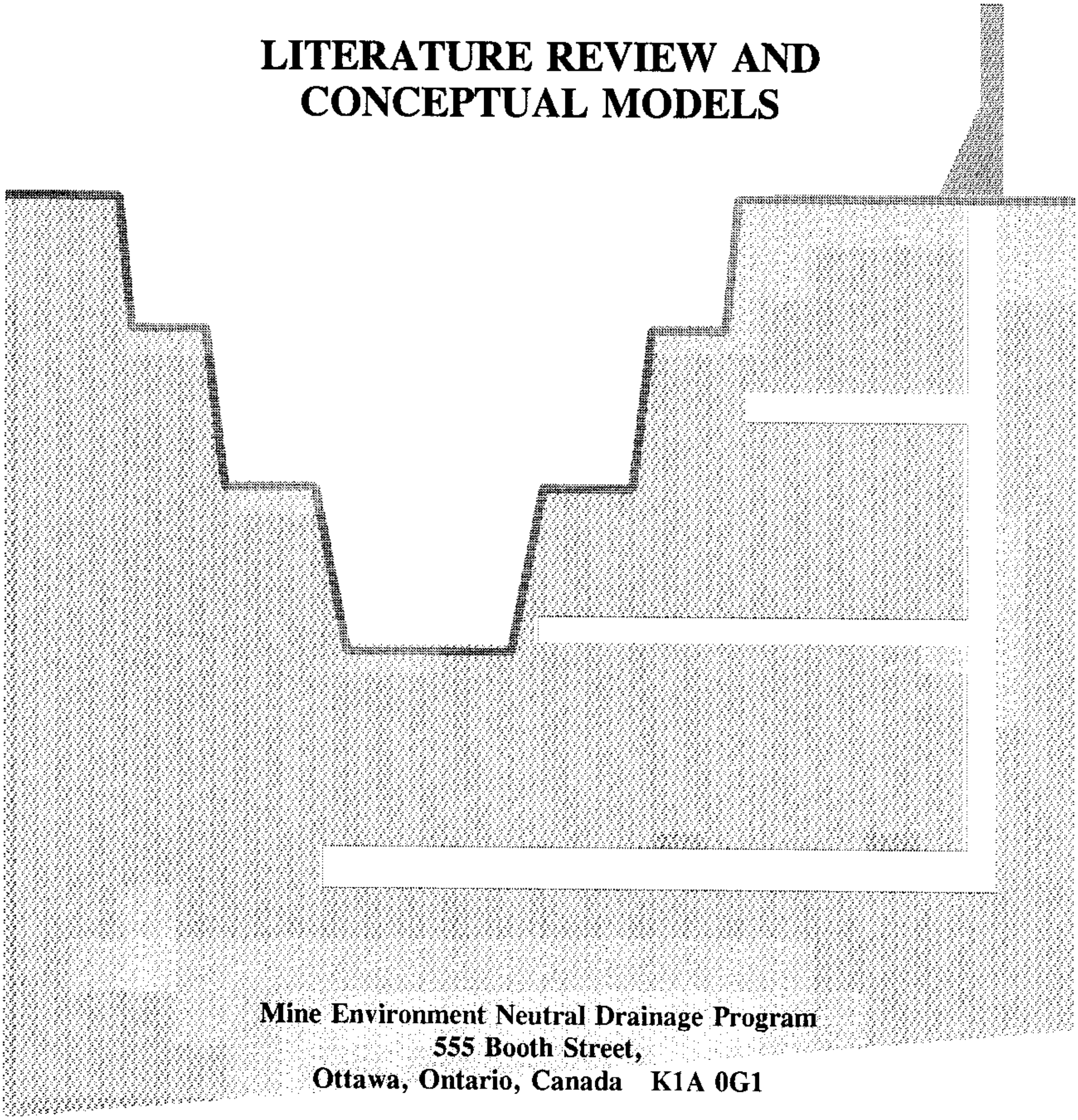
MINEWALL 2.0

LITERATURE REVIEW AND CONCEPTUAL MODELS



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1. INTRODUCTION AND OBJECTIVES

Minesites can consist of many components including the mine itself (pits and underground workings), tailings impoundments, waste-rock dumps, ore stockpiles, plant sites, building foundations, and roads. Any component can affect the chemistry of water flowing over, or through, it by various geochemical processes. These processes include the leaching of metals and nonmetals at neutral, acidic, and alkaline pH and the oxidation of sulfide minerals.

Case studies of water chemistry and geochemical processes in tailings impoundments are generally available (e.g., Jambor and Blowes, 1994). Similar studies for mine-rock piles including roads and foundations are less common, but still available (e.g., Morin et al., 1991). However, geochemical investigations of pits and underground workings ("mines") are rarer.

In order to better understand and predict water chemistry in and around mines in Canada, the Canadian Mine Environment Neutral Drainage (MEND) Program and the British Columbia Acid Mine Drainage Task Force sponsored a project now known as MINEWALL 1.0 (Morin, 1990). That study involved (1) a literature review, (2) a one-time geochemical assessment of the Main Zone Pit at Equity Silver Mines (British Columbia), (3) the development of a rudimentary site-specific computer program for predicting pit-water chemistry (MINEWALL 1.0), and (4) recommendations for conducting pit-water assessments.

MEND and the Task Force decided to expand and refine MINEWALL, leading to MINEWALL Version 2.0. This is one of four reports describing MINEWALL 2.0, which is both a simple technique for predicting water chemistry in mines and a computer program to assist with predictions for complex scenarios. MINEWALL 2.0 is based on literature reviews of relevant theory, testwork, and past studies, some over 30 years old (summarized in Morth et al., 1972). As a result, the technique and program were designed to be flexible and widely adaptable to many site-specific conditions.

The following sections of this report present data and observations gathered from published and unpublished literature. As a result, many relevant physical, geochemical, and biological principles are discussed. All of this information can be combined into generalized conceptual models that define and summarize each important factor in the simulation of open-pit and underground mines. To make the information easier to follow, the conceptual models are presented first (Sections 2 and 3), followed by illustrative and supporting studies (Sections 4 and 5). In the conceptual models, it is important to distinguish between *Operation* and *Closure* of a mine, because the MINEWALL computer program uses specific definitions for them.

MINEWALL can estimate water chemistry continuously through the *Operational* and *Closure Phases* of a mine. The Operational Phase encompasses the time from when a pit or underground working approaches a relatively stagnant size or its fullest extent to the beginning of Closure. Earlier stages of mining can be simulated by the computer program on a step-by-step basis, rather than continuously. The Closure Phase extends from the end of Operation into the future. The program can simulate up to 500 years of Operation (a mining company's greatest dream!) and Closure at one time.

In addition to this Literature Review, there are three other related reports. The first is a User's Manual for the computer program. In effect, this Literature Review describes the "why" behind the technique and code, whereas the User's Manual is limited to the "how-to" of code operation. However, for proper use of the computer program, the conceptual models in this document should be well understood. This becomes even more critical for simulations of particularly unusual minesites, at which several factors in the conceptual models must be (and can be) manipulated to fit the sites.

The second related report is the Programmer's Notes and Source Code. That document discusses some of the more technical aspects of MINEWALL's programming and contains a listing of MINEWALL 2.0's roughly 24,000 lines of code. The third related report is the Application of MINEWALL 2.0 to Three Minesites, which illustrates ways with which to simulate various conditions at a minesite.

2. PHYSICAL CONCEPTUAL MODELS

2.1 Open-Pit Mines

Based on the review of pertinent literature and applicable hydrogeologic principles, a conceptual model of surface-water and groundwater movement in and around an operating open pit can be defined (Figure 2-1). This model includes *precipitation* with subsequent *evaporation*. Also, any surface flow consists of *runoff* over the pit walls and benches, which is augmented by any seeps discharging from the pit walls. The seeps, in fact, represent an interaction of *groundwater* (discussed below) and *runoff*, and from a chemical perspective can be indicative of geochemical processes operating behind the walls (Section 3).

Any *precipitation* infiltrating into the ground becomes *unsaturated groundwater flow*. Some percentage of this unsaturated flow will eventually discharge from pit walls as seeps and thus be accounted for as *runoff*. The remainder can be expected to eventually reach the water table and join the regional *saturated flow* system which extends to the pit.

In addition to the aforementioned "inputs" of water, the important "outputs" during *Operation* (see Section 1 for definition) are usually (1) active *pumping* of water (or gravity drainage for pits not completely surrounded by walls in sloping terrain) and (2) *evaporation*. Without these factors, a typical pit would begin to fill with water.

As with any conceptual model, there is a myriad of variations. For example, Figure 2-1 shows the water table joining the pit near its base whereas Lopaschuk (1979) reported a drawdown of only 10 meters in a 90-150 meter deep pit (Section 5.1). This is easily handled by the MINEWALL 2.0 computer program, because it requires only the input of groundwater inflow, not the location of the water table. Additionally, Kipp et al. (1983) investigated pits located above the water table so that no groundwater flow from the saturated zone reported to them, but pit water was lost to the underlying unsaturated zone. MINEWALL 2.0 can easily handle this situation by using zero values for *saturated groundwater flow* and negative values

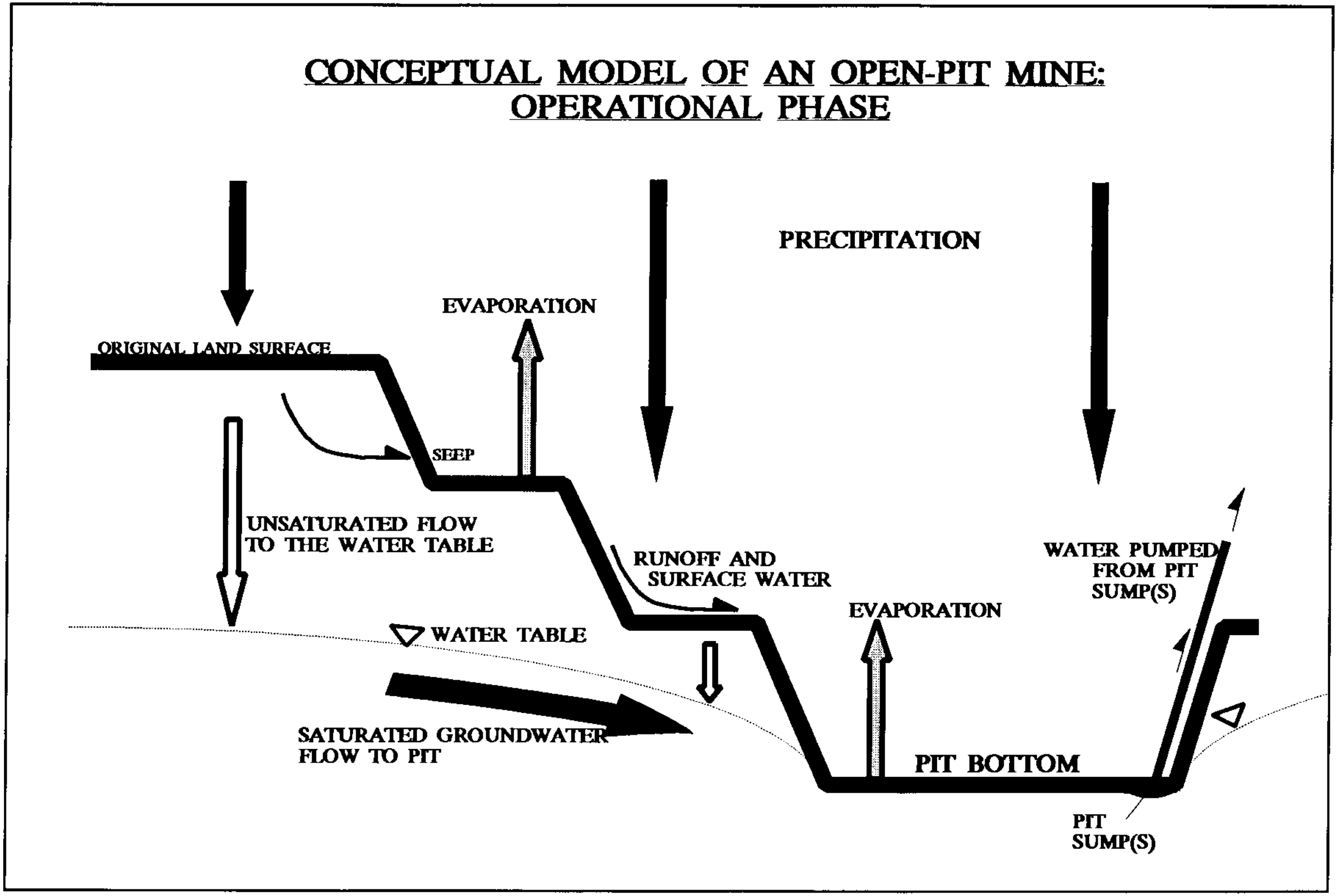


FIGURE 2-1. Conceptual MINEWALL Model of Water Movement In and Near Pit Walls During Operation.

(water lost from the pit) for *unsaturated flow*, or vice versa. Other flows can even be "re-assigned": *pit pumping* during a simulation, for example, can be used to account for evaporation of ponded water while *evaporation* accounts only for water on exposed rock. The MINEWALL 2.0 program has been coded in a flexible manner so that virtually any combination of site-specific factors can be handled and simulated.

Upon cessation of mining and initiation of decommissioning (*Closure* - see definition in Section 1), pit pumps may be turned off or gravity drainage may be halted. This causes the water level in the pit to rise out of the sump(s), cover the pit bottom, and progressively submerge the pit walls. As an example, Davis and Ashenberg (1989) reported that the water level in the large Berkeley Pit in Montana was rising 22 meters a year and that total water storage was expected to reach 5×10^8 m^3 of acidic water. Such filling of a pit can be accomplished through natural processes (Figure 2-2) or enhanced by pumping of water into the pit (Figure 2-3). The "assisted" pumping of Figure 2-3 may reflect the intent to quickly submerge the pit and any tailings/rock in it for water-quality control or may reflect the intent for long-term storage of acidic or metal-laden water until treatment can begin.

The primary difference between natural and assisted filling during Closure lies in the local direction of groundwater movement near the pit. Because the quality of water contacting the wall rock could be degraded, the direction of local groundwater movement would thus determine whether this affected water would directly enter the pit or be temporarily carried away from the pit to a temporary stagnant zone in the groundwater system (Figures 2-2 and 2-3). This temporary zone is not explicitly considered by MINEWALL 2.0.

As the input of water continues, the water level rises until a "static" level is attained (Figure 2-4). At that point, the rate of water entering the pit is generally balanced with water leaving the pit, and the water level remains relatively steady from year to year. However, "static" level is only meant in a general sense and some season-to-season and year-to-year variations can be expected. As with the Operational phase (Figure 2-1), there is likely a myriad

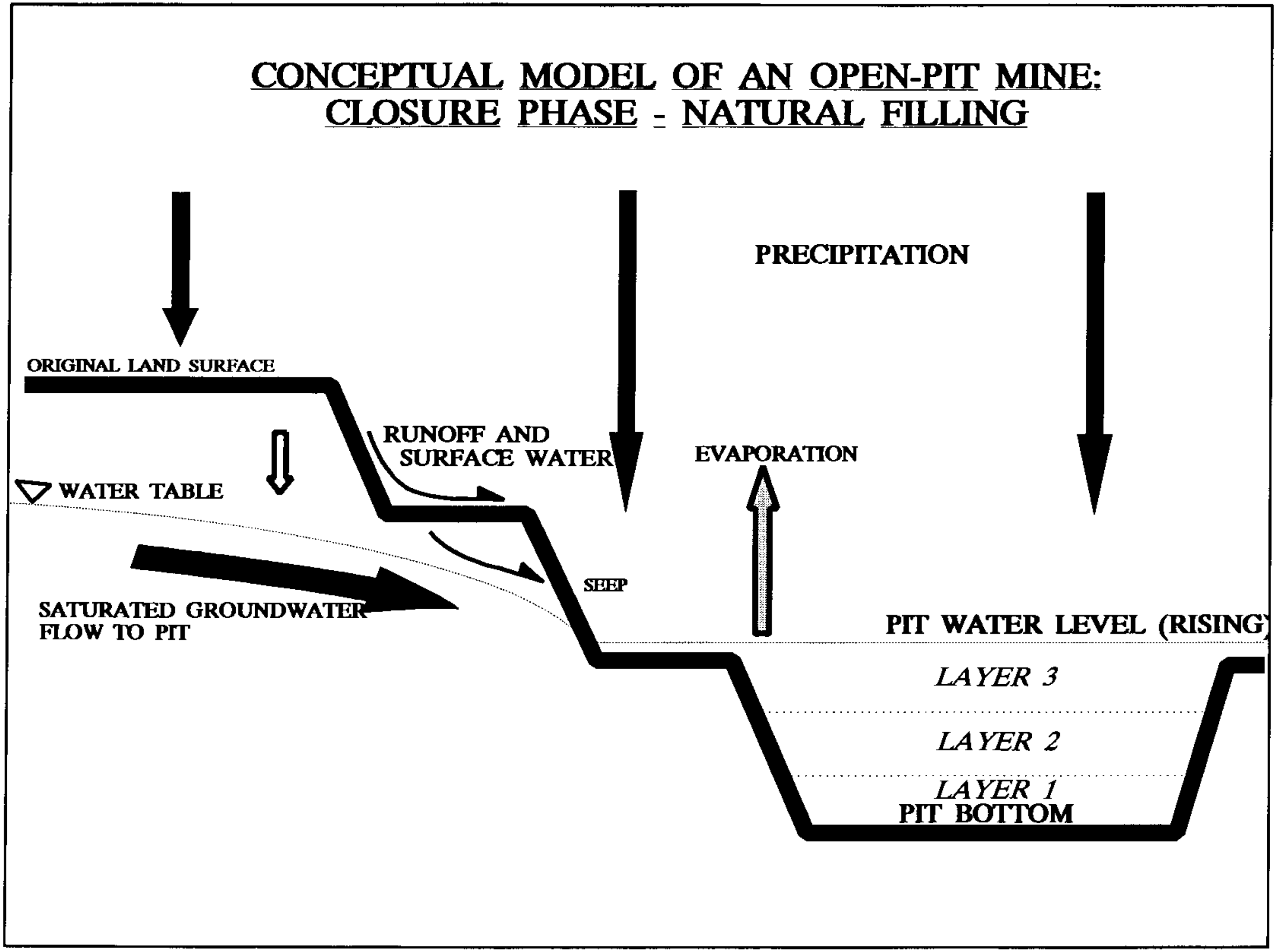


FIGURE 2-2. Conceptual MINEWALL Model of Pit Filling by Natural Processes During Closure.

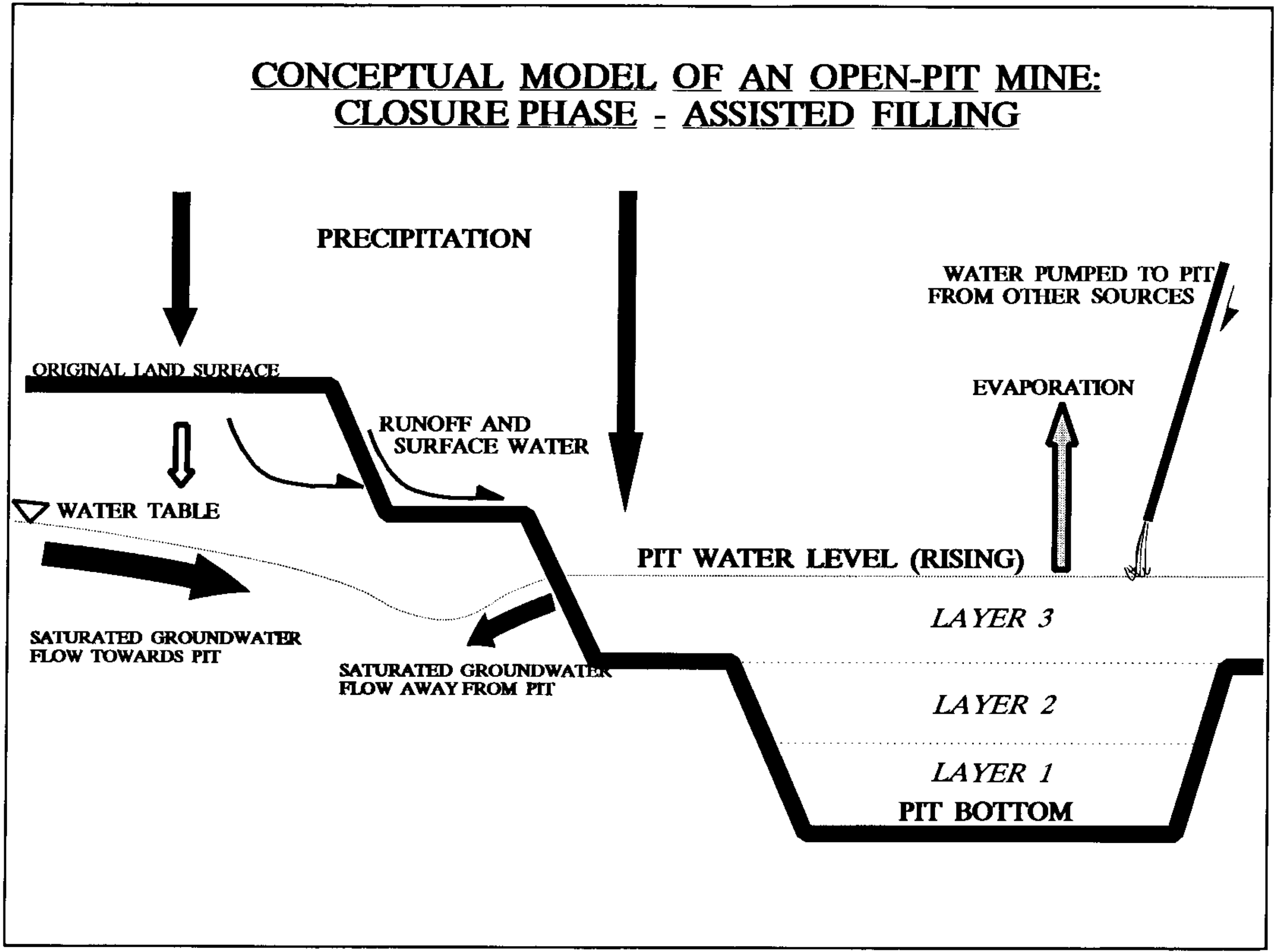


FIGURE 2-3. Conceptual MINEWALL Model of Pit Filling by Assisted Methods During Closure.

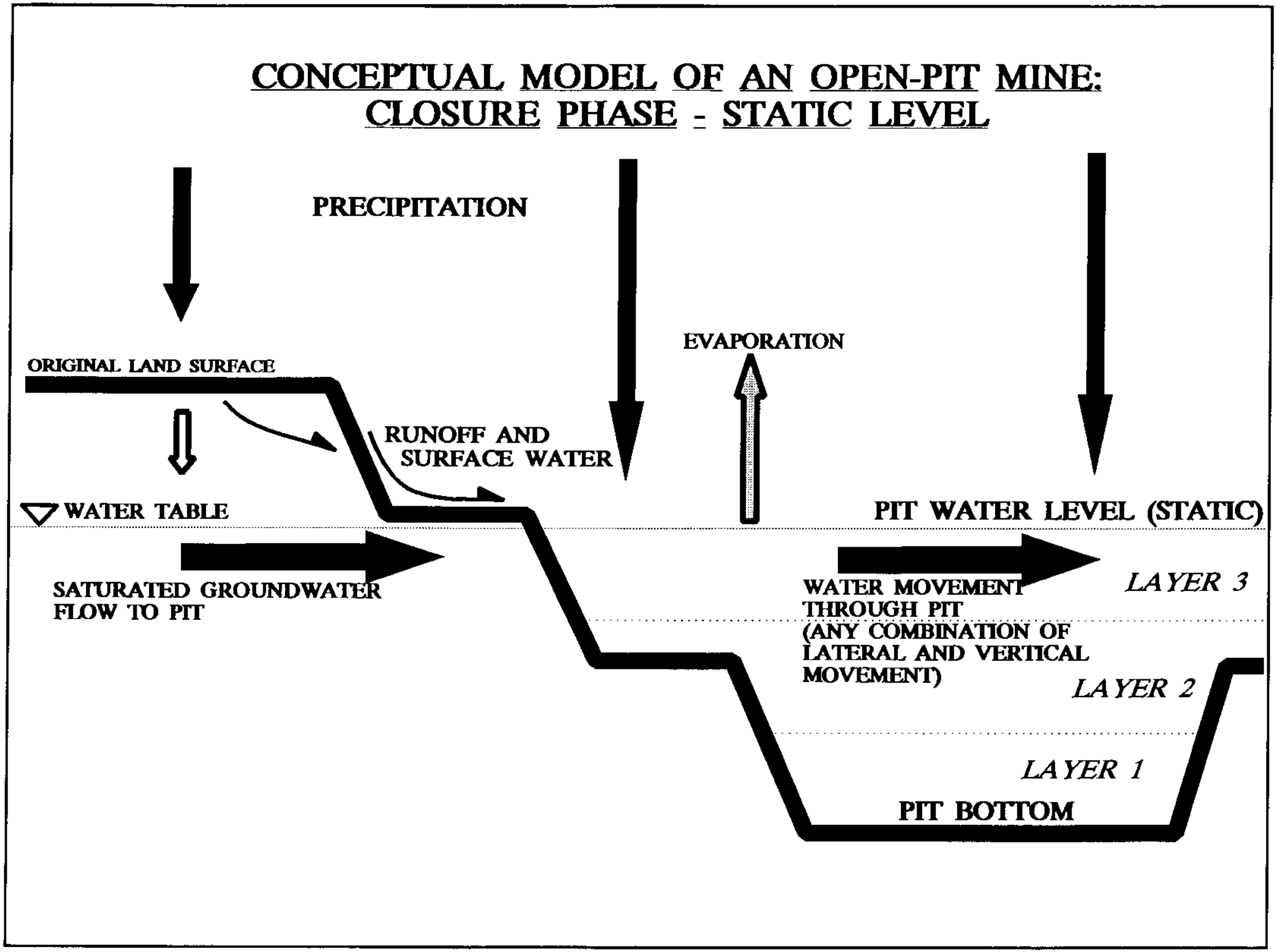


FIGURE 2-4. Conceptual MINEWALL Model of a Pit Filled to its Static Level.

of variations. For example, if pit-water chemistry is unacceptable for discharge, ongoing pumping may be required to create an artificial static water level below the equilibrium static level so that no pit water is lost to the groundwater system. The coding of the MINEWALL 2.0 program provides sufficient flexibility to handle this scenario and virtually all site-specific combinations of factors for Closure.

Although pit water can be thought of as one physical mass, the water can actually consist of several layers, divided on the basis of chemical and/or thermal characteristics. The circulation of water is sometimes limited to each layer or water may circulate freely among the layers. In fact, if the pit water experiences physical "turnover" due to thermal instability, some or all layers may occasionally mix together and thus be temporarily indistinguishable. This is discussed further in Section 3.

For the MINEWALL computer program, the flows associated with all arrows of Figures 2-1 through 2-4 must be known or estimated for proper calculations and simulations. However, due to mass-balance relationships (Section 3.1), the flow associated with one arrow can be estimated if all other flows are known. In this situation, the MINEWALL 2.0 program can "fill in the blanks" for a missing flow as explained in the User's Manual.

2.2 Underground Mines

The physical conceptual models of water movement through underground mines are both simpler and more complex than those for open-pit mines (Section 2.1). They are simpler because underground mines are not usually subjected to highly variable inputs of water, such as caused by precipitation and snowmelt. This is not to say all underground mines have steady flows. For example, mines with vertical openings to the surface or in highly fractured, shallow rock can have significant variations (Section 5). In any case, MINEWALL 2.0 can handle both steady and variable flows.

On the other hand, the conceptual models are more complex than those for open-pit mines because underground workings can be oriented in three dimensions with many hydraulic connections between drifts and levels (Section 5). In some cases, each level can act as an independent system. The MINEWALL 2.0 program handles this complexity by requiring the user to carefully design input values in light of the mine's three dimensional complexity. For example, if an underground level is free-draining through a portal and loses some water down a vertical pass, then this level can be simulated separately from the remainder of the mine. MINEWALL will then estimate the chemistry of water flowing through the portal and the pass. The underlying level of the mine can then be simulated by using the flow and chemistry from the vertical pass as an input ("*pumping*") factor. On the other hand, for levels that are all connected and drain to one shaft, then these levels and the shaft can be combined in one MINEWALL simulation.

Based on the preceding information, the physical conceptual models of water movement in underground mines can now be presented. The two basic types, or end points, of underground mines from the perspective of water movement are (1) those that have portals through which minewater can drain by gravity ("gravity-drained mines") and (2) those containing one or more vertical shafts to which lateral levels are connected and to which water drains for pumping to the surface ("pumped mines"; Figures 2-5 to 2-7). As with many physical and chemical factors, there is a myriad of variations between these endpoints.

During *Operation* (see Section 1 for definition), the source of water entering gravity-drained and pumped mines is primarily through the surrounding groundwater system (Figure 2-5). In some cases, the local water table or piezometric surface may remain high and thus saturated groundwater flow is the primary contributor. In other cases, the water table or piezometric surface may lie below, or eventually fall below, the mine and thus unsaturated flow will provide most of the inflow. Other potential sources of water include (1) surface water from access/ventilation/glory holes excavated near or to the land surface, (2) long drillholes extending to either the surface, the saturated groundwater zone, or isolated and pressurized fault/fracture systems, and (4) drainage from any backfilled material (Brealey, 1965; Emrich and Merritt,

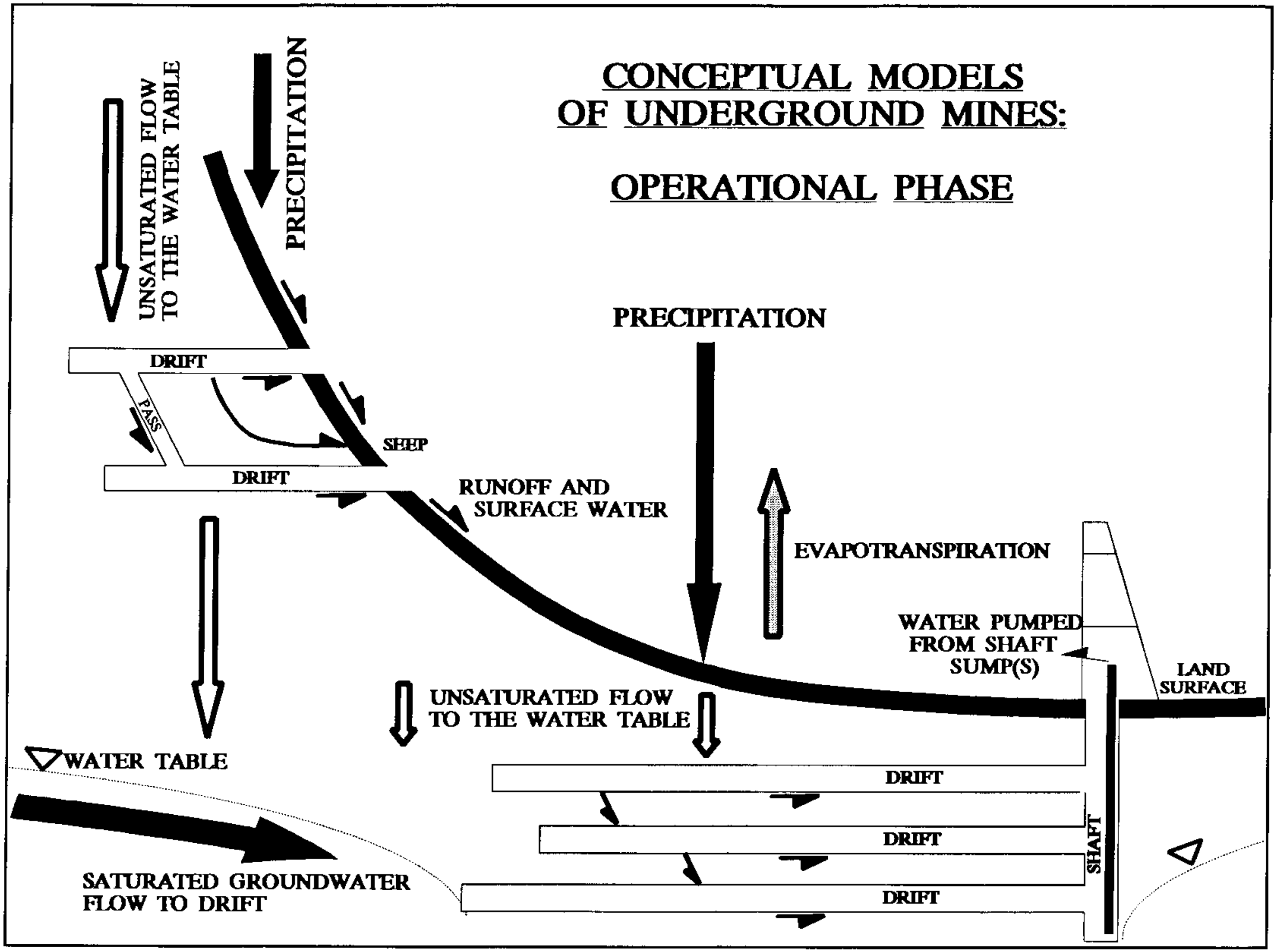


FIGURE 2-5. Conceptual Models for Underground Mines Free Draining to Surface or A Shaft.

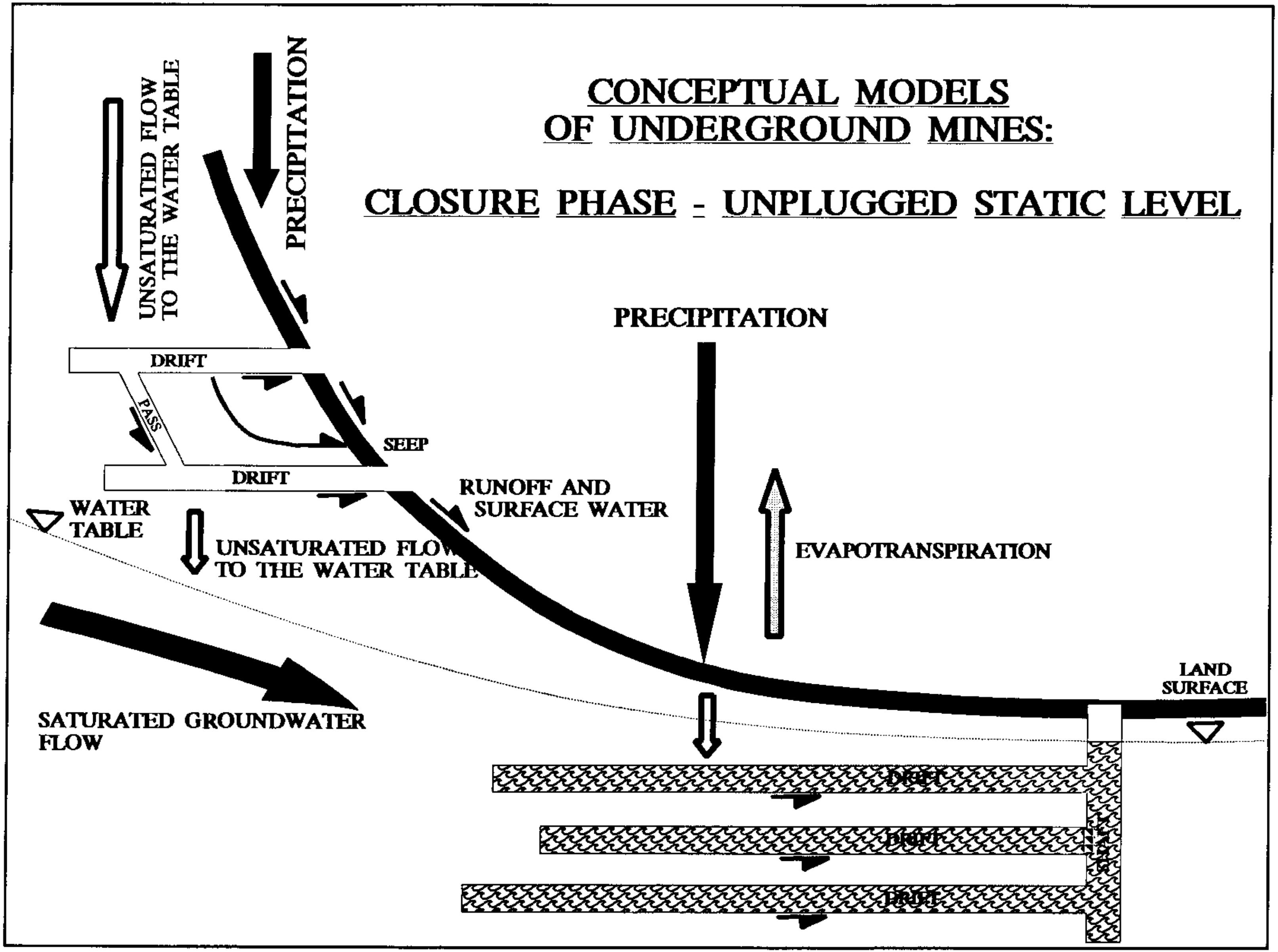


FIGURE 2-6. Conceptual Models for Underground Mines During Closure with No Portal Plugging.

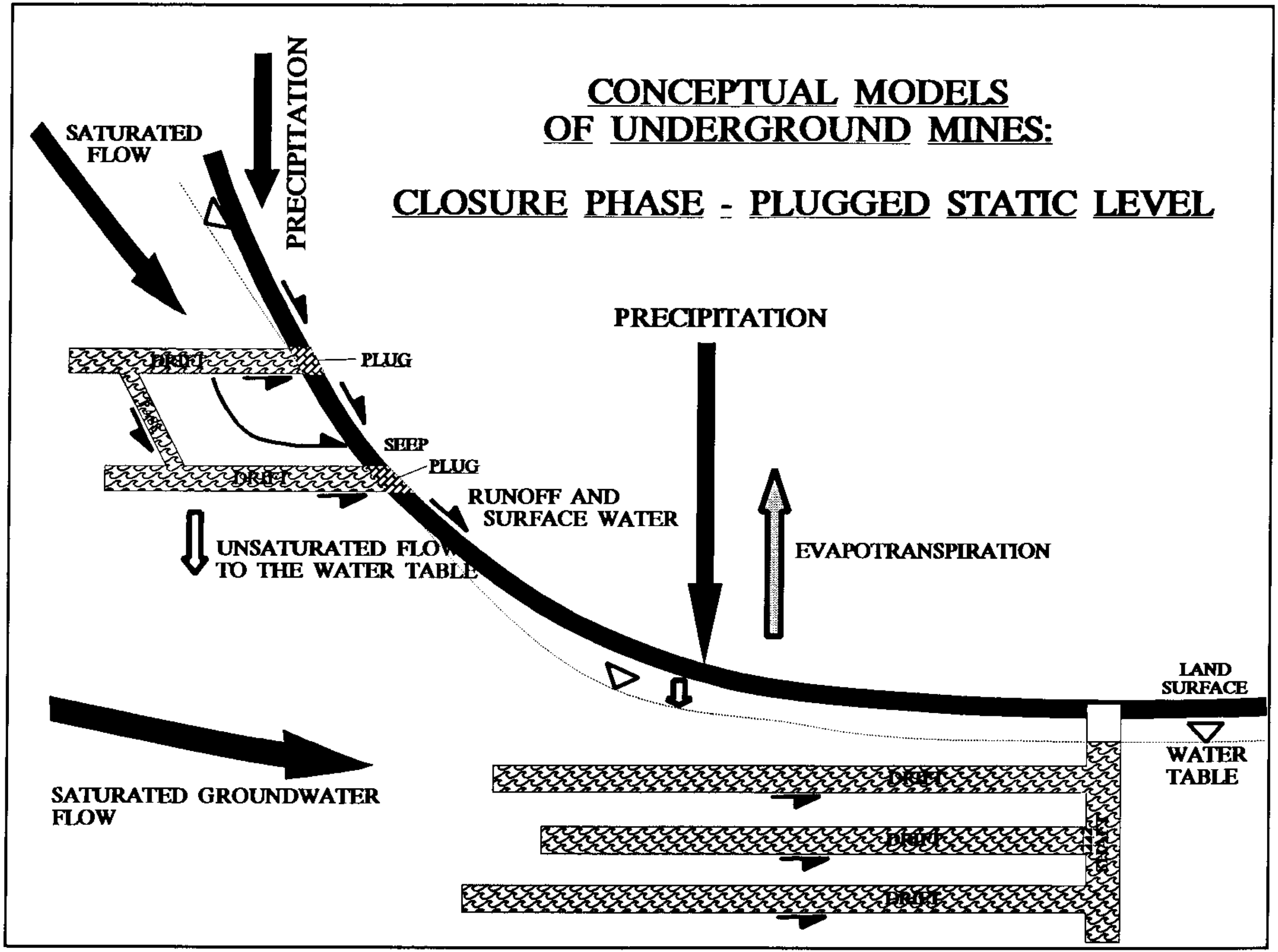


FIGURE 2-7. Conceptual Models for Underground Mines During Closure with Fully Plugged Portals.

1969; Williams et al., 1979; Trexler, 1979).

In many cases, the aforementioned classic concept of distinct and vertically stratified unsaturated and saturated groundwater zones may not apply, especially in rock masses where groundwater primarily flows through fractures (e.g., studies compiled in Hutt and Morin, 1994). In these cases, fractures not connected, or poorly connected, to the main fracture system may, for example, remain "saturated" even after the local water table has fallen due to mining. If mining activity eventually intercepts such a fracture, the groundwater will quickly drain, appearing as saturated flow, and little if any water will follow as "unsaturated" flow. Because of these and other complexities, the differentiation of flow into distinct categories like saturated and unsaturated flow may not be possible, and are not important for a MINEWALL simulation. All potential sources of flow can, in fact, be grouped into one category of flow for convenience. Potential sources can be unplugged drillholes, ventilation shafts, a pass from a higher level, and draining backfill. Similarly, all potential losses of water can be simulated as one category if desired. Potential sinks can be a pass extending to a deeper level and general loss of water into fractures on the mine floor which may then contribute to chemical loadings on a deeper level.

During *Closure* (see Section 1 for definition), pumped mines will partially or fully fill with groundwater through time (Figures 2-6 and 2-7), generally reflecting the pre-mining groundwater level modified by any surficial changes in infiltration and runoff. Gravity-drained mines will not fill unless naturally or artificially plugged. Although portal plugging has been reported to be generally successful at limiting flow from the mine and raising the internal water level, there has reportedly been less success from the chemical perspective. This is apparently because (1) the physical limitation of water movement does not necessarily correlate with limitation of reaction rates and (2) flooding of an underground mine can cause massive removal of accumulated soluble metals and nonmetals (Section 3).

3. CHEMICAL CONCEPTUAL MODELS

This section describes the conceptual models that lie behind the predictions of water chemistry in the MINEWALL 2.0 computer program. The chemistry associated with water in mines (pits and underground workings) is the result of three basic processes:

❶ the chemical load brought into a mine by the various sources of water flowing into it (discussed in Section 2),

❷ the additional load provided by the rock exposed on the mine walls and in fractures behind the walls (discussed in Section 3.2), and

❸ the geochemical evolution of the water chemistry during its residence in the mine (discussed in Section 3.3).

For MINEWALL, the first process is easily addressed simply by including chemical concentrations for each inflow of water, such as precipitation and saturated groundwater flow (Section 2). MINEWALL can accept both steady-state and variable chemistry for every inflow and, if desired, for outflow such as pumping of a selected water source in the mine.

The second and third processes involve various physical, geochemical, and biological mechanisms. These mechanisms and their influences on minewater chemistry are discussed in following subsections. However, the basic mass-balance equations that lie behind the three processes will first be presented.

3.1 Chemical Mass-Balance Equations

During Operation, the prediction of concentrations in the drained or pumped minewater is relatively easy using the units of milligram (mg), liters (L), and weeks (wk):

$$\mathrm{Conc}_m = \{[\mathrm{Flow}_1*\mathrm{Conc}_1] + [\mathrm{Flow}_2*\mathrm{Conc}_2] + \ldots + [\mathrm{Flow}_x*\mathrm{Conc}_x] + \mathrm{MW}_1\} / \{\mathrm{Flow}_1 + \mathrm{Flow}_2 + \ldots + \mathrm{Flow}_x\} \quad (3\text{-}1)$$

where Conc_m = concentration in pumped/drained mine water (mg/L);

Flow_x = positive (inflow) or negative (outflow) flowrate such as precipitation or groundwater (L/wk);
Conc_x = concentration associated with Flow_x (mg/L); and
MW_1 = loading (mg/wk) from occasional flushing of mine walls (Section 3.2).

A negative value for Conc_m in Equation 3-1 indicates there is a net loss of water from the mine.

During Closure, the prediction of concentrations in the water filling the mine is more difficult due to the accumulation of mine water, additional contributions from the mine walls, and the potential for flow reversals. With units of mg, L, and time steps of one week, the general equation for predicting the concentrations in minewater during Closure is:

$$\text{Conc}_{m,t} = \{[\text{Conc}_{m,t-1}*\text{Evol}_{m,t-1}*\text{Volume}_{m,t-1}] + [\text{Flow}_1*\text{Conc}_1] + [\text{Flow}_2*\text{Conc}_2] + \ldots + {} + [\text{Flow}_x*\text{Conc}_x] + \text{MW}_1 + \text{MW}_2 + \text{MW}_3\} \,/\, \text{Volume}_{m,t} \qquad (3\text{-}2)$$

$$\text{and } \text{Volume}_{m,t} = \text{Volume}_{m,t-1} + \text{Flow}_1 + \text{Flow}_2 + \ldots + \text{Flow}_2 \qquad (3\text{-}3)$$

where $\text{Conc}_{m,t}$ = Closure concentration (mg/L) in ponded minewater at current week, t
$\text{Conc}_{m,t-1}$ = Closure concentration (mg/L) in ponded minewater at previous week, t-1
$\text{Evol}_{m,t-1}$ = factor representing evolution of minewater chemistry between times t-1 and t (see Section 3.3)
$\text{Volume}_{m,t}$ = volume of ponded mine water at current week (L)
$\text{Volume}_{m,t-1}$ = volume of ponded mine water from previous week (L)
MW_1 = loading (mg/wk) from occasional flushing of mine walls (Section 3.2)
MW_2 = loading (mg/wk) from major flushing of recently submerged mine walls (Section 3.2)
MW_3 = loading (mg/wk) from previously submerged mine walls (Section 3.2).

All Flow_x and Conc_x in Equations 3-1 to 3-3 must be known for predictions of minewater chemistry. Most of this required information on flows and concentrations, such as for precipitation and groundwater, often comes from routine monitoring of proposed, operating, and closed mines. Also, concentrations in precipitation and evaporation are often negligible and can thus be ignored. Furthermore, concentrations in runoff and unsaturated flow can be set to zero and their contributions can be more easily estimated as part of MW_x. As a result, the most critical Conc_x is often for groundwater flow, which can come from monitor wells at the minesite. Therefore, most of the data requirements for MINEWALL, except the MW_x factors (Section

3.2) and $Evol_{m,t-1}$ (Section 3.3), can be relatively easily obtained or estimated. Because Operating mines often monitor flow and chemistry ($Conc_m$) in mine drainage/pumpage, Equation 3-1 can sometimes be solved in reverse to obtain an unknown $Conc_x$ or $Flow_x$.

Again, the primary unknown factors in Equations 3-1 to 3-3 are often the MW_x factors, which reflect loadings from the rock surfaces on and behind the mine walls. These loadings are considered to be *release rates* into the minewater, such as 300 mg Ca/week. These rates are determined by three factors which are discussed in Section 3.2: (1) unit-surface-area production rates, (2) total reactive rock-surface area in a mine, and (3) reaction-product retention and flushing. In other words, for one week of time:

Release rate (MW_x, mg/wk) = {[Production rate (mg/m^2 surface/wk)*Total rock surface (m^2)] + [Previously retained products (mg)]} * {%Flushed from surface/100%} (3-4)

3.2 Effect of Mine Walls on Water Chemistry

Since mine walls are composed of rock, they may contain minerals capable of releasing metals and nonmetals to water passing over their surfaces. Geochemical processes such as metal leaching and acid generation can occur on the surfaces, or outer boundaries, of rock particles and fracture planes, as well as within grains or blocks of rock. Consequently, there are two localities where the geochemical reactions take place: *surficial* and *internal* relative to the smallest divisible particle or block at a site (Figure 3-1). The overall effects of surficial reactions are generally a function of exposed surface area relative to the water volume, whereas the effects of internal reactions are functions of exposed surface area, distance to the center of a grain, and the volume of the grain relative to the water volume. Perhaps due to practicality and simplicity, surficial reactions seem to be more often addressed in field-oriented studies (e.g., Morth et al., 1972), while internal reactions appear more often in theory-oriented studies (e.g., Cathles, 1982).

In "soft-rock" mines such as bituminous coal, the rock is typically composed of indurated mineral grains which can have elevated grain-surface areas per unit weight or rock (Figure 3-1).

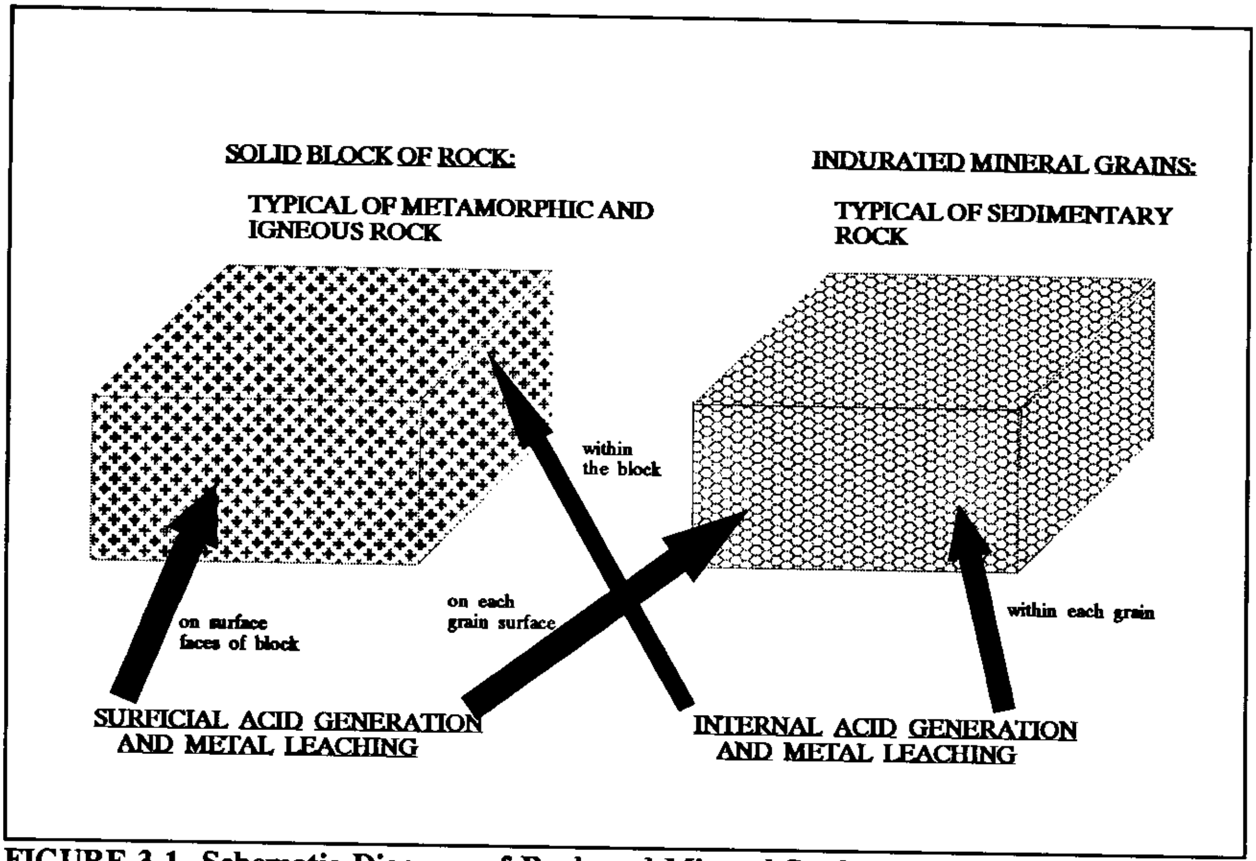


FIGURE 3-1. Schematic Diagram of Rock and Mineral Surfaces in Various Rock Types.

In "hard-rock" mines such as silicified epithermal copper, the fracture surfaces along which the rock has broken into blocks will provide the exposed area for minerals. This area will often be less per unit weight than for soft rock. Therefore, soft-rock mines may have a greater impact on their water chemistry per unit volume of material. However, there are other factors that can affect this ranking as discussed below.

For the MINEWALL studies to date, only hard-rock mines have been examined and these mines typically have relatively large blocks of rock separated by fractures. Consequently, the current work focusses on surficial, rather than internal, reactions. These surficial reactions can be addressed simply by isolating and monitoring portions of rock surfaces such as with MINEWALL Stations (Section 4).

A primary objective of MINEWALL monitoring and testwork (Section 4) is to obtain unit-area reaction rates (Equation 3-4), such as 1.5 mg Cu/m^2 of rock surface/wk (1.5 mg Cu m^{-2} wk^{-1}). A second objective is to describe the manner in which the reactions rates change with time. Literature on acidic drainage contains many reports on theoretical and empirical trends of rates through time (e.g., Ferguson and Morin, 1991, contains a summary). On the basis of elapsed time (e.g., DAYS), the variation in normalized rates (rate at DAY 0 = 1.0) can often be described through equations of the form: DAYS^{FACTOR}, $10^{(FACTORS*DAYS)}$, and 1/log$_{10}$(FACTOR*DAYS). Examples using various values for FACTOR are portrayed in Figures 3-2 to 3-4.

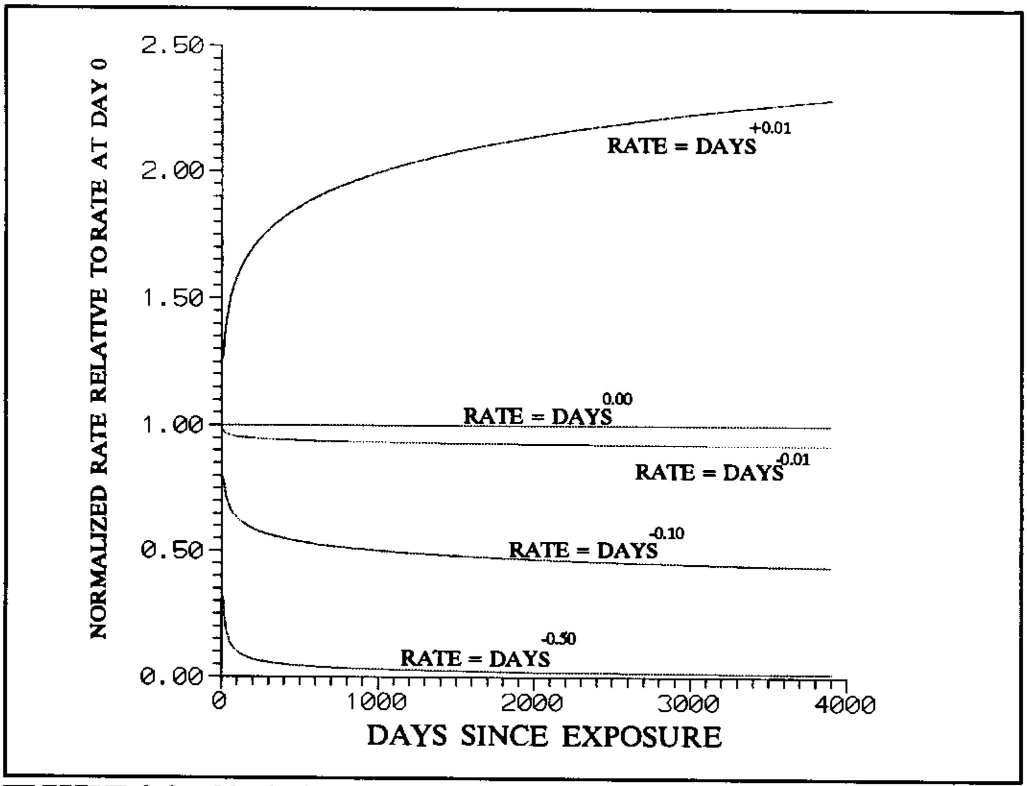


FIGURE 3-2. Variation of Reaction Rates With Time as TIME^{FACTOR}.

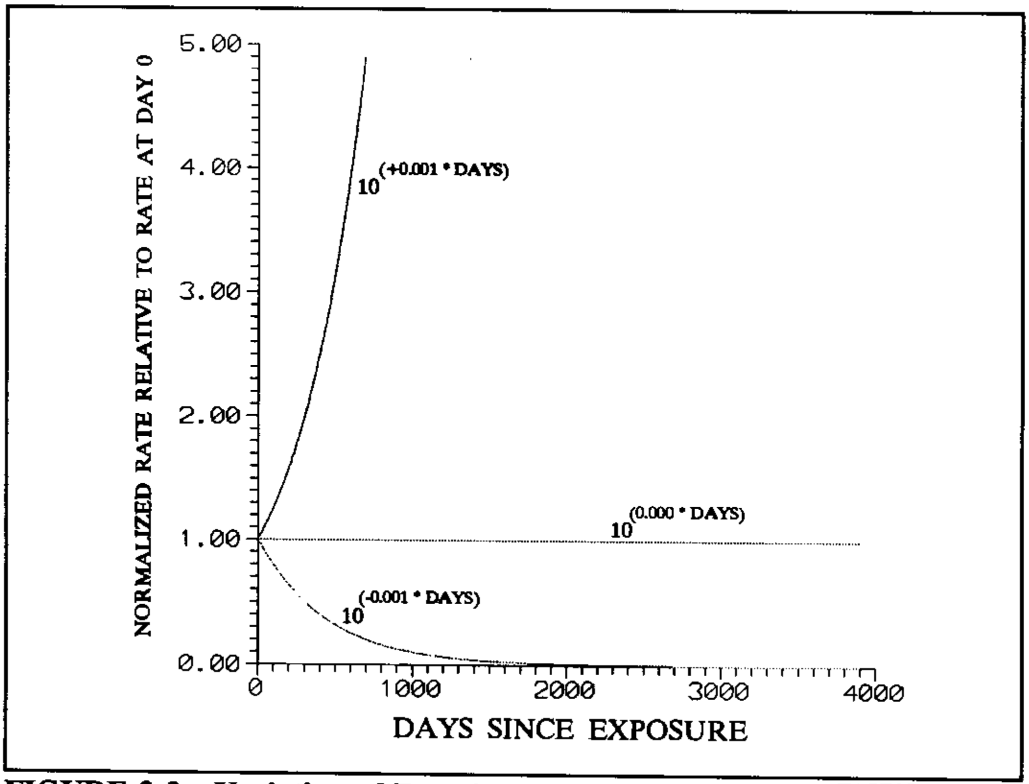


FIGURE 3-3. Variation of Reaction Rate with Time as ($10^{(FACTOR * TIME)}$).

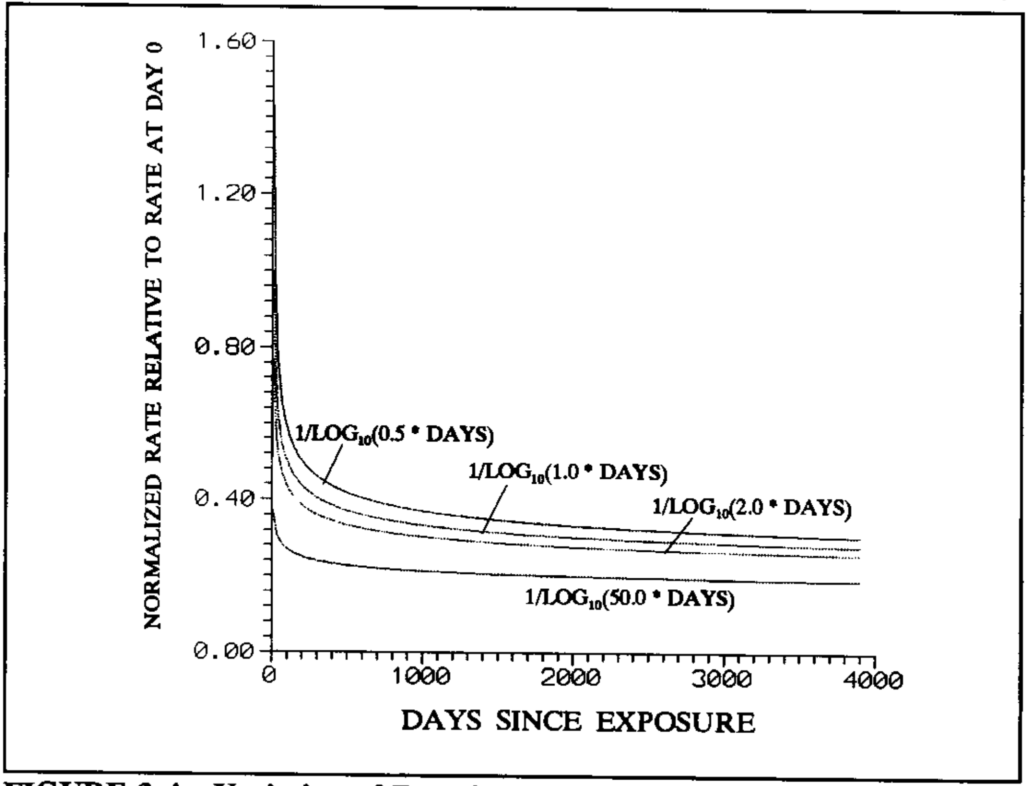


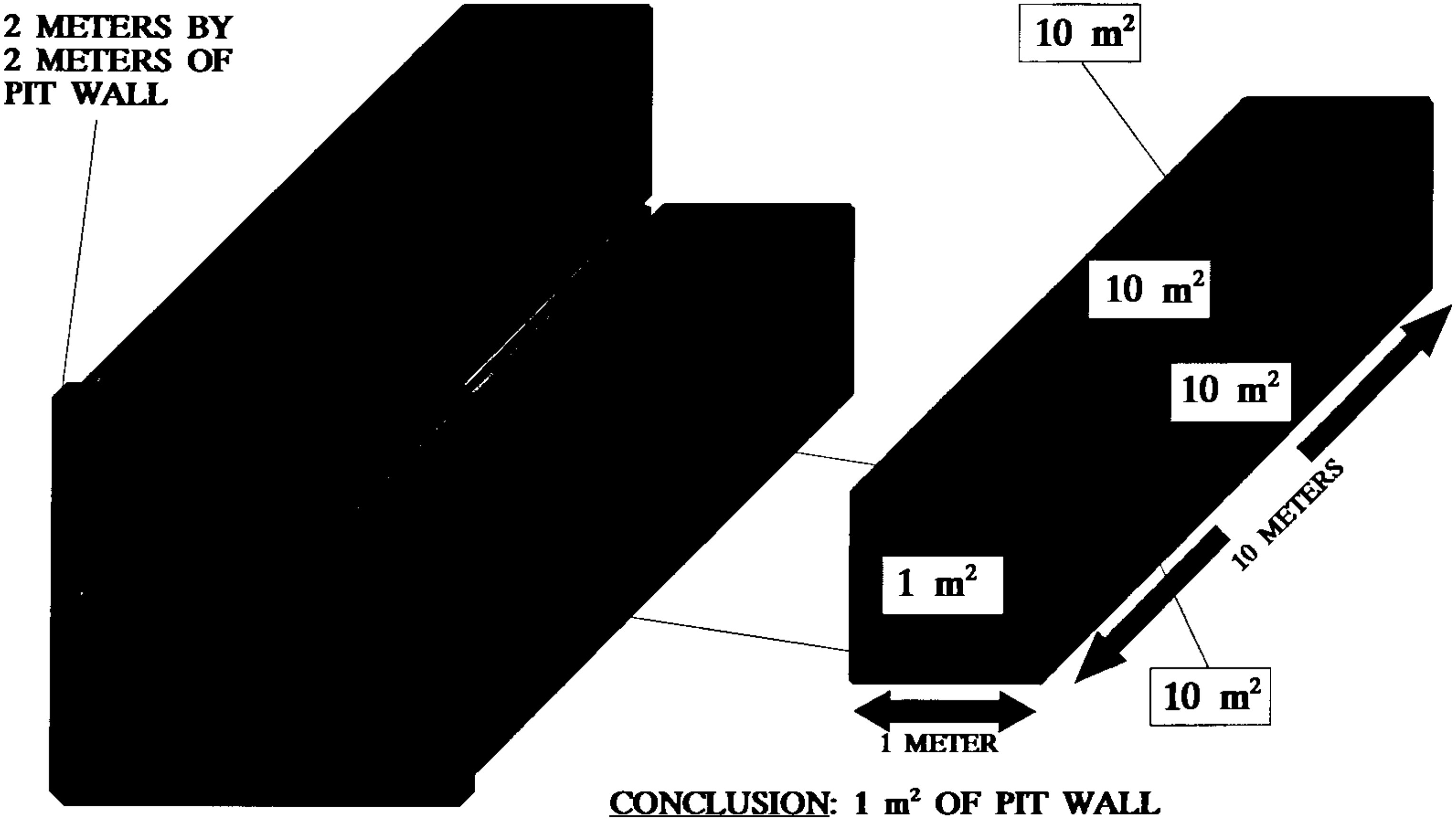
FIGURE 3-4. Variation of Reaction Rate with Time as $1/\log_{10}$(FACTOR * TIME)

After geochemical reaction rates are obtained for unit surface areas, the next major issue in MINEWALL becomes the amount of rock surface that is reactive (Equation 3-4). A first impression might be that the total reactive surface is equal to the exposed walls of a mine. However, the fractures that are invariably present in mine walls naturally and induced by blasting and excavation (Evans, 1987; Pusch, 1989; Toran and Bradbury, 1988) provide additional reactive surfaces. Morth et al. (1972), for example, found reactive fracture surfaces that extended as far as 15 meters from mine walls. Also, visual observations at Island Copper Mine in British Columbia (Section 4.3) during a pushback of a pit wall revealed fractures oxidized as far back as 10 meters from the wall. Consequently, the fracture surfaces behind the mine walls can provide a much greater reactive surface than that indicated simply by the exposed walls. As a numerical example, a pit wall that has (1) spacings for vertical and horizontal fractures of 1 meter and (2) oxidation occurring to 10 meters behind the wall will have 41 m^2 of reactive surface for each m^2 of exposed wall (41:1; Figure 3-5). For the MINEWALL field studies (Section 4), the average estimated ratio varied from 27:1 to 161:1. Any waste rock, ore rock, tailings, or backfill placed in the mine can add to this reactive surface area and is handled in the MINEWALL program by labelling it as one or more additional "rock units".

For rock with on-going sulfide oxidation, acid generation, and metal leaching (Figures 3-2 to 3-4), the products of these reactions usually remain in a relatively soluble form on the rock surfaces. These products will only be removed from the surfaces if water contacts and dissolves them. This leads to the third and last factor in Equation 3-4 for estimating MW_x (Equations 3-1 and 3-2).

For ease of calculations, MINEWALL assumes that during Operation of an open-pit mine (Figure 2-1) certain percentages of the total reactive rock surfaces are flushed (1) regularly by precipitation, condensation, etc., (2) periodically such as during snowmelt or a high-precipitation month, and (3) not at all during Operation. During Closure (Figures 2-2 and 2-3), the latter, third category can have a significant effect on water chemistry as pit-water levels rise and flush reaction products accumulated over decades from flooded surfaces. Nevertheless, the first two categories continue to play a role in pit-water chemistry if any pit walls remain exposed to air.

EXAMPLE OF THE EFFECT OF FRACTURE PLANES ON REACTIVE SURFACE AREAS PER UNIT AREA OF MINE WALL



CONCLUSION: 1 m^2 OF PIT WALL
REPRESENTS 41 m^2 OF REACTIVE SURFACE

FIGURE 3-5. Schematic Effect of Fracture Planes on Reactive Surface Area Per Unit Area of Mine Wall.

For underground mines, the removal of reaction products is simulated the same as for pits. To reconcile this approach with that of Morth et al. (1972), their terminology will be discussed in light of MINEWALL's terminology. Morth et al. defined three types of reaction-product removal, involving *diffuse leaching*, *trickle leaching*, and *inundation leaching*. These three processes are direct parallels to the flushing of rock surfaces regularly, periodically, and not at all during Operation. *Diffuse leaching* is synonymous with the "condensation leaching" defined by Morth et al., but condensation is not necessarily the only cause for a slow, non-point-source leaching of reaction products into minewater. *Trickle leaching* is a point-source removal of reaction products, such as at a borehole or fracture through which a stream of water flows into the mine. *Inundation leaching* implies full saturation and submergence of fractures, a portion of the mine, or even the entire mine. Inundation leaching in an underground mine on a large scale is usually prevented during Operation through active water control such as pumping, grouting, and bulkheads. However, during Closure, inundation leaching will often prevail in a pumped mine (Section 2.2) when pumping ceases. Such leaching may occur in a gravity-drained mine only when a plug is installed or wall collapse creates a natural plug.

3.3 Geochemical Evolution of Water Chemistry Within a Mine

Once the chemical loadings from the inflows and mine walls have been added to the water, the water chemistry may begin to evolve due to various physical, chemical, and biologic mechanisms (greatly simplified as $\mathrm{Evol}_{m,t-1}$ in Equation 3-2). During Operation, this evolution is not considered by the MINEWALL program, because the residence time in the mine is relatively short before it is pumped or drained. However, the accumulation of water in an open-pit or underground mine during Closure can provide sufficient residence time for geochemical evolution to become important. For example, mineral precipitation/dissolution, controlled by equilibrium or kinetic processes, can change concentrations in ponded water. Also, redox reactions can change concentrations of some aqueous species through transformation to other ions. Sulfate-reducing bacteria, for example, can reduce sulfur from sulfate (SO_4^{2-}) to sulfide (S^{2-}). MINEWALL 2.0 does not attempt to distinguish among all the possible reactions and transformations, but provides an empirical approach for simulating their time dependent effects.

In addition to reactions that can affect water chemistry throughout an entire water column, there are also physical and chemical mechanisms that lead to chemical differences along a water column (e.g., Figures 2-2 to 2-4). It is not the purpose of this document to be a primer on limnology, but flooded pits and underground mines, as parallels to lakes, can become complex physically and chemically in three dimensions. This complexity is often reduced to two dimensions in a vertical plane in which horizontal masses of water are separated by near-horizontal contact zones (Wetzel, 1983). For example, "thermoclines" separate water layers with different temperature and "chemoclines" separate those with different chemistries. Due to unstable thermal stratification that can arise over an annual period, some or all of the water layers mix vertically ("turnover") and the thermocline and chemoclines essentially disappear (Stevens et al., 1994). MINEWALL 2.0 does not simulate stratification due to memory limitations, but its modular code is designed to allow the addition of stratification at a later time.

When one or more of the deeper layers do not participate in turnover at any time, the minewater is termed "meromictic". The concept of a meromictic pit with sulfate-reducing bacteria (SRB) in the deepest, unmixed "monolimnion" has become a target of research in Canada (e.g., Horne, 1993). The objective of this research is to determine if fresh minewater can be fed to the monolimnion without destabilizing it, allowing the sulfide generated by bacteria to precipitate metals from the minewater.

4. MINEWALL-BASED STUDIES

As part of MINEWALL 2.0 development, field studies were conducted at Noranda Mineral's Bell Mine and BHP Canada's Island Copper Mine, two open-pit mines in British Columbia. Also, the Main Zone Pit at Equity Silver Mines in British Columbia was characterized as part of MINEWALL 1.0 studies (Morin, 1990). These three mines form the MINEWALL field database at this time and are discussed in greater detail in the related document, Application of MINEWALL 2.0 to Three Minesites.

4.1 Equity Silver Mines

Equity Silver Mines, now closed, is located in central British Columbia and consisted of several open pits, waste-rock dumps, and a tailings impoundment (Equity Silver Mines, 1989; Patterson, 1990). The Main Zone Pit was the target of MINEWALL 1.0 studies (Morin, 1990).

At Equity Silver's Main Zone Pit, four MINEWALL Stations were established with an additional Station located in the adjacent Southern Tail Pit in order to define unit-area reaction rates. These stations were rinsed only once, so variations of reaction rates through time remain unknown. Based on the single rinse event, unit-area reaction rates were calculated (Table 4-1) from chemical analysis and volume of the rinse water, isolated area of each station, and the elapsed time since last rinse (rainfall). Values of other relevant parameters were based on measured data or estimated from related data (Morin, 1990) and some values are reproduced in Table 4-2 to illustrate the data required for a MINEWALL simulation. This case study is discussed further in the companion MINEWALL 2.0 report, Application of MINEWALL 2.0 to Three Minesites, but some notable aspects of it are discussed below.

The major parameter that could not be estimated or measured was the percentages of rock surfaces that were flushed regularly, occasionally, or not during Operation (Section 3). Consequently, these percentages were obtained by calibrating MINEWALL 1.0 to Operational

TABLE 4-1
Unit-Area Production Rates from the MINEWALL Stations at Equity Silver Mines[1]
(rates in $mg/m^2/wk$)

Parameter[4]	---------- Station ----------			Average of Nos. 1,4,5	Station No. 3[2]	Station No. 2[3]
	No. 1	No. 4	No. 5			
Sulfate	49.9	48.3	133	77.1	539	204
Acidity	54.1	15.9	16.7	28.9	3.63	10.2
Alkalinity	26.4	6.99	73.3	35.6	24.0	8.82
Nitrate	0.330	0.111	0.833	0.425	3.18	0.801
Field pH	4.9	5.0	6.0	5.3	6.1	5.2
Aluminum[5]	0.132	0.0731	0.0722	0.0924	0.0291	0.0131
Antimony	0.0144	0.0264	0.0122	0.0177	0.0109	0.0005
Arsenic	0.00180	0.000318	0.00111	0.00108	0.000182	0.000219
Cadmium	0.00421	0.0226	0.00611	0.0110	0.0327	0.000947
Copper	0.0601	2.35	0.0333	0.814	0.0327	0.00874
Iron[5]	0.360	0.0953	<0.167	0.180	0.0272	0.0219
Zinc	0.721	3.88	0.777	1.79	2.80	1.50

[1] based on data in Morin (1990); estimated time from last rinse (by precipitation) = 8 days

[2] the rates for Station 3 are based on the 8-day accumulation, but actual time from last flush is probably much greater

[3] Station No. 2 is located in the Southern Tail Pit, now filled with waste rock

[4] Metal rates are for dissolved metals in units of $mg/m^2/wk$

[5] Reported rates are not significantly different from the Blank sample and thus actual rates are probably less than the reported values

TABLE 4-2
Partial Input Data for the Simulation of the Main Zone Pit at Equity Silver Mines (Morin, 1990)

PARAMETER	ROCK UNIT #1	ROCK UNIT #2	ROCK UNIT #3
Rock Type	Gabbro	Volcanics	Volcanics
NNP from Acid-Base Accounting (t $CaCO_3$/1000 t)	NNP > 0.0	-40 < NNP < 0.0	NNP < -40
Orientation on Pit Wall	Vertical	Vertical	Vertical
Exposed Surface Area (m^2)	315,000	234,000	214,000
Sloping Walls - Percentage of Total Area	35%	35%	35%
Sloping Walls - Angle from Horizontal (degrees)	70	70	70
Exposed Acid-Generating Sulfur (g S/m^2)	48	64	144
Exposed Neutralization Potential (g $CaCO_3/m^2$)	225	100	25
Reactive Surface (m^2) for Each m^2 of Pit Wall	21	41	41
Fracture Flushed*: - percentage flushed monthly	28%	28%	28%
- percentage flushed once a year	2%	2%	2%
- percentage not flushed during Operation	70%	70%	70%

* These best-fit values were obtained from calibration: see text.

conditions and adjusting the percentages until approximate matches to Operation water chemistry were obtained. For example, an unreasonably high percentage of frequently flushed and yearly flushed surfaces created anomalously low pH values in the pit sump, which had never been measured. Conversely, an unreasonably low percentage created anomalous, continually alkaline pH values, which did not mimic the periodic flushing of accumulated reaction products. The best-fit calibration by MINEWALL 1.0 and 2.0 indicated that 70% of the rock surfaces were not being flushed during Operation (Table 4-2).

After the calibration to Operational data, MINEWALL was then used in predictive mode for Closure. The MINEWALL 1.0 simulations indicated there would be an initial flushing of accumulated acidity, depressing pH to below 6.0, but the pH would afterwards recover and remain alkaline. MINEWALL 2.0 gave the same prediction, except pH would not be initially depressed at Closure. As a result, as the pit-water level continue to rise slowly, the estimated 70% of reactive surfaces that had accumulated reaction products year after year were not flushed at a sufficient rate to offset the alkalinity brought into the pit by saturated groundwater flow.

4.2 Noranda Mineral's Bell Mine

Bell Mine is located near Granisle in central British Columbia. This open-pit mine closed in 1993 after operating for about twenty years and a formal Closure Plan was submitted in 1993 for regulatory approval (Noranda Minerals Inc., 1993; McArthur and Gallinger, 1994). A portion of the Closure plan involved an assessment of acidic drainage and metal leaching based on over 200 acid-base accounts, seven humidity cells, four sizes of leach testwork (1 kg to 30 t), and over 10 years of monitoring data around the minesite (Morin and Hutt, 1993a and b; Morin et al., 1995). This information played a key role in MINEWALL simulations.

As part of MINEWALL 2.0 studies, six MINEWALL Stations were established on various rock types and underwent four rinses over a period of two months. Average rates from the MINEWALL stations are provided in Table 4-3. Reactive rock-surface area in this pit was estimated at 10,500,000 m^2, which represented a ratio of 27:1 of reactive surface to visible surface. This site and its MINEWALL simulation are discussed further in the companion report, Application of MINEWALL 2.0 to Three Minesites.

4.3 BHP Minerals' Island Copper Mine

Island Copper Mine is located on the northern end of Vancouver Island, near Port Hardy, British Columbia. This mine has operated since 1971 and is expected to close in 1995 or 1996 (Horne, 1993; Island Copper Mine, 1994).

Five MINEWALL Stations were created at the site and rinsed 16 times over 1.5 years. Average rates from each station are compiled in Table 4-4. Reactive rock-surface area was estimated at 244,000,000 m^2 with a 161:1 ratio of reactive to visible surface. This minesite and its MINEWALL simulation are discussed further in the companion report, Application of MINEWALL 2.0 to Three Minesites.

TABLE 4-3
Average Production Rates for Bell Mine MINEWALL Stations ($mg/m^2/wk$)

Parameter ($mg/m^2/wk$)	Station BFP	Station BFP2	Station BBFP	Station TUFF	Station QFP	Station SED
pH (units)	7.25	6.95	7.28	6.43	7.20	7.00
Particulates	878	1060	1250	3930	821	622
Alkalinity	85.5	276	252	21.4	118	62.6
Acidity	2.14	2.47	2.15	8.23	1.36	21.6
Sulfate	181	122	334	2670	200	195
Silicate	<0.3	<1.0	<0.9	<0.6	<0.3	<0.8
Nitrate	0.407	0.00823	1.53	0.360	0.168	0.0807
Aluminum	<0.06	<0.3	<0.2	<0.12	<0.06	<0.16
Cadmium	<0.003	<0.014	<0.008	<0.006	<0.003	<0.008
Calcium	98.1	214	249	69.1	127	40.8
Copper	0.0086	0.0211	0.0244	0.308	0.0423	0.881
Magnesium	17.3	28.7	44.2	525	21.0	51.5
Manganese	0.0149	0.109	0.0346	2.10	0.0522	0.0384
Molyb.	0.240	<0.04	0.356	0.038	0.00828	<0.025
Potassium	1.97	<3.0	2.31	3.80	1.09	<3
Sodium	1.28	<3.0	2.45	4.18	1.21	<3
Strontium	0.929	0.840	1.46	0.563	1.45	0.344
Zinc	0.0061	0.00829	0.00712	0.0915	0.00219	0.0174

TABLE 4-4
Average Production Rates for MINEWALL Stations at Island Copper Mine (mg/m^2/wk)

Parameter (mg/m^2/wk)	Station 1	Station 2	Station 4	Station 5	Station 6
pH (pH units)	7.31	6.61	6.25	6.23	6.45
Alkalinity	25.3	9.95	4.82	5.66	16.9
Sulfate	79.8	70.7	8.90	42.1	187
Silicon	11.0	-	1.48	1.86	10.6
Aluminum	0.500	1.07	0.130	0.157	0.420
Cadmium	<0.01	<0.01	<0.01	<0.01	<0.01
Calcium	51.2	32.0	9.23	23.1	75.2
Copper	0.0102	0.0191	0.00886	0.0143	0.0619
Magnesium	1.84	0.388	0.153	0.824	3.30
Zinc	<0.1	<0.1	<0.1	0.11	0.170

5. OTHER STUDIES NOT RELATED TO MINEWALL

5.1 Compiled Data on Groundwater Drawdowns Near Pits

As depicted in Figure 2-1, the local groundwater level is drawn down around pits, generally forming a "drawdown cone" and causing groundwater on a regional scale to be drawn towards and into the pit. Additionally, most infiltration within the cone can be expected to eventually report to the pit.

A compilation of data on the lateral extent and maximum depth of drawdown around pits of various sizes is presented in Table 5-1. Maximum reported extent and depth of drawdown in the Table is 50 km and 130 m, respectively, both from a coal mine in Australia after 22 years of pumping at 68,000 m^3/day. The drawdown at each pit is unique, and reflects factors such as the width and depth of the pit, the pumping rate, the length of pumping, and complexities in the hydrogeologic system including hydraulic boundaries such as rivers and relatively impermeable faults.

5.2 Compiled Information on Fractures and Their Effects on Groundwater

As indicated in Equation 3.4 and discussed in Section 3.2, fractures can play the dominant role in determining total reactive surface area as well as the rate of groundwater inflow in a pit or underground mine. It is therefore appropriate to highlight the near-random occurrence, shape, orientation, and surface of fractures (Figures 5-1 and 5-2) in order to show that any resulting estimates of minewater flow and

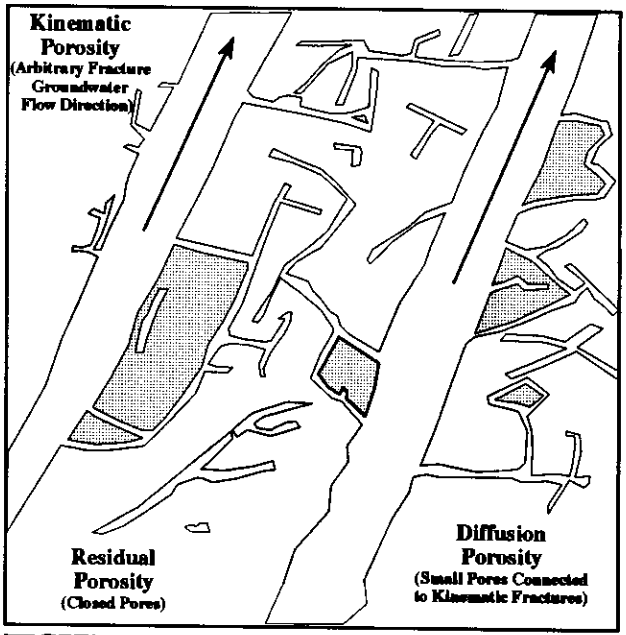


FIGURE 5-1. Kinematic, Diffusion, and Residual Porosities from Fractures (Nordstrom et al., 1989).

TABLE 5-1
Case Studies of Groundwater Flow to Pits

Location & Type of Mining	Approximate Pit		Maximum Drawdown (m)	Lateral Extent of Drawdown (m)	Elapsed Time	Pumping Rate (m^3/d)	Reference & Miscellaneous
	Area (m^2)	Depth (m)					
Faro Mine, Yukon - base metal & Ag	840,000	90 - 150	10	-	1-2 wks	2180	Lopaschuk,1979; 380 m^3/d from lateral drains, 1800 as inflow to pit
Whitewood, Alberta - coal	110,000	20 - 40	20	1200	> 6 yrs	410 to 685	Sumer et al., 1987; recharge = 5% of 50 cm/yr; dewater wells; model $K = 10^{-6} - 5*10^{-4}$ m/s, Sy = 0.05, S = 0.001
Twin Buttes, Arizona - base metal(?)	1,170,000	230	90	300	1.3 yrs	generally < 5	Pentz, 1979; underground galleries with lateral drill holes for dewatering; $K = 10^{-8}$
Shirley Basin, Wyoming - uranium	150,000	90	21	> 300	1 yr	> 1440	Straskraba & Kissinger, 1982; dewater wells/lateral drains; $K = 2*10^{-5}$ m/s, S = 0.00054
Southern Illinois - coal	-	-	> 24	> 300	-	-	Oertel & Hood, 1983; $K < 10^{-7}$ m/s
Kentucky - coal	-	-	10 & <u>rise</u> of 1.5	-	< 1 month	-	Kipp et al., 1983; pits opened in unsaturated zones above wells with positive and negative effects on recharge
Haywood Pit, Scotland - coal	225,000	71	45	-	-	26,000	Norton, 1982; dewater wells
Blindwells Pit, Scotland - coal	1,200,000	60	30	< 1,000	2 yrs	35,000 to 69,000	Norton, 1982; area contains abandoned, flooded underground mines; water table stable prior to pit mining; dewatering wells
Piaski, Poland - coal	-	124	> 100	8,000	-	560,000	Seweryn, 1982; dewater wells

Location & Type of Mining	Approximate Pit		Maximum Drawdown (m)	Lateral Extent of Drawdown (m)	Elapsed Time	Pumping Rate (m^3/d)	Reference & Miscellaneous
	Area (m^2)	Depth (m)					
Nyirad, Hungary - bauxite	2 pits: 5,000,000 2,500,000	 > 100 > 100	100	45,000	21 yrs	430,000	Bocker and Vizy, 1982; Paris, Karst limestone; K = 10^{-5} - 10^{-4} m/s, porosity = 2 - 4%; S = 0.00002; dewatering wells; recharge = 25% of 63 cm/yr
Queensland, Australia - limestone	10,000	27	25	> 2,000	7 yrs	1,400	Dudgeon, 1987
Victoria, Australia - coal	3,000,000	-	130	50,000	22 yrs	68,000	Evans, 1987; aquifer K = $6*10^{-6}$ - $2*10^{-4}$ m/s, S = 0.00002 - 0.0003; K of aquitard = 10^{-11} m/s
South Australia - coal, trial pit	200,000	60	40	-	1 yr	17,300	Armstrong, 1982; aquifer K = 6 - $9*10^{-5}$ m/s; aquitard K = 10^{-8}; series of emergency wells for water control
Neyveli, India - coal	7,770,000	70	49	11,000	18 yrs	160,000 to 320,000	Hofedank, 1979; Brealey, 1965; transmissivity = 0.01 m^2/s; dewatering wells

chemistry carry a high degree of uncertainty.

For example, Jakubick et al. (1989) examined fracture occurrence perpendicular to walls of underground workings (Figures 5-3 and 5-4), and found discrete fractures with variable permeability (and thus variable size, extent, and surface area). In particular, Jakubick et al. were searching for enhanced fracturing close to the minewalls, but apparently did not find it.

Also, Witherspoon et al. (1981) measured groundwater pressures in discrete intervals of boreholes extending 30 m from an underground working (Figure 5-5). The boreholes contained packers that isolated the fractured intervals, which had been free draining for years. When flow from Borehole R01 was blocked, pressures in some intervals of other boreholes increased. This was taken as an indication of the strength of hydraulic connection (Figure 5-1) between the intervals.

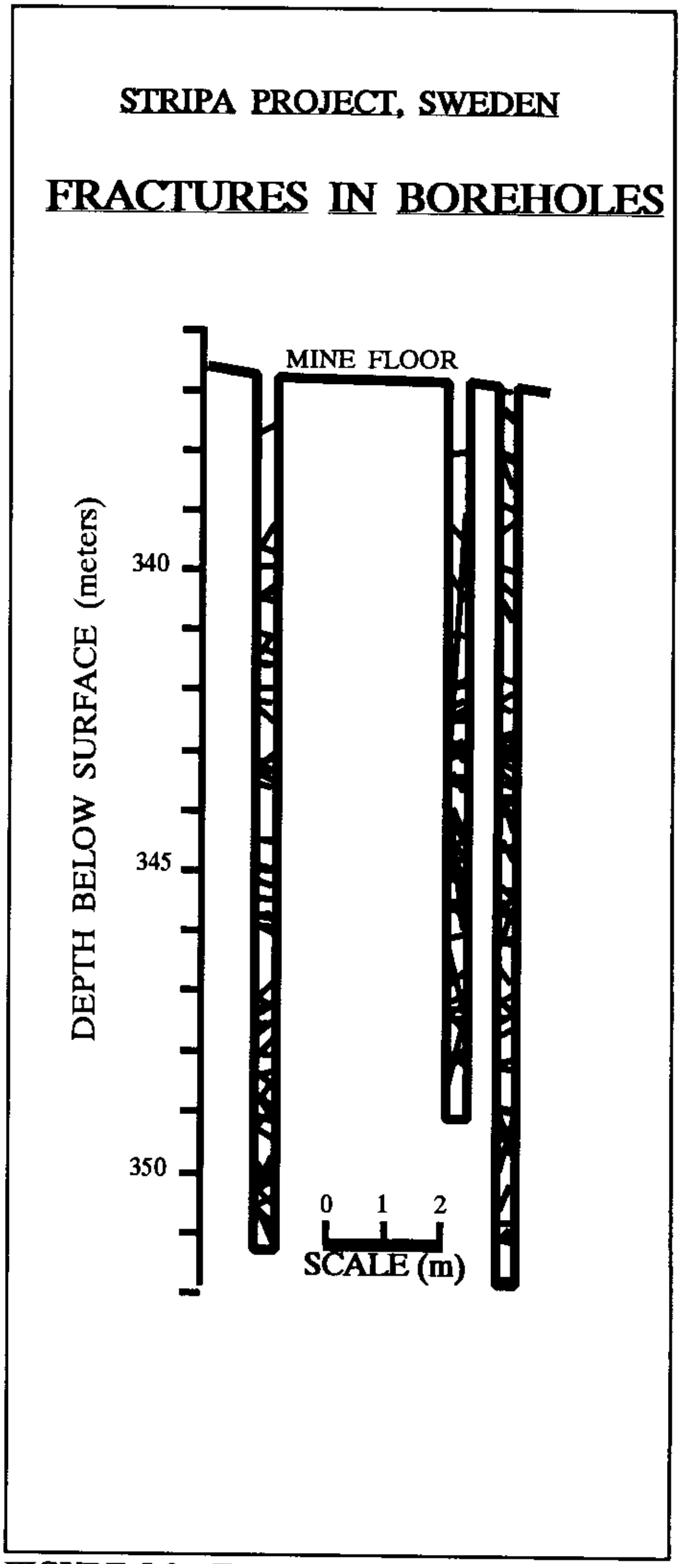


FIGURE 5-2. Fracture Patterns in Closely Spaced Boreholes at the Stripa Project (adapted from Witherspoon et al., 1981).

Many of the studies mentioned in this section on fractures were conducted

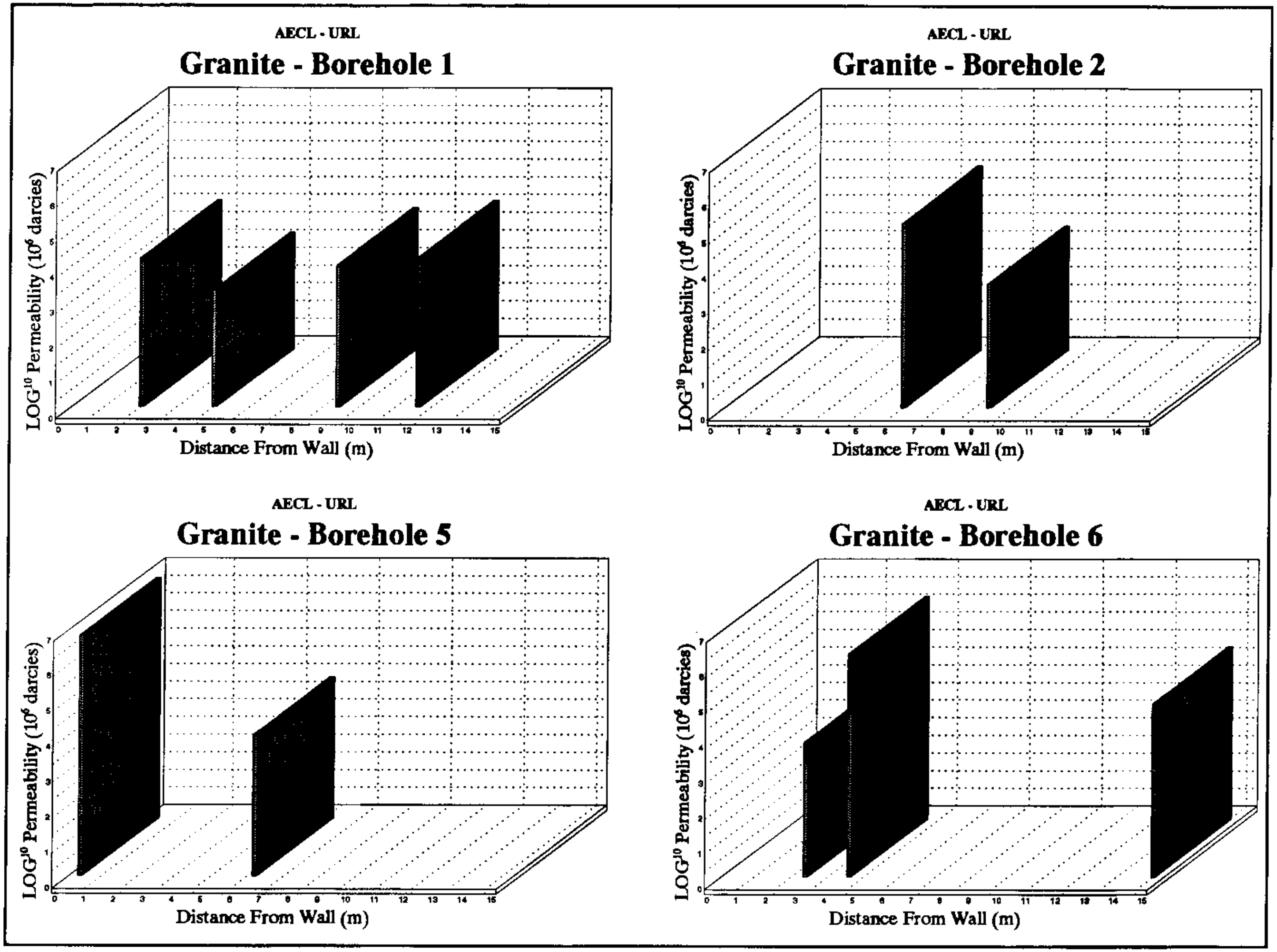


FIGURE 5-3. Discrete Fracture Permeabilities at 15 m depth at Canada's Underground Research Laboratory (adapted from Jakubick et al., 1989).

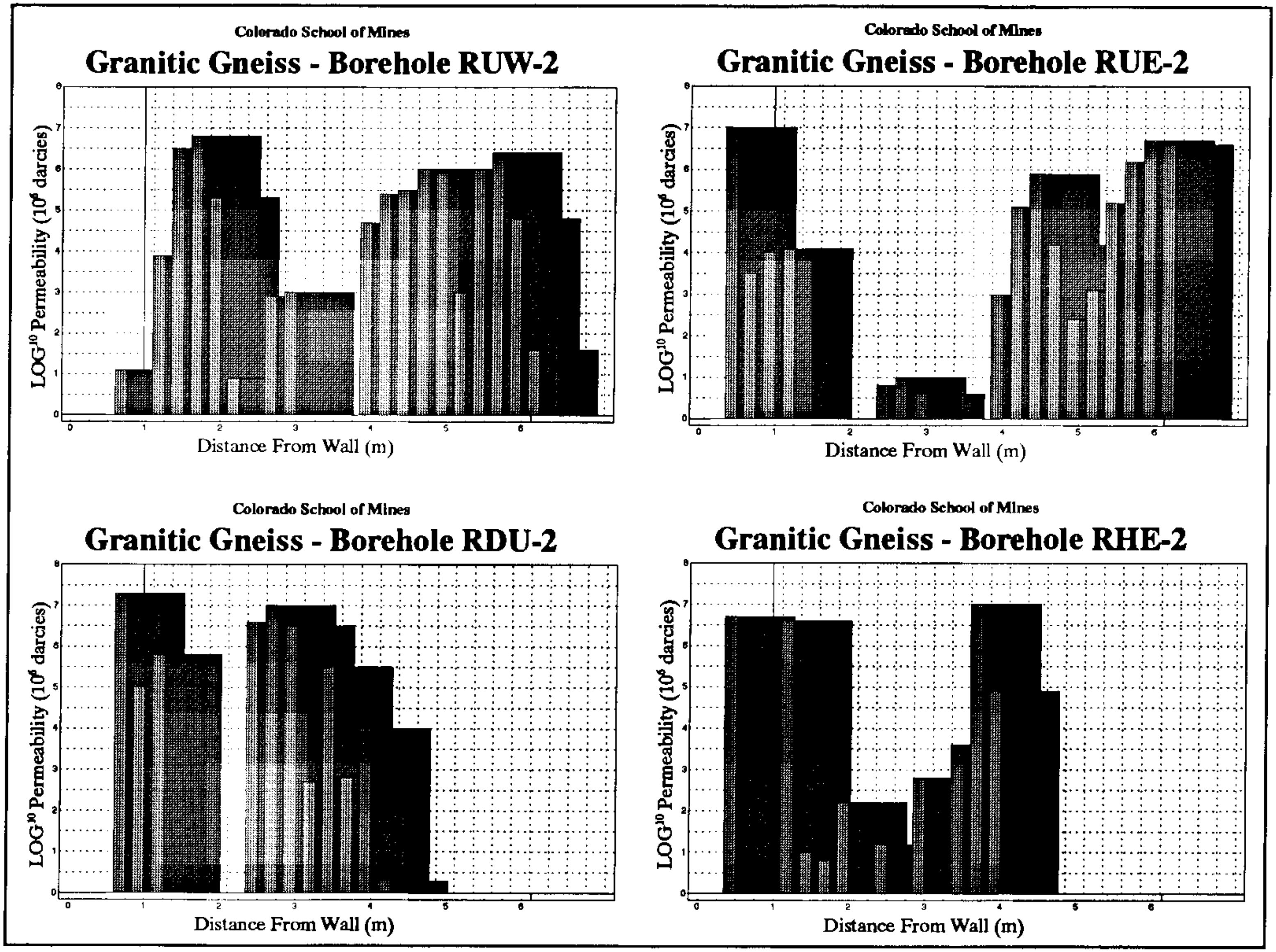


FIGURE 5-4. Fracture Permeabilities at Colorado School of Mines' Experimental Mine (adapted from Jakubick et al., 1989).

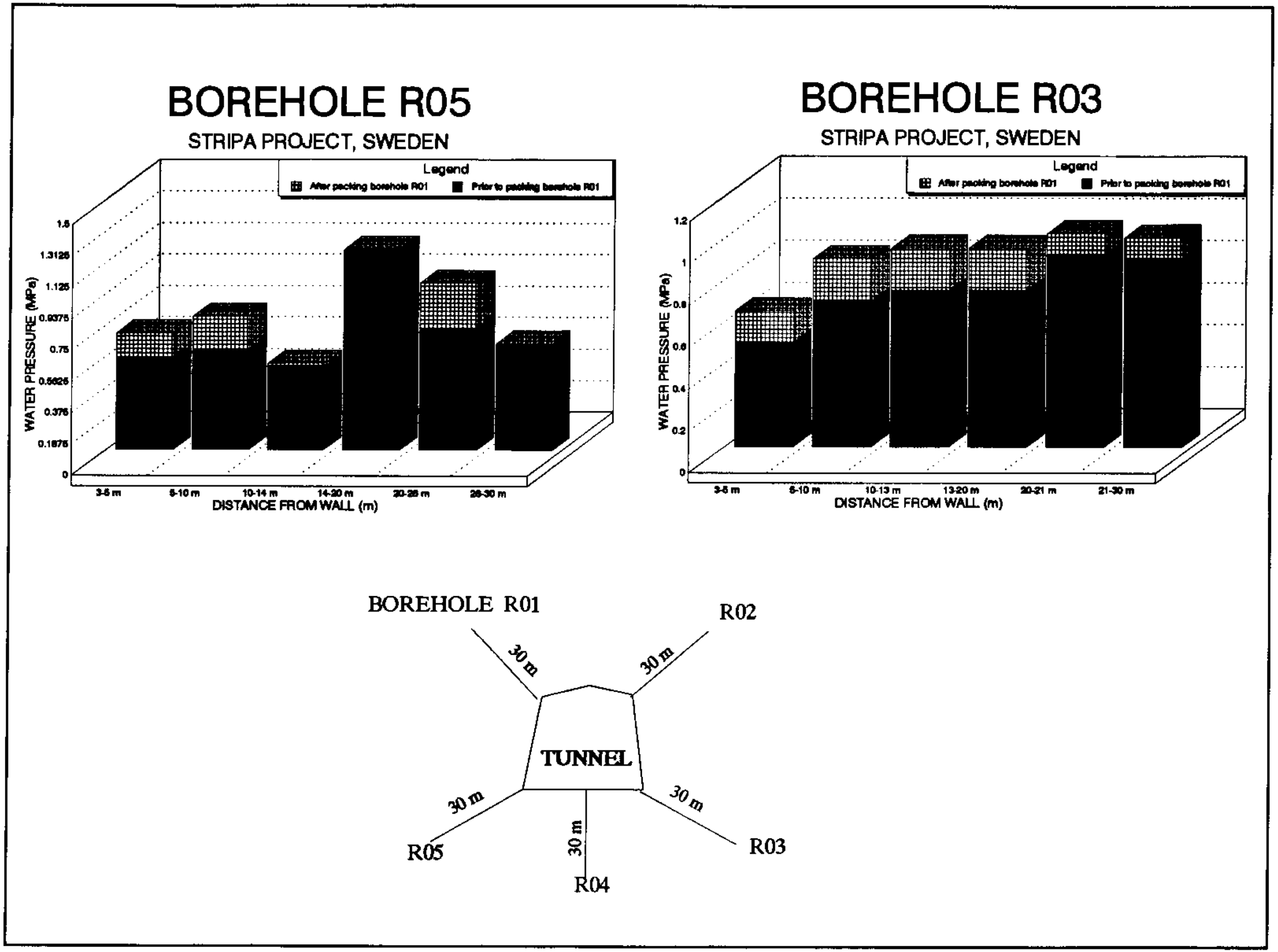


FIGURE 5-5. Changes in Water Pressure within Selected Fractures at the Stripa Project, Sweden (adapted from Witherspoon et al., 1981).

for the safe disposal of high-level nuclear waste. When this and other information was assembled into a computer program, predictions were made on the changes in fracture characteristics and the movement of groundwater as an underground working is extended during mining. When an extension in Canada's Underground Research Laboratory was modelled beginning with a pilot tunnel and then expanded (Lang, 1989), the discrepancies between model-predicted and actual pressure loss was notable (Figure 5-5). This indicated that "the models used for predicting the hydrogeologic response did poorly". No doubt a similar conclusion can be drawn on the ability to define and predict other aspects of fractures, including reactive surface area. As a result, a clear recognition of uncertainty should be built into every MINEWALL simulation.

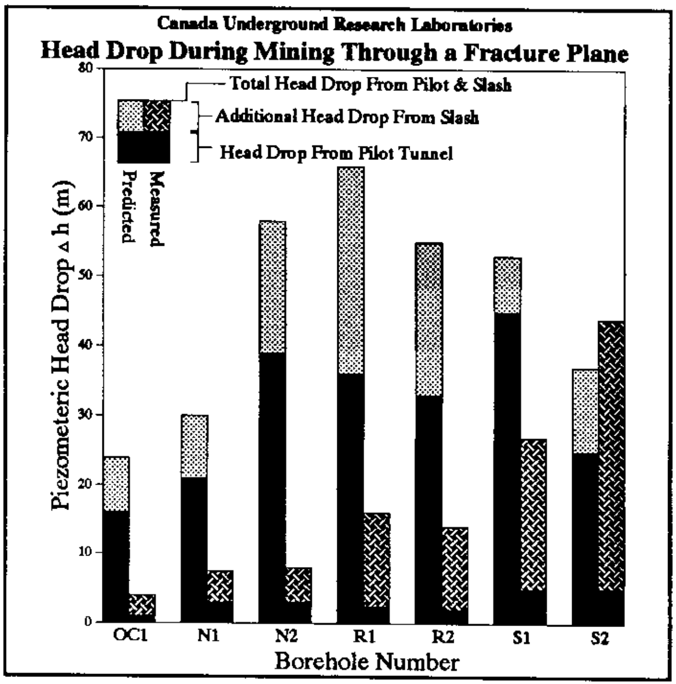


FIGURE 5-6. Predicted and Measured Head Losses During Mining Through a Fracture Plane at URL (Lang, 1989).

5.3 Rum Jungle, Australia

Flooding of pits, particularly the Whites and Intermediate Pits, were reported at the closed Rum Jungle uranium mine in Australia (Goodman et al., 1981; Northern Territory Department of Mines and Energy, 1986; Water Resources Division, 1986; Henkel and Alcock, 1986 and 1987). Whites Pit has a lateral area of 10.5 ha, a maximum depth of 50 m, and a volume of 2,700,000 m^3. This pit flooded to its equilibrium static level (Figure 2-4) with the assistance of diverted river water (Figure 2-3) and at that time had a pH of 4.75 and a sulfate concentration of 180 mg/L. However, unneutralized tailings and treatment raffinate were dumped into the pit through the 1960's so that by 1974 pH had fallen to 2.4 and sulfate increased to 9000 mg/L. The annual flow of water through Whites Pit was estimated at 19,000,000 m^3, but did not include subsurface inflow and outflow.

The upper 15 m of pit water was found to have thermoclines and chemoclines. Consequently, treatment of pit water involved pumping of the deeper, denser acidic water and returning the less dense treated water to the top. Although some mixing of the two layers hindered initial treatment efforts, eventually most of the acidic water was treated, pH stabilized at 6.0, and sulfate stabilized at 200 mg/L. Water chemistry varied noticeably over a year, improving when the flow of pH-neutral surface water into the pit increased and worsening when evaporation and groundwater inflow (which was found to be acidic) increased.

The Intermediate Pit has a lateral area of 4 ha, a maximum depth of 78 m, and a volume of 1,100,000 m^3. Like Whites Pit, this pit also flooded to equilibrium level within a year and in 1974 pH was 3.5 with sulfate at 2000 mg/L. The annual flow of water through the Intermediate Pit was estimated at 18,000,000 m^3, but did not include subsurface inflow and outflow.

Unlike Whites Pit, the Intermediate Pit did not contain significant chemoclines and thermoclines, but the reasons for this are not given. Therefore, a different treatment system was designed whereby lime was applied to the top of the pit-water column and the settled sludge removed. As a result, pH increased to 5.9 and sulfate fell to 200 mg/L. Like Whites Pit, pit-water chemistry varied noticeably through the year.

5.4 Compiled Data on Groundwater Chemistry Near Pit and Underground Mines

As a parallel to physical effects (Sections 5.1), McCurry and Rauch (1986) and O'Steen and Rauch (1983) evaluated over 100 groundwater samples from wells near pit and underground mines in West Virginia, USA. Although drawdown should prevent the migration of any contaminated water from a mine, these researchers found impacts by pits as far away as 460 m laterally and 30 m vertically based on sulfate concentrations. The concentrations were found to decrease exponentially with distance from the pits. However, because even upgradient wells showed impacts, the cause of increasing sulfate may not be migration from pits, but the exposure of fresh rock to air in the drawdown cone (e.g., Figure 2-1). Water-chemistry effects of underground mines were less than those from pits.

5.5 Research by The Ohio State University (Morth et al., 1972)

Through the 1960's and 1970's, many graduate degrees were awarded by the Department of Geology at The Ohio University based on studies of acidic drainage from underground mines. Many of these studies are summarized by Morth et al. (1972), which is the earliest integrated physical, chemical, and biological study of acidic drainage identified in this literature review. Although some equations in this report may be erroneous based on dimensional analysis, it was the cornerstone of the MINEWALL conceptual models for underground mines and contributed significantly to the conceptual models for pits. Morth et al. was virtually "lost" to researchers, since it was discovered by chance on a library shelf during the MINEWALL 1.0 study and did not appear in any computer-based searches of databases.

In addition to opposing some views on acid generation still held today, Morth et al. (1972) report on field and laboratory tests involving the occasional rinsing of exposed coal. The field tests involved the isolation of a face of coal in the mine by cutting channels 15-30 cm deep around the selected face and filling the channels with polyurethane foam. This is similar to the MINEWALL Stations (Section 4), but MINEWALL Stations involve only the isolation directly on the face, since small channels are difficult to cut in hard rock and may cause fracturing on the face. The laboratory tests were similar to today's kinetic tests known as "humidity cells".

Rates from the field tests, based on rinses every 4-8 weeks, were 140-5250 mg acidity/m^2/wk, which are higher than those from the MINEWALL studies (Tables 4-1 to 4-3) possibly because the coal was more porous (Figure 3-1). Laboratory rates, based on condensation leaching of reaction products on coal blocks 1x2x4 inches, were 21-56 mg acidity/m^2/wk, 9-16 mg sulfate/m^2/wk, and 13-51 mg iron/m^2/wk. The lower rates from the laboratory testwork may have represented the retention of some reaction products on grain surfaces from incomplete condensation leaching.

Based on cores into the mine walls, Morth et al. indicated that sulfide oxidation was occurring at distances of up to 15 m from the walls. An equation was developed for calculating the rate of acid generation in a porous block of coal, although dimensional analysis suggests the equation contain errors. The equation was normalized to 1 g of pyritic material at 60-mesh grain size which was found to consume 25 μg O_2/hr/g at 25°C. This equation was later adapted under MINEWALL 1.0 to two impermeable blocks of rock separated by an oxygen-limited fracture with a fracture trace (length exposed on the wall) of L meters and unlimited depth into the wall:

$$RATE = 2\ \{R/F - (R/F * \exp(-F * L))\} \tag{5-1}$$

where RATE = rate of acid generation in one fracture (mg SO_4/day)

R = maximum oxidation rate under atmospheric oxygen (mg SO_4/m^2/day)

F = $(K/D)^{1/2}$

K = reaction rate (1/day; = $2.0\times10^{-6}/V$, Morth et al.)

V = gas volume in fracture exposed to 1 g of pyrite (m^3)

D = 0.6 * DA

DA = normal oxygen diffusivity in air in appropriate units

L = length of fracture trace on mine wall (m)

Equation 5-1 has not been tested or verified, and thus is not used in MINEWALL. Additionally, Equation 5-1 does not include other effects, such as barometric pumping of oxygen into the fracture, and may therefore underestimate rates.

Morth et al. identified three primary mechanisms for water movement and leaching:

❶ unsaturated flushing of rock surfaces by trickling water,

❷ saturated flushing of channels by inundation of the channel, temporarily halting oxidation, and

❸ migration of acidic condensation, originating from moist air in the mine and the hygroscopic nature of concentrated acidic solutions around pyrite.

Based on the three mechanisms, simulations were performed on the small McDaniels Mine test drift (12-14 m long), on large-diameter holes near the McDaniels Mine known as Auger Holes 1, 3, 4, 5, and 6, and on the Decker Mine with an acid output in 1964 of 76,000 kg/yr. For MINEWALL 1.0, the raw data from Morth et al. was normalized to time and the reported surface area "providing water", yielding the unit-area rates in Table 5-2. The rates for acidity production are similar to those measured in the aforementioned field tests, supporting their validity. However, these rates are notably higher than those measured in the less-porous MINEWALL studies of Section 4, presumably due to less reactive surface within a 1 m^3 block of rock.

Morth et al. concluded that hundreds of kilograms of acidity could be stored within the walls of small mines like the McDaniels (one drift of 12-14 m length). These stored reaction products can then be released slowly through time or released quickly upon flooding, which could be mistaken for ongoing acid generation.

5.6 Solubec Pit, Quebec, Canada

The Solbec Copper-Zinc-Lead Mine in Quebec was operated from 1962 to 1970 with a total of 1.5×10^6 metric tonnes (mt) extracted through underground cut-and-fill. An additional 4.0×10^5 mt was removed from a pit in 1964 and 1965 (Figure 5-1). The mill continued operation with ore from the nearby Cupra, Weedon, and Clinton Mines until 1977, processing a total of 4.9×10^6 mt. In 1988, 276,000 m^3 of sulfide-bearing waste rock and mine wastes, mostly of sand to cobble size, was placed into the dewatered open pit. A till dyke installed around the pit perimeter allowed the pit-water level to be raised to 1.5 m above the backfilled rock after a 5-cm layer of non-dolomitic limestone was placed as a cap.

TABLE 5-2
Adjusted Rates of Acid Leaching and Flow Dependencies
(raw data from Morth et al., 1972 normalized to surface area and time)

	McDaniels	---------------- Auger Hole ----------------					Decker #3
		No. 1	No. 3	No. 4	No. 5	No. 6	
Surface area providing (m^2)	736	574	438	338	94	632	255,000
Trickle leaching, mg acid/m^2/wk	770	630	770	880	1040	2190	-
Inundation, mg acid/m^2/wk	77	710	990	886	550	360	-
Condensation, mg acid/m^2/wk	390	180	31	84	250	190	-
Minimum flows below which:							
- inundation removed no acidity (L/m^2/wk)	10.9	5.3	3.6	6.3	0	4.0	31
- trickle leach removed no acidity (L/m^2/wk)	5.4	3.9	1.8	2.4	0	2.9	3.6
- condensation leaching was determined to be independent of flowrate							

Ross et al. (1994) described a study of the physical and chemical hydrogeology of the backfilled Solbec pit (Figure 5-7). This study showed that groundwater was flowing laterally through the backfilled rock and that elevated concentrations of acidity and metals were found downgradient of the pit. The elevated concentrations were attributed to two mechanisms: (1) flushing of accumulated acidity and metals from the backfilled rock and (2) slow oxidation in the submerged rock. Data over the short time since backfilling could not reveal the relative contribution of these two mechanisms to the downgradient chemistry. From the MINEWALL perspective, there may also be contributions from the pit walls as well as the mine workings beneath the pit.

The pit walls and floor exposed chloritic schist and sericite. "Since the rock was extracted using dynamite, it is highly probable that the pit walls and the floor have fractured" (Ross et al., 1994). Drillholes revealed that up to 6 m of fine material had accumulated in the pit prior to backfilling. A small proportion of this material was attributed to deposition of $1\text{x}10^4$ m^3 of mine wastes into the pit in the last stages of milling.

The underground workings were at least 10 m below the pit floor. Some shafts extended from the pit floor into the workings, but these shafts may have been filled and covered with till before placement of waste rock. Other shafts extended from ground surface around the pit to the workings.

Ross et al. (1994) drilled five boreholes: two inside the pit and three around the pit perimeter (Figure 5-7). Multilevel piezometers were placed in four of the five holes. Groundwater flow was found to be generally lateral from north to south, with some variable vertical gradients along the flowpaths. Based on hydraulic conductivities of $1\text{x}10^{-6}$ to $1\text{x}10^{-7}$ m/s in the surrounding fractured bedrock, maximum porosity of 0.01, and hydraulic gradient of 0.03, the average linear groundwater velocity was estimated at 10-95 m/yr. Total volume of groundwater flow was estimated at $1.2\text{x}10^4$ m^3/yr. The effect of the underground workings on groundwater movement could not be determined.

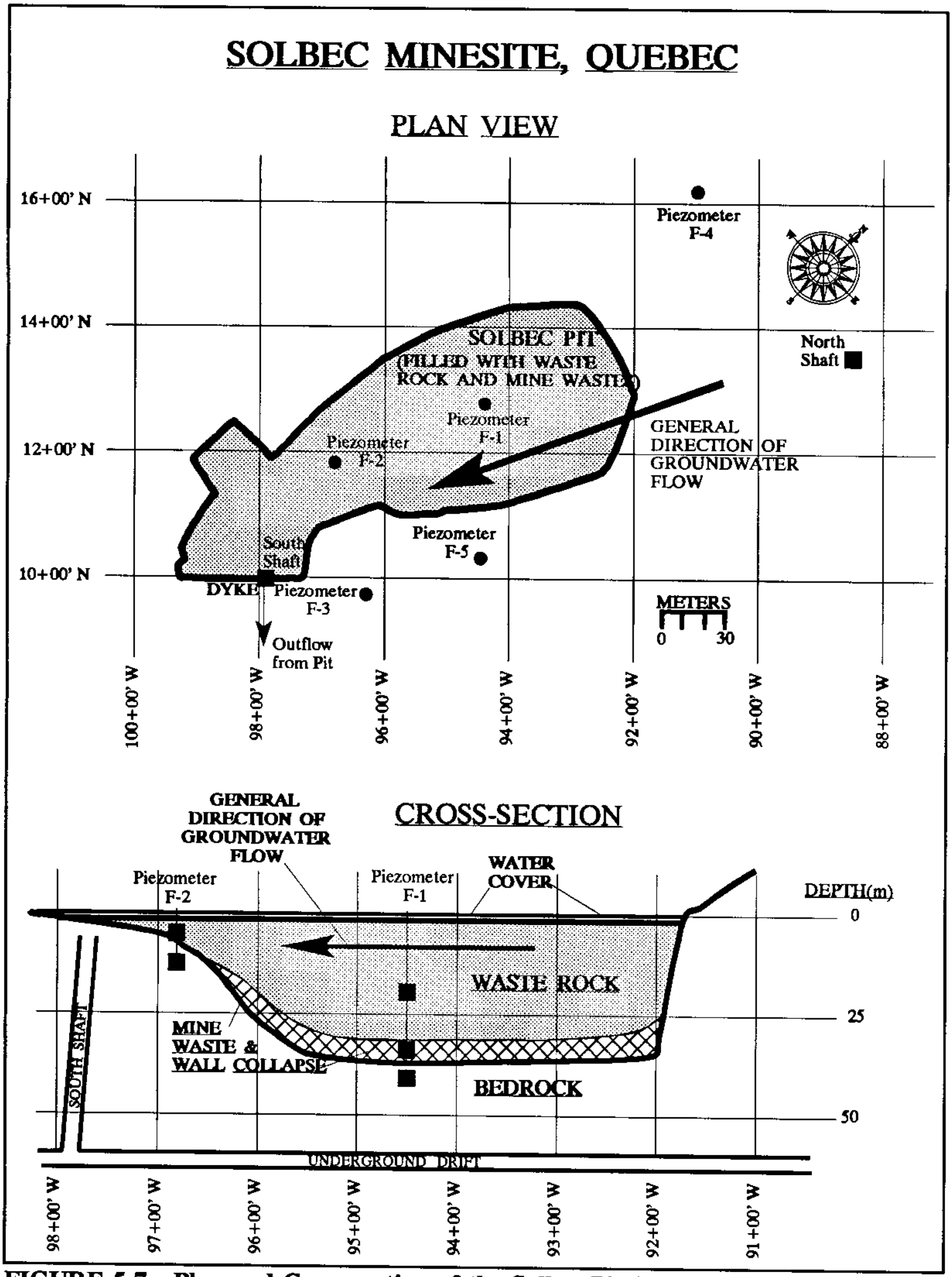


FIGURE 5-7. Plan and Cross-section of the Solbec Pit (adapted from Ross et al., 1994).

An electromagnetic survey indicated groundwater contamination extended 120-160 m south from the pit perimeter, which was consistent with piezometer samples to the east of the pit. Porewater in the submerged waste rock contained more than 1 mg/L dissolved oxygen and thus ongoing oxidation of sulfide minerals was considered a possible source of the groundwater contamination. Also, since the waste rock was known to be a source of acidic drainage and leached metals prior to placement in the pit, the simple flushing of accumulated reaction products could account for the downgradient contamination. Further monitoring was recommended to distinguish between the two causes.

Ross et al. (1994) provided little water-chemistry data. However, bar charts that are presented for a few parameters suggest the concentrations are regulated by chemical equilibrium. As a result, the calculation of reaction rates and further chemical evaluations are not possible from data in the reference.

5.7 Brunswick Mining No. 6 Minesite, New Brunswick, Canada

The No. 6 Minesite, located approximately 27 km southwest of Bathurst, New Brunswick, began operating in 1966 and closed in November, 1983 (St-Arnaud and Aiken, 1991). Ore containing zinc, lead, copper, and silver was mined from a pit with limited underground workings, to a total of 1.21×10^7 tonnes.

Approximately 3.0×10^6 m^3 of waste rock and waste sulfides from the operation were placed in the North, East, and Pyrite Waste Dumps. An additional 2.1×10^5 m^3 was used for roads and foundations and lesser amounts were placed in the South waste pad. Roughly 8×10^4 m^3 of tailings from a pilot-scale operation are also located on the site.

The geology of the minesite is dominated by coarse-grained, quartz-rich meta-gabbro of Ordovician age. The orebody contained massive interlayered sulfide beds consisting mostly of pyrite, pyrrhotite, chalcopyrite, galena, and sphalerite. Local faulting was identified with up to 30 m of displacement which also brecciated the nearby sulfide beds up to a width of 1 m. Overburden covering the rock is predominantly compact sandy silty basal till.

Maximum depth of the No. 6 Pit was 185.4 m and lateral surface area at the overflow elevation of 2581.7 m was estimated at 11.1 ha. The underground workings joined the south side of the pit near the base. Total volume of the pit excluding the workings was set at $8.5x10^6$ m^3. The pit was approximately 50 m below its overflow elevation, but expected to reach it by the Year 2000.

Identified or suspected sources of acidic drainage at the site were: mine walls, the waste-rock dumps and pads, the pyrite dump, the tailings pond, acid-water holding and settling ponds, a natural on-site pond, roads, and building foundations. Acidic drainage from the site was treated seasonally with lime and a average of $3.7x10^5$ m^3 of treated water was annually discharged into the pit for settling of precipitants. Additionally, high-pyrite acidic waste rock was dumped into the southeast corner of the flooded pit.

St-Arnaud and Aiken (1991) reported on a hydrogeologic study of the minesite, involving 36 monitoring wells and a few samples of pit water. Most groundwater was expected in the more fractured, shallow bedrock, which had hydraulic conductivities between 10^{-4} and 10^{-7} m/s with a mean (presumably geometric) of $1.27x10^{-5}$ m/s. Vertical hydraulic gradients were negligible so lateral flow was expected to dominate at calculated average velocities of 100-1000 m/yr.

Groundwater levels in the wells indicated drawdown caused groundwater within 100-200 m of the pit to move into it. However, the shape of the drawdown cone shows marked anisotropy with relatively steep hydraulic gradients to the north and west of the pit. When the pit is filled, groundwater is expected to migrate southward through the pit area towards a brook over 500 m away.

The rate of groundwater flow into the pit was calculated through the "equivalent well approach" attributed to Singh and Reed (1988) and modified from the Dupuit Equation for lateral flow in an unconfined aquifer:

$$Q(t) = \pi K \left(H^2 - h^2(t)\right) \ln(R(t)/r) \tag{5-2}$$

$$r = (2/\pi)\,(Y * W)^{0.5} \tag{5-3}$$

where $Q(t)$ = flow ("t" accompanying a variable indicates its value changes through time)
K = bulk hydraulic conductivity
H = regional elevation of water table
$h(t)$ = pit-water level at a specific time
$R(t)$ = radius of groundwater drawdown caused by the pit
r = "equivalent-well radius" of the pit
Y = length of the pit
W = width of the pit

Based on flows from this model plus precipitation, treatment discharge, and the addition of waste rock, and subtracting evaporation, the June 1990 water-level elevation was accurately predicted and an additional average volume of 470,000 m^3 was being added to the pit each year. As a result, the pit-water level was expected to reach the regional water-table level in 1995 and continue rising until the pit overflows in 1999. At that time, flow from the pit would follow both subsurface and surface pathways.

Pit-water analyses indicated zinc exceeded 59 mg/L and pH was less than 4. All physical and chemical parameters were relatively constant laterally, but pH and total iron were consistently lower at depths less than 4.3 m. This difference at shallow depths was attributed to iron oxidation and acidic rain.

Around most of the minesite, aqueous sulfate concentrations in groundwater exceeded 2000 mg/L, indicating the effect of sulfide oxidation was pervasive. However, groundwater with pH less than 5.0 and zinc greater than 100 mg/L was limited to the north and east of the pit, apparently reflecting subsurface acidic drainage from the North, East, and Pyrite Waste Dumps and the tailings area. Maximum reported zinc concentration in groundwater was 3240 mg/L beneath the North Waste Dump.

5.8 Midnite Mine, Washington, USA

This inactive uranium mine is located on the Spokane Indian Reservation, approximately 40 miles northwest of Spokane, Washington. The minesite has approximately 130 ha of disturbed land at a altitude of roughly 850 m. Uranium was discovered at the site in 1954, and mining was carried out between 1956 to 1982 except for a shutdown period from 1962 to 1969. The site contains two pits (Pit 4 and the larger Pit 3), one ore stockpile, and two waste-rock dumps (Figure 5-8).

In 1978, a yellow-white precipitant was noted in a creek bed draining the site, extending several hundred meters below the mine boundary. This precipitant was found to contain aluminum-bearing minerals, gypsum, and 3-6% uranium oxide, which was taken as evidence of significant metal leaching. A retention pond and dam was constructed and water from this pond and Pit 4 was occasionally pumped to Pit 3. A small flow of surface water with total dissolved solids of approximately 6,000 mg/L still bypassed the dam.

Sumioka (1991) reports on a detailed physical and chemical study of the hydrogeology and hydrology of the minesite. The site has warm, dry summers and clear, cold winters with snow. Average precipitation at a nearby town was 49 cm/yr over a 29 year period. The mine is mostly limited to one topographic basin which is drained by three small streams that converge.

The geology of the area consists of Precambrian metasedimentary phyllites and calc-silicate hornfels. This rock has been intruded by Cretaceous porphyritic quartz monzonite in portions of the minesite. Quaternary alluvium covers the rock in places. Uranium ore is located at contacts between the metasedimentary and granitic rock. Near the surface, uranium occurs as autunite and meta-autunite, whereas at greater depth it occurs as uraninite and coffinite as is associated with pyrite and marcasite.

Pit 3 is located near the center of the minesite. It has a maximum depth of approximately 170 m, a lateral area of 3.6 ha at full water level, and an estimated volume of

4.4×10^5 m^3. Pit 4 at the north end of the site has a maximum depth of roughly 140 m, a lateral area of 2.4 ha at full water level, and an estimated volume of 1.8×10^5 m^3. The southern rim of Pit 4 was raised 12-15 m with fill in 1982 to prevent overflow of the pit. The retention pond below these pits has a maximum depth of 9 m, a surface area of 0.4 ha, and a volume of 1.8×10^4 m^3.

Most of the study conducted by Sumioka (1991) focussed on site-wide water balance and downstream water chemistry. However, there were chemical analyses of the water columns in Pits 3 and 4, and in the retention pond (Tables 5-3 to 5-8).

In Pit 4 (Tables 5-3 and 5-4), conductance and pH (~7) are generally steady through time and depth. However, with increasing depth, temperature and dissolved oxygen clearly decrease, while total dissolved solids, ^{226}Ra, and U increase. These trends indicate a relatively stable water column with chemoclines and thermoclines (Section 3.3), and more reducing conditions mobilizing uranium with depth.

In contrast to Pit 4, Pit 3 (Tables 5-5 and 5-6) shows relatively few or minor trends of any parameters through time and depth. The measured pH remains around 4.5. As a result, Pit 3 appears to remain well mixed with little stratification.

The retention pond (Tables 5-7 and 5-8) shows some decrease of dissolved oxygen and increase of suspended solids with depth. However, most parameters remain generally steady with pH around 3.6.

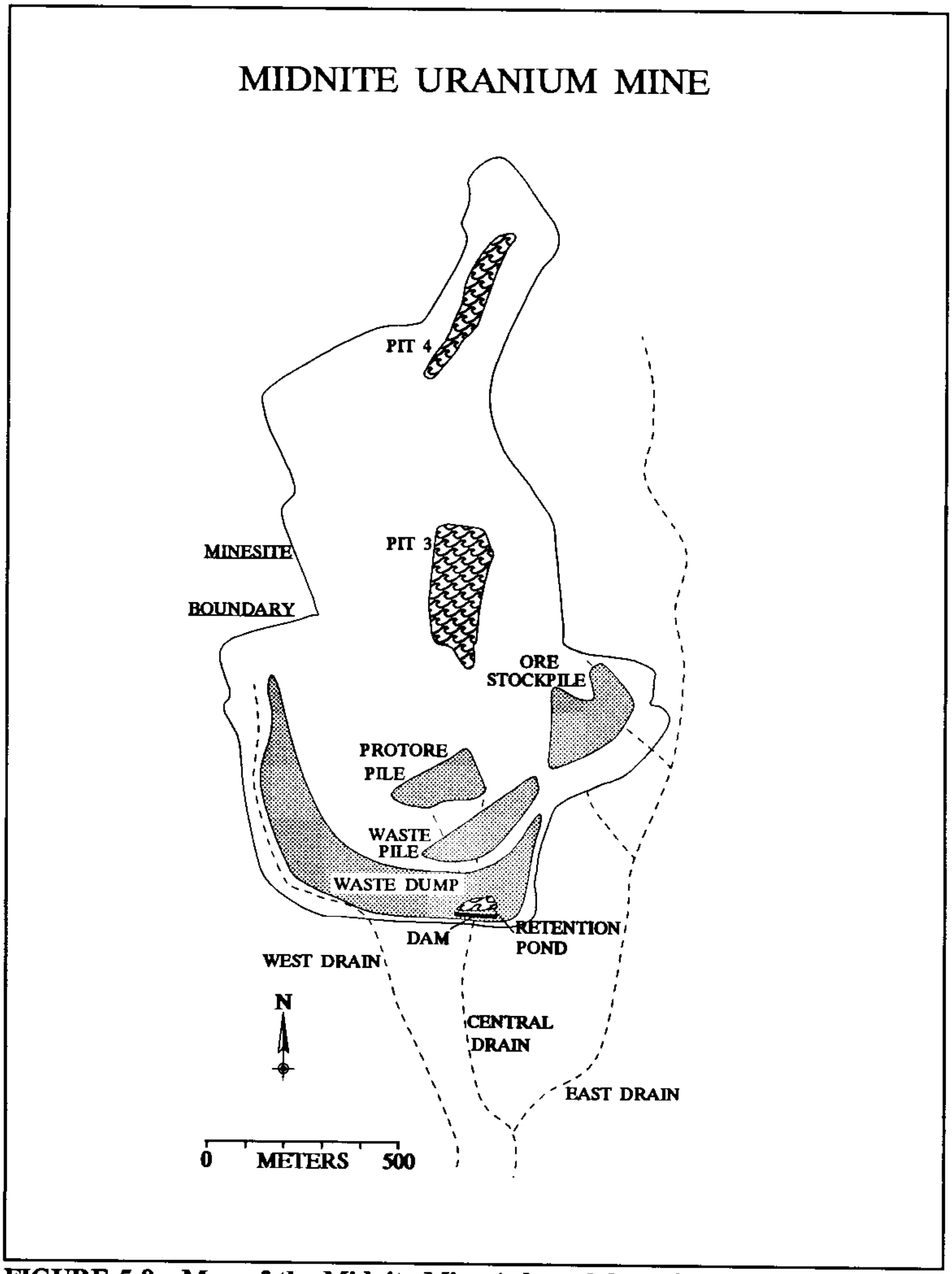


FIGURE 5-8. Map of the Midnite Mine (adapted from Sumioka, 1991).

TABLE 5-3
Conductance and pH with Depth from Pit 4, Midnite Mine

Depth (m)	pH	Conductance (μS/cm, 25°C)
31 August 1983		
3	7.6	1210
16	6.9	1350
25	6.9	1550
50	6.8	1580
73	6.9	1600
1 May 1984		
3	6.9	933
12	6.9	1120
32	6.8	1440
50	6.7	1470
65	7.0	1550
5 September 1984		
3	7.0	1070
12	7.1	1070
21	7.0	1410
40	6.9	1550
67	6.8	1600

TABLE 5-4
Pit 4 Water Chemistry with Depth, Midnite Mine

Parameter (mg/L unless noted)[1]	31 August 1983			1 May 1984			6 September 1984		
	0.9 m	4.9 m	22.3 m	0.9 m	9.8 m	19.8 m	0.9 m	6.4 m	20.4 m
Cond	1210	1350	1600	933	1440	1550	1070	1410	1600
pH	7.6	6.9	6.9	6.9	6.8	7	7	7	6.8
°C	19	10.5	5.2	7.6	5.2	5.8	17	9.3	6.3
O_2 d	8.1	6.4	2.1	10.2	3.6	0.4	8.6	9	0.1
COD	-	-	-	12	-	11	30	50	40
Hardness	650	660	850	450	740	800	540	660	810
Ca,d	200	200	260	130	220	240	160	200	250
Mg,d	36	39	49	28	43	47	33	39	45
Na,d	30	33	41	23	36	39	27	33	39
K,d	3.3	3.2	3.9	2.4	3.6	4	3.1	3.4	4
Alk, t	30	35	53	27	45	58	27	35	60
SO_4, d	560	610	770	430	710	780	460	580	740
Cl, d	1.4	1.4	1.7	1	1.4	1.5	1	1	1.4
F, d	0.5	0.4	0.3	0.3	0.3	0.3	0.3	0.2	0.2
SiO_2, d	22	20	19	20	19	19	24	22	19
Diss Solid	966	1050	1320	763	1230	1330	826	1010	1290
Susp Solids	-	-	-	2	3	<2	6	8	4
NO_2,N	-	-	-	0.03	<0.01	<0.01	0.1	0.04	<0.01
NO_2+ NO_3,N	19	23	30	16	28	29	14	20	27
P, total	-	-	-	<0.01	<0.01	<0.01	<0.01	<0.01	0.01
Al, d	<0.100	<0.100	<0.100	<0.100	<0.100	<0.100	<0.100	<0.100	<0.100
Al, t	-	-	-	-	0.11	0.1	<0.100	<0.100	<0.100
Be, d	-	-	-	0.002	<0.001	<0.001	<0.001	<0.001	<0.001

Parameter (mg/L unless noted)[1]	31 August 1983			1 May 1984			6 September 1984		
	0.9 m	4.9 m	22.3 m	0.9 m	9.8 m	19.8 m	0.9 m	6.4 m	20.4 m
Cd, d	0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Cd, t	-	-	-	<0.001	<0.001	<0.001	0.034	0.007	0.003
Cu, d	0.001	0.001	<0.001	<0.002	<0.001	<0.001	0.001	<0.001	<0.001
Cu, t	-	-	-	0.004	0.005	0.004	0.005	0.004	0.003
Fe, d	<0.003	<0.003	0.003	0.004	0.055	0.021	<0.003	<0.003	<0.003
Fe, t	-	-	-	0.05	0.06	0.07	0.03	0.04	0.03
Mn, d	1.3	2.9	3	1.9	3	3.3	1.2	2.4	3.1
Mn, t	-	-	-	2.1	2.9	3.2	1.7	2.4	3.1
Ni, d	0.061	0.085	0.082	0.057	0.067	0.063	0.038	0.052	0.058
Ni, t	-	-	-	0.075	0.09	0.084	0.056	0.058	0.051
Sr, d	0.7	0.78	1	5	8.8	9.6	0.59	0.79	0.95
Zn, d	0.023	0.05	0.05	0.046	0.055	0.042	0.016	0.03	0.045
Zn, t	-	-	-	0.06	0.05	0.04	0.03	0.03	0.03
^{226}Ra,d pCi/L	1.7	5.2	8.3	7	7.5	6.5	1	2.6	6.7
U, diss	3.9	6.2	8.4	4.3	10	8	1.7	5	8.6

[1] Conductivity in μS/cm; "d" = dissolved; "t" = total; hardness and alkalinity as $CaCO_3$

TABLE 5-5
Conductance and pH with Depth from Pit 3, Midnite Mine

Depth (m)	pH	Conductance (μS/cm, 25°C)
1 May 1984		
3	4.6	2220
10	4.6	2220
20	4.7	2870
32	4.7	2870
42	4.7	2870
6 September 1984		
3	4.5	3000
25	4.5	3000
45	4.5	3050
55	4.5	3050
67	4.6	3150

TABLE 5-6
Pit 3 Water Chemistry with Depth, Midnite Mine

Parameter (mg/L unless noted)	1 May 1984			6 September 1984		
	0.9 m	6.1 m	12.8 m	0.9 m	13.7 m	20.4 m
Cond, μS/cm, 25°C	2220	2870	2870	3000	3050	3150
pH	4.6	4.7	4.7	4.5	4.5	4.6
Temp, °C	7.4	7.9	6.8	17.3	14.1	11.9
Oxygen, diss	10.5	9.6	9.8	8.4	8.2	4.8
COD	82	-	83	140	140	140
Hardness, $CaCO_3$	1100	1300	1300	1300	1100	1600
Acidity, total, $CaCO_3$	-	-	-	2.5	2.5	2.0
Ca, diss	240	270	270	290	250	370
Mg, diss	130	150	150	130	110	170
Na, diss	59	160	160	120	120	130
K, diss	2.6	3.0	3.2	3.3	3.1	3.3
Alkalinity, total, $CaCO_3$	<1	<1	<1	-	-	-
SO_4, diss	1500	1800	1900	2000	2200	2200
Cl, diss	3.3	5.4	5.4	3.8	4.0	4.0
F, diss	1.2	1.1	1.1	1.4	1.5	1.6
SiO_2, diss	23	20	19	22	20	17
Dissolved Solids	2330	2870	2910	3050	3110	3150
Suspended Solids	12	3	7	1	1	1
NO_2 as N	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
NO_2+NO_3 as N	7.7	8.9	8.7	7.7	7.0	8.0
P, total, as P	<0.01	0.01	0.01	<0.01	<0.01	<0.01
Al, diss	30	-	20	35	41	40
Al, total	31	-	20	30	36	31
Be, diss	0.02	0.01	0.02	0.03	0.03	0.02
Cd, diss	0.040	0.041	0.022	0.100	0.100	0.100
Cd, total	0.043	0.042	0.039	0.050	0.060	0.060

Parameter (mg/L unless noted)	1 May 1984			6 September 1984		
	0.9 m	6.1 m	12.8 m	0.9 m	13.7 m	20.4 m
Cu, diss	0.21	0.15	0.15	-	0.30	0.30
Cu, total	0.22	0.16	0.17	0.23	0.24	0.19
Fe, diss	0.15	0.11	0.10	0.18	0.13	0.10
Fe, total	0.19	0.10	0.09	0.13	0.11	0.09
Mn, diss	61	65	67	80	84	85
Mn, total	61	65	64	76	80	81
Ni, diss	1.10	0.97	0.86	-	1.00	-
Ni, total	1.20	1.10	0.83	1.10	1.00	1.00
Sr, diss	1.6	2.0	2.0	1.7	1.8	1.8
Zn, diss	2.4	1.8	1.8	2.8	3.1	2.8
Zn, total	2.5	1.8	1.8	2.7	2.9	2.6
^{226}Ra, diss, pCi/L	23	22	20	38	33	47
^{222}Rn, diss, pCi/L	-	-	-	-	-	1240
U, diss	16.0	14.0	14.0	4.3	3.7	4.5

TABLE 5-7
Conductance and pH with Depth from Retention Pond, Midnite Mine

Depth (m)	pH	Conductance (μS/cm, 25°C)
2 May 1984		
3	3.7	7130
8	3.5	8630
12	3.8	9630
5 September 1984		
3	3.6	8600
7	3.7	8700
11	3.7	9200

TABLE 5-8
Retention Pond Water Chemistry with Depth, Midnite Mine

Parameter (mg/L unless noted)	1 May 1984		6 September 1984	
	0.9 m	3.7 m	0.9 m	3.4 m
Cond, μS/cm, 25°C	7130	9630	8600	9200
pH	3.7	3.8	3.6	3.7
Temp, °C	11.2	12.9	19.5	19.5
Oxygen, diss	13.0	0.8	11.6	0.5
COD	670	850	-	-
Hardness, $CaCO_3$	4400	5700	5600	5800
Acidity, total, $CaCO_3$	76	96	41	41
Ca, diss	440	480	590	490
Mg, diss	810	1100	1000	1100
Na, diss	37	72	45	50
K, diss	4.4	4.3	4.9	4.0
Alkalinity, total, $CaCO_3$	<1	<1	-	-
SO_4, diss	8000	10000	10000	12000
Cl, diss	4.4	5.6	2.8	2.8
F, diss	11	12	<0.1	<0.1
SiO_2, diss	84	91	99	79
Dissolved Solids	12300	15600	15200	15700
Suspended Solids	21	41	18	26
NO_2 as N	<0.1	<0.1	<0.01	<0.01
NO_2+NO_3 as N	4.3	2.4	9.7	7.0
P, total, as P	0.01	0.01	0.01	0.01
Al, diss	490	710	510	470
Al, total	540	720	660	640
Be, diss	0.18	0.24	0.11	0.13
Cd, diss	<0.001	-	0.500	0.500
Cd, total	<0.001	0.550	0.450	0.500

Parameter (mg/L unless noted)	1 May 1984		6 September 1984	
	0.9 m	3.7 m	0.9 m	3.4 m
Cu, diss	0.002	3.300	3.300	3.500
Cu, total	2.800	3.500	7.000	7.000
Fe, diss	4.3	4.8	3.2	4.5
Fe, total	4.7	5.9	3.1	4.5
Mn, diss	500	630	630	640
Mn, total	530	650	600	610
Ni, diss	0.056	-	11.0	11.0
Ni, total	9.90	7.80	25.0	18.0
Sr, diss	0.95	0.93	0.92	0.89
Zn, diss	26.0	33.0	33.0	34.0
Zn, total	28.0	34.0	32.0	34.0
^{226}Ra, diss, pCi/L	55	48	63	61
U, diss	160.0	180.0	160.0	180.0

5.9 Bunker Hill Mine, Idaho, USA

The underground Bunker Hill Lead-Zinc Mine is located near Kellogg and the Coeur d'Alene River in northern Idaho. Mining began in 1885 and, by 1979, included over 240 km of workings to depths of approximately 1.6 km, representing roughly 180 km^3 of rock. The yearly rate of new drifting and drilling were roughly 6 km and 18 km, respectively. The rock is highly faulted and fractured metamorphic quartzites with interbedded argillites.

Annual precipitation in the area was approximately 0.76 m, mostly as snow. Discharge from the mine was 0.17 m^3/s at an average pH of 3.3 (Trexler et al., 1974) and 0.13-0.16 m^3/s at pH 4.0-4.7 with minimum of 3.3 (Trexler, 1979). The sources of this flow were: surface-water inflow through fractures and faults (69.1%), groundwater inflow (19.0%), and drainage of sand backfill (11.9%).

Portions of the underground workings have been backfilled with sand since 1961 and with rock from drifting prior to 1961. Some of the drift rock contained elevated levels of lead, zinc, and pyrite, which contributed to acidic drainage and metal leaching from the upper levels.

Trexler et al. (1974) graphically reported more than 400 pH measurements on a simplified two-dimensional vertical profile of the mine (Figure 5-9). This showed that acidic pH was encountered most often in the upper levels and near the No. 2 Shaft, although at least one acidic pH was found on most levels.

The availability of water to flush reaction products of acid generation and metal leaching was identified as the major control on water chemistry. Accordingly, reclamation activities focussed on identification and elimination of surface-water access to the workings.

The main drainage pathways for the mine are the Cherry Ditch which drains the upper levels and the Kellogg Tunnel. Hydrographs for the Ditch and Tunnel, and for Milo Creek which drains a portion of the land surface above the mine, showed a close correlation (Figure

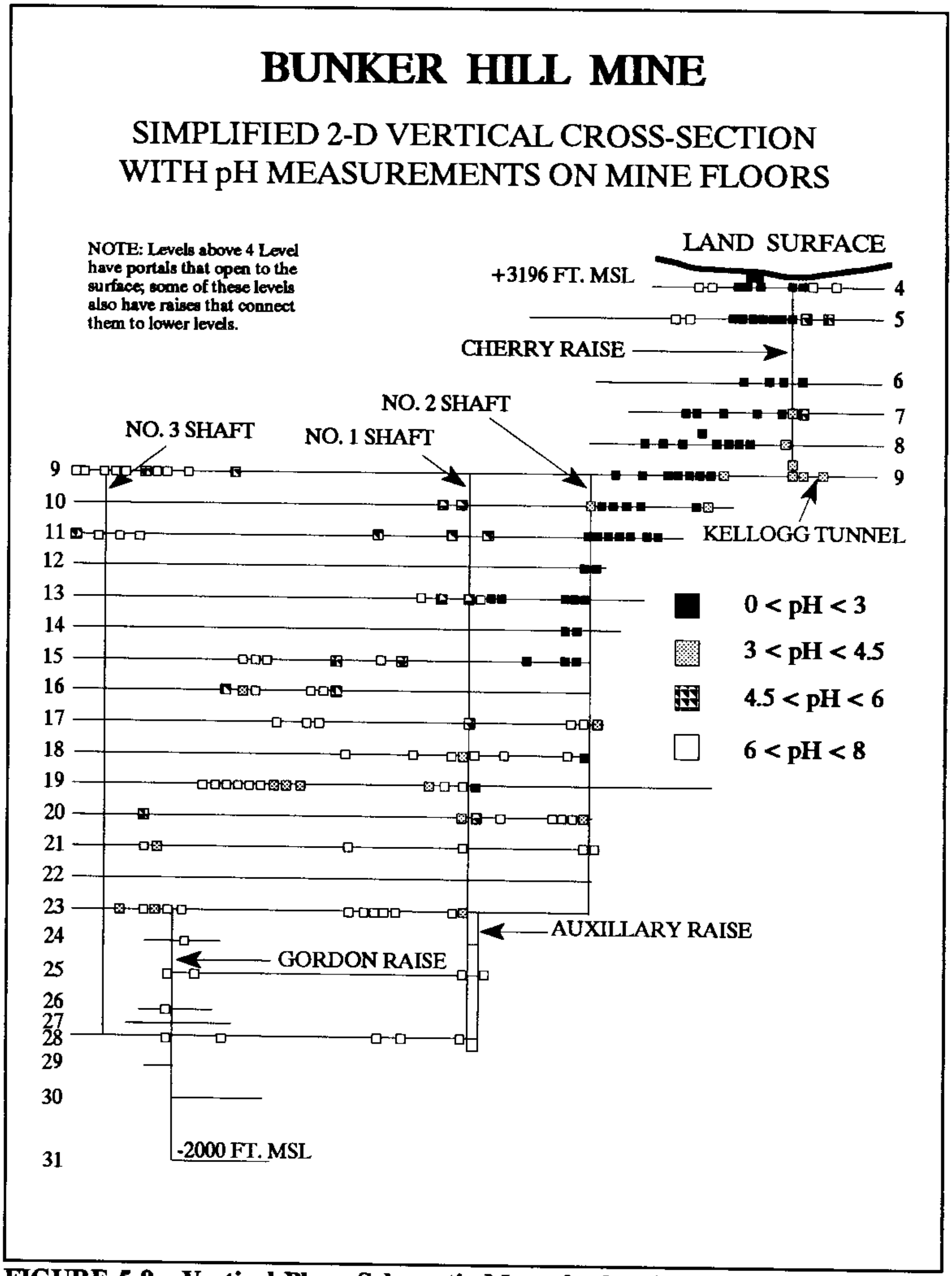


FIGURE 5-9. Vertical-Plane Schematic Map of a Portion of the Bunker Hill Mine (adapted from Trexler et al., 1974).

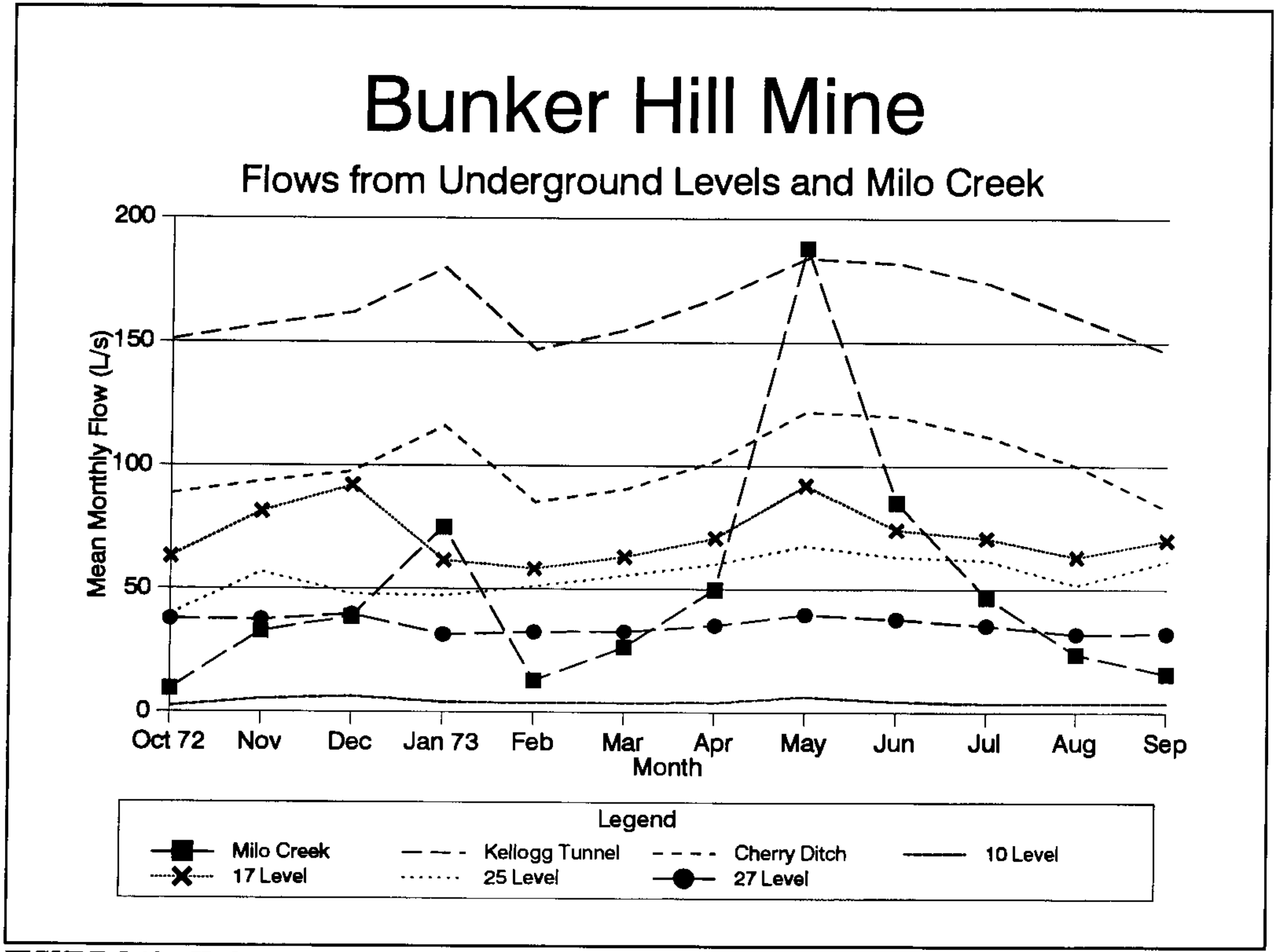


FIGURE 5-10. Hydrographs of Flows in Underground Levels and Milo Creek, Bunker Hill Mine (adapted from Trexler et al., 1974; Trexler, 1979).

5-10), although variations were more pronounced in Milo Creek. General, but more subdued, correlations were noted for deeper levels. The correlations indicated the underground flow was hydraulically connected to surface water with the connection decreasing with depth. Potential connections were then identified: (1) fractures between the surface and the upper workings due to surface subsidence, (2) faults between the surface and the upper workings, and (3) a drift and stope on 4 level extending close to surface.

The subsided "Caving Area", measuring 60 m in diameter and 12 m deep, apparently received water from tributaries of Milo Creek. During winter months, a "vapor cloud" of mine air could be seen issuing from this area. A dye tracer test in the tributaries through the Caving Area then showed that creek water reached 4 level within 15 minutes and 5 level in 30 minutes. Continuous flow monitoring in January 1973 indicated that at least 99% of drainage water was caught by the depression and surrounding fractures.

In another portion of Milo Creek, constant dye-release tests in May and July 1973 indicated that creek water was lost into fractures and a fault zone. Losses ranged from 29-76% of streamflow (0.028-0.093 m^3/s). In the adjacent Deadwood Creek, dye tracer tests showed that creek water required 12-36 hours to flow to 9 level, located 370 m below and 610 m laterally away from the creek.

5.10 Berkeley Pit, Montana, USA

This pit is located in southwestern Montana and is reportedly the second largest pit in the USA (Davis and Ashenberg, 1989). It has a maximum depth of 542 m with lateral dimensions of 1.8 km by 1.4 km. The lower 38 m had been filled to a flat surface due to the collapse of wall material. Underground workings are connected to the pit.

In 1987, the lower 340 m of the pit had been flooded and the water level was rising 22 m/yr due to inflow of groundwater and surface waters (David and Ashenberg, 1989). Total inflow was estimated at 28,700 m^3/day including an estimated loss to evaporation of 300 m^3/day.

Depth-specific samples from the 130 m water column indicated many parameters generally increased with depth including pH ($\sim$3), suspended solids, conductance, aluminum, calcium, total and ferrous iron, and sulfate (e.g., Table 5-9). On the other hand, dissolved oxygen and Eh decreased with depth. This indicated the presence of chemoclines and a thermocline, and the development of reducing conditions.

Due to metal leaching, concentrations of cadmium, copper, and zinc reached concentrations of 1.9, 214, and 500 mg/L, respectively. Concentrations of iron, calcium, aluminum, potassium, and sulfate were apparently regulated by mineral solubility of ferric hydroxide, gypsum, jurbanite, and jarosite. The presence of reduced sulfur (sulfide), which may lower metal concentrations through precipitation of relatively insoluble sulfide minerals, was not mentioned and thus the pit may not be meromictic (Section 3.3) or sufficiently reducing.

Changes in appearance and chemistry of the shallowest pit water was noted seasonally. For example, formation of ice cover resulted in the reduction of ferric iron to ferrous iron, with the reverse reaction occurring during summer months. Below a depth of 30 m, water chemistry was relatively steady.

5.11 D Pit, Amok Minesite, Saskatchewan, Canada

The Amok uranium mine is located near Cluff Lake in northwestern Saskatchewan. One relatively small, shallow orebody at Amok was named "D", and mining of it began in April of 1980 (Saskatchewan Environment, 1990 and 1993). Excavation of the pit was temporarily halted in October of that year and began again in May 1981. By September 1981, mining was completed after removal of approximately 107,800 tonnes of ore grading roughly 7% U_3O_8. The final pit had a lateral extent of 200 meters by 90 meters with a maximum depth of 23 meters.

TABLE 5-9
Water Chemistry in the Flooding Berkeley Pit
(from Davis and Ashenberg, 1989)

Depth (m)	pH	Eh (V)	Fe^{2+} (mg/L)	Fe^{3+} (mg/L)	Total Fe (mg/L)
0	2.76	0.82	0.25	196	196
1	2.72	0.72	60	142	202
3	2.84	0.64	262	14	276
15	2.95	0.57	622	28	650
31	3.08	0.50	900	10	910
66	3.15	0.46	938	-	854
100	3.15	0.47	944	14	958
130	3.14	0.46	962	24	986

Dewatering of D Pit continued until November 1982, when filling began. However, during the spring of the following year, a nearby creek overflowed and filled D Pit in 24 hours. A berm was later built to prevent creek water from entering the pit. Monitoring of the flooded pit began in 1983. Results through 1988 are reported by Saskatchewan Environment (1990) and partial results through 1992 are reported by Saskatchewan Environment (1993).

D pit lies in a faulted metamorphic zone which reportedly contains uraninite, selenides, gold, tellurides, nickel-sulfur arsenides, jordiste, minor pyrite, and chalcopyrite. The north wall of D Pit above water level exposes 10 meters of rock composed of garnet-rich aluminous gneiss with seven fault zones. The dominant fault zone is F-1 which contains a one-meter thickness of clays, slightly mineralized breccias, and gersdorffite and is assumed to extend to the base of the pit. F-1 has been identified as a source of metals for pit water.

A portion of the east side of the pit is covered with dumped waste rock, which has also been identified as a source of metals to pit water. On the south and west sides, no rock is exposed above the water level. Iron precipitant on the overburden in the southeast wall indicates groundwater is entering the pit there and apparently exiting through the southwestern wall of overburden.

The cold, boreal climate at Cluff Lake produces cool, short summers and long, cold winters with relatively dry subhumid moisture. Mean monthly temperatures are usually below zero from November to April. Total annual precipitation is approximately 400 mm. Evaporation occurs primarily from May through September, and potential evaporation exceeds total precipitation based on data from an evaporation pan. The average precipitation and evaporation represent about 3.8% and 6.5% of the pit volume, respectively.

Ice cover usually appears around October 20-30 and open water usually appears around April 17 to May 9. Snowmelt from the surrounding watershed causes a volume of water exceeding 1000 m^3 to flow into the pit, causing the pit to overflow.

All surface water and groundwater around D Pit is generally westward towards Cluff Lake. Most of the groundwater is believed to move through the shallow overburden due to its elevated hydraulic conductivity (1 to 5×10^{-4} m/s) over the fractured bedrock (2×10^{-7} to less than 2×10^{-9} m/s). The relationship between groundwater and pit water is discussed in more detail later in this section.

Based on mine surveys, pit volumes and lateral areas were determined for specific depth intervals (Table 5-10). At the overflow elevation of 332.5 m, the pit contained an estimated 177,400 m^3.

Monitoring of pit-water chemistry commenced in March 1986 at five sampling locations. At each location, depth-specific samples were collected at five-meter intervals. The depth-specific monitoring ceased in December 1988 and was replaced by periodic sampling of the top of the water column. Beginning in March 1992, depth-specific sampling was again initiated.

TABLE 5-10
Depth-Specific Areas and Volumes for D Pit
(from Saskatchewan Environment, 1993)

Elevation (masl)	Depth (m)	Area (m^2)	Volume (m^3)	Cumulative Volume (m^3)	Percent of Total Volume
332.5	0.0	16,565	8,081	177,383	4.6%
332.0	0.5	15,561	36,190	169,302	20.4
329.5	3.0	13,191	56,482	133,112	31.8
324.5	8.0	9,402	40,589	76,629	22.9
319.5	13.0	6,833	24,867	36,040	14.0
314.5	18.0	3,113	10,412	11,172	5.9
309.5	23.0	1,051	632	760	0.4

In addition to pit water, samples were also collected of runoff from the nearby waste rock into the pit, the nearby Boulder Creek, groundwater located 500 m upgradient and 400 m downgradient of the pit, water and rock within the F1 fault, and pit-bottom sediment.

Some of the conclusions reached by Saskatchewan Environment, based on the pit-water chemistry, were as follows. First, a thermocline formed during early spring and summer. As winter approached an ice cover forms and the thermal gradient approaches equilibrium across the water column. Second, major ions, metals and radionuclides concentrated at depth after spring runoff and rainstorm events. Third, the sources of the contaminants were likely seepage from the waste rock dump and dissolution of materials in the faults and wall rock along the north side of the pit. Another potential source that required further study was the possible resuspension of bottom sediments into the water column due to spring and fall turnover.

A sample of bottom sediments obtained with an Eckman dredge showed that the sediment was predominately composed of silicon, aluminum, and iron (Table 5-11). Under the assumptions that silicon occurs as silicon dioxide (quartz or an amorphous phase), and aluminum and iron occur as hydroxides ($Al(OH)_3$ and $Fe(OH)_3$), these three mineral phases would comprise

95% of the bottom sediment and would likely indicate the presence of significant mineral precipitants. The relatively high moisture content and loss on ignition suggested the mineral phases are amorphous or associated with water.

The presence of mineral precipitants was further supported by an analysis of suspended sediment (Table 5-11), which showed enrichment of arsenic and uranium over the underlying sediments. Because these two elements are more often regulated by solubility and redox conditions than radium, their higher levels suggested they may be precipitating in, and settling from, the water column. In agreement, water-column analyses also revealed a redox "front" at a depth of 19 m in the pit.

TABLE 5-11
Chemical Analyses of Bottom and Suspended Sediment in D Pit
(from Saskatchewan Environment, 1993)

Parameter	Concentration	Parameter	Concentration
Bottom Sediment			
Al (%)	4.6	Si (%)	35.2
As (μg/g)	192	Zn (μg/g)	60
Carbon, inorg (%)	0.009	L.O.I (%, @ 550°C)	4
Carbon, organic (%)	0.80	Moisture (%)	47.8
Cu (μg/g)	30	Pb 210 (total, Bq/g)	4.0
Fe (%)	3.1	Ra 226 (total, Bq/g)	4.5
Mn (μg/g)	300	Th (μg/g)	30
Mo (μg/g)	14	U (total, μg/g)	279
Ni (μg/g)	170		
Suspended Sediment (just above pit bottom)			
As (μg/g)	1090	U (total, μg/g)	922
Ra 226 (total, Bq/g)	7.0		

Prior and during mining of D Pit, piezometers were installed about 500 m upgradient of D Pit, on the southwest perimeter, and 400 m downstream. The two piezometers on the southwest perimeter were likely located in the water-table drawdown caused by excavation and pumping of the pit during mining (Section 5.1). As a result, these two piezometers probably reflected the chemistry of groundwater drawn into the pit until it filled in April of 1983.

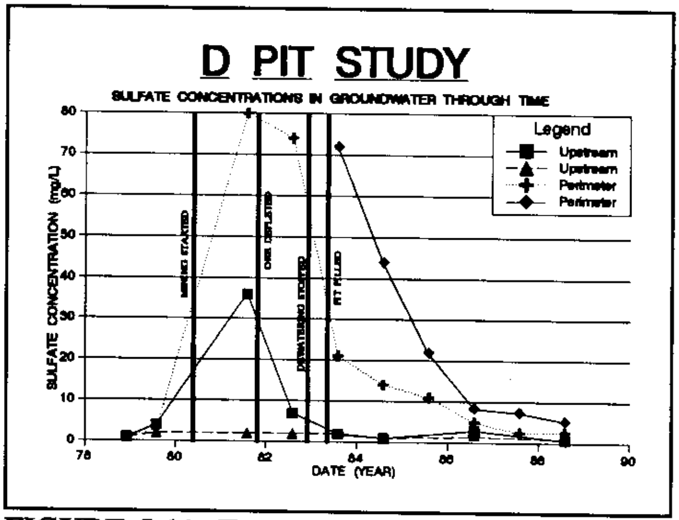


FIGURE 5-11. Temporal Trend of Sulfate in Groundwater Around D Pit (500 m Upgradient and Pit Perimeter).

A comparison of sulfate concentrations between the upstream and perimeter piezometers show the effect of mining (Figure 5-11). Shortly after the pit flooded in April 1983, the sulfate concentrations began decreasing to background values. The source of the sulfate could be Boulder Creek, since the two piezometers lie between the creek and the pit, or oxidation of sulfide-bearing minerals. However, because Boulder Creek reportedly carried less than 10 mg/L of sulfate, the more likely source was mineral dissolution. Interestingly, this impact on water chemistry occurred outside the pit perimeter and thus either groundwater was still migrating away from the pit during mining or, more likely, the drawdown of the water table and the resulting entry of oxygen reached several meters beyond the pit perimeter. This was also noted in Section 5.4 of this review.

There was a 1-2 year delay in the decrease of sulfate in the farther perimeter piezometer (roughly 10 m from the pit edge) relative to the closer piezometer on the pit edge. Additionally, two of three piezometers located 400 m downgradient of the pit showed a peak and then a decrease in sulfate about 3 years after the pit-edge piezometer. With the reported land-surface gradient, the highest reported hydraulic conductivities for the overburden, and an assumed porosity of 0.3, the downgradient piezometers should respond to non-retarded species about 4 years after the pit-edge piezometer, in agreement with the observed time. Thus, these downgradient piezometers were apparently installed in overburden. However, the 1-2 year delay in the perimeter piezometer suggested it was completed in a strata of relatively low permeability of about 2-5 x 10^{-5} m/s. This range was less than overburden, but greater than rock.

5.11 Myra Falls Operations, British Columbia, Canada

As part of closure planning, Northwest Geochem (1992) conducted a detailed hydrogeologic assessment of the upper levels of the Lynx Underground at the Myra Falls Operations (Figure 5-12). Each of eight upper levels were examined once to measure pH and estimate flow from stopes, drillholes, fractures, and along the mine floor (e.g., Figure 5-13), similar to work at the Bunker Hill Mine (Section 5.9).

On selected levels, weirs were installed on the mine floor and periodically monitored for flow and water chemistry (Figure 5-14). This allowed mass-balance calculations along the length of the mine floor to locate specific areas of diffuse (non-apparent) gain and loss of water (Figure 5-15). Based on the diffuse gains of water on several levels, a bulk hydraulic conductivity of $1x10^{-6}$ m/s was calculated for the rock under an assumed hydraulic gradient of 1.0. Within 100 m of the portals, inside the perimeter of the pit that lies above the workings, diffuse gains suggested that hydraulic conductivity increased by one order of magnitude.

Interestingly, this study highlighted the potential for erroneous interpretations based solely on flow and chemistry at a portal (Figure 5-16). The Level 8 Portal suggested that, as flow increased during early winter months, there was a flushing of accumulated acidity from the level and pH then recovered as flow increased. In reality, most of the flow originating on Level 8 was pH-neutral through the period, but each of two stopes bottoming on Level 8 were generating flows at volumes equivalent to those monitored at the portal. In other words, flow was twice as great on portions of Level 8 than indicated by portal flow. The loss of flow between the stopes and the portal was primarily down a raise that connected to the lower Level 9 just before

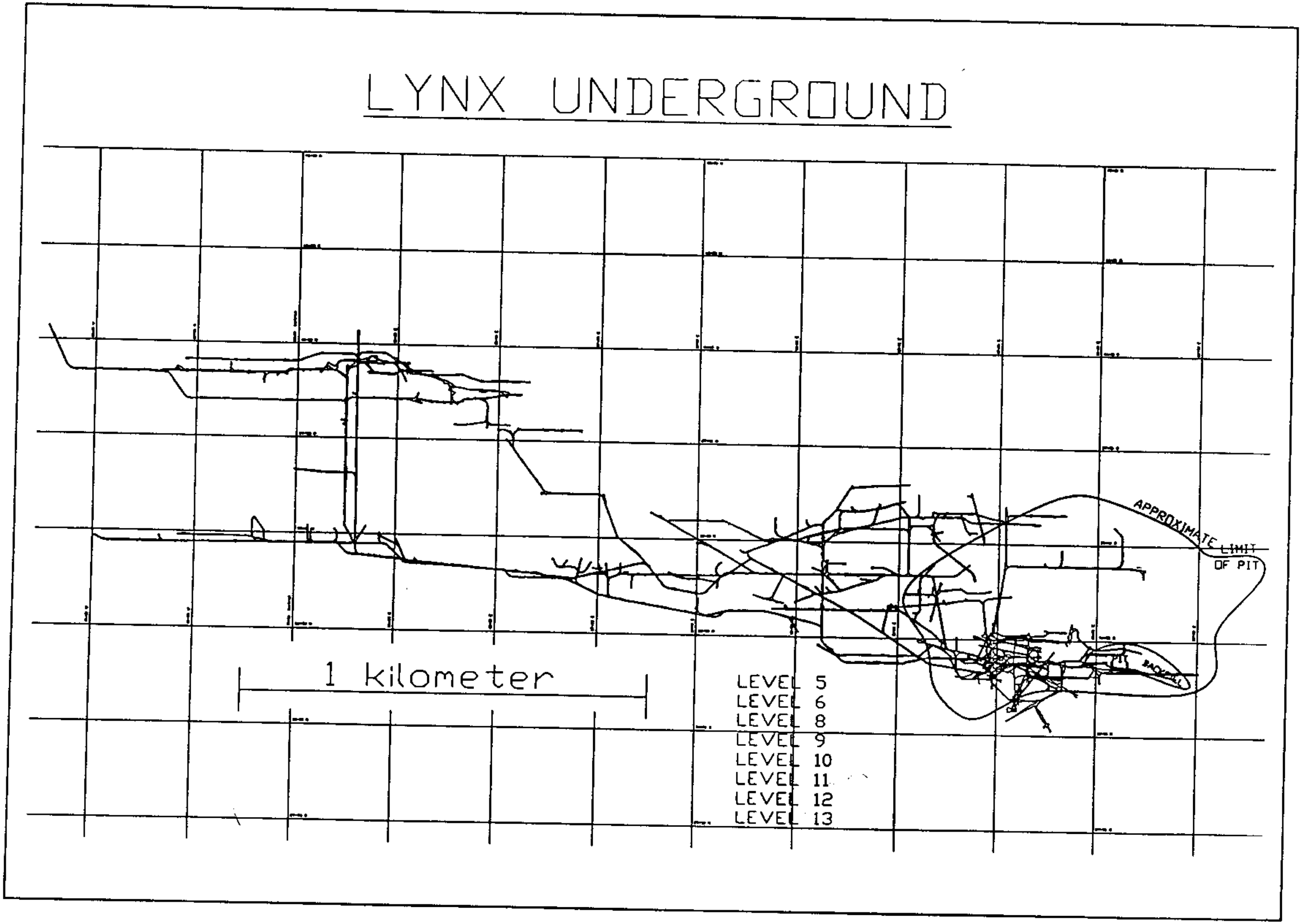


FIGURE 5-12. Composite Map of Upper Levels of the Lynx Underground, Myra Falls Operations.

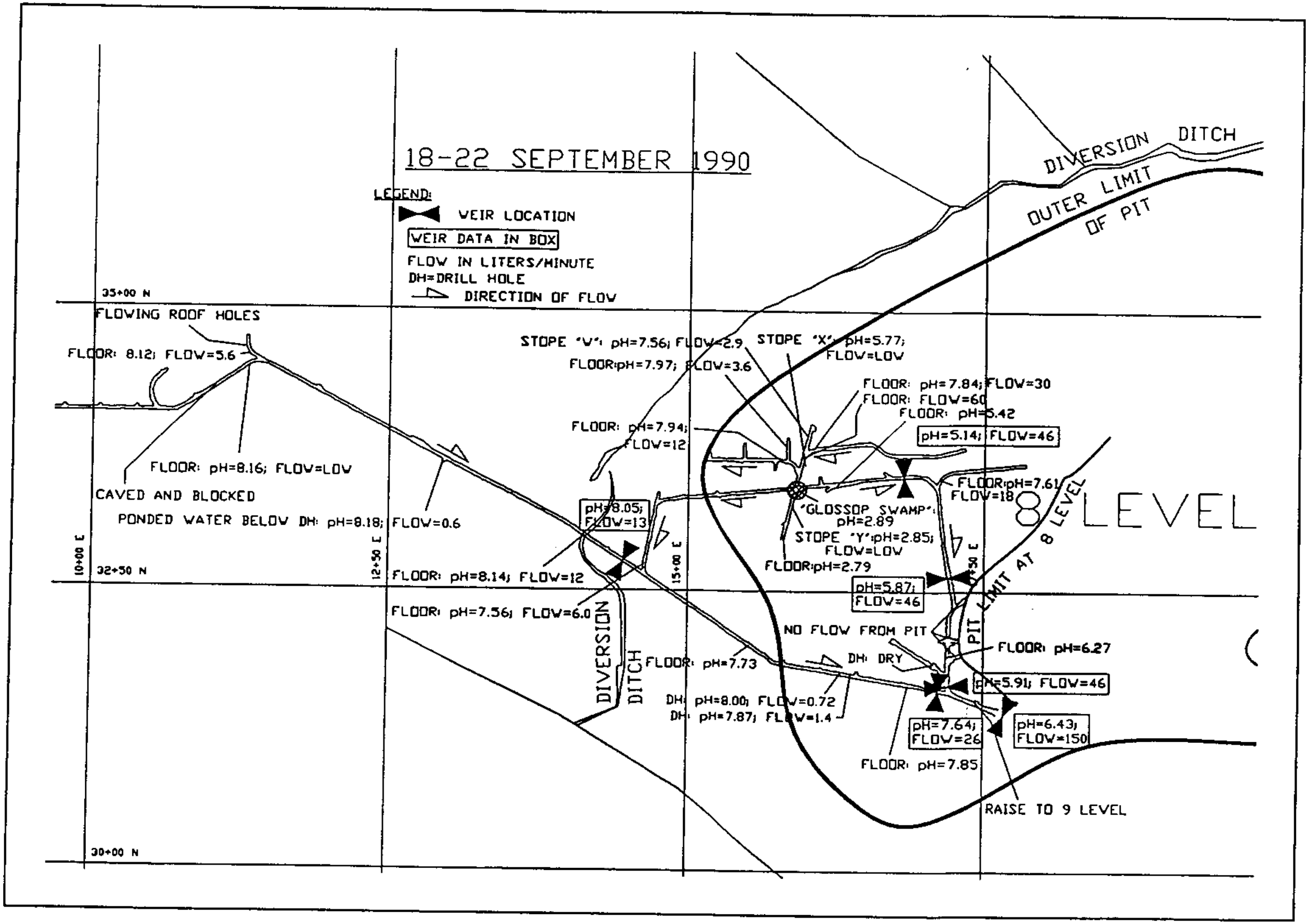


FIGURE 5-13. Monitoring Campaign for Flow and pH on Level 8, Myra Falls Operations.

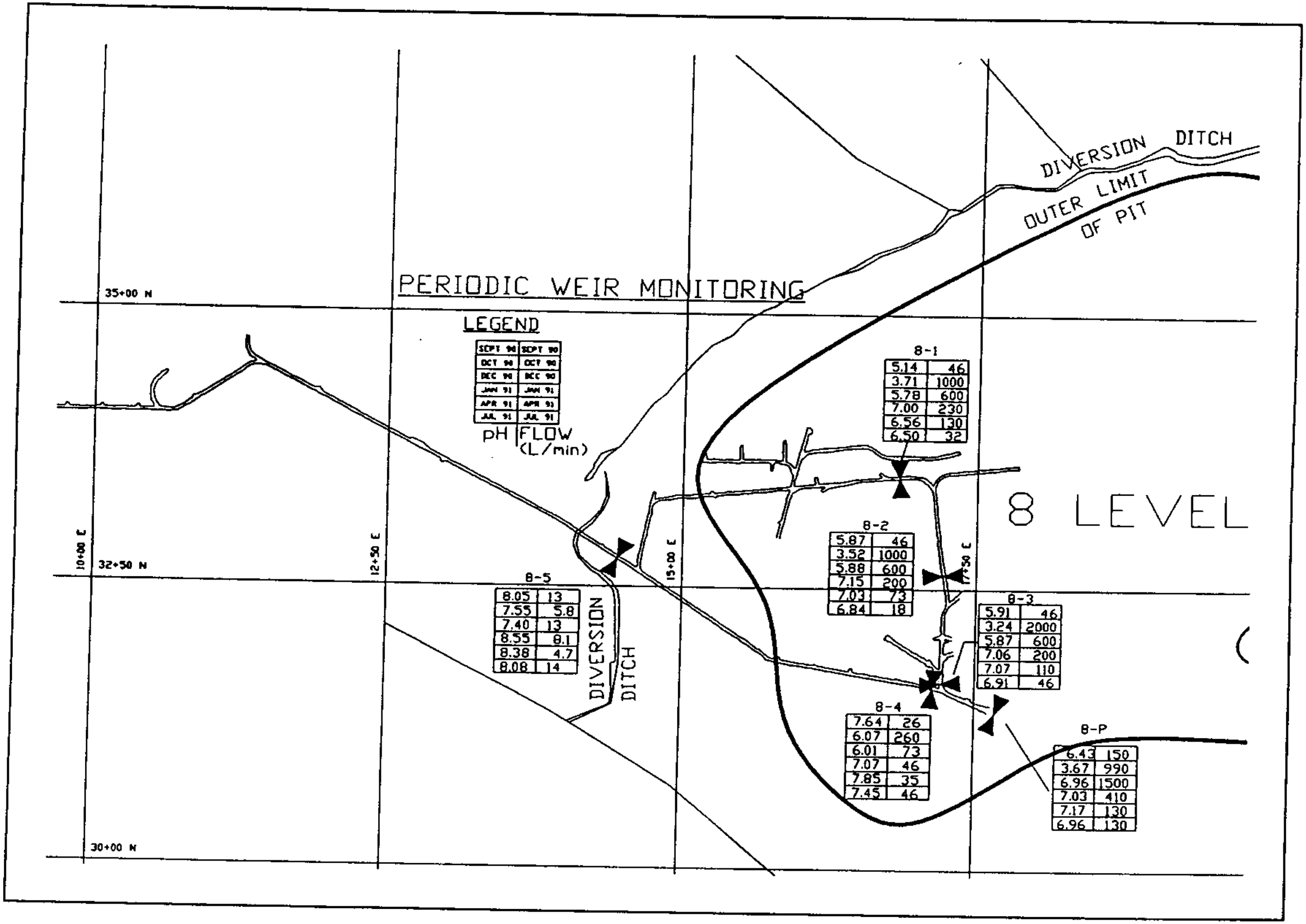


FIGURE 5-14. Periodic Monitoring of Flow and pH at Weirs on Level 8, Myra Falls Operations.

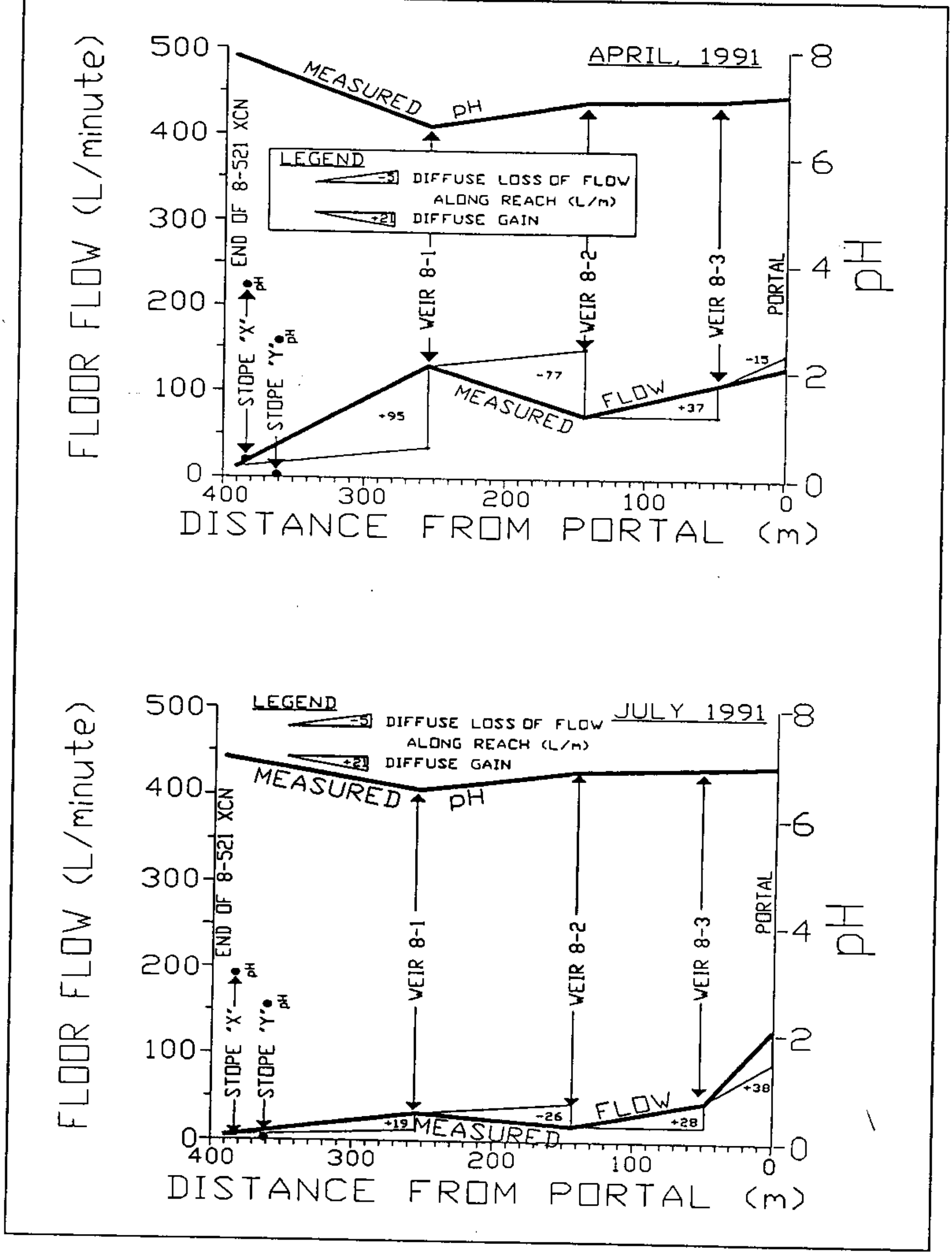


FIGURE 5-15. Non-visible Gain and Loss of Minewater Along Level 8 Mine Floor, Myra Falls Operations.

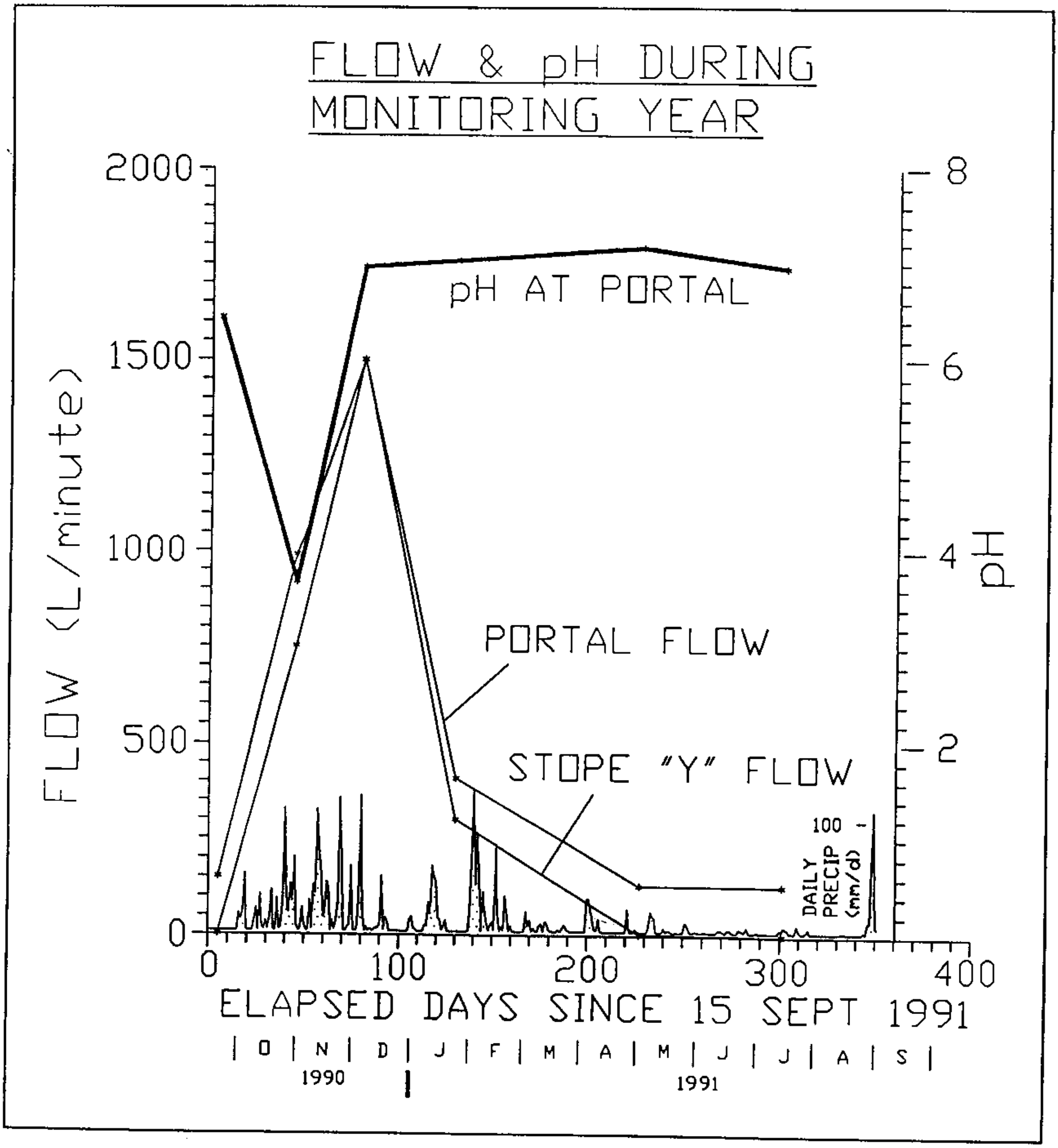


FIGURE 5-16. Monitoring of Flow and pH at Level 8 Portal, Myra Falls Operations.

the portal. The acidic water from Stope Y mixed with, and overwhelmed, the pH-neutral water as it flowed towards the portal.

As portal flow and each stope flow continued to increase towards 1500 L/minute in December (Figure 5-16), most of the acidic flow from Stope Y (pH 2.33) reversed and poured down a raise further back in the level, causing pH to rise at the portal. This hydraulic behavior could not have been deduced from the portal data alone.

Based on water and chemical balances, and of spatial distributions of minewater pH on various levels, Northwest Geochem concluded that there were two primary causes of acidic drainage from the workings. First, stagnation of water on a level allows it to accumulate acidity and eventually become acidic. The most acidic pH was measured on an abandoned level with no visible flow. Second, there was a zone of rock with a lateral extent of roughly 10,000 m^2 that was often associated with depressed pH, even in moving water. This zone extended diagonally downwards from one level to the next. The aforementioned acidic Stope Y was located in this zone. Such a zone may also exist at the Bunker Hill Mine based on the pattern of $pH < 3$ in Figure 5-9.

6. CONCLUSION

This report, entitled Literature Review and Conceptual Models, is one of four documents describing the MINEWALL 2.0 technique and computer program for predicting water chemistry in pits and underground workings ("mines"). These predictions can be made for both Operational and Closure Phases of a mine.

Based on case studies and general theory, conceptual models of water movement through mines were developed in Section 2. The conceptual models for chemistry associated with flow were then developed in Section 3. Simplified equations for predicting minewater chemistry were developed in Section 3.1, and complicating factors were clarified in Sections 3.2 and 3.3. Flexibility was emphasized in all conceptual models so that the models, the technique, and the computer program could be adapted to virtually any minesite.

Although the conceptual models were based on many decades of previous work, they were presented first so that various lessons learned from previous work would be more obvious as they were presented in Sections 4 and 5. In particular, Section 4 briefly described three field studies closely adhering to the MINEWALL technique. These studies were discussed in the companion report, Application of MINEWALL 2.0 to Three Minesites. Section 5 described many studies, but none contain all information to perform a reasonably detailed simulation with MINEWALL.

In addition to this Literature Review, there are three other related reports. These are (1) the User's Manual for the computer program, (2) the Programmer's Notes and Source Code, and (3) Application of MINEWALL 2.0 to Three Minesites. These reports should be consulted for additional details not presented in this document.

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