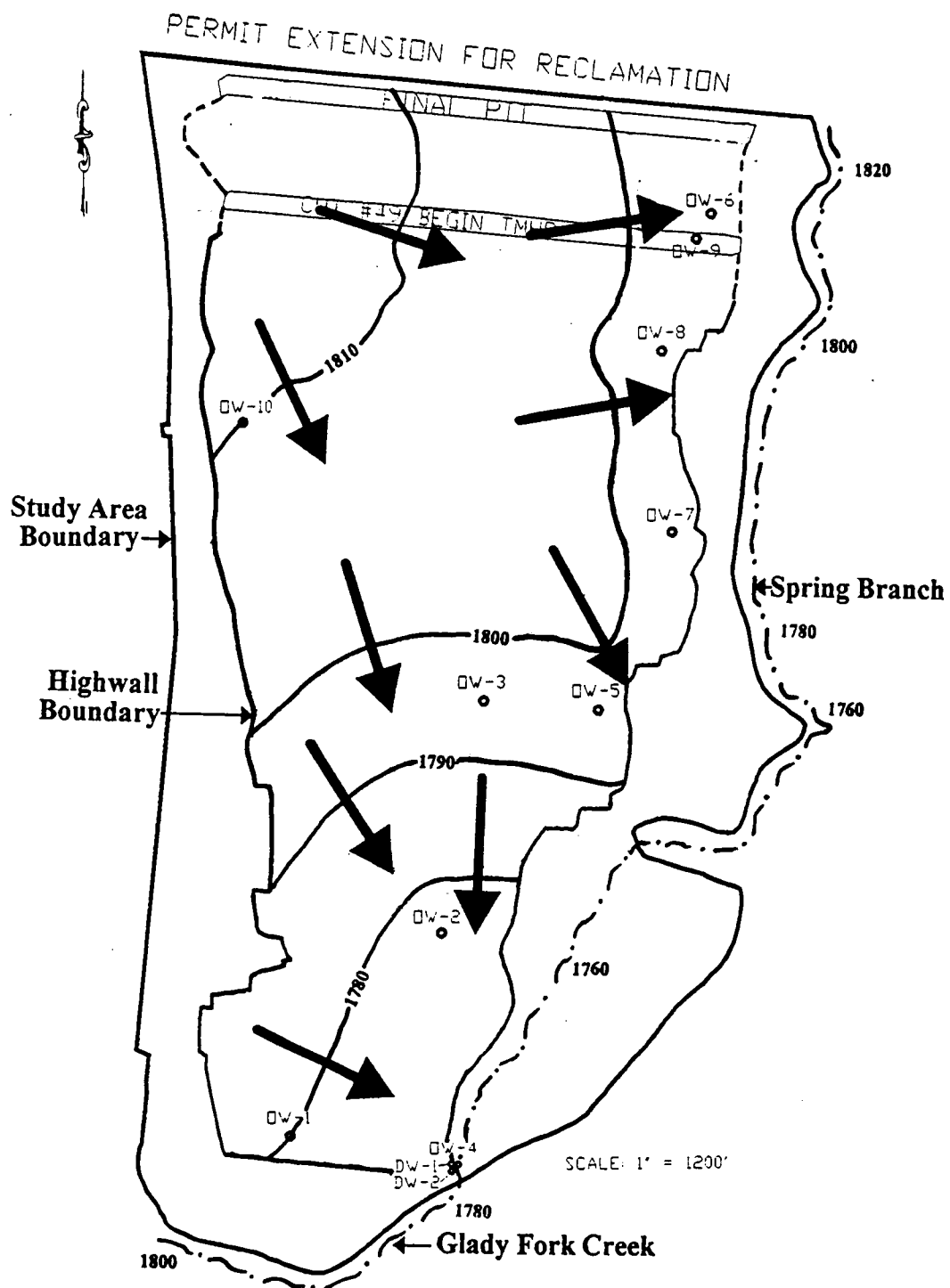


Figure 16: Potentiometric map of backfill, with stream elevations, indicating directions of groundwater flow in the saturated zone of the backfill in April, 1996.

the study area are higher than the elevation of Spring Branch and Glady Fork Creek, therefore indicating the possibility of discharge through the Newton sandstone in this area (Figure 14). This was evident in cross-sections B-B' and C-C', but not D-D', which was strongly influenced by dewatering wells DW-1 and DW-2. If pumping of these wells is stopped it is likely that water levels will rise in the southeast corner of the backfill and seepage towards Glady Fork Creek in this area will resume.

Downward migration of groundwater below the pit floor elevation is expected to be relatively slow. An average of 15 feet of Whitwell Shale, left relatively undisturbed between the pit floor and the underlying Sewanee Conglomerate, is anticipated to provide a low hydraulic conductivity boundary between backfill material and the underlying Sewanee Conglomerate. There is a possibility of stress relief fractures within the Whitwell Shale providing fracture flow to the underlying Sewanee Conglomerate, but softer units (e.g., shale and claystone) tend to be less permeable because the material is somewhat plastic and when fractures do occur they are relatively small or tend to close (Hawkins, et al., 1996).

Groundwater Summary

Groundwater flow within the study area occurs in backfill material with an expected hydraulic conductivity ranging from 1.6×10^{-3} to about 5×10^{-7} ft/s. Surface water infiltration is the primary source of recharge to the backfill. Groundwater flow within the backfill is predominately to the east and southeastern portions of the study area with discharge from the backfill expected to occur through the Newton sandstone along the eastern boundary of the study area, into Spring Branch and into Glady Fork Creek north of the dewatering wells. Some off-site discharge through the Sewanee coal seam is also possible. Recent and present flow conditions have been strongly influenced by pumping from the dewatering wells and from the mine pit when it was in operation. This makes it difficult to determine whether water levels in the pit will continue to rise, although the 1996 hydrograph data suggests that levels have stabilized, assuming that the present system of pumping continues. If pumping from the dewatering wells is stopped, as could occur sometime in the future, water levels are expected to rise in the backfill to levels above the elevation of Glady

Fork Creek, which could cause increased discharge into the creek in the southeast corner of the site and the seeps (previously described) would reappear.

Additional information necessary to determine the long-term flow conditions and locations of off-site flow from the study area, include continuing to monitor observation wells currently emplaced in the backfill on a regular basis. This monitoring would help determine whether the backfill has finished filling up with water and would help in assessing changes that could occur if the dewatering wells (DW-1 and DW-2) are removed or shutdown. This study indicates that flow from the backfill is entering Spring Branch and parts of Glady Fork Creek, so investigations to locate and measure discharge in these areas is suggested. Monitoring Spring Branch and Glady Fork Creek for evidence of seeps into the creeks will not only indicate locations of discharge from the backfill but will also enable water samples to be collected and analyzed to determine the water quality emanating from the backfill. The possibility of off-site discharge through the Sewanee coal seam could also be investigated, but this is a less likely pathway of off-site migration than flow through the Newton sandstone. Investigations in the coal seam might involve measurement of hydraulic conductivity of the Sewanee coal and monitoring of water levels and water chemistry within the coal.

Investigations could also be done to determine if the compacted layers formed by the dragline are preferentially diverting flow away from problematic material in the backfill. Observation wells completed just above the elevation of the dragline, and monitored after storm events could provide useful information in determining if infiltration is occurring through the dragline bench.

Chapter 9.

Water Quality

Surface Water Quality

In compliance with the Notice of Violation (NOV) issued to the mine operators, stream samples from Glady Fork Creek are collected on a monthly basis below and above the seep locations related to the NOV (see Figure 5). Table 14 represents surface water quality analyses downstream of the location of former seeps 1 and 2 from August, 1995 to April, 1996 (excluding September, 1995). The samples were collected by the mine operators and analyzed by an EPA certified laboratory. As illustrated in Table 14, pH values are near neutral and are below values considered to represent acidic drainage (i.e. $\text{pH} < 4.5$). Metal concentrations, including Total and Dissolved Fe and Mn are below EPA standards for discharge into receiving streams (i.e. Fe daily maximum of 6 mg/l, monthly average of 3 mg/l, Mn daily maximum of 2 mg/l, monthly average of 4 mg/l). The inactivity of the seeps and the improved water quality attest to the success of the dewatering wells in abating pollution of Glady Fork.

Groundwater Monitoring

Monitoring of groundwater quality, has been conducted on a monthly basis by mine operators beginning with the installation of the first observation wells in September, 1990. Water quality analyses for all samples collected at all monitoring visits are performed in the laboratory and in the field. Field measurements of water quality parameters were conducted by the writer from May, 1995-August, 1996 using an Aqua-Check® Water Analyzer. The Aqua-Check simultaneously measures pH, dissolved oxygen, conductivity and temperature. Groundwater samples were collected using a check-valve bailer and brought to the surface for field analyses and collection of samples for laboratory analyses. Water quality data are

presented in Appendix E. Appendix F contains groundwater sampling procedures, sample preservation techniques and analytical accuracy of the "Aqua-Check".

Table 14: Surface water quality analyses downstream of seeps 1 and 2. Units are in mg/l with the exception of pH. Alkalinity units in mg/l as CaCO₃.

Downstream of Seep 1:

Date	pH	Alkalinity	Total Fe	Diss. Fe	Total Mn	Diss. Mn	Sulfate
Aug/95	6.91	95	1.31	0.27	0.46	0.21	6.0
Oct/95	6.86	20	0.82	0.17	0.10	0.06	7.0
Nov/95	7.13	17	0.56	0.20	0.13	0.10	5.0
Dec/95	6.14	15	0.08	0.05	0.10	0.05	9.5
Jan/96	7.29	11	0.10	0.06	0.10	0.10	6.5
Feb/96	6.99	12	0.48	0.17	0.13	0.08	5.0
Mar/96	6.71	12	0.36	0.17	0.10	0.06	5.5
Apr/96	7.59	19	0.41	0.12	0.08	0.05	5.0

Downstream of Seep 2:

Date	pH	Alkalinity	Total Fe	Diss. Fe	Total Mn	Diss. Mn	Sulfate
Aug/95	6.95	57	0.82	0.19	1.22	0.91	7.0
Oct/95	6.80	15	0.34	0.10	0.08	0.05	5.0
Nov/95	6.25	12	0.44	0.17	0.10	0.05	5.5
Dec/95	6.22	14	0.13	0.06	0.14	0.10	18
Jan/96	7.42	11	0.58	0.11	0.12	0.06	6.0
Feb/96	7.36	13	0.34	0.12	0.11	0.07	5.0
Mar/96	6.58	13	0.34	0.17	0.14	0.10	6.5
Apr/96	7.40	7	0.30	0.10	0.04	0.50	4.1

Groundwater monitoring records of OW's 1, 2, and 4, span a five and a half year period. Groundwater monitoring records for OW's 5, 7, 8, and 10 include three and a half years of data. OW-3, installed in 1990, remained dry during pumping from the active mine pit, however, after the final pit had been closed in August of 1995, groundwater levels in OW-3 began to rise. Groundwater samples have been analyzed and groundwater elevations have been monitored for OW-3 since December, 1995, the first month groundwater levels reached a measurable height. Monitoring of OW-9, installed as the last observation well in January, 1996, began in February, 1996 when groundwater elevations reached measureable levels in the area of OW-9.

Backfill Groundwater Quality

The geochemical characteristics of groundwater samples from OW's 2, 3, 5, 7, 8, and 9 will be evaluated in this section. Groundwater characteristics of OW's 1 and 4 will not be evaluated due to their proximity to and related influence from the dewatering wells, DW-1 and DW-2. As previously illustrated, DW-1 and DW-2 have influenced groundwater levels in OW's 1 and 4 relative to other observation wells. The dewatering by DW-1 and DW-2 can effectively "purge" groundwater volumes in OW's 1 and 4 which influences the concentration of the geochemical parameters being monitored. Thus, the quality of groundwater in OW's 1 and 4 in the monthly samples are not correlative with the undisturbed water in the other observation wells.

Groundwater quality data from OW-10 will not be evaluated in this study due to the relative isolation of OW-10 in regards to groundwater flow in the study area. It is the writer's opinion that the relative isolation of OW-10 will not allow water quality in the well to reflect the increases and/or decreases in alkalinity associated with the TMHP.

In order to evaluate and characterize changes in groundwater quality of the selected observation wells, graphs of specific chemical parameters were prepared. Appendix G includes the groundwater quality data used to prepare the graphs. The chemical parameter graphs constructed to aid in this evaluation, include graphs of:

- 1) pH
- 2) Total and Dissolved Iron (Fe)
- 3) Total and Dissolved Manganese (Mn)
- 4) Alkalinity
- 5) Sulfate

pH

The pH graphs of OW's 2, 3, and 5 (Figure 17) and 7, 8, and 9 (Figure 18), indicate that pH in all observation wells has remained above levels considered to represent acid drainage (i.e. pH <4.5). The pH

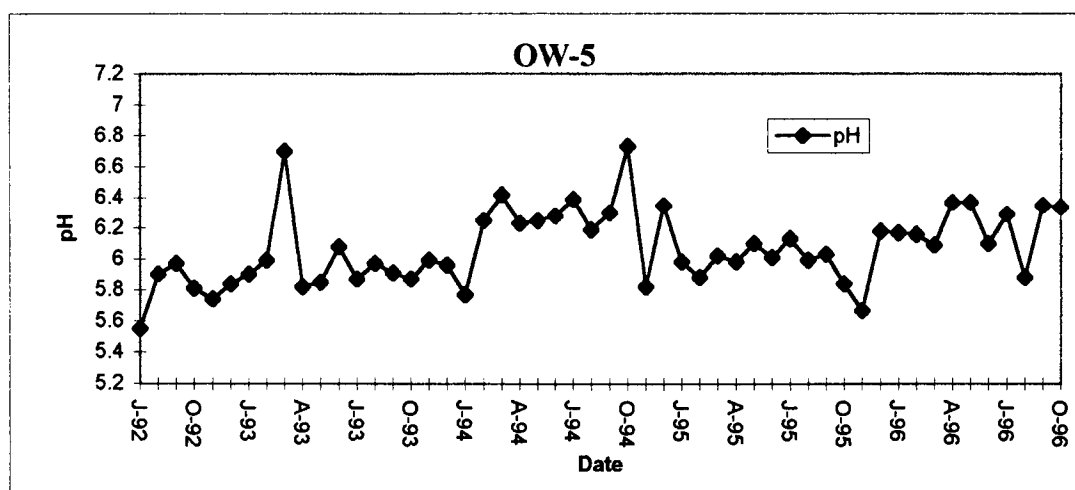
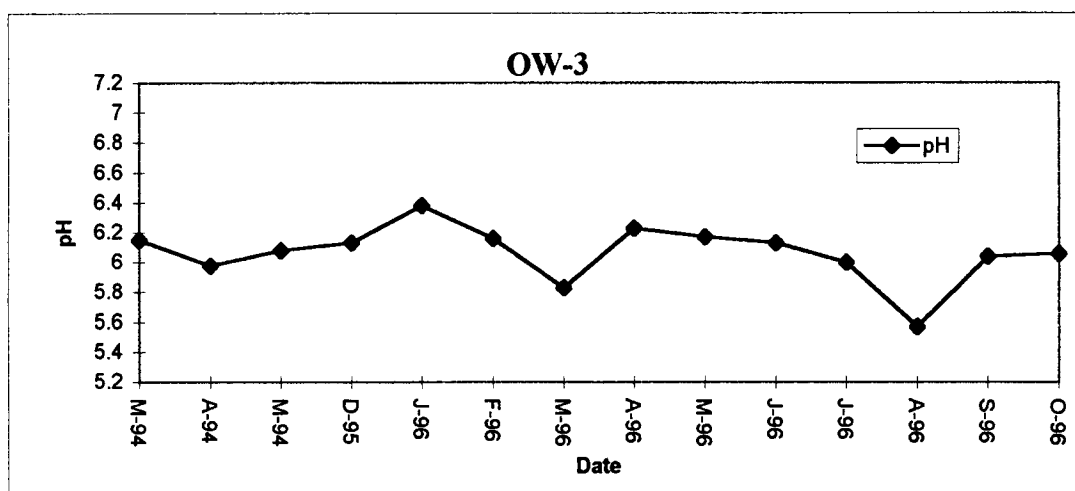
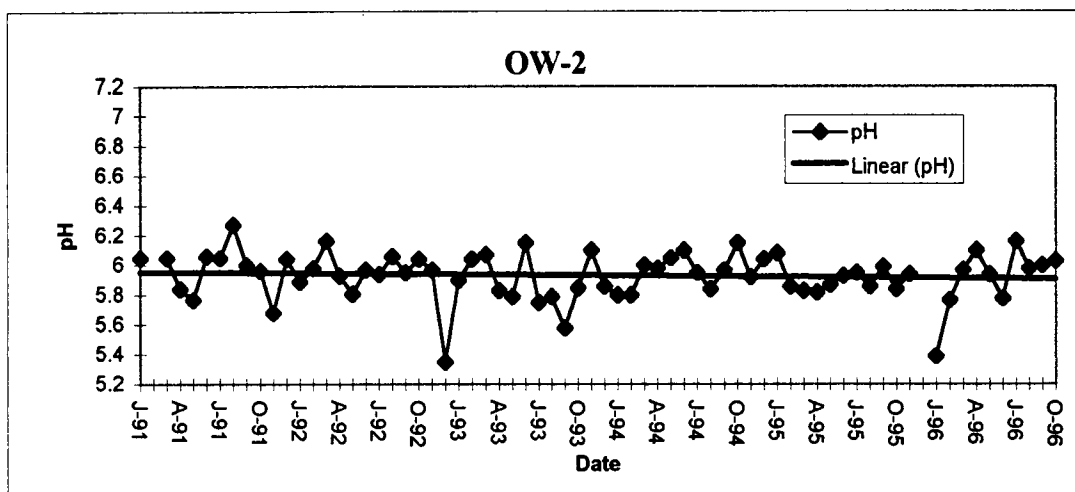


Figure 17: pH Graphs of OW's 2, 3, and 5.

values have remained near neutral in all observation wells, with the exception of OW-3, since the first month of monitoring. Values, excluding OW-3, are increasing or remaining relatively constant in all observation wells.

The pH values are relatively the highest in OW's 7, 8, and 9 and lowest in OW's 2, 3, and 5. The high pH values represented in OW's 7, 8, and 9 are believed to indicate that the dissolution of calcium carbonate (Equation 8 and 9) associated with limestone amendments in the TMHP, is effectively removing H^+ ions from solution, thereby increasing pH.

Decreasing pH in OW-3, located in close proximity to OW-5 where pH is increasing, suggests that the neutralization of acidity by the alkaline recharge structure (ARS #2), located adjacent to OW-3, has not been successful in increasing pH through the dissolution of limestone associated with ARS #2. The decrease in pH in OW-3 could be related to acid production in the backfill during water table recovery. The rising water table could effectively "flush" the backfill during recovery, dissolving sulfosalts formed during conditions prior to inundation. The rising water table would serve as a medium, allowing for the migration of acidity that was generated as a result of oxidation of sulfide minerals prior to water table recovery.

The influence of the recovering water table on pH following pit closure is reflected in Figures 17 and 18. Figures 17 and 18 indicate that following pit closure in August, 1995, pH values initially decreased as the water table recovered in the backfill followed by an increase in OW's 2, 5, 7, and 8. The decrease in pH following pit closure is most likely a result of the "flushing" of acidity (i.e. in the case of OW-3) generated during oxidizing conditions prior to groundwater recovery. As the water table stabilized, in February, 1996, pH values began to increase. The increase is greatest in observation wells located closest to the TMHP boundary, namely OW's 7 and 8. The pH values increase in OW's 2 and 5 following pit closure, however the increase is not as large as in OW's 7 or 8. The increase in pH indicates that after the water table had recovered and groundwater elevations had stabilized, alkaline-enriched groundwater was enabling the removal of H^+ from solution.

Total and Dissolved Fe

Graphs of Total and Dissolved Fe concentrations for OW's 2, 3, and 5, (Figure 19), and OW's 7, 8, and 9 (Figure 20) indicate Total and Dissolved Fe concentrations are highest in observation wells located farthest from the TMHP boundary and lowest in observation wells located closest to the TMHP boundary. The graphs of Total and Dissolved Fe concentrations are plotted on the same vertical scale as a means to compare Fe concentrations in observation wells located closest to and farthest from the TMHP boundary.

Total and Dissolved Fe concentrations are highest in OW-2, located farthest from the TMHP boundary, and lowest in OW's 7, 8, and 9 which are located closest to the TMHP boundary. Concentrations in OW-2 have decreased over time and are approaching concentrations similar to those in OW's 7, 8, and 9. Fe concentrations in OW-3 are lower relative to OW-2, however Fe concentrations are increasing in OW-3 compared to decreasing or remaining relatively constant in other observation wells. Increasing Fe concentrations in OW-3 correspond with decreasing pH values in OW-3. OW-3 is the only observation well that shows a distinct relationship between decreases in pH and increasing metal concentrations.

Fluctuations in Total and Dissolved Fe concentrations are most evident in OW's 2, 3, and 5 compared to relatively stable concentrations in OW's 7, 8, and 9. The stable Fe concentrations in OW's 7, 8, and 9 compared to fluctuating conditions in OW's 2, 3, and 5, can be associated with the neutralization of mineral acidity from limestone amendments of the TMHP.

The alkalinity generated from the dissolution of the limestone amendments can eliminate the initiation of acid producing reactions that can result in increased Fe concentrations. Caruccio (1996) has indicated that if the initiation of acid reactions can be eliminated and/or minimized, acid production can be reduced; however, once acid reactions are initiated, the production of acidity is difficult to control. The limestone additions related to the TMHP appear to have inhibited the initiation of acidity in areas closest to the TMHP area, where Fe concentrations are relatively low. In OW-2, however Fe concentrations are highest, indicating that acid reactions (i.e. Equations 1-4) resulting from oxidizing conditions have allowed the dissolution of sulfide minerals, and siderite, contributing Fe to the groundwater. Fe concentrations in OW-2 are decreasing, indicating that the neutral pH of the groundwater is removing Fe ions from solution.

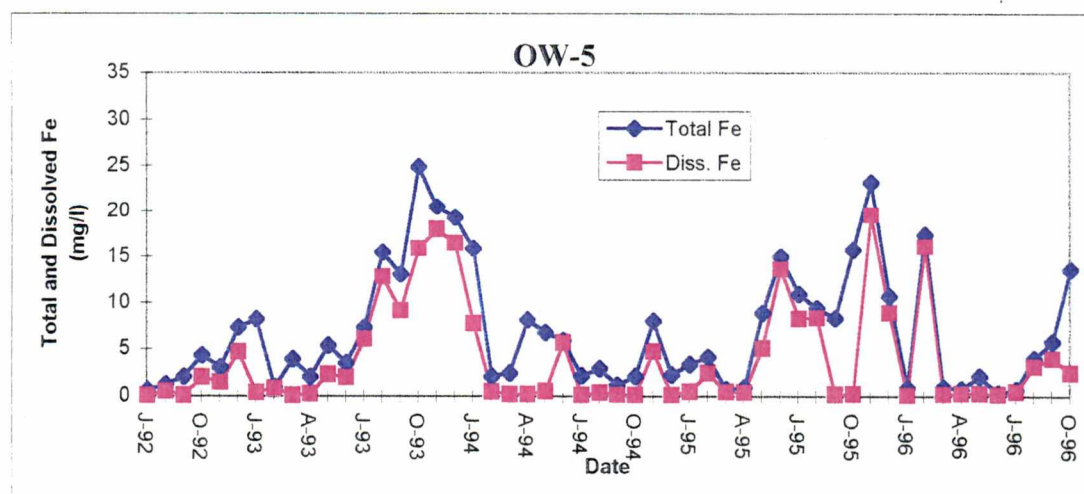
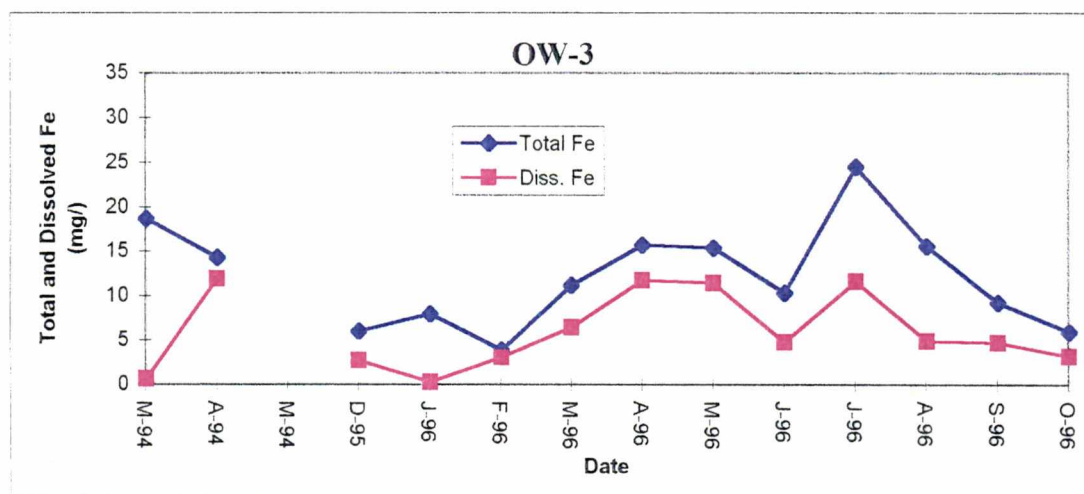
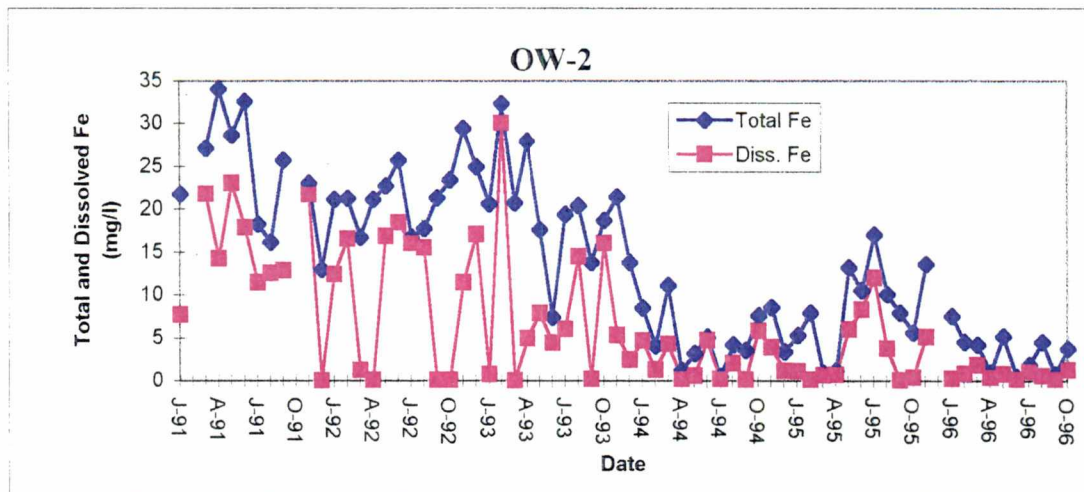


Figure 19: Total and Dissolved Fe Graphs for OW's 2, 3, and 5.

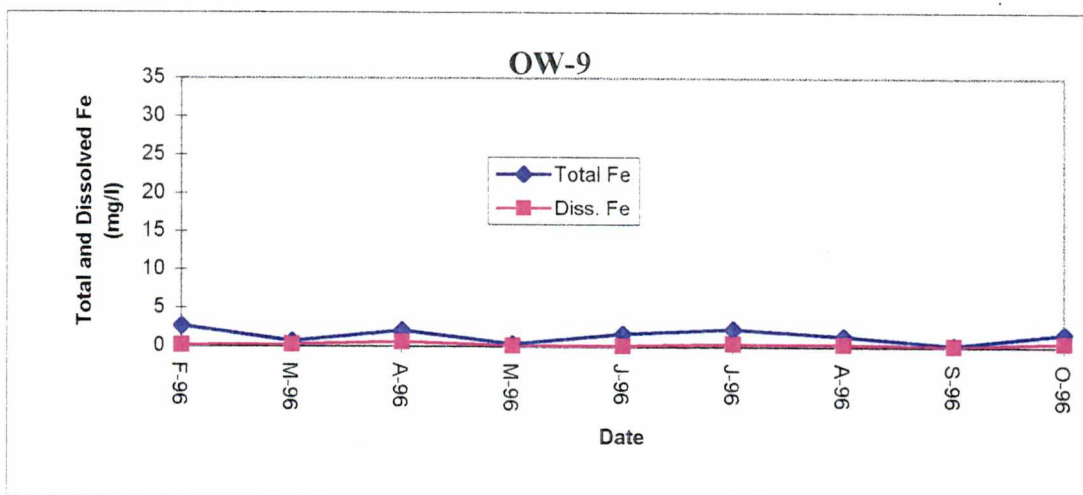
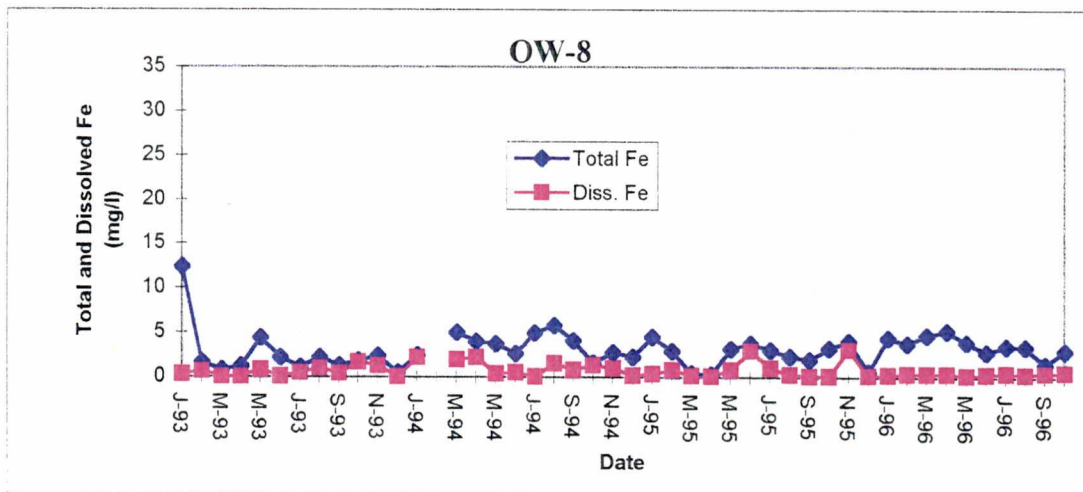
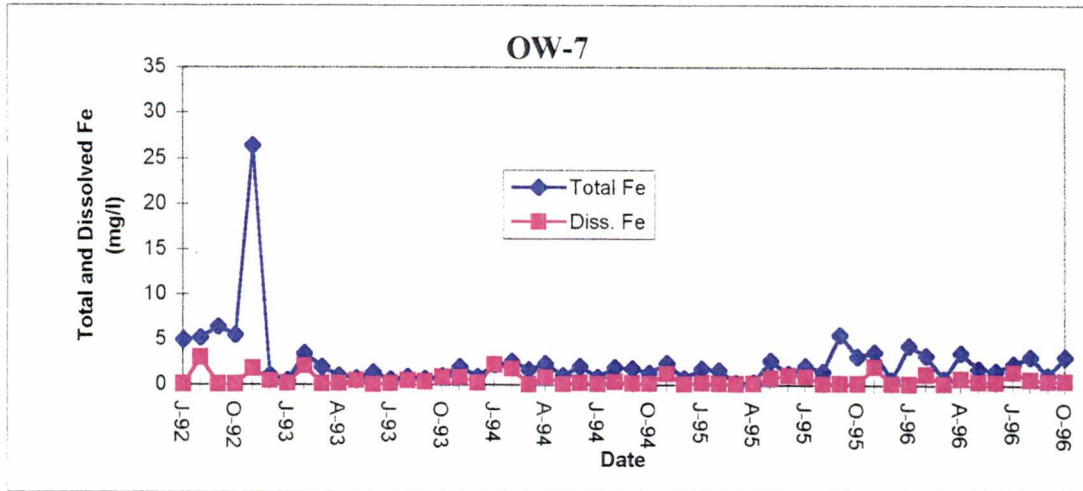


Figure 20: Total and Dissolved Fe Graphs for OW's 7, 8, and 9.

Fe concentrations in all observation wells, with the exception of OW-3, are decreasing and approaching EPA discharge effluent standards. Fe concentrations have fluctuated in OW's 7 and 8 after pit closure, however after initially increasing, Fe concentrations are presently decreasing. The increasing Fe concentrations in OW's 7 and 8 following pit closure could be related to oxidation of sulfide minerals prior to pit closure. The final pit was left open for two years after mining, enabling the oxidation of sulfide minerals to generate elevated levels of Fe in the backfill.

Total and Dissolved Mn

Graphs of Total and Dissolved Mn (Figures 21 and 22), indicate that Mn concentrations are highest in OW-2 and lowest in OW's 8 and 9. Total and Dissolved Mn concentrations are similar to Fe concentrations, in that observation wells located closest to the TMHP boundary reflect lower metal concentrations than observation wells located furthest from the TMHP boundary.

Mn concentrations are decreasing in all observation wells, with the exception of OW's 8 and 9. The increasing Mn concentrations in OW's 8 and 9 suggest that the limestone amendments, as of yet, have not been effective in reducing Mn concentrations. Even though the Mn levels in OW's 8 and 9 have been increasing, the concentrations are lower relative to other observation wells. The Mn concentrations in OW-8 were below 5 mg/l until April, 1994 when they began to increase. This increase in Mn concentrations beginning in April, 1994 could be correlated with the final pit filling with water after mining was completed in May, 1993. The rising water level in the pit resulted in groundwater flow into the backfill, similar to the "flushing" of the backfill that resulted in lowering of the pH in OW-3 and initial lowering of pH following pit closure in OW's 8 and 9. The "flushing" effect generated dissolution of toxic material, increasing Mn concentrations in the groundwater. Pumping from the final pit to prepare for final reclamation was initiated in March, 1994, however, increasing Mn concentrations beginning in May, 1994 indicate that oxidation of backfill material had occurred while the pit remained open and was subsequently "flushed" during elevated water levels in the final pit. Mn concentrations also increased after the final pit was closed, indicating that the "flushing" of Mn occurred during water table recovery in the backfill.

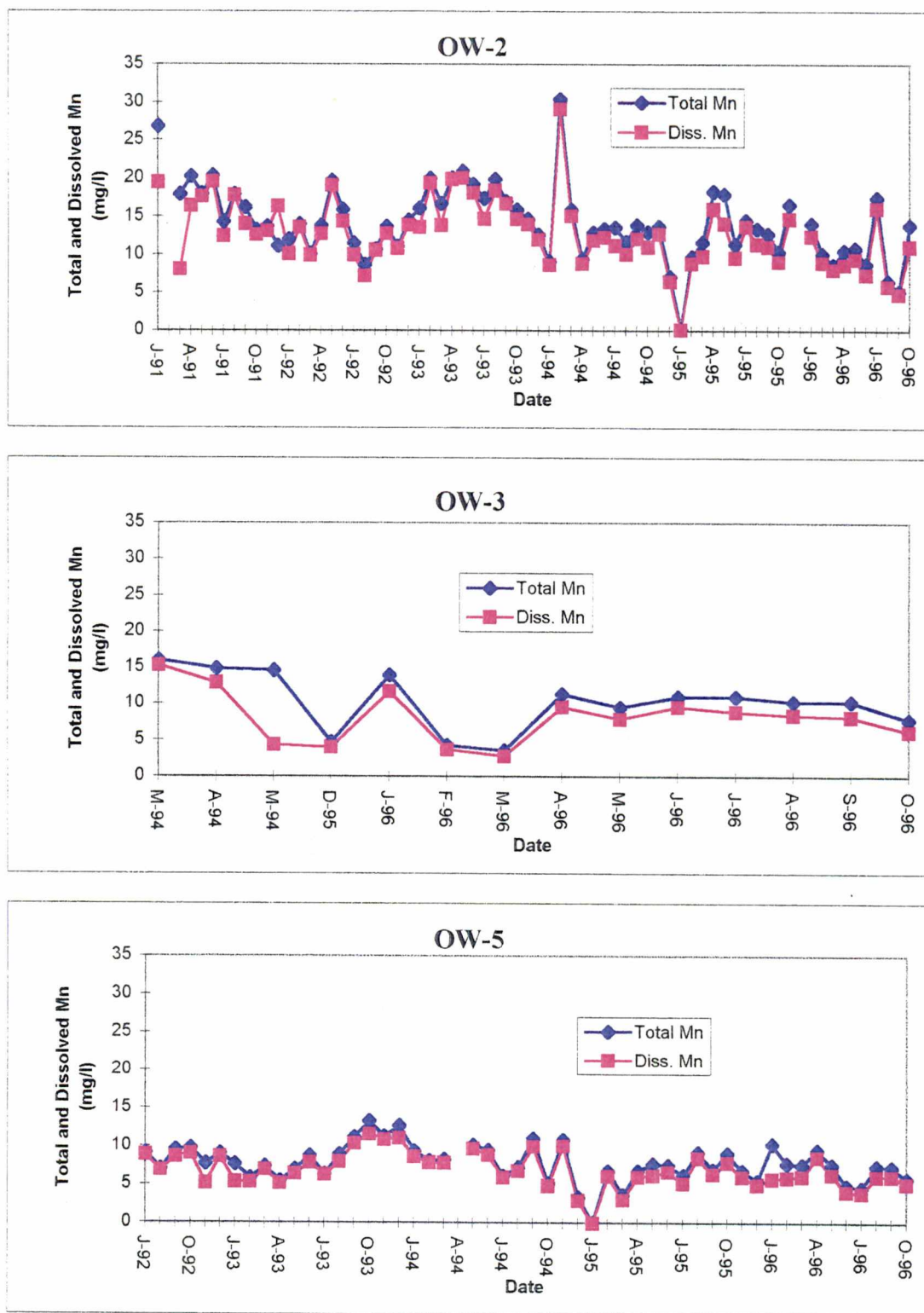


Figure 21: Total and Dissolved Mn Graphs for OW's 2, 3, and 5.

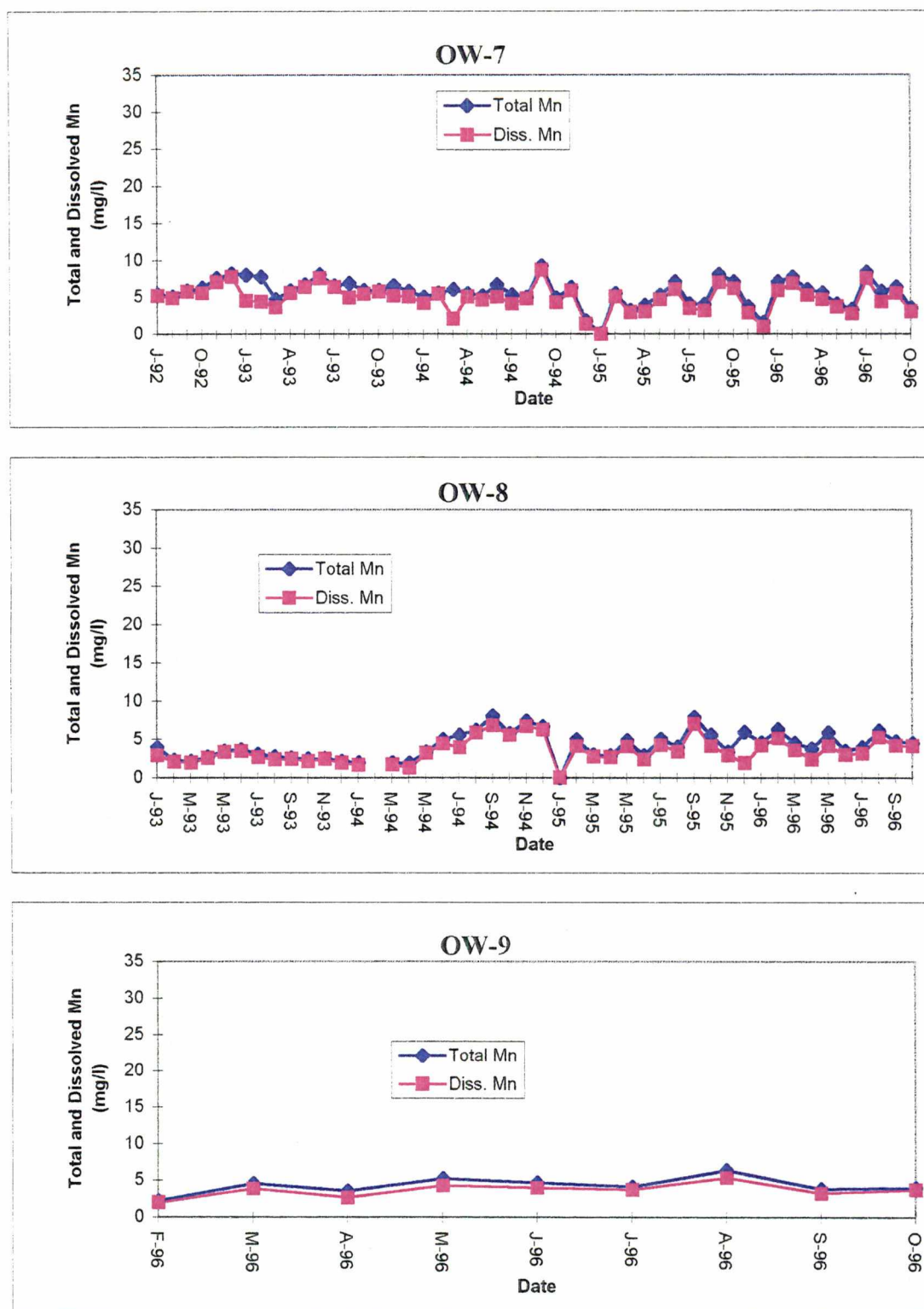


Figure 22: Total and Dissolved Mn Graphs for OW's 7, 8, and 9.

Although the Mn concentrations in OW-9 are increasing, the concentrations are relatively low, as are the increases, increasing from 2mg/l in February to 5.5 mg/l in August, 1996. Although increasing, the relatively low Mn concentrations are close to EPA effluent standards of daily maximum concentrations of 4 mg/l with a monthly average of 2 mg/l. In February, 1996, the Mn concentration equaled the monthly standard concentration for Mn at 2 mg/l. Since OW-9 is a relatively new observation well, it might be expected that Mn concentrations fluctuate as groundwater stabilizes. Once the groundwater stabilizes, the Mn concentrations in OW-9 should decrease to EPA standards, which would assure that any surface discharge would certainly be in compliance.

The decreasing Mn concentrations in OW-3 are at odds with increasing Fe concentrations in OW-3. Relative to Fe, Mn is difficult to remove from solution, therefore Fe concentrations should be expected to decrease while Mn concentrations should increase and/or remain relatively constant; this, however, is not the case in OW-3.

As previously mentioned the oxidation of Mn^{+2} results in the precipitation of the relatively insoluble MnO_2 . The relative insolubility of the Mn oxide precipitate would indicate that Mn concentrations would not decrease, but remain constant or increase. The relative insolubility of the Mn oxide precipitate would explain the similarity in Total and Dissolved Mn concentrations. The similar values between Total and Dissolved Mn concentrations indicates that the dissolution of Mn is being limited and that the majority of the Mn in the groundwater exists as Total Mn.

Mn concentrations are higher than Fe concentrations in all observation wells, with the exception of OW's 2 and 3. The elevated Mn concentrations could be related to the soluble nature of the siderite deposit. As previously mentioned, siderite is 14 times more soluble than pyrite. The soluble nature of the siderite would increase the release of Mn into solution. The Mn concentrations are decreasing, however, suggesting that as time progresses, the high pH of the groundwater favors removal of Mn from solution.

Alkalinity

Alkalinity concentrations (as mg/l $CaCO_3$) are highest in observation wells located closest to the

TMHP boundary and lowest in observation wells located farthest from the TMHP boundary. Alkalinity graphs of OW's 2, 3, and 5, included in Figure 23, indicate low alkalinity concentrations relative to alkalinity of OW's 7, 8, and 9 (Figure 24). Alkalinity concentrations are, however, increasing in OW's 2 and 5.

Referring to Figure 23, alkalinity concentrations were relatively constant in OW-2 until October, 1993 when alkalinity begins to increase. The increase in alkalinity began in OW-2 exactly one year after the initiation of the TMHP. The increase in alkalinity is most likely related to flow of alkaline-enriched groundwater from the TMHP area. The groundwater flow-path map (see Figure 16) indicates groundwater flow from the TMHP in the direction of OW's 8, 7, 5, and 2.

Figure 24 indicates that alkalinity concentrations in OW's 7, 8, and 9 have maintained consistently high levels of alkalinity since the first month of monitoring. The elevated initial alkalinity concentrations in OW's 7, and 8 are related to the addition of 10 tons/acre of "crusher run" limestone that was added to the topsoil/spoil interface beginning in August, 1992.

The migration of a plume of alkaline-enriched groundwater from the TMHP area as a result of elevated water levels in the backfill is evident by referring to Figure 25. Figure 25 compares groundwater elevations and alkalinity concentrations in OW's 2, 5, 7, and 8. As indicated in Figure 25, increased alkalinity concentrations corresponds with elevated groundwater levels. The increased alkalinity is believed by the author to be related to alkaline-enriched groundwater migrating from the TMHP area.

Migration of a plume of alkaline-enriched groundwater is also evident in the graph of OW-5. As indicated in the graph of OW-5, alkalinity concentrations remained relatively constant until April, 1993 followed by increasing alkalinity concentrations. The increase in alkalinity in OW-5 occurred within six months after the initiation of the TMHP, whereas in OW-2, alkalinity concentrations began to increase one year after the initiation of the TMHP. OW-5 is positioned closer than OW-2 to the TMHP boundary, therefore the plume of alkaline-enriched groundwater would reach OW-5 before OW-2.

The movement of alkaline-enriched groundwater within the study area is also evident by comparing alkalinity concentrations in April, 1994 in OW's 2, 5, 7, and 8. Alkalinity concentrations peaked in these wells during April, 1994, the same month that water table elevations peaked in response to the final pit filling

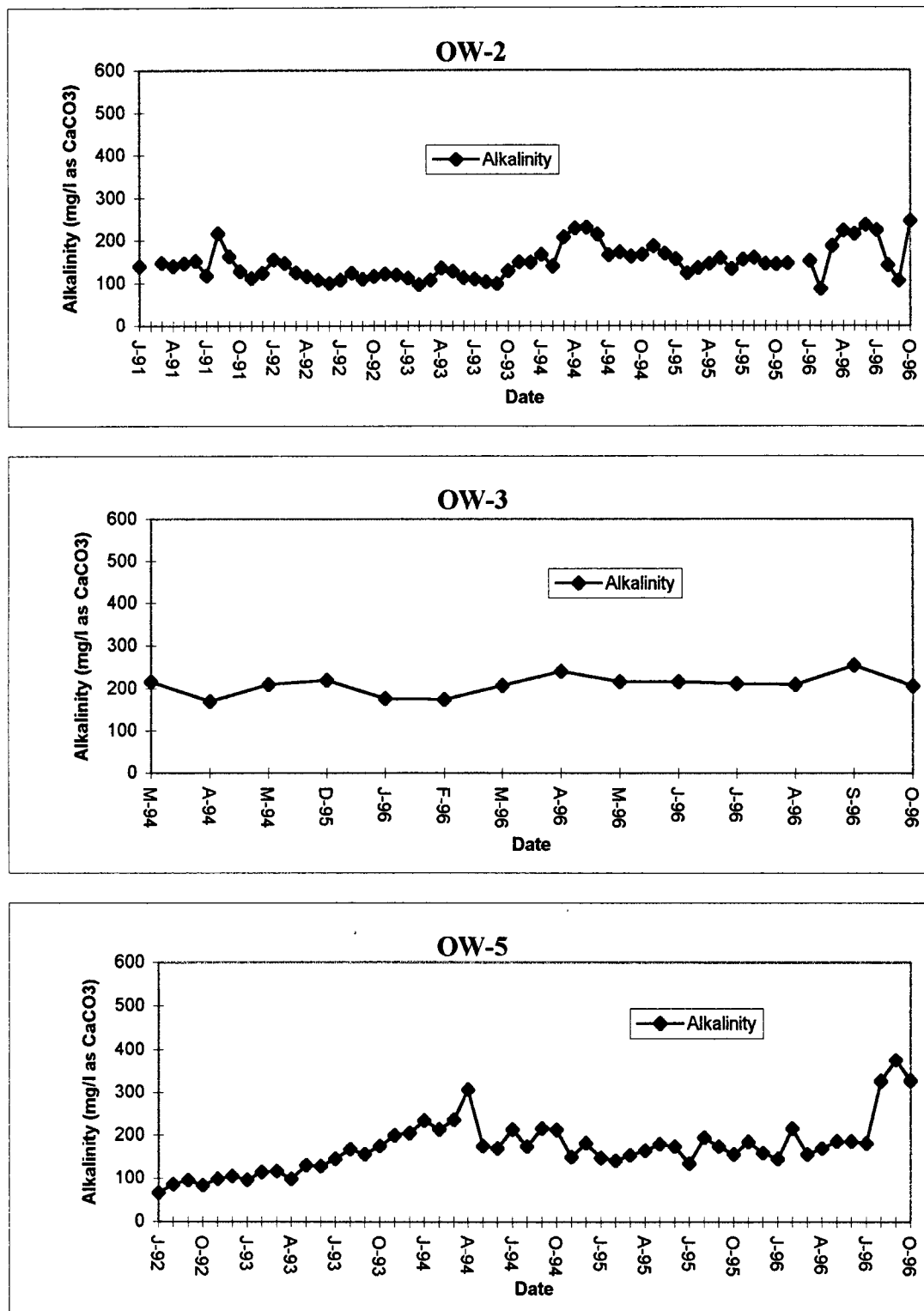


Figure 23: Alkalinity Graphs for OW's 2, 3, and 5.

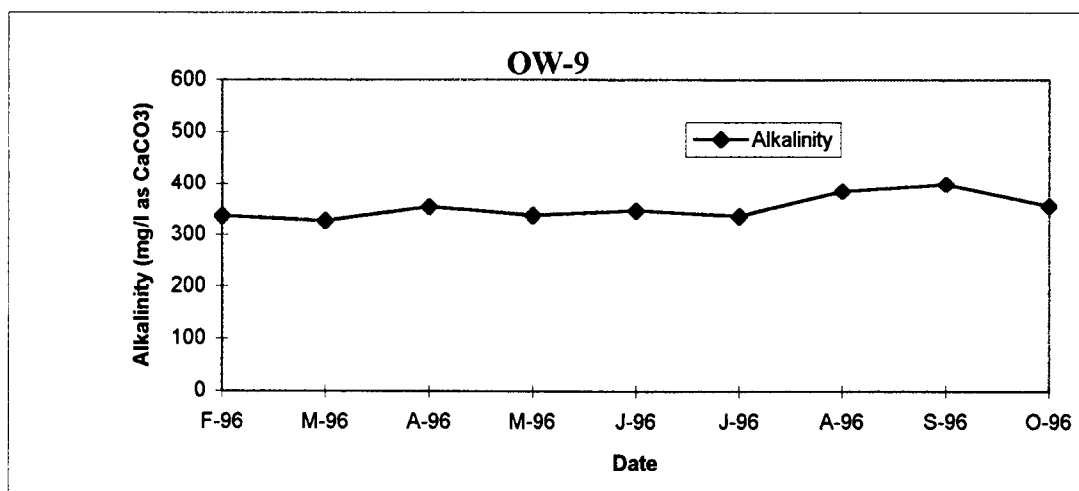
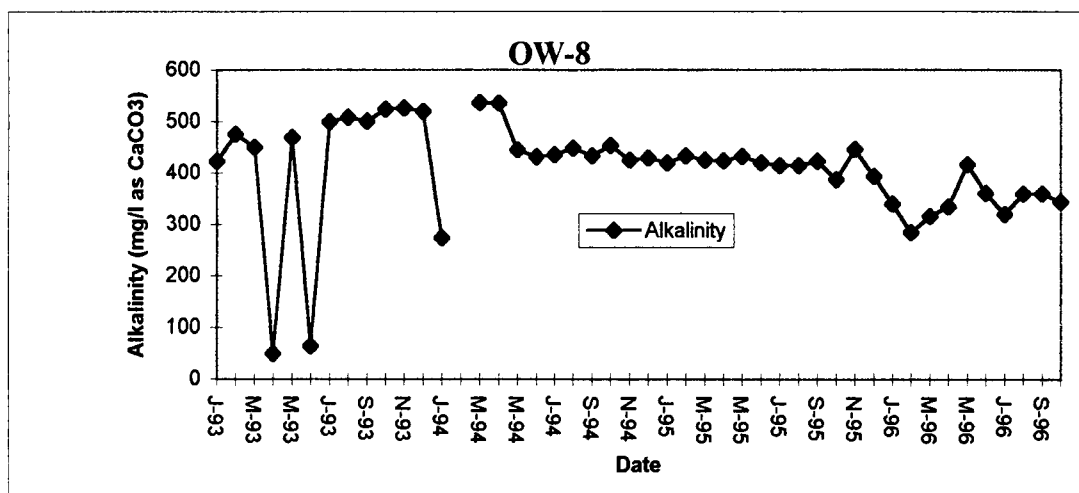
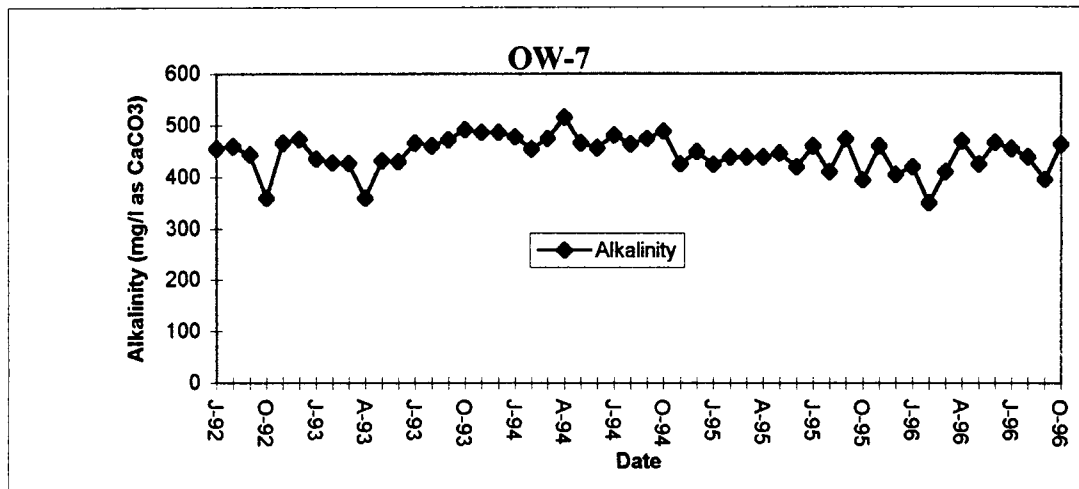


Figure 24: Alkalinity Graphs for OW's 7, 8, and 9.

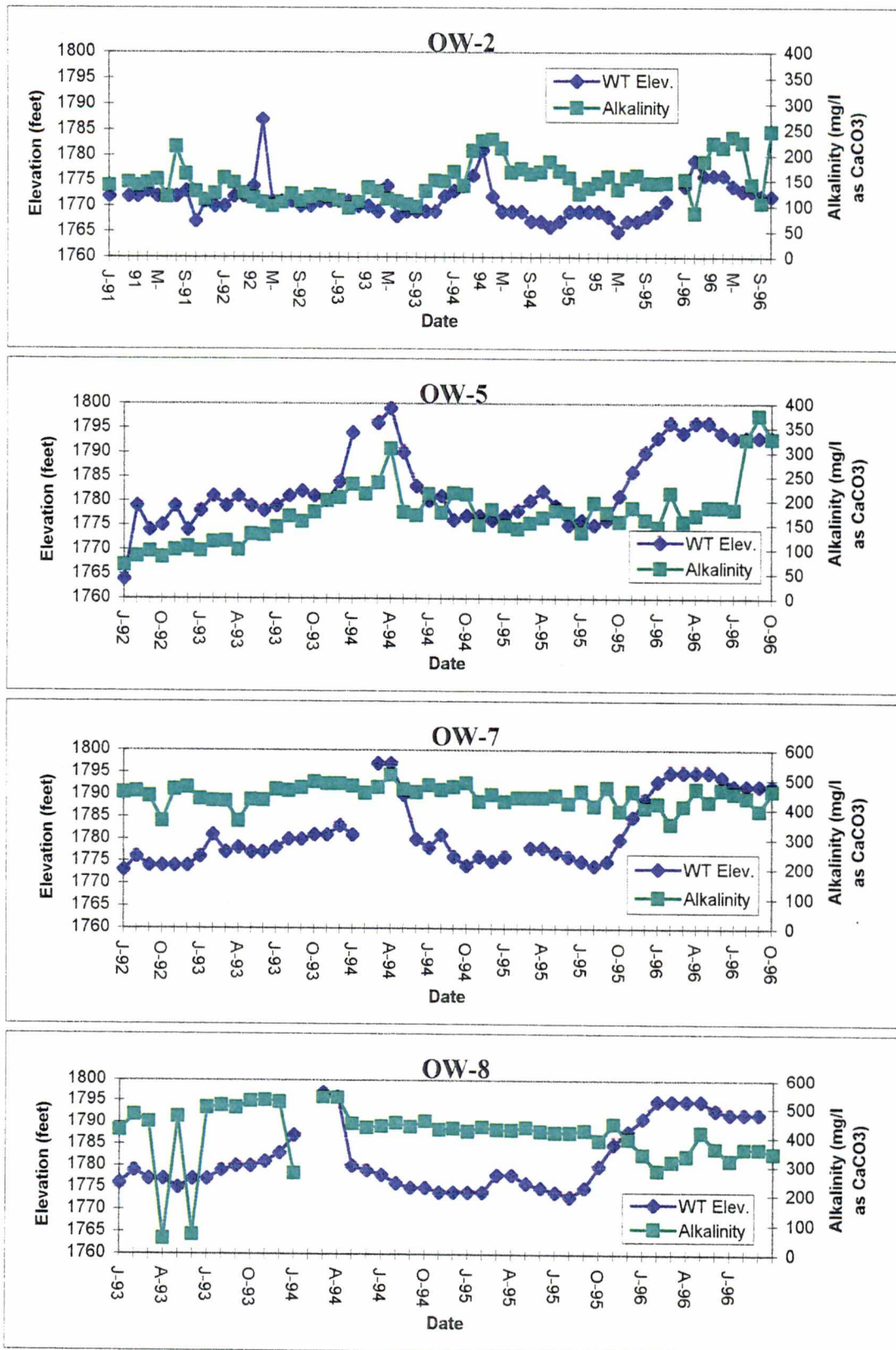


Figure 25: Water Table Elevations relative to Alkalinity for OW's 2, 5, 7, and 8.

with water. The elevated water levels in the final pit created a hydraulic head that resulted in groundwater flow from the final pit in the direction of Glady Fork Creek.

Alkalinity concentrations in OW's 7 and 8, however, have decreased following pit closure. Figure 24 indicates that after August, 1995 alkalinity concentrations in OW's 7 and 8 began to fluctuate and, in general, are decreasing. Although alkalinity concentrations in OW's 7 and 8 are decreasing, they are still nearly twice as high as alkalinity concentrations in OW's 2, 3, and 5. The decrease in alkalinity concentrations in OW's 7 and 8 indicates that alkalinity is being consumed by acidity in the groundwater as the water table recovers. As the water table recovers following pit closure, the rising water table effectively "flushes" oxidations products generated prior to water table recovery. The acidic material will consume alkalinity as indicated in Equations 8 and 9, resulting in a decrease in alkalinity.

Relative constant alkalinity concentrations in OW's 2 and 5 following pit closure suggests that alkalinity was not being consumed during water table recovery. Alkalinity would not be consumed as groundwater recovered due to the isolation of OW's 2 and 5 from oxidation that occurred while the pit remained open. Evidence suggesting the migration of an alkaline-enriched groundwater plume from the TMHP area is the increase in alkalinity following pit closure in OW's 2 and 5.

Alkalinity graphs do not indicate a relationship between the Alkaline Recharge Structures (ARS's) and alkalinity concentrations in the backfill. This conclusion was reached by considering the date the ARS's were constructed and alkalinity concentrations in observations wells downgradient from ARS's. The alkalinity graphs of OW's 2, 5, and 8, located downgradient from ARS's #'s 1, 2, and 3 respectively, indicate increases in alkalinity, however the increases are evident several months after the installation of the ARS's. The increases in alkalinity are believed by the writer to have resulted from a flow of alkaline-enriched groundwater, as previously discussed (Figure 25), and not from the ARS's.

Sulfate

Graphs of sulfate concentrations depicted in Figures 26 and 27, indicate that sulfate concentrations are increasing in all observation wells with the exception of OW-9. Increasing sulfate concentrations

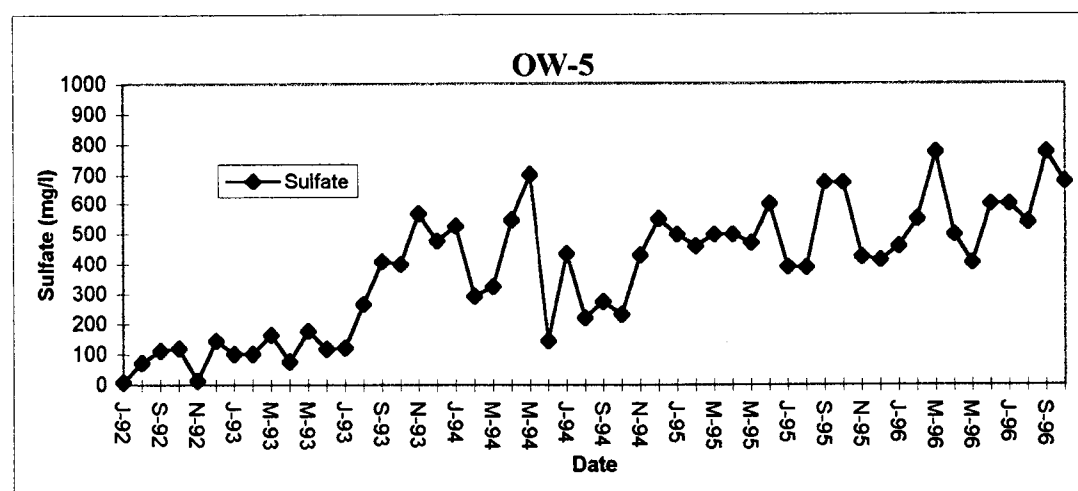
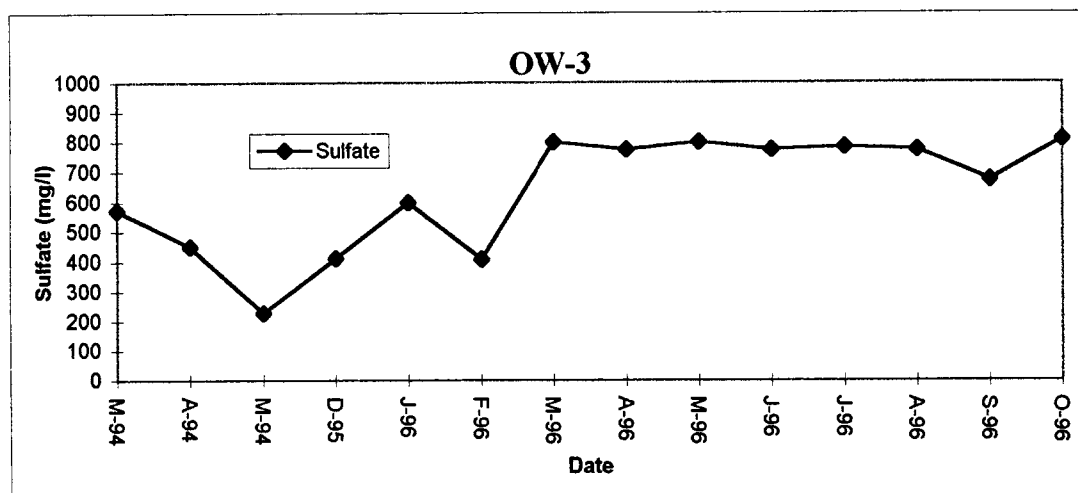
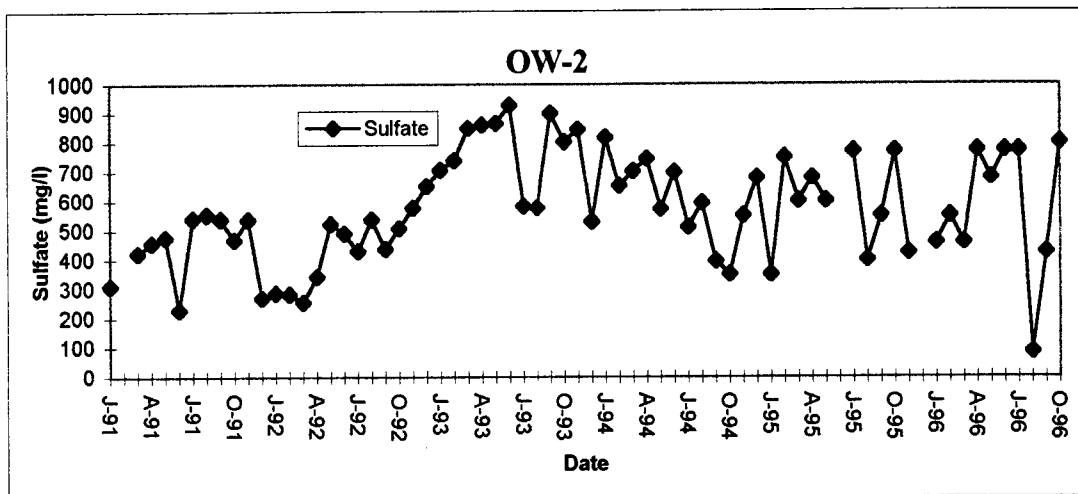


Figure 26: Sulfate Graphs for OW's 2, 3, and 5.

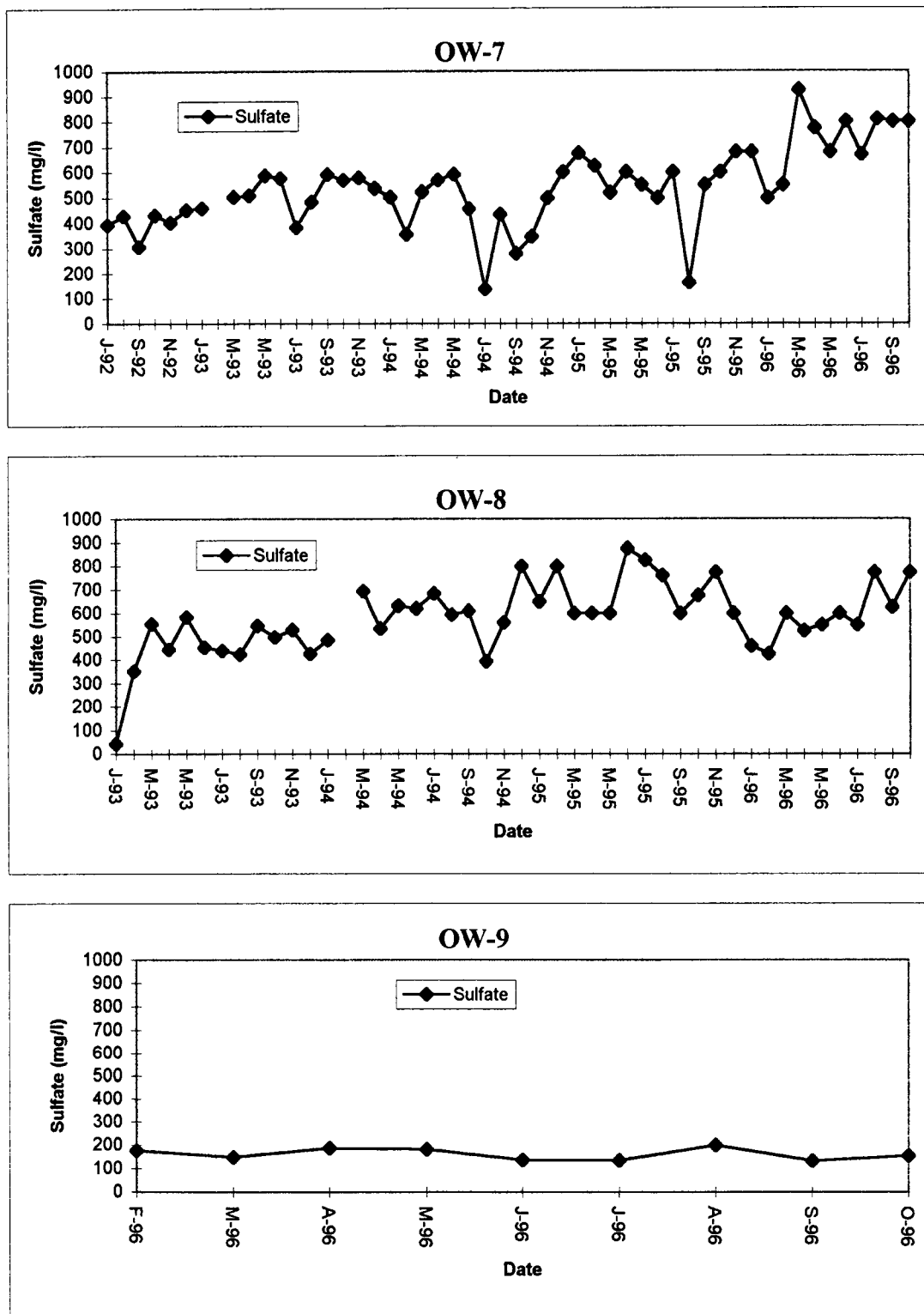


Figure 27: Sulfate Graphs for OW's 7, 8, and 9.

indicates that oxidation of sulfidic material is occurring in the backfill. Increasing sulfate concentrations can be related to the precipitation (i.e. rainfall) and groundwater fluctuations in the study area. As indicated in sulfate graphs of OW's 2, 5, and 7, sulfate concentrations increased until mid-1993 followed by a general decrease in sulfate concentrations until October, 1994. After October, 1994, sulfate concentrations increased in OW's 2, 5, 7, and 8. The increase in sulfate concentrations can be related to 59.96" of precipitation prior to October, 1994. The relative "wet" period prior to October, 1994 appears to have generated a "piston" flow of recharge in the study area. The increase in recharge "flushed", sulfosalts from the backfill resulting in increased sulfate concentrations in the groundwater. Sulfate concentrations peaked in December, 1994, followed by a general decrease during a relatively "dry" year in 1995.

Increasing sulfate concentrations in response to groundwater recovery following pit closure is evident in sulfate graphs of OW's 5, and 7. Sulfate concentrations in OW's 5, and 7 remained relatively constant until the final pit was closed in August, 1995, however after the pit was closed sulfate concentrations began to increase. The increasing groundwater levels during water table recovery following pit closure generated a "wet edge" that "flushed" the backfill, resulting in an increase in sulfate concentrations. Sulfate concentrations in all observation wells are generally increasing as Fe concentrations decrease, an anomaly that indicates the inexact nature of chemical reactions in mine backfill.

Chapter 10.

Evaluation of TMHP and Expected Future Performance

Effectiveness of the TMHP is suggested by evaluation of chemical parameter graphs. The TMHP may be, at least in part, cause for: 1) decreasing metal concentrations, 2) increasing alkalinity concentrations, and 3) increasing pH values in backfill groundwater of the study area. The influence of the TMHP on groundwater quality in the backfill is most evident in water quality of observation wells located closest to the area where the TMHP was implemented. Water quality in observation wells located closest to the TMHP boundary (OW's 7, 8, and 9) indicate lower metal concentrations, higher alkalinity and higher pH values relative to observation wells located farthest from the TMHP boundary (OW's 2, 3, and 5).

Evaluation of water quality data also suggests that the TMHP could be related to decreasing metal concentrations and increasing alkalinity in areas of the backfill where the TMHP was not implemented. Groundwater quality parameters in observation wells located farthest from the TMHP area indicate trends of decreasing metal concentrations, increasing alkalinity, and increasing pH values (with the exception of OW-3). The influence of the TMHP on groundwater quality in areas where the TMHP was not implemented, is considered to be related to groundwater flow in the backfill. As indicated in Chapter 8, groundwater flow in the backfill is primarily to the southeast portion of the study area (downgradient from the TMHP area) and in the direction of OW-2 with the groundwater most likely discharging into Gladly Fork Creek and Spring Branch. The flow of groundwater from the TMHP area is expected to be saturated with respect to calcium carbonate which could explain increased alkalinity and decreasing metal concentrations in observation wells located farthest from the TMHP area.

The future performance of the TMHP can be evaluated by considering the groundwater quality of OW-9 which was completed in the TMHP area. Total and dissolved Fe concentrations measured in OW-9 from February to October, 1996, averaged 1.55 and 0.31 mg/l respectively, while total and dissolved Mn

concentrations measured 4.1 and 3.4 mg/l respectively. The Fe concentrations are below EPA effluent discharge limits and the Mn concentrations are decreasing to the EPA limit. Also, pH values in OW-9 are near neutral (averaging 6.42) and alkalinity concentrations are maintaining a consistent concentration of nearly 325 mg/l as CaCO_3 . Sulfate concentrations in OW-9 are also much lower than observation wells completed outside of the TMHP area, suggesting that if the TMHP had been implemented throughout the entire study area, the oxidation of sulfidic materials could have been abated.

An important factor in this evaluation is that observation well data from the study area represents groundwater quality from mine backfill (not from mine effluent) where the final pit has been closed for less than two years. The writer considers the water quality within the backfill of the study area as exceptional when considering literature reviews of other recently closed mine operations. In the writer's opinion, backfill water quality values used to evaluate the impact of mining that are close to and/or below EPA standards within a recently closed mining operation should be taken as indications of success in reclamation practices. Continuing to monitor the groundwater quality in all observation wells emplaced in the backfill is suggested to determine the long-term effectiveness of the TMHP in abating adverse environmental impacts associated with surface coal mining.

Chapter 11.

Conclusions

Evaluation of geochemical and hydrological investigations have concluded that:

- 1) The siderite bearing lithology is composed of nearly 1 weight percent Mn and can be considered a potential source for elevated levels of Mn concentrations in the backfill groundwater of the study area.
- 2) The sandy shale unit within the Whitwell shale contains an average of 1.35 weight percent sulfur, indicating that it is capable of acid production in terms of lowering pH in backfill groundwater and mine effluent.
- 3) In order to determine an accurate prediction of the acid potential of mine overburden, the siderite bearing lithology should be recognized as a potential acid-producer and proper analytical procedures should be used during Neutralization Potential analyses of siderite-bearing lithologies.
- 4) Hydrogeologic cross-sections of the study area indicate that groundwater flow within the backfill is to the east and southeast portion of the study area and discharge from the backfill is primarily to the east into Spring Branch and into Glady Fork Creek, north of the dewatering wells. Discharge into Glady Fork Creek in the southeastern corner of the site, where the seeps had been noted previously are not evident at this time due to pumping from the dewatering wells in this area that have lowered water levels in the backfill below the elevation of Glady Fork Creek. It is expected that shut down of the dewatering wells would cause the seeps to reappear and these could have a negative impact on the water quality of the creek.
- 5) Evaluation of data obtained from monthly monitoring of observation wells installed in the backfill of the study area suggests that the TMHP is having a positive affect on groundwater quality in the backfill of study area in terms of increasing alkalinity, reducing metal concentrations, and increasing and maintaining high pH values. The effectiveness of the TMHP, in regards to increasing alkalinity and

decreasing metal concentrations, can be determined by noting that alkalinity concentrations are highest and metal concentrations lowest in observation wells closest to the TMHP.

6) Increasing sulfate concentrations in all observation wells (with the exception of OW-9) indicates that sulfide oxidation continues to occur in the study area. However sulfate concentrations in OW-9, located within the TMHP area, are nearly three times as low as concentrations in OW-8 (located closest to the TMHP area), which suggests that if the TMHP had been implemented throughout the entire study area, sulfate concentrations would most likely be lower.

7) Taking into consideration that mining within the study area has been completed for a relatively short period of time and that the backfill groundwater quality indicates trends of decreasing metal concentrations, increasing alkalinity, and increases in pH, the TMHP is a method to be considered during reclamation activities.

8) In order to evaluate the long-term effectiveness of the TMHP as a reclamation technique, further monitoring of backfill groundwater quality and discharge into the local creeks is recommended.

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APPENDICES

Appendix A:
Sample Selection and Preparation

Sample Selection

Representative overburden samples were collected and sealed in ziploc storage bags until they were prepared for analyses conducted at the Department of Geological Sciences at the University of Tennessee-Knoxville.

EMP Sample Selection and Preparation

A siderite clast from the siderite conglomerate of the Newton sandstone has been chosen for analysis in this study because of the sharp contact between siderite clasts and the sandstone matrix. Siderite beds within the Whitwell shale were not chosen for analyses because the contact is gradational, not providing a clear distinction between the siderite beds and the shale.

A representative siderite clast was cut and mounted on a one-inch diameter circular glass slide for EMP analyses. The sample was trimmed, ground on a grinding platform and then polished using progressively finer grit sandpaper. The sample was then placed on a rotary polisher and progressively polished with 6, 3 and 1 micron diamond polishing paste. The sample was examined in reflected light during the process to judge the quality of the polishing.

The next stage of preparation involved examination with a petrographic microscope using plane-polarized light to determine the petrology of the sample. Thin-section microscopy provided a preliminary identification of what appear to be four distinct phases.

The four phases include:

- 1) Quartz.
- 2) A carbonate phase that is yellowish-white under plane-polarized light.
- 3) A clay-like phase (i.e., grain size $<1\mu\text{m}$) that is black to dark brown under plane-polarized light.
- 4) A framboidal iron sulfide phase comprising $<1\%$ of the thin-section sample area.

After completing the petrographic analysis, areas were chosen within the sample for EMP analyses. The area was outlined using a rapidograph ink pen to facilitate the location of the sample area when in the EMP. The sample was then carbon coated for EMP analyses.

XRF Sample Preparation (for all samples including siderite clast)

The XRF analytical preparation involved grinding a 10 gram sample of the siderite clast in a Shatterbox using a few drops of freon as a grinding agent for six minutes. Two, boric-acid backed pellets were then prepared using a pellet die and hydraulic press. The samples were pressed for 30 seconds at 10,000 psi followed by one minute of pressing at 30,000 psi. The samples were removed and stored in a dessicator to preserve sample integrity until XRF analysis.

XRD Sample Preparation

XRD analytical preparation involved size-separation of a siderite clast and preparing a smear-mount from a shale sample. In order to determine the clay phase contained in the siderite clast, size-separation was performed. 45 grams of a representative siderite clast were ground for 3 minutes in a Shatterbox with a tungsten carbide mill. The sample was then mixed with distilled water in a quart container for particle size separation. The sample was allowed to settle for 35 minutes in order for the particles larger than 5 μ m to settle to the bottom of the container. The distilled water was then pipetted from the container to a predetermined level and transferred to another quart container that was then filled with distilled water. The solution was mixed and allowed to settle for 35 minutes in order for the 2-5 μ m size particles to settle from suspension. The solution was then pipetted to the predetermined level and transferred to a third quart container. The third container was filled with water, mixed and allowed to settle for 3 hours and 40 minutes in order for the 2 μ m size particles to settle. After 3 hours and 40 minutes the solution was separated into four, pint-sized bottles and placed in a Heraeus Instruments Megafuge 1.0 to obtain the 2 μ m size particles. The solution was centrifuged at 1,700 rpm for 22 minutes. After 22 minutes the four containers were decanted and the 2 μ m size particles were transferred to a container. The 2 μ m size particles were transferred with an eyedropper to a 3/4" x 1 1/2" glass slide and allowed to dry overnight prior to XRD analysis.

A smear mount was made using a 10 gram sample that was ground in a Shatterbox for 6 minutes. After grinding, the sample was removed and a portion of the sample was mixed with distilled water, placed on a glass slide and allowed to dry prior to XRD analysis.

Appendix B:
Analytical Techniques and Methods

Analytical Techniques

Various analytical techniques were employed to provide an overall geochemical characterization of the representative overburden samples. Analytical techniques included EMP, XRD, and XRF analyses. In order to compare analytical results using EMP and XRF analyses, a siderite clast was separated from a siderite conglomerate and cut in half. One-half of the clast was analyzed using EMP, the other half was analyzed using XRF. The analytical techniques listed below were utilized to provide a quantitative geochemical characterization of the representative samples.

Electron Microprobe Analysis (EMP)

EMP analyses were used to provide geochemical characterization of the siderite clast using a four-spectrometer CAMECA SX-50 located in the Geological Sciences Department on the University of Tennessee-Knoxville (UTK) campus. The EMP was used exclusively on the siderite clast. The accelerating voltage was 15 kV with beam currents of 10 and 20nA. A 10nA beam current was used for analyses of the siderite clast as a carbonate and a 20nA current was used for analyses of the siderite as an oxide. The beam size was varied for individual phases. Beam sizes ranged from 2 μ m for sulfide analyses to 15 μ m for broad-beam analyses. Counting times of 20 seconds were used. All data were corrected using ZAF procedures.

After completion of x-ray mapping and energy dispersive spectrometer (EDS) analyses, standards were selected for analysis of the siderite clast. The standards were selected based on x-ray mapping and EDS results. Analytical crystals were then selected for sample and standard analysis on the basis of the characteristic wavelength for each element. Standards were calibrated prior to sample analyses and were checked after sample analyses were completed.

X-ray Fluorescence (XRF) Analysis

An energy dispersive, EG&G ORTEC TEFA III (Tube Excited Fluorescence Analyzer) System was employed for elemental analyses of all samples. ORTEC's ATAC program combined with FECLAY,

LSDOL, and SIAL protocols prepared in the Department of Geological Sciences were used for quantitative elemental analyses. Two different protocols were used to analyze the siderite clast as a clay (FECLAY) and as a carbonate (LSDOL). The SIAL protocol was used to analyze the sandstone samples.

X-ray Diffraction (XRD) Analysis

X-ray Diffraction (XRD) was used to determine the mineralogical phases present in the siderite and shale lithologies. XRD analysis of the siderite clast and Whitwell shale, including both bulk and $<2\mu\text{m}$ fractions, were performed using a Siemens Difract-5000 XRD instrument located in the X-ray Laboratory in the UTK Department of Geological Sciences. The analytical angles used during XRD analysis ranged from $2-60^\circ$ 2 theta. A current beam of 30nA was used during analyses.

Analytical Methods

Due to the heterogeneous nature of the siderite clast, various analytical methods were applied during EMP analyses. The various analytical methods were deemed necessary in order to provide a comprehensive geochemical analysis of the siderite clast. Back-scattered electron (BSE) imaging was employed to assist in choosing sample analysis points prior to EMP analysis. BSE imaging provided superior resolution of the sample which aided in the selection of areas to perform sample analyses. BSE mapping indicates elemental intensities as a direct function of average atomic numbers.

The analytical methods performed with the EMP included:

a. Broad-beam Analysis

A broad-beam ($15\mu\text{m}$), seventeen-point traverse analysis using the CAMECA SX-50 was made across the siderite clast in order to obtain an average composition of the siderite clast. The traverse step size was $1.16\mu\text{m}$. The accelerating voltage was 15 kV. Counting times of 20 seconds were used and all data were corrected using ZAF procedures. The siderite clast was analyzed as an oxide assuming CO_2 by difference.

b. Single-Phase Analysis

An individual point analysis of the siderite phase and the clay-like phase was conducted using a beam diameter of 5 μ m and a beam current of 10nA. The counting time and accelerating voltages were the same as those used for the broad-beam traverse analyses. The individual point analyses were used in order to determine the bulk elemental compositions for the two phases. The individual point analysis analyzed the siderite clast as an oxide assuming CO₂ by difference.

c. Energy Dispersive Spectrometer (EDS) Analysis

Energy Dispersive Spectrometer (EDS) analysis using an Oxford Instruments EDS located in the Geological Sciences Department on the UTK campus was used to identify major elements contained in the siderite clast prior to EMP analyses.

EDS was used to identify major elements and construct an x-ray map of areas selected for EMP analyses.

Multi-image Phase Analysis

A multi-image phase analysis using the EDS was used to provide a Reconstructed Whole-Rock Analysis (RWRA) of the siderite clast. The multi-image phase analysis identifies the specific type and abundance of the distinct mineralogical phases present in the siderite clast. The four phases previously classified were identified during the multi-image phase analysis. The multi-image phase analysis determined the areal percentage of each individual phase. Seven areas of the siderite sample were analyzed using multi-image phase analysis. The total area analyzed was 217,464 sq. μ m. The areal percentage for each phase combined with the average elemental concentrations of the individual phases was used to construct the RWRA.

Appendix C:
RWRA, Sulfide Analyses and Discussion of
Analytical Differences Between EMP and XRF

RWRA and Sulfide Analyses

Table C.1 is the RWRA from EMP, Tables C.2 and C.3 are results from the sulfide analysis. The XRF analysis using the FECLAY protocol, disregarding Mn concentrations, did result in elemental analyses that were similar to EMP datum. For example, the MgO concentration of 3.87 weight % determined by XRF was similar to the 3.51 weight % MgO determined by EMP. Another similarity, the Al_2O_3 concentration of 3.64 weight %, is close to the 3.29 weight % determined by EMP. The Al concentration determined by EMP and XRF can be related to modal percentages determined during EMP analyses to substantiate the presence of illite in the siderite clast. Pure illite has been found to be composed of roughly 25% Al_2O_3 . The modal percentage of the clay phase in the siderite clast was determined to be nearly 10%. The average Al_2O_3 content determined by EMP and XRF was found to be 3.5% Al_2O_3 , or nearly one-third of the clay phase. Considering the analytical precision and the heterogeneity of the siderite clast, the 3.5% Al_2O_3 contained in the clay phase is comparable to the 25% Al_2O_3 contained in pure illite.

Total elemental percentages determined by XRF analyses averaged 84.7% using the FECLAY protocol. The RWRA generated using EMP totaled 101.07%. If the assumption of CO_2 by difference is ignored in the RWRA, the 36.9% CO_2 value can be subtracted from the 101.07% average to total 64.2%. This produces a range of 84.7% determined by XRF to 64.2% for the total elemental concentrations determined by the two analytical methods. The reasons for low analytical totals have been discussed and would therefore explain the discrepancy between the RWRA and XRF modal percentages.

Table C.2 contains datum generated as a result of the representative sulfide analysis. This analysis selected analytical points on framboidal pyrite phases.

Table C.3 includes the recalculated formula moles for the sulfide phase. The trace elements including Co, Ni, Cu, and Zn were not included in formula mole calculations because the measured levels for these elements were close to EMP detection limits. Table 8 indicates that in some cases, the trace elements were not detected.

Table C.1: Reconstructed Whole-Rock Analysis.

Cmpd.	Phase Elemental Average			(Phase Elemental Avg.)(Phase %)			
	Siderite	Quartz	Clay	Siderite	Quartz	Clay	RBRC
FeO	49.1	0	4.72	43.7	0	0.42	44.1
MnO	1.11	0	0.18	0.99	0	0.02	1.01
CaO	3.88	0	0.39	3.45	0	0.04	3.49
MgO	3.67	0	4.94	3.27	0	0.44	3.71
K ₂ O	0.11	0	8.11	0.10	0	0.73	0.83
SiO ₂	0.66	100	44.0	0.59	3	3.96	7.55
Al ₂ O ₃	0.45	0	28.6	0.40	0	2.57	2.97
Na ₂ O	0.07	0	0.14	0.06	0	0.01	0.07
P ₂ O ₅	0.46	0	0.35	0.41	0	0.03	0.44
CO ₂	40.4	0	11.1	35.9	0	0.99	36.9
				Note:		Total: 101.07	
				Phase	Phase %		
				Siderite	0.89		
				Quartz	0.03		
				Clay	0.11		

Table C.2: Representative Sulfide Analysis.

Element	Sample			Average
	Sam. 1.3	Sam. 1.4	Sam. 1.5	
Fe	44.9	45.1	45.9	45.3(0.5)
S	50.7	50.9	51.3	51.1(0.3)
Co	<0.03	<0.03	<0.03	NA
Ni	0.05	0.10	0.05	0.07
Cu	0.05	<0.03	<0.03	0.05
Zn	0.08	<0.03	<0.03	0.08
Total	95.8	96.1	97.3	96.5

Table C.3: Sulfide Formula Moles.

Element	Average Wt. %	Atomic Weight	Atomic Proportion	Atomic Ratios
S	51.1	32.07	1.59	2
Fe	45.3	55.85	0.8111	1
Co	NA	NA	NA	NA
Ni	0.07	58.7	0.0011	NA
Cu	0.05	63.54	0.0006	NA
Zn	0.05	65.38	0.0008	NA

Discussion of Analytical Variations Between EMP and XRF Analyses

EMP and XRF analyses revealed Mn concentrations ranging from 0.28 to 1.3 wt % in the siderite clast. This range in Mn concentrations is believed to be most likely attributable to the analytical accuracy inherent in the analytical methods used.

EMP

EMP analyses reveal variation in elemental concentrations, as well as, low analytical totals. The wide range in elemental concentrations is reflected in the sigma values. The sigma values, or standard deviation, is a measure of the dispersion of data in statistics. It is the writer's opinion that the wide range in dispersion and low total concentrations can be attributed to several factors. These include:

- 1) Sample heterogeneity.
- 2) The presence of elements not comprising a significant portion of the siderite clast and were therefore not detected during analysis (i.e. trace elements, etc.).
- 3) The ZAF (i.e. atomic number, absorption and fluorescence) correction procedure assumes uniform composition in total volume excited by electron beam and x-rays. For example, the selection of a grain size $<5\mu\text{m}$ may have up to $15\mu\text{m}$ diameter interaction volume due to the interaction of electron beam and x-ray currents.
- 4) The presence of fine-grained clay inclusions. The siderite phase often included and/or was surrounded by a clay phase. The penetration of the phase by the electron beam or the overlap onto the clay-like phase by the electron beam during analyses might have resulted in "edge effects". For this reason, phase boundaries were avoided, and analyses were conducted in the middle of seemingly pure phases. However the thickness of the phase might have been penetrated by EMP analyses to a depth where another phase might have been encountered. In order to account for the mixing of two phases, the siderite was analyzed using standards that would detect elements common in clay minerals.
- 5) The presence of water in the clay-like phase can also explain the low reported totals. The presence of water in the clay phase would not be detected during EMP analyses. The water content could

comprise a significant percentage of the clay-like phase which would explain why elemental concentrations did not total 100%.

6) The presence of oxidized iron (Fe^{+3}) in the siderite clast could also account for low reported totals. EMP analyses are based on analyzing the reduced form of iron (Fe^{+2}). The presence of Fe^{+3} would disrupt the software used to calculate FeO during EMP analyses. EMP software is based on calculating Fe^{+2} rather than Fe^{+3} which could explain why totals were less than 100% for the siderite clast. For example, Table 4 includes siderite clast analyses that were conducted assuming the siderite clast as a carbonate. The variations in the $\text{Fe}(\text{CO}_3)$ ranged from 74.4% to 81.2%. The $\text{Fe}(\text{CO}_3)$ values reflect roughly an 8% variation between the low value and the high value. This variation could possibly be attributed to the presence of Fe^{+3} in the siderite clast. The presence of Fe^{+3} , as well as interference from the clay phase could explain the low reported totals in Table 4.

XRF

The XRF analyses revealed a range of Mn concentrations from 0.28 to 1.38wt %. The range of Mn concentrations can be attributed to the analytical accuracy of the XRF. The two protocols that were used to analyze the siderite clast did not contain standard samples that contained either a matrix or Mn concentration similar to the siderite clast. The FECLAY protocol was used because the siderite clast has a similar matrix as clay minerals, as well as containing an abundance of Fe. The FECLAY protocol, however, did not use standards that contained a Mn concentration similar to the siderite clast. The best-fit line that the XRF uses to determine elemental concentrations is based on standard samples that contain elemental concentrations that "bracket" (i.e. contain concentrations between the expected lowest and highest elemental concentrations) concentrations of the unknown sample. In this case, the Mn concentration in the siderite clast was higher than the Mn concentration of the standard sample. This would call for extrapolation of the best-fit line above the maximum standard concentration. For this reason, the Mn concentration determined using the FECLAY protocol is questionable and can be considered as a low-end concentration of Mn in the siderite conglomerate.

The Mn concentrations reported using the LSDOL protocol averaged 1.38 weight % Mn. The standard selection contained in the LSDOL protocol bracketed the Mn concentration detected in the sample, however, the best-fit line was drawn between a cluster of standard samples with Mn concentrations averaging 0.09 weight % to a single point (rhodocrosite) with a Mn concentration of 45.7 weight %. The extrapolation of the best-fit line between a cluster of standard samples with very low Mn concentrations (i.e. 0.09 weight %) to a single point with an extremely high Mn concentration (i.e. 45.7 weight %) results in a questionable best-fit line.

The SIAL protocol had not been recently run and was ready for recalibration. The high elemental totals can be attributed to high normalization factors calculated during elemental analyses. The normalization factors were out of range (i.e. 1.04-1.08), however the machine was to be turned off and it was not known when it would be up again. The silica concentration was high, however major elements were within 2 sigma. The reported values can be considered accurate within $\pm 10\%$ of the reported value (i.e. relative error is about 10%).

Appendix D:
Groundwater Elevations for OW's 1, 2, 3, 4, 5, 7, 8, 9, and 10.

Table D.1: Water Table Elevations for OW-1 (left) and OW-2 (right).

Date	WT Elev.	Date	WT Elev.	Date	WT Elev.	Date	WT Elev.
J-91	1779	J-94	1781	J-91	1772	J-94	1773
F-91	1779	F-94	1780	F-91		F-94	1774
M-91	1779	M-94	1780	M-91	1772	M-94	1776
A-91	1780	A-94	1779	A-91	1772	A-94	1781
M-91	1780	M-94	1779	M-91	1773	M-94	1772
J-91	1780	J-94	1779	J-91	1772	J-94	1769
J-91	1780	J-94	1784	J-91	1772	J-94	1769
A-91	1779	A-94	1779	A-91	1772	A-94	1769
S-91	1779	S-94	1779	S-91	1773	S-94	1767
O-91	1780	O-94	1779	O-91	1767	O-94	1767
N-91	1779	N-94	1779	N-91	1771	N-94	1766
D-91	1779	D-94	1779	D-91	1770	D-94	1767
J-92	1779	J-95	1779	J-92	1770	J-95	1769
F-92	1781	F-95	1780	F-92	1772	F-95	1769
M-92	1789	M-95	1781	M-92	1772	M-95	1769
A-92	1781	A-95	1780	A-92	1774	A-95	1769
M-92	1791	M-95	1779	M-92	1787	M-95	1768
J-92	1779	J-95	1777	J-92	1771	J-95	1765
J-92	1784	J-95	1779	J-92	1771	J-95	1767
A-92	1782	A-95	1779	A-92	1771	A-95	1767
S-92	1780	S-95	1779	S-92	1770	S-95	1768
O-92	1780	O-95	1779	O-92	1770	O-95	1769
N-92	1780	N-95	1780	N-92	1771	N-95	1771
D-92	1780	D-95	1780	D-92	1771	D-95	
J-93	1786	J-96	1780	J-93	1771	J-96	1774
F-93	1779	F-96	1780	F-93	1771	F-96	1779
M-93	1777	M-96	1780	M-93	1770	M-96	1776
A-93	1777	A-96	1780	A-93	1770	A-96	1776
M-93	1779	M-96	1786	M-93	1769	M-96	1776
J-93	1786	J-96	1781	J-93	1774	J-96	1774
J-93	1779	J-96	1780	J-93	1768	J-96	1773
A-93	1779	A-96	1780	A-93	1769	A-96	1773
S-93	1779	S-96	1780	S-93	1769	S-96	1772
O-93	1779	O-96	1780	O-93	1769	O-96	1772
N-93	1779			N-93	1769		
D-93	1779			D-93	1772		

*Units in feet.

Table D.2: Water Table Elevations for OW-3 (left) and OW-4 (right).

Date	WT Elev.
M-94	1796
A-94	1797
M-94	1789
D-95	1789
J-96	1792
F-96	1795
M-96	1794
A-96	1795
M-96	1795
J-96	1794
J-96	1791
A-96	1791
S-96	1791
O-96	1791

Date	WT Elev.	Date	WT Elev.
J-91	1769	J-94	1760
F-91		F-94	
M-91	1759	M-94	1779
A-91	1759	A-94	1760
M-91	1759	M-94	1758
J-91	1760	J-94	1758
J-91	1759	J-94	1758
A-91	1759	A-94	1758
S-91	1759	S-94	1758
O-91	1774	O-94	1758
N-91	1769	N-94	1771
D-91	1771	D-94	1778
J-92	1775	J-95	1768
F-92	1772	F-95	
M-92	1763	M-95	1780
A-92	1771	A-95	1776
M-92	1765	M-95	1771
J-92	1765	J-95	1768
J-92	1766	J-95	1758
A-92	1759	A-95	1758
S-92	1758	S-95	1758
O-92	1758	O-95	1765
N-92	1759	N-95	1767
D-92	1758	D-95	
J-93	1759	J-96	1770
F-93	1759	F-96	1774
M-93	1759	M-96	1770
A-93	1763	A-96	1772
M-93	1759	M-96	1760
J-93	1758	J-96	1760
J-93	1758	J-96	1759
A-93	1758	A-96	1762
S-93	1758	S-96	1762
O-93	1758	O-96	1761
N-93	1759		
D-93	1760		

*Units in feet.

Table D.3: Water Table Elevations for OW-5 (left) and OW-7 (right).

Date	WT Elev.	Date	WT Elev.	Date	WT Elev.	Date	WT Elev.
J-92	1764	J-95	1777	J-92	1773	J-95	1776
A-92	1779	F-95	1778	A-92	1776	F-95	NA
S-92	1774	M-95	1780	S-92	1774	M-95	1778
O-92	1775	A-95	1782	O-92	1774	A-95	1778
N-92	1779	M-95	1779	N-92	1774	M-95	1777
D-92	1774	J-95	1775	D-92	1774	J-95	1776
J-93	1778	J-95	1776	J-93	1776	J-95	1775
F-93	1781	A-95	1775	F-93	1781	A-95	1774
M-93	1779	S-95	1776	M-93	1777	S-95	1775
A-93	1781	O-95	1781	A-93	1778	O-95	1780
M-93	1779	N-95	1786	M-93	1777	N-95	1785
J-93	1778	D-95	1790	J-93	1777	D-95	1789
J-93	1779	J-96	1793	J-93	1778	J-96	1793
A-93	1781	F-96	1796	A-93	1780	F-96	1795
S-93	1782	M-96	1794	S-93	1780	M-96	1795
O-93	1781	A-96	1796	O-93	1781	A-96	1795
N-93	1780	M-96	1796	N-93	1781	M-96	1795
D-93	1784	J-96	1794	D-93	1783	J-96	1794
J-94	1794	J-96	1793	J-94	1781	J-96	1792
F-94	NA	A-96	1793	F-94	NA	A-96	1792
M-94	1796	S-96	1793	M-94	1797	S-96	1792
A-94	1799	O-96	1793	A-94	1797	O-96	1792
M-94	1790			M-94	1790		
J-94	1783			J-94	1780		
J-94	1780			J-94	1778		
A-94	1781			A-94	1781		
S-94	1776			S-94	1776		
O-94	1777			O-94	1774		
N-94	1777			N-94	1776		
D-94	1776			D-94	1775		

*Units in feet.

Table D.4: Water Table Elev. for OW-8 (top), OW-9 (middle) and OW-10 (bottom).

Date	WT Elev.	Date	WT Elev.	Date	WT Elev.	Date	WT Elev.
J-93	1776	J-94	1787	J-95	1774	J-96	1791
F-93	1779	F-94	NA	F-95	1774	F-96	1795
M-93	1777	M-94	1797	M-95	1778	M-96	1795
A-93	1777	A-94	1796	A-95	1778	A-96	1795
M-93	1775	M-94	1780	M-95	1776	M-96	1795
J-93	1777	J-94	1779	J-95	1775	J-96	1793
J-93	1777	J-94	1778	J-95	1774	J-96	1792
A-93	1779	A-94	1776	A-95	1773	A-96	1792
S-93	1780	S-94	1775	S-95	1775	S-96	1792
O-93	1780	O-94	1775	O-95	1780	O-96	1792
N-93	1781	N-94	1774	N-95	1785		
D-93	1783	D-94	1774	D-95	1788		

Date	WT Elev.
J-96	1791
F-96	1794
M-96	1793
A-96	1794
M-96	1794
J-96	1792
J-96	1790
A-96	1790
S-96	1790
O-96	1790

Date	WT Elev.	Date	WT Elev.	Date	WT Elev.	Date	WT Elev.
J-93	1808	J-94	1810	J-95	1810	J-96	1810
F-93	1808	F-94	1810	F-95	1810	F-96	1810
M-93	1808	M-94	1810	M-95	1810	M-96	1810
A-93	1809	A-94	1810	A-95	1809	A-96	1810
M-93	1809	M-94	1808	M-95	1808	M-96	1810
J-93	1809	J-94	1811	J-95	1809	J-96	1810
J-93	1809	J-94	1810	J-95	1809	J-96	1810
A-93	1808	A-94	1810	A-95	1809	A-96	1810
S-93	1808	S-94	1810	S-95	1809	S-96	1810
O-93	1808	O-94	1810	O-95	1810	O-96	1810
N-93	1809	N-94	1809	N-95	1810		
D-93	1809	D-94	1809	D-95	1810		

*Units in feet.

Appendix E:
Groundwater Analyses Obtained in the Field

Table E.1: Field analyses of groundwater from May, 1995-August, 1996.

Date	OW #	Groundwater Elevation (ft.)	Temp.	pH	Conductivity (microsiemens)	Dissolved Oxygen
5/15/95	1	1780	63.7	6.17	1651	2.9
	2	1768	65.1	5.91	1708	2.6
	3	Dry	NA	NA	NA	NA
	4	1770	62.7	5.17	355	2.5
	5	1779	65.2	6.35	1360	1.9
	7	1777	64.4	6.21	1880	0.9
	8	1776	66.1	6.11	2010	1.1
	10	1808	69.1	6.02	2170	1.5
6/5/95	1	1778	64.3	6.2	1695	2.7
	2	1765	65.6	5.95	1805	2.2
	3	Dry	NA	NA	NA	NA
	4	1767	63.2	5.24	780	2.8
	5	1775	66.1	6.36	1410	1.7
	7	1777	65.2	6.32	1920	1.1
	8	1775	66.7	6.19	1990	1.5
	10	1808	69.6	6.07	2205	1.2
7/14/95	1	1779	65.5	6.1	1710	2.6
	2	1766	66.2	5.9	1885	2.3
	3	Dry	NA	NA	NA	NA
	4	1758	64.3	5.19	890	2.6
	5	1776	66.5	6.31	1510	1.5
	7	1775	65.9	6.29	1980	1.3
	8	1774	66.5	6.15	2010	1.5
	10	1809	69.7	6.12	2190	1.3
8/4/95	1	1779	64.9	6.12	1700	2.5
	2	1766	66.5	5.89	1845	2.1
	3	Dry	NA	NA	NA	NA
	4	1758	64.7	5.22	850	2.4
	5	1775	66.8	6.28	1490	1.5
	7	1774	66.2	6.31	1940	1.2
	8	1773	66.3	6.19	2050	1.5
	10	1809	69.6	6.2	2140	1.4
9/11/95	1	1779	60.9	6.45	1410	2.1
	2	1767	62.4	6.36	1350	1.9
	3	Dry	NA	NA	NA	NA
	4	1758	62.6	6.81	1290	2.2

Table E.1: Field analyses of groundwater from May, 1995-August, 1996.

	5	1776	64.7	6.54	1310	2.2
	7	1776	61.2	6.59	1640	1.8
	8	1775	64.4	6.58	1580	2.1
	10	1809	63.7	6.56	1710	2.3
10/11/95	1	1779	61.1	7.03	1250	4
	2	1768	63.4	6.97	1300	3.4
	3	Dry	NA	NA	NA	NA
	4	1764	58.9	6.8	404	4.5
	5	1781	62.5	7.11	1250	1.9
	7	1781	63.3	7.09	1490	2.3
	8	1780	64.9	6.9	1400	2.5
	10	1810	65.7	7	1640	4.4
11/9/95	1	1780	55.4	6.44	1205	4.2
	2	1770	56.4	5.97	1430	2.7
	3	1785	NA	NA	NA	NA
	4	1767	54.7	5.96	846	3
	5	1786	55.6	6.14	1271	2.3
	7	1786	55.5	6.54	1630	3.2
	8	1785	58	6.5	1640	1.1
	10	1810	58.1	6.16	1740	3
12/6/95	1	1780	55.1	6.39	1250	4.1
	2	NA	NA	NA	NA	NA
	3	1788	55.5	6.1	1311	2.4
	4	NA	NA	NA	NA	NA
	5	1790	55.8	6.19	1229	2.8
	7	1789	55.2	6.5	1620	3.1
	8	1788	55.4	6.09	1310	4.1
	10	1809	57.8	6.11	1690	2.6
1/10/96	1	1780	61.2	6.65	1213	4.1
	2	1774	62.3	6.36	1300	5
	3	1792	60.9	6.46	1350	4.3
	4	1770	59.2	6.37	680	3.1
	5	1793	60.3	6.63	1230	4.3
	7	1793	61.9	6.23	1540	4.5
	8	1791	62.1	6.95	1120	3.1
	9	1791	63.2	6.75	983	5.4
	10	1810	64.3	6.53	1740	4.5
2/14/96	1	1780	54.6	6.35	1344	3.8

Table E.1: Field analyses of groundwater from May, 1995-August, 1996.

	2	1779	54.8	5.69	1335	4.9
	3	1795	54.8	6.08	1340	4.1
	4	1774	54	5.81	942	2.8
	5	1796	54.4	6.26	1300	4.5
	7	1796	55	6.53	1620	4.5
	8	1795	57.3	6.8	1290	2.8
	9	1794	53.4	6.39	934	5.6
	10	1810	55.6	6.13	1790	4.7
3/19/96	1	1780	54.2	6.3	1230	3.5
	2	1776	55.1	6.03	1278	3.8
	3	1794	54.3	5.96	1309	3.9
	4	1770	54.5	5.75	814	3
	5	1795	54.8	6.15	1117	3.7
	7	1795	55.3	6.51	1510	3.9
	8	1795	57.6	6.87	1314	3.2
	9	1793	53.8	6.35	826	3.9
	10	1810	55.9	6.03	1605	4.1
4/9/96	1	1780	54.5	6.47	1390	NA
	2	1776	55.7	6.12	1460	NA
	3	1794	55.3	6.19	1350	NA
	4	1772	56.5	5.51	1280	NA
	5	1796	53.4	6.33	1211	NA
	7	1796	55.3	6.57	1660	NA
	8	1795	55.8	6.92	1310	NA
	9	1793	55.1	6.48	1060	NA
	10	1810	57.6	6.49	1740	NA
5/16/96	1	1786	63.2	6.5	1120	2.2
	2	1776	64.5	6.02	1040	2.5
	3	1795	64.9	6.21	992	1.9
	4	1760	63.1	5.83	842	2
	5	1796	65.3	6.38	920	2.8
	7	1796	65.6	6.41	1210	2.5
	8	1795	66.3	7.07	1140	2.8
	9	1794	65.9	6.31	675	3.1
	10	1810	67.8	6.15	1420	3.1
6/12/96	1	1781	64.3	6.23	975	2.3
	2	1774	65.8	6.08	1120	2.4
	3	1793	66.6	6.17	1180	2.2

Table E.1: Field analyses of groundwater from May, 1995-August, 1996.

	4	1759	63.5	5.82	761	2.1
	5	1794	66.5	6.19	987	2.5
	7	1794	65.8	6.53	1310	2.5
	8	1793	66.9	6.84	1210	2.4
	9	1792	67.1	6.35	687	2.8
	10	1810	68.3	6.01	1470	2.8
7/13/96	1	1781	59.6	6.25	1760	2.6
	2	1772	61.7	6.06	1700	2.7
	3	1791	61.5	5.96	1620	1.8
	4	1758	60.4	5.84	1105	2
	5	1793	60.8	6.32	1252	3.3
	7	1792	61.7	6.56	1810	2.8
	8	1791	62.8	6.73	1530	3.6
	9	1790	61.6	6.36	803	3.8
	10	1809	64.9	6.01	2090	3.8
8/2/96	1	1781	59.2	6.25	1770	2.8
	2	1773	62.5	5.86	643	3.3
	3	1791	64.4	5.97	1540	3.4
	4	1761	59	5.71	883	3.9
	5	1793	62.5	6.22	1430	3.1
	7	1792	62	6.57	1840	4.1
	8	1791	61.7	6.76	1530	3.9
	9	1790	61.6	6.33	969	6.6
	10	1810	66.9	6.07	2150	3.8

Appendix F:
Groundwater Sampling Procedure
Groundwater Sample Preservation
Disclaimer
Analytical Accuracy of “Aqua-Check”

Groundwater Sampling Procedure

Monthly groundwater samples were collected using a one-litre, check-valve bailer. The observation wells were not purged and samples were collected from undisturbed groundwater wells. Groundwater samples were brought to the surface using the bailer where they were transferred to a one-litre, plastic, screw-cap container.

Groundwater Sample Preservation

Groundwater samples were not acid-treated and were transferred to a cooler in the one-litre container "as is". After collecting all groundwater samples, the samples were refrigerated at the study area until they were picked up by the laboratory responsible for analyses. Samples were generally picked up within one working day after they were collected.

Disclaimer

The geochemical characteristics of the groundwater samples are considered to generally represent the groundwater quality in the study area. The interpretation of the groundwater characteristics is considered to be reasonably representative of the quality of the groundwater, considering sampling procedures. The author is not responsible for any discrepancies that might arise upon questioning of sampling procedures.

Table F.1: Aqua-check Analytical Accuracy

<u>SENSOR</u>	<u>UNIT</u>	<u>RESOLUTION</u>	<u>RANGE</u>	<u>ACCURACY</u>	<u>COMPENSATION</u>
pH:	pH	.01	1-13	.02	TEMP (AUTO)
	mV	1	± 500	± 2%	NONE
TEMP:	°C	.1°	0-50	1	NONE
	°F	.1°	32-122	1.8	NONE
COND:	MHOS	.1 umho	0-99.9	± 3% ¹	TEMP (AUTO) (20 or 25°C) OR NONE
		1 umho	0-999	± 3%	
		.01 mmho	0-9.99	± 3%	
		.1 mmho	0-99.9	± 3%	
¹ MINIMUM SIGNAL IS 10 umhos					
	SIEMANS	.1 uS	0-99.9	± 3% ²	TEMP, AUTO (20 or 25°C) OR NONE
		1 uS	0-999	± 3%	
		.01 mS	0-9.99	± 3%	
		.1 mS	0-99.9	± 3%	
² MINIMUM SIGNAL IS 10 uS					
TDS		.1 ppm	0-49.8	± 3% ³	TEMP, AUTO (20 or 25°C) OR NONE
		1 ppm	0-498	± 3%	
		.01 ppt	0-4.98	± 3%	
		.1 ppt	0-49.8	± 3%	
³ MINIMUM SIGNAL IS 5 ppm					
OXYGEN:	ppm	.1 ppm	0-20.0	± 2 TO 5% ⁴	TEMP, AUTO SALINITY, OPT.
	mg/L	.1 mg/L	0-20.0	± 2 TO 5% ⁴	TEMP, AUTO SALINITY, OPT.
	%SAT	.1 %	150.0 % sat	± 2 TO 5% ⁴	TEMP, AUTO SALINITY, OPT. PRESS, AUTO

⁴ ± 2% within 5°C of calibration temperature, ± 5% from 0 - 35°C.

Appendix G:
Water Quality Data for OW's 2, 3, 5, 7, 8 and 9

Table G.1(a): pH Values for OW-2 (left) and OW-5 (right).

Date	pH	Date	pH
J-91	6.05	J-94	5.8
F-91	NA	F-94	5.8
M-91	6.05	M-94	6
A-91	5.84	A-94	5.98
M-91	5.77	M-94	6.05
J-91	6.06	J-94	6.1
J-91	6.05	J-94	5.95
A-91	6.27	A-94	5.84
S-91	6	S-94	5.97
O-91	5.96	O-94	6.15
N-91	5.68	N-94	5.92
D-91	6.04	D-94	6.04
J-92	5.89	J-95	6.08
F-92	5.98	F-95	5.86
M-92	6.16	M-95	5.83
A-92	5.93	A-95	5.82
M-92	5.81	M-95	5.87
J-92	5.97	J-95	5.93
J-92	5.94	J-95	5.95
A-92	6.06	A-95	5.86
S-92	5.95	S-95	5.99
O-92	6.04	O-95	5.84
N-92	5.97	N-95	5.94
D-92	5.35	D-95	NA
J-93	5.9	J-96	5.39
F-93	6.04	F-96	5.77
M-93	6.07	M-96	5.97
A-93	5.83	A-96	6.1
M-93	5.79	M-96	5.94
J-93	6.15	J-96	5.78
J-93	5.75	J-96	6.16
A-93	5.79	A-96	5.98
S-93	5.58	S-96	6
O-93	5.85	O-96	6.03
N-93	6.1		
D-93	5.86		

Date	pH	Date	pH
J-92	5.55	J-95	5.98
A-92	5.9	F-95	5.88
S-92	5.97	M-95	6.02
O-92	5.81	A-95	5.98
N-92	5.74	M-95	6.1
D-92	5.84	J-95	6.01
J-93	5.9	J-95	6.13
F-93	5.99	A-95	5.99
M-93	6.7	S-95	6.03
A-93	5.82	O-95	5.84
M-93	5.85	N-95	5.67
J-93	6.08	D-95	6.18
J-93	5.87	J-96	6.17
A-93	5.97	F-96	6.16
S-93	5.91	M-96	6.09
O-93	5.87	A-96	6.37
N-93	5.99	M-96	6.37
D-93	5.96	J-96	6.1
J-94	5.77	J-96	6.29
F-94	6.25	A-96	5.88
M-94	6.42	S-96	6.35
A-94	6.23	O-96	6.34
M-94	6.25		
J-94	6.28		
J-94	6.39		
A-94	6.19		
S-94	6.3		
O-94	6.73		
N-94	5.82		
D-94	6.35		

Table G.1(b): pH Values for OW-7 (left) and OW-8 (right).

Date	pH	Date	pH
J-92	6.6	J-95	6.52
A-92	6.54	F-95	6.37
S-92	6.75	M-95	6.62
O-92	7.2	A-95	6.34
N-92	6.67	M-95	6.02
D-92	6.27	J-95	6.39
J-93	6.25	J-95	6.43
F-93	6.54	A-95	6.7
M-93	6.46	S-95	6.39
A-93	6.35	O-95	6.57
M-93	6.27	N-95	6.48
J-93	6.56	D-95	6.49
J-93	6.17	J-96	6
A-93	6.38	F-96	6.43
S-93	6.27	M-96	6.49
O-93	6.37	A-96	6.41
N-93	6.42	M-96	6.45
D-93	6.27	J-96	6.69
J-94	6.25	J-96	6.72
F-94	6.56	A-96	6.73
M-94	6.64	S-96	6.61
A-94	6.58	O-96	6.53
M-94	6.05		
J-94	6.65		
J-94	6.47		
A-94	6.42		
S-94	6.45		
O-94	6.7		
N-94	6.27		
D-94	6.56		

Date	pH	Date	pH
J-93	6.5	J-95	6.27
F-93	6.64	F-95	6.18
M-93	6.37	M-95	6.24
A-93	6.38	A-95	6.22
M-93	6.35	M-95	6.22
J-93	6.57	J-95	6.26
J-93	6.36	J-95	6.23
A-93	6.39	A-95	6.22
S-93	6.4	S-95	6.37
O-93	6.43	O-95	6.34
N-93	6.48	N-95	6.32
D-93	6.44	D-95	5.97
J-94	6.54	J-96	6.55
F-94	NA	F-96	6.78
M-94	6.43	M-96	6.8
A-94	6.57	A-96	7.08
M-94	6.41	M-96	7.05
J-94	6.66	J-96	6.72
J-94	6.31	J-96	6.82
A-94	6.29	A-96	6.57
S-94	6.31	S-96	6.84
O-94	6.43	O-96	6.87
N-94	6.22		
D-94	6.4		

Table G.2(a): Fe Concentrations for OW-2.

Date	Total Fe	Date	Total Fe
J-91	21.77	J-94	8.5
F-91	NA	F-94	4.07
M-91	27.1	M-94	11.11
A-91	34.05	A-94	1.23
M-91	28.6	M-94	3.19
J-91	32.6	J-94	5.1
J-91	18.25	J-94	0.52
A-91	16.1	A-94	4.19
S-91	25.68	S-94	3.59
O-91	NA	O-94	7.6
N-91	23.0	N-94	8.55
D-91	13.0	D-94	3.4
J-92	21.1	J-95	5.3
F-92	21.2	F-95	7.9
M-92	16.7	M-95	0.91
A-92	21.1	A-95	1.2
M-92	22.7	M-95	13.2
J-92	25.7	J-95	10.5
J-92	16.7	J-95	17
A-92	17.7	A-95	10.1
S-92	21.3	S-95	7.9
O-92	23.4	O-95	5.6
N-92	29.4	N-95	13.6
D-92	24.9	D-95	NA
J-93	20.6	J-96	7.5
F-93	32.3	F-96	4.51
M-93	20.7	M-96	4.22
A-93	27.9	A-96	0.94
M-93	17.6	M-96	5.17
J-93	7.42	J-96	0.46
J-93	19.4	J-96	1.82
A-93	20.4	A-96	4.5
S-93	13.8	S-96	0.75
O-93	18.7	O-96	3.78
N-93	21.4		
D-93	13.8		

Date	Diss. Fe	Date	Diss. Fe
J-91	7.68	J-94	4.65
F-91	NA	F-94	1.26
M-91	21.77	M-94	4.25
A-91	14.2	A-94	0.19
M-91	23	M-94	0.55
J-91	17.84	J-94	4.7
J-91	11.4	J-94	0.19
A-91	12.48	A-94	2.02
S-91	12.8	S-94	0.15
O-91	NA	O-94	5.83
N-91	21.7	N-94	3.87
D-91	0.03	D-94	1.15
J-92	12.35	J-95	1.08
F-92	16.5	F-95	0.15
M-92	1.24	M-95	0.64
A-92	0.07	A-95	0.67
M-92	16.8	M-95	6.00
J-92	18.4	J-95	8.3
J-92	16.0	J-95	12.0
A-92	15.5	A-95	3.78
S-92	0.07	S-95	0.08
O-92	0.05	O-95	0.41
N-92	11.4	N-95	5.11
D-92	17.0	D-95	NA
J-93	0.72	J-96	0.24
F-93	30	F-96	0.85
M-93	0.03	M-96	1.88
A-93	4.9	A-96	0.41
M-93	7.84	M-96	0.79
J-93	4.39	J-96	0.20
J-93	5.97	J-96	1.09
A-93	14.5	A-96	0.6
S-93	0.21	S-96	0.22
O-93	16	O-96	1.26
N-93	5.29		
D-93	2.42		

*Units in mg/l.

Table G.2(b): Fe Concentrations for OW-5.

Date	Total Fe	Date	Total Fe
J-92	0.61	J-95	3.4
A-92	1.2	F-95	4.16
S-92	2.06	M-95	0.72
O-92	4.32	A-95	0.96
N-92	3.01	M-95	8.9
D-92	7.3	J-95	15
J-93	8.2	J-95	10.9
F-93	0.94	A-95	9.4
M-93	3.94	S-95	8.3
A-93	2.04	O-95	15.8
M-93	5.39	N-95	23.2
J-93	3.53	D-95	10.7
J-93	7.32	J-96	0.9
A-93	15.5	F-96	17.4
S-93	13.1	M-96	0.94
O-93	24.9	A-96	0.75
N-93	20.5	M-96	2.1
D-93	19.3	J-96	0.32
J-94	15.9	J-96	0.7
F-94	2.13	A-96	4
M-94	2.43	S-96	5.8
A-94	8.1	O-96	13.6
M-94	6.8		
J-94	5.9		
J-94	2.19		
A-94	2.9		
S-94	1.19		
O-94	2.09		
N-94	8		
D-94	2.29		

Date	Diss. Fe	Date	Diss. Fe
J-92	0.08	J-95	0.44
A-92	0.5	F-95	2.45
S-92	0.06	M-95	0.47
O-92	1.97	A-95	0.4
N-92	1.48	M-95	5.1
D-92	4.7	J-95	13.7
J-93	0.34	J-95	8.22
F-93	0.79	A-95	8.34
M-93	0.03	S-95	0.17
A-93	0.25	O-95	0.21
M-93	2.3	N-95	19.6
J-93	1.97	D-95	8.9
J-93	6.05	J-96	0.1
A-93	12.8	F-96	16.1
S-93	9.1	M-96	0.26
O-93	15.9	A-96	0.27
N-93	18	M-96	0.29
D-93	16.5	J-96	0.19
J-94	7.73	J-96	0.46
F-94	0.49	A-96	3.17
M-94	0.15	S-96	3.95
A-94	0.19	O-96	2.44
M-94	0.51		
J-94	5.64		
J-94	0.09		
A-94	0.33		
S-94	0.17		
O-94	0.12		
N-94	4.71		
D-94	0.1		

*Units in mg/l.

Table G.2(c): Fe Concentrations for OW-7.

Date	Total Fe	Date	Total Fe
J-92	4.97	J-95	1.82
A-92	5.24	F-95	1.64
S-92	6.4	M-95	0.21
O-92	5.5	A-95	0.32
N-92	26.4	M-95	2.7
D-92	1.1	J-95	1.26
J-93	0.45	J-95	2.07
F-93	3.46	A-95	1.52
M-93	1.97	S-95	5.6
A-93	1.04	O-95	3.19
M-93	0.71	N-95	3.68
J-93	1.4	D-95	0.6
J-93	0.62	J-96	4.38
A-93	0.89	F-96	3.3
S-93	0.64	M-96	0.72
O-93	0.96	A-96	3.6
N-93	1.97	M-96	1.87
D-93	0.9	J-96	1.61
J-94	2.19	J-96	2.4
F-94	2.56	A-96	3.19
M-94	1.69	S-96	1.14
A-94	2.31	O-96	3.2
M-94	1.02		
J-94	2.02		
J-94	0.81		
A-94	1.89		
S-94	1.81		
O-94	1.33		
N-94	2.42		
D-94	0.7		

Date	Diss. Fe	Date	Diss. Fe
J-92	0.04	J-95	0.21
A-92	3	F-95	0.16
S-92	0.05	M-95	0.09
O-92	0.06	A-95	0.17
N-92	1.8	M-95	0.79
D-92	0.46	J-95	1.1
J-93	0.19	J-95	0.94
F-93	2.12	A-95	0.19
M-93	0.13	S-95	0.16
A-93	0.17	O-95	0.15
M-93	0.48	N-95	2.01
J-93	0.07	D-95	0.17
J-93	0.17	J-96	0.11
A-93	0.47	F-96	1.19
S-93	0.34	M-96	0.19
O-93	0.81	A-96	0.7
N-93	0.81	M-96	0.43
D-93	0.26	J-96	0.36
J-94	2.19	J-96	1.47
F-94	1.78	A-96	0.67
M-94	0.06	S-96	0.47
A-94	0.81	O-96	0.48
M-94	0.14		
J-94	0.25		
J-94	0.14		
A-94	0.34		
S-94	0.2		
O-94	0.17		
N-94	1.24		
D-94	0.11		

*Units in mg/l.

Table G.2(d): Fe Concentrations for OW-8.

Date	Total Fe	Date	Total Fe	Date	Total Fe	Date	Total Fe
J-93	12.4	J-94	2.45	J-95	4.5	J-96	4.4
F-93	1.64	F-94	NA	F-95	2.92	F-96	3.81
M-93	0.79	M-94	5.02	M-95	0.42	M-96	4.7
A-93	1.25	A-94	3.98	A-95	0.28	A-96	5.19
M-93	4.38	M-94	3.7	M-95	3.18	M-96	3.89
J-93	2.2	J-94	2.61	J-95	3.8	J-96	2.83
J-93	1.12	J-94	4.97	J-95	3.06	J-96	3.46
A-93	2.19	A-94	5.81	A-95	2.35	A-96	3.44
S-93	1.29	S-94	4.1	S-95	1.95	S-96	1.48
O-93	1.85	O-94	1.56	O-95	3.24	O-96	2.99
N-93	2.36	N-94	2.72	N-95	4.02		
D-93	0.48	D-94	2.29	D-95	0.59		

Date	Diss. Fe	Date	Diss. Fe	Date	Diss. Fe	Date	Diss. Fe
J-93	0.33	J-94	2.2	J-95	0.36	J-96	0.27
F-93	0.69	F-94	NA	F-95	0.77	F-96	0.39
M-93	0.05	M-94	1.98	M-95	0.17	M-96	0.39
A-93	0.06	A-94	2.17	A-95	0.1	A-96	0.39
M-93	0.8	M-94	0.36	M-95	0.8	M-96	0.16
J-93	0.06	J-94	0.5	J-95	2.99	J-96	0.28
J-93	0.51	J-94	0.09	J-95	1.02	J-96	0.42
A-93	0.98	A-94	1.55	A-95	0.31	A-96	0.29
S-93	0.41	S-94	0.78	S-95	0.1	S-96	0.51
O-93	1.62	O-94	1.33	O-95	0.1	O-96	0.58
N-93	1.28	N-94	0.97	N-95	3.11		
D-93	0.09	D-94	0.17	D-95	0.17		

*Units in mg/l.

Table G.3(a): Mn Concentrations for OW-2.

Date	Total Mn	Date	Total Mn
J-91	26.85	J-94	9.1
F-91	NA	F-94	30.4
M-91	17.91	M-94	15.9
A-91	20.25	A-94	9.5
M-91	18	M-94	12.9
J-91	20.4	J-94	13.4
J-91	14.3	J-94	13.6
A-91	17.9	A-94	11.7
S-91	16.24	S-94	13.9
O-91	13.2	O-94	13
N-91	13.7	N-94	13.7
D-91	11.13	D-94	7.1
J-92	12	J-95	0.18
F-92	14	F-95	9.7
M-92	10.1	M-95	11.6
A-92	13.8	A-95	18.3
M-92	19.7	M-95	17.9
J-92	15.9	J-95	11.4
J-92	11.5	J-95	14.5
A-92	8.7	A-95	13.4
S-92	10.7	S-95	12.7
O-92	13.7	O-95	10.4
N-92	11.1	N-95	16.6
D-92	14.5	D-95	NA
J-93	16.1	J-96	14.1
F-93	20	F-96	10.1
M-93	16.7	M-96	8.7
A-93	20.1	A-96	10.5
M-93	21	M-96	10.8
J-93	19.2	J-96	8.8
J-93	17.4	J-96	17.5
A-93	19.9	A-96	6.54
S-93	17	S-96	5.2
O-93	15.9	O-96	13.9
N-93	14.7		
D-93	12.6		

Date	Diss. Mn	Date	Diss. Mn
J-91	19.44	J-94	8.7
F-91	NA	F-94	29.1
M-91	8.08	M-94	15.1
A-91	16.4	A-94	8.9
M-91	17.6	M-94	11.9
J-91	19.6	J-94	12.3
J-91	12.45	J-94	11.2
A-91	17.76	A-94	10.1
S-91	14	S-94	12.1
O-91	12.6	O-94	11
N-91	13.1	N-94	12.7
D-91	16.36	D-94	6.5
J-92	10.1	J-95	0.12
F-92	13.6	F-95	8.9
M-92	9.9	M-95	9.8
A-92	12.7	A-95	16
M-92	19.1	M-95	14.1
J-92	14.4	J-95	9.6
J-92	10	J-95	13.7
A-92	7.2	A-95	11.4
S-92	10.58	S-95	11.1
O-92	12.8	O-95	9.1
N-92	10.8	N-95	14.7
D-92	13.9	D-95	NA
J-93	13.6	J-96	12.4
F-93	19.4	F-96	9
M-93	13.85	M-96	8.1
A-93	19.9	A-96	8.7
M-93	20.1	M-96	9.4
J-93	18.1	J-96	7.4
J-93	14.7	J-96	16.1
A-93	18.4	A-96	5.89
S-93	16.7	S-96	4.86
O-93	14.7	O-96	11.1
N-93	14		
D-93	12		

*Units in mg/l.

Table G.3(b): Mn Concentrations for OW-5.

Date	Total Mn	Date	Total Mn
J-92	9.2	J-95	0.05
A-92	7.1	F-95	6.81
S-92	9.6	M-95	3.7
O-92	9.8	A-95	6.8
N-92	7.78	M-95	7.7
D-92	9.1	J-95	7.5
J-93	7.7	J-95	6.2
F-93	5.9	A-95	9.2
M-93	7.4	S-95	7
A-93	5.5	O-95	9.1
M-93	7.03	N-95	6.8
J-93	8.8	D-95	5.5
J-93	6.39	J-96	10.3
A-93	9	F-96	7.8
S-93	11.2	M-96	7.6
O-93	13.3	A-96	9.5
N-93	11.3	M-96	7.6
D-93	12.7	J-96	4.8
J-94	9.4	J-96	4.6
F-94	8.12	A-96	7.4
M-94	8.3	S-96	7.3
A-94	NA	O-96	5.8
M-94	10.2		
J-94	9.5		
J-94	6.17		
A-94	7.3		
S-94	11		
O-94	5.17		
N-94	10.85		
D-94	3.4		

Date	Diss. Mn	Date	Diss. Mn
J-92	8.9	J-95	0.02
A-92	6.95	F-95	6.17
S-92	8.68	M-95	3.1
O-92	9.1	A-95	6
N-92	5.2	M-95	6.2
D-92	8.7	J-95	6.7
J-93	5.35	J-95	5.19
F-93	5.36	A-95	8.5
M-93	6.99	S-95	6.4
A-93	5.17	O-95	7.9
M-93	6.43	N-95	6
J-93	7.9	D-95	5.01
J-93	6.3	J-96	5.7
A-93	8	F-96	5.9
S-93	10.4	M-96	6.1
O-93	11.6	A-96	8.6
N-93	10.9	M-96	6.3
D-93	11.1	J-96	4.1
J-94	8.7	J-96	3.94
F-94	7.91	A-96	6.1
M-94	7.81	S-96	6.1
A-94	NA	O-96	5.1
M-94	9.7		
J-94	8.9		
J-94	5.97		
A-94	6.8		
S-94	10		
O-94	4.87		
N-94	10.05		
D-94	2.97		

*Units in mg/l.

Table G.3(c): Mn Concentrations for OW-7.

Date	Total Mn	Date	Total Mn
J-92	5.4	J-95	0.09
A-92	5.08	F-95	5.5
S-92	5.9	M-95	3.2
O-92	6.3	A-95	3.9
N-92	7.6	M-95	5.3
D-92	8.2	J-95	7.1
J-93	8	J-95	4.1
F-93	7.8	A-95	4
M-93	4.7	S-95	8.1
A-93	5.9	O-95	7.1
M-93	6.7	N-95	3.7
J-93	8.1	D-95	1.56
J-93	6.58	J-96	7.1
A-93	6.92	F-96	7.7
S-93	5.98	M-96	6
O-93	5.98	A-96	5.56
N-93	6.6	M-96	4.1
D-93	5.86	J-96	3.3
J-94	4.99	J-96	8.4
F-94	5.6	A-96	5.8
M-94	6.1	S-96	6.4
A-94	5.51	O-96	3.6
M-94	5.19		
J-94	6.7		
J-94	5.32		
A-94	5.01		
S-94	9.3		
O-94	4.9		
N-94	6.35		
D-94	1.8		

Date	Diss. Mn	Date	Diss. Mn
J-92	5.22	J-95	0.03
A-92	4.92	F-95	5.1
S-92	5.8	M-95	2.97
O-92	5.6	A-95	3.08
N-92	7.1	M-95	4.7
D-92	7.8	J-95	6.1
J-93	4.53	J-95	3.5
F-93	4.38	A-95	3.17
M-93	3.66	S-95	7
A-93	5.6	O-95	6.2
M-93	6.4	N-95	2.9
J-93	7.6	D-95	1.01
J-93	6.37	J-96	5.9
A-93	4.97	F-96	6.9
S-93	5.48	M-96	5.26
O-93	5.77	A-96	4.7
N-93	5.2	M-96	3.69
D-93	5.07	J-96	2.77
J-94	4.19	J-96	7.6
F-94	5.45	A-96	4.38
M-94	2.04	S-96	5.61
A-94	5.07	O-96	3.1
M-94	4.63		
J-94	5.1		
J-94	4.17		
A-94	4.86		
S-94	8.7		
O-94	4.3		
N-94	5.9		
D-94	1.41		

*Units in mg/l.

Table G.3(d): Mn Concentrations for OW-8.

Date	Total Mn	Date	Total Mn	Date	Total Mn	Date	Total Mn
J-93	4	J-94	2.01	J-95	0.05	J-96	4.59
F-93	2.32	F-94	NA	F-95	4.99	F-96	6.33
M-93	2.19	M-94	2	M-95	3	M-96	4.53
A-93	2.75	A-94	1.94	A-95	2.9	A-96	3.78
M-93	3.51	M-94	3.41	M-95	4.9	M-96	5.91
J-93	3.69	J-94	5.03	J-95	2.9	J-96	3.5
J-93	3.09	J-94	5.6	J-95	5.1	J-96	3.99
A-93	2.76	A-94	6.2	A-95	4.06	A-96	6.19
S-93	2.6	S-94	8.1	S-95	7.9	S-96	4.76
O-93	2.49	O-94	5.82	O-95	5.63	O-96	4.55
N-93	2.54	N-94	7.45	N-95	3.4		
D-93	2.2	D-94	6.7	D-95	6		

Date	Diss. Mn	Date	Diss. Mn	Date	Diss. Mn	Date	Diss. Mn
J-93	2.98	J-94	1.72	J-95	0.04	J-96	4.21
F-93	2.1	F-94	NA	F-95	4.17	F-96	5.17
M-93	2	M-94	1.76	M-95	2.79	M-96	3.6
A-93	2.6	A-94	1.34	A-95	2.7	A-96	2.41
M-93	3.4	M-94	3.24	M-95	4.1	M-96	4.17
J-93	3.52	J-94	4.5	J-95	2.4	J-96	3.04
J-93	2.72	J-94	4.01	J-95	4.27	J-96	3.22
A-93	2.41	A-94	5.9	A-95	3.43	A-96	5.29
S-93	2.45	S-94	6.9	S-95	7.1	S-96	4.26
O-93	2.17	O-94	5.6	O-95	4.19	O-96	4.17
N-93	2.46	N-94	6.8	N-95	2.99		
D-93	2.01	D-94	6.3	D-95	1.92		

*Units in mg/l.

Table G.4(a): Alkalinity for OW-2 (left) and OW-5 (right).

Date	Alkalinity	Date	Alkalinity
J-91	140	J-94	167
F-91	NA	F-94	140
M-91	147	M-94	208
A-91	140	A-94	228
M-91	146	M-94	231
J-91	152	J-94	214
J-91	118	J-94	166
A-91	216	A-94	173
S-91	163	S-94	163
O-91	128	O-94	167
N-91	112	N-94	187
D-91	124	D-94	169
J-92	155	J-95	156
F-92	146	F-95	124
M-92	125	M-95	135
A-92	116	A-95	145
M-92	107	M-95	158
J-92	100	J-95	133
J-92	107	J-95	155
A-92	123	A-95	160
S-92	109	S-95	145
O-92	116	O-95	144
N-92	121	N-95	146
D-92	119	D-95	NA
J-93	111	J-96	152
F-93	96	F-96	86
M-93	107	M-96	187
A-93	136	A-96	223
M-93	128	M-96	215
J-93	113	J-96	236
J-93	109	J-96	224
A-93	103	A-96	142
S-93	98	S-96	106
O-93	129	O-96	246
N-93	150		
D-93	149		

Date	Alkalinity	Date	Alkalinity
J-92	67	J-95	148
A-92	86	F-95	142
S-92	96	M-95	154
O-92	84	A-95	165
N-92	99	M-95	180
D-92	105	J-95	175
J-93	97	J-95	135
F-93	115	A-95	195
M-93	117	S-95	175
A-93	99	O-95	157
M-93	131	N-95	185
J-93	129	D-95	160
J-93	146	J-96	146
A-93	168	F-96	216
S-93	157	M-96	157
O-93	176	A-96	170
N-93	200	M-96	186
D-93	206	J-96	186
J-94	234	J-96	182
F-94	214	A-96	326
M-94	237	S-96	375
A-94	307	O-96	327
M-94	176		
J-94	170		
J-94	214		
A-94	175		
S-94	216		
O-94	213		
N-94	150		
D-94	182		

*Units in mg/l as CaCO₃.

Table G.4(b): Alkalinity Concentrations for OW-7 (left) and OW-8 (right).

Date	Alkalinity	Date	Alkalinity
J-92	456	J-95	425
A-92	460	F-95	439
S-92	444	M-95	439
O-92	360	A-95	439
N-92	466	M-95	446
D-92	473	J-95	420
J-93	435	J-95	460
F-93	428	A-95	410
M-93	427	S-95	473
A-93	360	O-95	394
M-93	432	N-95	460
J-93	430	D-95	405
J-93	466	J-96	420
A-93	461	F-96	350
S-93	472	M-96	410
O-93	491	A-96	469
N-93	486	M-96	425
D-93	486	J-96	466
J-94	478	J-96	454
F-94	454	A-96	439
M-94	474	S-96	395
A-94	516	O-96	463
M-94	466		
J-94	457		
J-94	481		
A-94	464		
S-94	475		
O-94	488		
N-94	426		
D-94	449		

Date	Alkalinity	Date	Alkalinity
J-93	423	J-95	420
F-93	476	F-95	434
M-93	451	M-95	425
A-93	50	A-95	424
M-93	470	M-95	433
J-93	64	J-95	420
J-93	500	J-95	415
A-93	508	A-95	415
S-93	501	S-95	423
O-93	524	O-95	387
N-93	526	N-95	446
D-93	520	D-95	394
J-94	274	J-96	340
F-94	NA	F-96	284
M-94	537	M-96	316
A-94	536	A-96	335
M-94	445	M-96	417
J-94	432	J-96	361
J-94	436	J-96	320
A-94	448	A-96	360
S-94	434	S-96	360
O-94	454	O-96	344
N-94	425		
D-94	430		

*Units in mg/l as CaCO₃.

Table G.5(a): Sulfate Concentrations for OW-2 (left) and OW-5 (right).

Date	Sulfate	Date	Sulfate
J-91	312	J-94	816
F-91	NA	F-94	653
M-91	423	M-94	702
A-91	458	A-94	745
M-91	476	M-94	572
J-91	229	J-94	699
J-91	541	J-94	512
A-91	553	A-94	592
S-91	539	S-94	395
O-91	469	O-94	350
N-91	536	N-94	550
D-91	270	D-94	680
J-92	286	J-95	350
F-92	283	F-95	750
M-92	255	M-95	600
A-92	342	A-95	680
M-92	523	M-95	600
J-92	491	J-95	NA
J-92	429	J-95	770
A-92	538	A-95	400
S-92	437	S-95	550
O-92	507	O-95	770
N-92	577	N-95	424
D-92	651	D-95	NA
J-93	705	J-96	460
F-93	738	F-96	550
M-93	849	M-96	460
A-93	860	A-96	775
M-93	865	M-96	680
J-93	928	J-96	775
J-93	581	J-96	775
A-93	576	A-96	85
S-93	900	S-96	425
O-93	802	O-96	800
N-93	844		
D-93	530		

Date	Sulfate	Date	Sulfate
J-92	9	J-95	500
A-92	73	F-95	460
S-92	112	M-95	500
O-92	119	A-95	500
N-92	14	M-95	470
D-92	144	J-95	600
J-93	101	J-95	390
F-93	102	A-95	388
M-93	163	S-95	675
A-93	77	O-95	675
M-93	176	N-95	424
J-93	117	D-95	413
J-93	121	J-96	460
A-93	266	F-96	550
S-93	407	M-96	775
O-93	399	A-96	500
N-93	569	M-96	405
D-93	477	J-96	600
J-94	528	J-96	600
F-94	292	A-96	540
M-94	325	S-96	775
A-94	548	O-96	680
M-94	702		
J-94	144		
J-94	436		
A-94	221		
S-94	275		
O-94	233		
N-94	430		
D-94	550		

*Units in mg/l.

Table G.5(b): Sulfate Concentrations for OW-7 (left) and OW-8 (right).

Date	Sulfate	Date	Sulfate
J-92	393	J-95	675
A-92	428	F-95	625
S-92	306	M-95	520
O-92	432	A-95	600
N-92	403	M-95	550
D-92	452	J-95	500
J-93	460	J-95	600
F-93	NA	A-95	162
M-93	505	S-95	550
A-93	511	O-95	600
M-93	588	N-95	680
J-93	577	D-95	680
J-93	383	J-96	500
A-93	484	F-96	550
S-93	591	M-96	925
O-93	570	A-96	775
N-93	578	M-96	680
D-93	538	J-96	800
J-94	502	J-96	670
F-94	355	A-96	810
M-94	523	S-96	800
A-94	570	O-96	800
M-94	591		
J-94	457		
J-94	136		
A-94	434		
S-94	280		
O-94	345		
N-94	500		
D-94	600		

Date	Sulfate	Date	Sulfate
J-93	44	J-95	650
F-93	352	F-95	800
M-93	553	M-95	600
A-93	445	A-95	600
M-93	584	M-95	600
J-93	454	J-95	875
J-93	439	J-95	825
A-93	423	A-95	760
S-93	547	S-95	600
O-93	497	O-95	675
N-93	530	N-95	775
D-93	426	D-95	600
J-94	486	J-96	460
F-94	NA	F-96	425
M-94	692	M-96	600
A-94	534	A-96	525
M-94	631	M-96	550
J-94	619	J-96	600
J-94	684	J-96	550
A-94	594	A-96	775
S-94	610	S-96	625
O-94	395	O-96	775
N-94	560		
D-94	800		

*Units in mg/l.

Table G.6: Total and Dissolved Mn and Fe, Alkalinity, pH and Sulfate for OW-3.

Date	Total Mn
M-94	16
A-94	14.9
M-94	14.6
D-95	4.7
J-96	14
F-96	4.3
M-96	3.6
A-96	11.4
M-96	9.5
J-96	11
J-96	11
A-96	10.3
S-96	10.3
O-96	7.8

Date	Diss. Mn
M-94	15.3
A-94	12.9
M-94	4.35
D-95	4.01
J-96	11.7
F-96	3.71
M-96	2.79
A-96	9.6
M-96	7.9
J-96	9.6
J-96	8.9
A-96	8.4
S-96	8.2
O-96	6.2

Date	Alkalinity
M-94	216
A-94	170
M-94	210
D-95	220
J-96	176
F-96	174
M-96	207
A-96	240
M-96	216
J-96	216
J-96	212
A-96	210
S-96	255
O-96	206

Date	Total Fe
M-94	18.7
A-94	14.3
M-94	NA
D-95	6
J-96	7.95
F-96	3.82
M-96	11.2
A-96	15.7
M-96	15.4
J-96	10.3
J-96	24.5
A-96	15.6
S-96	9.2
O-96	5.91

Date	Diss. Fe
M-94	0.64
A-94	11.9
M-94	NA
D-95	2.7
J-96	0.26
F-96	3.08
M-96	6.42
A-96	11.7
M-96	11.4
J-96	4.76
J-96	11.6
A-96	4.9
S-96	4.7
O-96	3.15

Date	pH
M-94	6.15
A-94	5.98
M-94	6.08
D-95	6.13
J-96	6.38
F-96	6.16
M-96	5.83
A-96	6.23
M-96	6.17
J-96	6.13
J-96	6
A-96	5.57
S-96	6.04
O-96	6.06

Date	Sulfate	Date	Sul.	Date	Sul.	Date	Sul.
M-94	572	J-96	600	M-96	800	S-96	675
A-94	452	F-96	410	J-96	775	O-96	810
M-94	229	M-96	800	J-96	785		
D-95	413	A-96	775	A-96	775		

*Units in mg/l with alkalinity in mg/l as CaCO₃.

Table G.7: Total and Dissolved Mn and Fe, Alkalinity, pH, and Sulfate for OW-9.

Date	Total Mn
F-96	2.19
M-96	4.6
A-96	3.58
M-96	5.29
J-96	4.69
J-96	4.17
A-96	6.4
S-96	3.81
O-96	4.04

Date	Diss. Mn
F-96	2
M-96	3.9
A-96	2.7
M-96	4.36
J-96	4.02
J-96	3.79
A-96	5.41
S-96	3.27
O-96	3.76

Date	Alkalinity
F-96	336
M-96	327
A-96	354
M-96	337
J-96	346
J-96	336
A-96	386
S-96	400
O-96	356

Date	Total Fe
F-96	2.67
M-96	0.68
A-96	2.11
M-96	0.38
J-96	1.69
J-96	2.32
A-96	1.4
S-96	0.2
O-96	1.73

Date	Diss. Fe
F-96	0.17
M-96	0.31
A-96	0.6
M-96	0.17
J-96	0.16
J-96	0.44
A-96	0.38
S-96	0.17
O-96	0.54

Date	pH
F-96	6.28
M-96	6.26
A-96	6.39
M-96	6.28
J-96	6.39
J-96	6.47
A-96	6.5
S-96	6.39
O-96	6.4

Date	Sulfate
F-96	176
M-96	150
A-96	188
M-96	183
J-96	137
J-96	135
A-96	200
S-96	134
O-96	155

*Units in mg/l with Alkalinity in mg/l as CaCO₃. Water Table Elevation in feet.

Vita

Charles Grant Blackburn was born on September 16, 1962 in Winston-Salem N.C. Charles graduated from Tryon High School, Tryon, N.C. in 1980. He began his college education at the College of Charleston, Charleston, S.C. in 1980. Charles took a leave of absence from the College in the fall of 1984. In the summer of 1985, while working at an antique store in Charleston, S.C., Charles met his future bride, Margaret Pendlebury. They dated for 5 years and married in May, 1990. During their time together, Charles worked in the construction industry, where he operated his own painting/drywall and carpentry business. Charles and Margaret also spent summers together in the Outer Banks of North Carolina. It was here that Charles decided that he needed to return to the College and finish his degree. In 1989, Charles re-enrolled at the College of Charleston to finish his dream of earning a degree in geology. He graduated Cum Laude in 1992. After a year of contemplating he and his wife's future, Charles and Margaret decided for Charles to pursue a Master's degree in Geology at the University of Tennessee-Knoxville. Charles will be awarded his MS degree in December, 1996. During his time in Knoxville, Charles and Margaret were blessed with the birth of a son, Charles Grant, Jr. Charles Grant is the pride of both his mother and father and for whom his father strives to provide him with every opportunity that life has to offer. Charles's hobbies include spending time with his son, playing golf and spending time with his lovely wife, Margaret.