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ANVIL RANGE MINING COMPLEX -

INTEGRATED COMPREHENSIVE ABANDONMENT PLAN

Volume 2 of 3

Site Characterization

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Preface

This document contains an Integrated Comprehensive Abandonment Plan (ICAP) for the Anvil Range Mining Complex near Faro, Yukon Territory. This ICAP is a requirement by the water licences IN #89-001, Part D Section 8(a) (Faro minesite) and IN #89-002, Part G Section 4 (Vangorda Plateau minesite) issued by the Yukon Territory Water Board to Anvil Range Mining Corporation.

The Integrated Comprehensive Abandonment Plan presented in this document consists of three Volumes:

- Volume 1: Background
- Volume 2: Site Characterization
- Volume 3: Closure

Volume 1 provides background information relevant to the closure plan including:

Chapter 1. Anvil Range Mining Complex

Chapter 2. Closure Plan Philosophy

Chapter 3. Regional Setting

Volume 2 describes the physical and geochemical aspects of the two minesites including:

Chapter 4. Faro Mine - Site Characterization

Chapter 5. Vangorda Plateau Mine - Site Characterization

Chapter 6. Minesite Water Chemistry

Volume 3 discusses all issues related to minesite closure including:

Chapter 7. Faro Closure Plan

Chapter 8. Vangorda Plateau Closure Plan

Chapter 9. Post-Closure Water Management and Treatment

Chapter 10. Post-closure monitoring and Maintenance

Chapter 11. Schedule and Costing

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4. Faro Mine - Site Characterization

The Faro minesite consists of a mill site, three open pits (Zone 1, 2 and 3), an underground working, several rock dumps, a series of tailings impoundments, and several water management structures including a freshwater reservoir (Figure 4-1). The various mine components are described in more detail below.

Elevations used in Pit operations at Faro have historically been expressed in feet, and are often but not always given with respect to "Mine Datum". Elevations used in Tailings Pond operations at Faro have historically been expressed in meters, and are often but not always given with respect to "Down Valley Datum". The benchmarks for elevations used in the Pit and the Tailings Area are not the same (the differential being approximately 3.2 feet). The mean sea level datum (sometimes incorrectly referred to as the "UTM datum") is based on the North American Datum of 1927 (NAD 1927), which is also not consistent with either of the two benchmarks (see Appendix F for more details).

For clarity in this report all elevations quoted in the text of this report are given in metres above mean sea level (m AMSL). Elevations shown in parentheses in discussion of the Faro pit area are the equivalent elevations in feet above mean sea level (ft AMSL).

Note, that the same consistency does not apply to the drawings produced for the Faro minesite. In all drawings which are not based on the mean sea level datum (NAD, 1927) the respective datum is noted in the figure. The reader is advised to confirm the elevation data shown on the drawings before comparing elevations. The same caution applies to older technical reports and maps tabled before the Yukon Territory Water Board by Anvil Range Mining Corporation, its consultants and its predecessors (Appendix F).

The following conversion rules apply:

- to convert from mean sea level to mine datum add 33.3m or 109.2ft.
- to convert from mean sea level to Down Valley datum add 32.3m or 106.0 ft.

4.1 Mine Workings

4.1.1 History of Development

The Faro mine was developed in the late 1960's. Stripping of the pit began in 1968 and ore was milled on a commercial basis beginning in September 1969. The first pit mined was Zone 1 (in some documents the Faro pit zones are referred to with roman numerals - these are the same zones described herein) and waste rock was dumped in the Faro Valley and Northwest dumps (including Mt. Mungly and Fuel tank dumps). The pit was developed as a narrow, northwesterly elongate cut into the hill slope north west of Faro Creek initially. The pit was then broadened to the southwest in the early 1970's, dumping to the west side of the North west dumps and into the west Main Dump. Eventually the pit was expanded to the southeast across Faro Creek by the mid 1970's, following establishment of the initial Faro Creek diversion. The Main dump were used and

northeast dumps were started at that time. Zone 1 was mined into the early 1980's and was essentially completed by Cyprus Anvil. Curragh Inc. mined several small remnants of ore from the pit walls in the 1986 to 1992 period but dumps were internal to the pit. Cyprus Anvil deposited several million tonnes of oxidized ore from Zone 1 (and Zone 2, see below) near the mill.

Zone 2 was mined in the late 1970's and early 1980's, the intermediate dumps were started at that time. It is believed that the initial stripping of oxidized ore, metal enriched overburden and sulphide waste from the Zone 2 pit were deposited on the intermediate dump thus the lower lift of this dump can be expected to contain a significant stored load of soluble metal.

Zone 3 was a down-dropped block of ore which required considerable stripping. This stripping was begun by Cyprus Anvil in the mid 1970's in conjunction with mining of Zone 1 and using the northeast dumps. The major stripping effort by Cyprus Anvil was during the mid 1980's shutdown, the Main and intermediate dumps were in use then. The southeast slot access to the Zone 3 pit was developed at that time. Clean calc-silicate and schist waste from Zone 3 stripping was segregated on top of the east Main Dump for possible use in dam raising at a later date. Waste from Zone 3 stripping was also deposited by Cyprus Anvil in the mined out Zone 2 pit and in the Intermediate dump.

Curragh mined mainly in Zone 3 and considerable stripping was required. The major active dumps during Curragh's tenure at Faro were the Main, Intermediate and Zone 2 dumps. Curragh deposited most of their sulphide waste in a cell on the upper lift of the Intermediate dump but later in the mining also deposited sulphides on the upper lift of the Main dump, on top of the clean calc-silicate and schist placed by Cyprus Anvil. Calc-silicate breccia stripped from Zone 3 was used for the North Fork of Rose Creek rock drain. Schist, calc-silicate and minor intrusive was also used to build the haul road to Vangorda Plateau and a haul road to the mill on the southwest side of the Main and Intermediate dumps. Rock placed in the haul road southeast of the North Fork of Rose Creek was derived from stripping in Zone 3 and was hauled through the south slot access thus is entirely clean, non-sulphide, waste as that was all that was being mined in that part of the pit at the time. Curragh also placed considerable rock in the previously mined portions of the Zone 1 and 3 pits, much of this material was sulphide bearing.

A small extension of Zone 2, called the ramp zone was mined by Curragh in 1986 then backfilled. The Ramp zone was immediately southwest of the southeast slot access to the Zone 3 pit thus the pit wall there between the slot and the Zone 2 extension (i.e. the Ramp Zone) is thin. Sulphide material was also dumped over the southwest pit wall by Curragh with the intention that it would be pushed down into the pit later.

Curragh deposited low grade ore (3 to 5 % Pb + Zn) in two stockpiles, A and C, beside the main haul road from the Zone 1 pit. Curragh process the oxidized ore stockpiled by Cyprus Anvil after screening out the fine fraction of the ore. The oxidized fines still are present near the mill active fresh ore stockpiles.

Curragh mined 1.7 million tonnes of ore from a room and pillar underground mine developed through a portal into the southwest wall of the Faro Pit. All openings into this mine were internal to the Faro pit and are now flooded. Before flooding, Curragh prepared the underground workings for

possible use in water treatment at a later date. Tailings deposition into the mined out Faro Zone 1 and 3 pits began in August 1992 and continues.

Anvil Range has carried out no mining at the Faro site but continues to deposit tailings into the Faro pit.

4.1.2 Geology of the Faro Minesite

4.1.2.1 Introduction

The Faro minesite is the area bounded by the main stem of Rose Creek, the North Fork of Rose Creek, Next Creek and the southwest contact of the Anvil Batholith. The area is unusual in that the grade of metamorphism is higher than most other mining areas in the district, though typical for areas near the Anvil Batholith contact.

4.1.2.2 Surficial Geology

On the uplands adjacent to the Rose Creek Valley, bedrock is discontinuously covered with a veneer of morainal and colluvial deposits which increase in thickness towards Rose Creek. The valley is filled with a complex assemblage of fluvial, glacial and lacustrine deposits.

Drilling results from past studies indicate that bedrock in the valley bottom is overlain in most places by very dense glacial till, apparently continuous with till deposits found over bedrock on the valley slopes. In the valley bottom the till is overlain by a complex of fluvial, and glaciofluvial sands and gravels. Within the sand and gravel are irregular deposits of silts, sands and gravels which may have been deposited in lacustine or flood plain environments. These silty deposits appear to be discontinuous which may be a result of post deposition glacial or fluvial action. Maximum thicknesses of the sand and gravel are 35 to 40 m. Prior to mine development, Rose Creek had incised this deposit by up to 10 m in the region of the tailings impoundments.

4.1.2.3 Bedrock Geology

The rocks in the Faro mine-site were metamorphosed in the amphibolite facies of regional metamorphism. The result is that schists are more important at Faro than elsewhere and the calc-silicate gneiss, a rock type not seen at the other sites, is widespread as the major rock of the Vangorda formation. The increased metamorphism generally tended to coarsen mineral grain sizes, including sulphide minerals.

Deformation is so complex, and flattening into the metamorphic foliation is so pronounced, at the Faro site that the geology appears to be a simple layer cake of stratigraphy. The ore-body is concordant to formational boundaries and is stratigraphically approximately 75 m below the top of the Mt. Mye formation (Figure 4-2).

The Mt. Mye formation is represented by schists, the upper 100 m of the formation by a fine grained grey spotted schist (unit 1D0) and the lower parts by brown coloured, coarser, boitite - muscovite schists (units 1C0, 1CD). Mafic meta-igneous rocks, now amphibolites, are present locally but are volumetrically minor. Graphitic schists, locally with chaistolite, are another minor component of the

formation. A pure, white, calcite marble marker horizon occurs several hundred meters below the ore-body but has not been excavated by mining activities.

The Vangorda formation is represented by calc-silicates (unit 3D0) rather than the usual grey calcareous phyllite. The basal member of the formation is an assemblage of carbon rich schist, mafic schist and calc-silicates (unit 3A). This is overlain by hard, dense, purplish brown and green thinly and lenticularly banded calc-silicate gneiss. The principle minerals in the calc-silicates are diopside, tremolite, quartz, calcite, plagioclase and biotite. The calc-silicates grade upward into calcareous biotite bearing phylites and normal grey calcareous phyllite of the Vangorda formation. The mine excavations within the Vangorda formation are entirely within the calc-silicates. The calcareous phyllites (unit 5B0) are found extensively along the south side of the Rose Creek tailings impoundment. As elsewhere in the district, mafic meta-igneous rocks are important members of the Vangorda formation. In the pit these units are generally a one to a few meters thick and are amphibolotes (unit 5C). Higher in the section they are greenstones (also unit 5C) and range up to 10's of metres thick. These rocks are significant in that they are physically and chemically resistant and have been quarried extensively for rip-rap used to construct the Rose Creek diversion canal.

As noted below faults are significant In the area, both from the perspective of bounding the individual ore zones but also as ground water pathways. Most late faults are steeply dipping and trend north-south or northeast-southwest (Figures 4-3 and 4-4), while a few faults have many 10's of meters of throw most are relatively minor.

The minesite is near the Anvil Batholith but the rocks of the batholith (unit 10AB) do not actually come into contact with the ore-bdy. Two members of the Anvil dyke suite are present at the Faro site. The most abundant is a medium grained hornblende biotite diorite to quartz diorite (unit 10E), locally this rock grades into a monzonite. Less significant is a quartz feldspar porphyry (unit 10F) consisting of quartz, locally bi-pyrimidal, feldspar and minor biotite in an aphanitic white matrix. The porphyry is notable on the dumps in that it is bright white and the rock slakes severely. Porphyry is a minor component of the dumps. Diorite is a significant component of the Northwest and Faro Valley dumps but elsewhere is relatively minor.

Along the northeast edge of the Zone 3 pit was a distinctive, coarse silicified breccia consisting of very large clasts of calc-silicate in a fine siliceous matrix. The breccia appears to be related to emplacement of the quartz diorite dykes. The material was important at Faro since it is very hard, thus physically resistant, and unreactive thus it was used to build the North Fork rock drain and is found in rip-rap locally.

For the purposes of mapping rock types in the dumps the rocks have been lumped into 5 assemblages. These assemblages have been used to quantify the dump contents and are used to simplify the rock units for presentation of summary geochemical data. These assemblages are outlined in Table 4.1.

Table 4.1

Major rock assemblages of the Faro mine area

Assemblage	Rock unit	Rock Type	Fe sulphide	Carbonates
1	1C	tan to brown, coarse, quartz muscovite biotite staurolite garnet schist	.5 to 2 % py	nil
	1D	grey, fine graind, prophyroblastic, biotite muscovite andalusite schist	.5 to 2 % py	nil
	1CD	intermediate between 1C and 1D, biotite muscovite andalusite schist	.5 to 2 % py	nil
	1C6	Clotted biotite muscovite andalusite schist, clots are dark irregular andaulsite biotite amsses along the foliation	.5 to 2 % py	nil
	1D2	black, graphitic mica schist, commonly with chiastolitic andalusite	1 to 5% py	nil - local caclite
1*	1D4	White mica envelope, pale altered quartz muscovite schist, minor marcasite pyrite	1 to 7% py mc	nil
2	sulphides	massive and disseminated pyritic sulphides, 5 to 95 % pyrite	5-95% py, po	nil -local carbonates
3	3D0	calc-silicate and related schist, amphibolite and quartz diorite, calc-silicate breccia	.5 to 1% py/po	10 - 30 % calcite
4	10E	hornblende biotite quartz diorite	nil	nil
	10F	quartz feldspar porphyry	nil	nil
5	overburden	gravel and till, largely excavated fron the Faro Creek valley and over parts of Zone 2		

4.1.2.4 Ore Zone geology

The ore occurs as gently dipping stratiform massive sulphide zones (Figure 4-3). The main ore body is composed of two zones (Zones 1 and 3) vertically offset by a normal fault. This fault, the Faro fault, trends north east is sub-vertical and the east block is down dropped approximately 50 m, it bisects the main ore-body. An important diorite dyke is intruded along the fault zone (Figure 4-4). A separate ore body (Zone 2) is geologically part of the same overall sulphide lens but has been separated by normal fault offset. The Big Indian - North Fork fault system separates Zone 2 from Zone 3. This fault set consists of several strands with overall trend N 20° E and approximately 50° dip toward the west. The throw on the faults is on the order of 75 m, west block down. The main deposit was approximately 1500 m long, 500 m wide and averaged 46 m thick. The Zone 2 deposit was approximately 300 m long, 180 m wide and averaged 24 m thick.

The general nature of the ore deposit is described in Appendix G. Quartz and pyrite are the dominant minerals in the ore-body with sphalerite, galena, pyrrhotite, chalcopyrite and marcasite being other important minerals. Sulphides make up about half of the volume of the ore. The ore contains a consistent ratio of zinc to lead of about 60:40 by weight with low copper concentrations, characteristic of this type of deposit. The major ore types are described in Table 4.2

The sulphide bodies are surrounded by an irregular zone of bleached, pale coloured, altered phyllites and schists (unit 1D4, or the "white mica envelope") which is turn enclosed by unbleached phyllites and schists. The orebody is concordant with the enclosing metasediments, with its axis trending north-west, and is truncated, at it's north-west end by the steeply dipping, north-east trending contact of a hornblende quartz diorite (to monzonite) body belonging to the Anvil Dyke suite. The quartz diorite is kaolinite-altered up to 100 m from the ore-body. Smaller quartz feldspar porphyry dykes and diorite to quartz diorite dykes also cut the deposit.

Table 4.2

Sulphide rock types at the Faro Minesite

Unit	Name	Description	% pyrite	other sulphides	qangue	remarks
2A	Ribbon banded graphitic quartzite	bands of quartz and pyrite separated by graphitic schist to quartzite	5% to 30%	sphalerite, galena, minor pyrrhotite	quartz graphite	basal and peripheral ore type, considerable is sulphide waste, moderately soft, tendency to slake
2B	Quartzite	grey quartzite with minor sulphides, no graphite	0%	galena sphalerite	quartz	minor rock, main occurrence being a 1 to 3 m. thick layer across the top of the deposit
2CD	Pyritic quartzite	weakly banded grey quartzite, no carbon	10 to 60%	galena sphalerite chalcopyrite	quartz magnetite	hard quartz rich rock, abundant along north-east edge of the deposit, major component of sulphide waste
2E	Massive pyritic sulphides	massive pyrite with minor galena-sphalerite, weakly banded	60% to 100%	galena sphalerite	quartz barite magnetite	hard barren quartzose variety along the north-east edge of pit is major part of sulphide waste,
2F	Buckshot massive sulphided	massive sulphides, pyrite rich but high grade	60% to 90%	galena sphalerite chalcopyrite	quartz barite	characteristic texture on 1 mm pyrite porphyroblasts in galena-sphalerite matrix, high grade
2G	Baritic massive sulphides	slightly banded dense massive barite - sulphide rock, generally high grade	30% to 75%	galena sphalerite	barite Ba-Fe-Mg carbonates	locally with magnetite
2H	Pyrrhotitic massive sulphides		50% to 85%	pyrrhotite galena sphalerite	quartz	irregular bodies of massive pyrrhotite tend to occur most frequently along the south-west side of the pit, fine grained, common augen texture.
1D4	White mica alteration	muscovite quartz feldspar schist w/ minor marcasite	0 to 5%	pyrite, galena, sphalerite	quartz feldspar FeMg carbonate	soft slaking rock type, minor sulphides but acid generating component readily available, forms an irregular halo around the sulphide deposit, 10's m. thick

Pyrite within the ore is mainly formed of subhedral grains enveloped by other minerals, but also occurs as masses of small grains. Sphalerite occurs as envelopes around the pyrite grains and as granular aggregates. Galena is associated with sphalerite as single small grains interstitial to it, and pyrrhotite occurs as granular aggregates. Pyrite forms the largest grains in the ore with an average size of 0.56 mm and a maximum size of 5 mm.

Changes in the ore type and grade that was mined and milled, change the tailings composition over time. A brief description of the main ore body is provided to illustrate the spatial differences in mineralogy and the ways that the three ore zones may differ. Only very broad comparisons could be made with respect to the time of milling and the mineralogy of the ore types milled.

Key spatial differences in the main ore body, are as follows:

- the north-east edge of the massive sulphide lense is thicker, but of a lower grade than the south-west edge;
- the ore body dips gently towards the south,
- the central portion of the ore deposit contains coarse porphyroblasts of pyrite, with inclusions of sphalerite and galena in a fine grained matrix of sphalerite, galena and calcopyrite; the margins of the deposit contain fine grained aggregates of pyrrhotite with vienlets and fine disseminations of chalcopyrite, sphalerite and galena
- the massive sulphide component of the ore body overlies the more disseminated quartzite hosted sulphide component of the ore body.
- Zone 1 was high grade and barite rich, Zone 2 was low grade and rich in quartzite ore, Zone 3 was mineralogically intermediate.
- Zones 1, 2 and 3 are separated by large fault zones, and have the gradational differences discussed, but are otherwise mineralogically very similar.

During the early years of the mine operation, from 1969 to 1982, Zone 1 was mined. Ore types that were predominately milled were; 2EG, 2H, and 2EF in Zone 1. These differ in the following ways from those milled toward the end of the mine life; they were of higher grade, they contained more pyrrhotite, and a higher barite content. Zone 2 was mined from 1978 to 1981 and was very low grade, principle ore types were 2EF and 2A with lesser 2H. From 1986 to 1992, primarily Zones 1 and 3 were mined. The major ore types milled during this period were; 2EF, 2ED, 2BCD, 2EG, and 2A. The majority of the material milled during this time was still massive sulphides, however it was of a slightly lower grade the earlier massive sulphide ore, and a significant portion of the low density quartzite ore was milled. Generally the ore that was milled in more recent years has had a lower pyrrhotite content, a slightly lower density, due to the reduction in barite content, and has been of a lower grade than the earlier milled ore.

4.1.2.5 Rock Geochemical Program

4.1.2.5.1 Objectives

Since it is recognized that ARD is an issue for both Faro and Vangorda, the geochemical testing program focused on some very specific objectives:

- identify existing and potential water quality concerns in terms of both source and contaminants;
- evaluate how the chemistry of the site drainage will change over time, in terms of both acidity and metal content; and,
- obtain data required for the determining water treatment requirements in the longer term.

There continues to be considerable debate in the industry as to the extent to which the drainage water chemistry from a waste rock pile can be predicted, and the methods by which to so do. For this Closure Plan, the latter two objectives are critical. Thus, a suite of samples were collected from the different dumps and pits in an attempt to address the variability in:

- rock types (units);
- "ages" of material, i.e. time since mining or deposition;
- geochemical composition within the rock unit;
- weathering over time within each rock type; and,
- weathering over time for different rock units.

The focus of the program was on rock sampling at the Faro site, since it is the site for which there is the least geochemical information available (having been developed to a large extent before ARD and related water quality concerns were recognized or understood) and the site for which there is the most complexity in terms of prediction and control of drainage water quality. Emphasis in the sampling and testing was on the sulphides, and to a lesser extent the schists, which are recognized to be the primary sources of acidity and metals.

Both static and kinetic test results are available for samples from Vangorda, and to lesser extent for Faro and for Grum. These results are incorporated where appropriate. This is particularly important with respect to kinetic testing for waste rock and pit walls, where the methods and interpretation of results have changed considerably since the testing was done for these sites from 1988 to 1991.

4.1.2.5.2 Sampling Methods

To address the different sampling requirements, different types of samples were collected:

1. Channel samples from the (near) surface of individual piles or loads of a distinct rock unit or characteristic mix of rock units. These samples are termed a "composite" in

that they represent a sample from the entire load or pile, and not a selective hand cobbed sample;

2. Excavations within individual piles from which two types of samples were collected:
    - samples from discrete layers or zones;
    - samples representing a "composite" or vertical channel over the excavation.
3. Trenches in the top, or crest of a dump from which three types of samples were collected:
    - samples from discrete layers or zones (labeled "A" etc.);
    - samples representing a "composite" or vertical channel over the excavation; and,
    - samples of accumulations of salts or staining along a flow path.

4.1.2.5.3 Dump Composition and Sampling

The geology of the waste rock at Faro has been summarized in a review of the Faro waste dump composition by Pigage (1988). Five main rock units are identified:

- Non-Calcareous Schists (Unit 1)
- Sulphides (Unit 2)
- Calc-Silicate (Unit 3)
- Intrusives (Unit 4)
- Overburden (Unit 5)

The overall dump composition at the Faro site comprises:

- non-calcareous biotite-muscovite ± garnet ± staurolite schists (unit 1)
- intrusives - hornblende diorite, quartz feldspar porphyry, granite (unit 4)
- meta-intrusives - basaltic greenstone from Grum (also in unit 4)
- disseminated sulphides, quartzites with varying sulphide content (unit 2)
- massive sulphides - Py, Po (also in unit 2)
- calc-silicate (unit 3)

The distribution of samples, by rock type, for each of the dumps is summarized in Table 4.3 and shown in detail in the sampling logs provided in Appendix H. It was not possible to collect samples of clean overburden from the Faro site; the overburden representing approximately 8% of the total waste rock. The sampling locations are shown on Figure 4-5.

Table 4.3
Summary of Rock Samples

	Rock Unit			
	1 Schist	2 Sulphide (Ore)	3 Calc. silicate	4 Intrusives
Est. % of Faro Dumps	42	13	31	6
Variability	high	high	medium	med. low
Number of Samples:				
Main and Intermediate	11	25	8	-
Low Grade Stockpile	2	10	-	-
Northeast Dumps	3	5	5	3
Zone 2 Dumps	4	2	2	2
Faro Valley	8	6	1	7
Northwest Dumps	-	4	-	2
Main Pit	2	2	2	2
Total samples	**30**	**59**	**19**	**22**

As can be seen from the above table and the detailed sampling logs, it was possible to sample a variety of rock types, locations, and ages of exposure (deposition). It is recognized however that there are limitations on sampling the waste dumps - particularly dumps of the age and size of the Faro dumps - in that the samples represent a very small proportion of the mass of waste rock, and that the samples are collected from essentially the surface of the dump, or of a lift. This limitation is in part addressed by the number of dumps at Faro, enabling sampling of a wide range of ages and sources of rock.

4.1.2.5.4 Testing Methods

The testing programs on the rock samples are described in Appendix H and Appendix I, and can be summarized to include:

- sample collection from waste dump and, where possible, pit walls;
- standard static testing including multi-element metal analyses by ICP and acid base accounting tests (using modified Sobek procedure);
- short term distilled water extraction tests on selected samples from each rock unit representing different ages and/or deposition environments to identify the readily soluble component and to understand the controls on geochemical processes;
- mineralogical analyses on selected samples of each rock type;
- particle size distribution analyses;

- enhanced oxidation rate testing using hydrogen peroxide addition, to assist in assessing the future geochemical characteristics of each rock type, for both "aged" and "fresh" material where possible;
- longer term kinetic (column) testing to quantify water chemistry over time for the same samples on which oxidation rate testing was done.

The samples for the current testing program were sent to the CESL Laboratory (Cominco Engineering Services Laboratory) in Vancouver. All sample preparation was done at CESL, as were the paste pH and NP determinations, particle size analyses, and moisture content or water sampling of some of the waste rock samples. CESL coordinated all other analyses as follows:

- sulphur determinations at Chemex Laboratories in Vancouver;
- ICP analyses at Min-en Laboratories in Vancouver;
- solution analyses at Analytical Services Laboratory (ASL) in Vancouver;
- XRD analyses by CESL and petrography by Vancouver Petrographics.

All static ARD prediction testing was done using standard procedures as outlined in the B.C. Draft Acid Rock Drainage Technical Guide. Acid base accounting used the modified Sobek procedure.

Field testing was done to select sample locations and sample intervals, in the case of trenching. These tests are very simple field measurements of paste pH and conductivity or TDS, used to indicate the comparative extent to which oxidation and leaching has occurred in different materials or at different depths. These field tests are done using pen type probes to measure a slurried sample of the waste rock fines (mixed approximately 2:1 liquid to solid ratio).

Short term extraction tests were done on selected samples to evaluate the stored acidity and soluble metal content of different ages of samples, and different rock types. The minus 0.25" fraction of a sample was mixed with distilled water at a liquid to solid ratio of approximately 2:1 (w/w) gently agitated for 24 hours, followed by filtering and solution analyses.

Kinetic testing was done at CESL, and comprise both "standard" column testing and modified testing to evaluate long-term oxidation. Tests are described in more detail in Appendix I.

The majority of the laboratory testwork for the Vangorda Plateau evaluation was conducted at B.C. Research in Vancouver. The results of testwork on samples from the Vangorda Plateau are reported in the Vangorda Plateau IEE. The testwork on samples from Faro is discussed in the Curragh Resources Schedule D report. Test procedures for this work are as described in the B.C. Draft Acid Rock Drainage Technical Guide (B.C. MEMPR, 1989).

4.1.2.5.5 Results

The test results are discussed in detail in Appendices H and I. The use of these tests to evaluate post-closure drainage water quality is discussed in Chapter 6.

4.1.3 Faro Main Pit

4.1.3.1 General Description

The Faro main pit (Zone 1 and 3) measures about 1675 m long by 975 m wide. Its circumference is 4.2 km covering a surface area of approximately 1.06 km^2 (Figure 4-6). The total volume of excavated overburden and bedrock material is estimated to equal approximately 100 million m^3. The lowest point in the Faro pit has an elevation of 975 m (3200 ft) AMSL, 335 m below the highest point on the western pit wall.

The Faro main pit has two access ramps which constitute low points in the pit perimeter. One access ramp is located in the south-west wall in proximity of the old Faro Creek channel with an invert elevation at 1180.5 m (3872.9 ft) AMSL. The second access ramp is located in the south-east corner of the pit and has a somewhat lower invert at an elevation of 1174.5 m (3853.2 ft) AMSL.

Table 4.4 lists the incremental and cumulative volumes of material excavated per bench (toe elevation shown) for the Faro main pit. Shown are the height-capacity data for (I) the deepest pit after completion of excavation in early 1991, and (ii) after some backfilling with waste rock from the Faro underground workings (completed in 1992). The data indicate that approximately 3.4 million m^3 of mine rock were placed into the pit, with the majority (~85%) dumped below the "3650 ft bench", i.e. below an elevation of 1085 m (3561 ft) AMSL.

Table 4.4

Height-Capacity Data for Faro Main Pit (Zone I/3).

Bench (ft) Mine Datum	Crest Elevation (ft) AMSL (NAD 27)	Deepest Pit Mined		Backfilled Pit	
		Volume (cu. ft)	Cum. Volume ($\times 10^6$ m^3)	Volume (cu. ft)	Cum. Volume ($\times 10^6$ m^3)
3310	3221	266,771	0.01	257,806	0.01
3330	3241	2,590,835	0.08	1,869,612	0.06
3350	3261	5,434,690	0.23	3,663,632	0.16
3370	3281	8,484,670	0.47	5,042,437	0.31
3390	3301	11,721,644	0.81	7,149,333	0.51
3410	3321	15,542,477	1.25	9,317,436	0.77
3430	3341	19,076,322	1.79	12,204,404	1.12
3450	3361	23,312,450	2.45	16,150,849	1.58
3470	3381	26,260,687	3.19	19,839,219	2.14
3490	3401	30,163,923	4.05	23,599,137	2.81
3510	3421	33,021,737	4.98	25,534,622	3.53
3530	3441	36,036,394	6.00	28,231,365	4.33
3550	3461	40,497,887	7.15	31,892,821	5.23
3570	3481	43,301,256	8.37	38,,094,399	6.23
3590	3501	46,722,858	9.70	38,882,474	7.33
3610	3521	51,131,932	11.14	43,183,858	8.55
3610	3541	56,039,429	12.73	49,492,414	9.95
3650	3561	60,344,237	14.44	54,738,244	11.50
3670	3581	66,018,620	16.31	65,167,837	13.35
3690	3601	71,848,903	18.34	70,870,308	15.35
3710	3621	77,871,785	20.55	77,736,919	17.55
3730	3641	82,910,012	22.90	82,621,223	19.89
3750	3661	87,759,092	25.38	87,101,318	22.36
3770	3681	92,479,065	28.00	91,257,045	24.94
3790	3701	99,5193,05	30.82	97,803,038	27.71
3810	3721	105,313,350	33.80	102,484,384	30.62
3930	3741	110,862,504	36.94	109,061,432	33.70
3850	3761	115,784,136	40.22	113,997,734	36.93
3870	3781	123,909,390	43.73	123,588,664	40.43
3890	3801	129,621,536	47.40	128,846,337	44.08
3910	3841	276,754,844	55.23	274,565,679	51.86
3950	not known	73,700,686	57.32	73,541,419	53.94

4.1.3.2 Pit Geology

Zones 1 and 3 of the Faro ore-body (the main Faro ore-body of early terminology) were mined in the Faro Main Pit. The ore deposit consisted of a large lens of massive and disseminated sulphide concordant to the foliation and layering in the host meta-sediments. The bulk of the lens has been extracted however remnants of sulphides are found in the pit walls, locally the ore extends significantly into the pit wall and underground methods were used to mine the ore. Figure 4-7 shows the geology of the Main pit, the geology is approximate as an old map prior to completion of the pit was used. Since the pit is flooded and significant covering waste dumps were placed on the pit floor and walls it is no longer possible to map the pit.

The rock types in the Faro are described in section 4.1.2

The bulk of the exposed sulphide is on the lower benches. Only minor sulphide extends high on the pit walls, in the northwest corner of the pit along the contact of the diorite dyke where pyrrhotite from the ore-body is mobilized along the contact and along the northeast pit wall where there are remnant slices of sulphides in between the strands of the Big Indian Fault Zone. Figure 4-7 shows these areas of sulphides in the final pit wall above the flooding elevation.

The bulk of the faults in the Faro pit trend east-west or north-south they form a graben structure with Zone 3 in the graben and Zones 1 and 2 both in upthrown blocks. The fault zones are only significant pathways out of the pit in the southeast corner where the Big Indian North Fork Faults are a likely flow path from the main pit to Zone 2 pit.

The northeast wall of the Faro main pit has always been a source of stability issues. Final mining of the pit required stepping the wall in and lost a million tonnes of ore. This is due to the southwest dip of the foliation so that it dips into the pit along that wall. In genera the failures on that wall are considered to be surficial and not deep seated.

When the Faro pit mining was completed no cleanup other than of blasted ore piles, was done. There are also significant in pit rock dumps as outlined in Table 4.3. These dumps contain considerable sulphide and are certainly many times more significant then the pit walls. No special accounting of these dumps was complete but it is estimated that they are half sulphide and half schist calc silicate and intrusive.

No special blasting was used fir the final pit wall in most parts of the Faro main pit. Thus blast damage can be expected.

4.1.3.3 Hydrogeology

The Faro main pit and dumps are located on a gently sloping, south-facing hillside of Rose Creek valley (Figure 4-1). Much of the area is covered with only a thin veneer of compacted glacial till overlying low-permeability bedrock. Thicker deposits of overburden soils are only found in the area of Faro Creek valley. North of the Faro pit, these sediments consist of interbedded granular and clayey soils with thicknesses of up to 100 m. South of the pit and dumps, the valley sediments are much thinner (~ 8-12 metres).

On the valley sides, groundwater flow is typically limited to a narrow zone of more permeable glacial till and weathered bedrock. Groundwater flows in these areas is very limited. The only significant flows of groundwater in this area were observed in the permeable valley sediments of Faro Creek Valley. Dewatering records during mine operation indicated that shallow groundwater inflows from the overburden material of upper Faro Creek Valley constitutes a major source of inflows to the pit (~22.7 l/s, Piteau & Associates, 1986). Presently, the pit is allowed to fill and this shallow groundwater is discharging directly into the pit.

Seepage of deep groundwater from the surrounding bedrock into the open pit appears to be very limited (see section 4.1.3.4)

At present the Faro main pit is being filled with tailings from ore processing. The pit water level is still well below the lowest elevation of overburden soils in the pit wall and no significant seepage of pit water into the surrounding bedrock has been observed. The potential for seepage from the Faro Main pit and Zone 2 pit are discussed in more detail in sections 4.1.3.4 and 4.1.4.3, respectively.

4.1.3.4 Water Balance

Prior to mine development most of the pit area was draining towards Faro Creek which was flowing south in a hillside valley (Faro Creek valley) to discharge into Rose Creek downstream of the confluence of the North Fork and South Fork of Rose Creek (Figure 4-8). The total watershed area commanded by Faro Creek at the current location of the pit was in the order of 15 km^2.

The development of the Faro main pit, however, has altered local drainage patterns. Foremost has been the relocation of Faro Creek through a diversion ditch to the North Fork of Rose Creek (Figure 4-1). This diversion ditch plus two smaller diversions, i.e. the Faro Valley Interceptor Ditch and the Zone 2 Interceptor Ditch, have reduced the total area of the subcatchment contributing directly to the main pit to 2.15 km^2 (Figure 4-1).

Pumping records from dewatering of the main pit indicated that total inflows to the Faro pit averaged between 40 l/s from 1978 to 1980 (Piteau & Associates, 1986) and 55 l/s from 1988-1990 (SRK, 1992). Several sources of surface and subsurface flows have been identified as most likely contributing to these inflows (see e.g. Piteau & Ass. 1986; SRK 1991; SRK 1992):

- leakage from Faro Creek Diversion along east wall of main pit;
- seepage of shallow groundwater from Faro Creek Valley in north wall of main pit;
- seepage of deep groundwater from north-western and eastern pit walls; and,
- local runoff from sub-catchment of Faro main pit.

Leakage from the Faro Creek Diversion may be the largest single source of inflow to the Faro main pit.

An investigation of surface water drainage into Faro main pit in 1986 suggested that surface and shallow subsurface seepage originating from the Faro Creek Diversion along its reach parallel to

the east wall comprised ~22.5 l/s (Piteau & Associates, 1986). From 1987 to 1990, spot measurements of stream flow were at various locations along the diversion channel during ice-free periods by Curragh Resources Inc. These measurements indicated that average annual leakage was in the order of 40 l/s over the entire reach of the diversion (SRK, 60643). It was estimated that approximately 62% (or 25 l/s) of total ditch leakage occurred in its upper reach draining towards the Faro Creek valley (SRK, 60643). In other words, it is likely that leakage from the Faro Creek diversion is also contributing to inflows into the main pit via shallow subsurface flow in the sediments of the Faro Creek valley (see below).

Seepage from the Faro Creek Valley has long been identified as a major source of inflow to the Zone 1 pit and was collected in sump A30 on the 4030 bench and pumped to Rose Creek valley during active mining. The north wall of the pit in the vicinity of the Faro Creek valley was investigated for purpose of slope stability in 1976 (Piteau, Gadsby, MacLeod Limited, 1976). Bedded granular and clayey soils were encountered to a maximum depth of 250 feet, with a 150 foot average depth. Groundwater was intersected in the soils, with the majority of flow encountered in coarser materials towards the contact with bedrock, some 50 - 80 feet below surface. A number of near-horizontal drains were drilled in this area of the north wall to reduce water-levels and promote wall stability. The drains produced approximately 0.58 l/s (Piteau & Associates, 1978). It appears that the drain holes installed in the North wall intercept only a small portion of the total seepage in the overburden soils of Faro Creek valley.

Based on pumping records Piteau & Associates estimated that average annual flows originating from Faro Creek Valley comprised about 22.7 l/s in 1986. These estimates are consistent with piezometer observations taken in the overburden soils of Faro Creek Valley in 1975-78. Estimates of seepage in these overburden soils calculated from Darcy's Law using observed piezometric data also yield flows in the order of 20 l/s, assuming the average permeability for this interbedded overburden material is $K=1x10^{-3}$ cm/sec. Note that the piezometer readings also indicated elevated pore pressures on the east side of Faro Creek valley near the Faro Creek Diversion suggesting that the diversion was leaking and contributed to seepage flows in the Faro Creek Valley (see above).

Seepage of deep groundwater from the surrounding bedrock into the open pit appears to be very limited. The most likely location of inflows of deep groundwater into the pit is along the East wall of the main pit where a major fault zone system (Big Indian fault system) intersects the pit (Figure 4-6). A number of boreholes were drilled at various locations in the east wall as part of a slope stability investigation conducted in 1986 (Piteau & Associates, 1986). Piezometer readings indicated that the east wall is not freely draining. Packer tests and pump tests indicated that in-situ permeability ranged from $6x10^{-4}$ cm/sec in intensely fractured massive sulphides and quartzites to $7x10^{-6}$ cm/sec in phyllites. Packer tests in Diorite intrusives returned a permeability value of $5x10^{-7}$ cm/sec. Higher values were recorded in the dike contact zone. The investigation data indicated that, in terms of fracturing and the resulting secondary permeability, conditions are very heterogeneous in the east wall. The heterogeneity can be related to the location of competent massive sulphide and quartzite formations, as opposed to incompetent schists and phyllites. The former hosts open fractures where structural features occur, while the latter cannot sustain open fractures. Permeability is therefore correlated with the competent formations.

Overall inflows of deep groundwater from the east wall into the main pit were estimated to be less than 1.9 l/s (Piteau & Associates, 1986).

A few small seeps were also observed in the North-west wall of the Zone 1 pit. Piteau and Associates estimated that seepage into the north-west zone of the Zone 1 pit comprised ~3.8 l/s (Piteau & Associates, 1986). These seepage flows may represent deep groundwater which flows along the contact of a diorite dike and the host rock. Alternatively, it may represent shallow subsurface flow delivering local runoff from the small sub-watershed to the north-west of the pit (see Figure 4-10).

The last source of inflow to the pit is local runoff from the incremental catchment commanded by the pit, including direct precipitation onto the pit area itself (1.06 km^2). Assuming a mean annual runoff (MAR) of 260 mm/yr. total inflow to the pit was estimated to equal an average of 18 l/s (SRK, 1992) with approximately equal proportions coming from local runoff and direct precipitation into the pit. In the delineation of the subcatchment area it was assumed that local runoff is controlled by surficial topography and that the three diversion ditches (Faro Valley Interceptor, Faro Creek Diversion, Zone 2 Interceptor) intercept all runoff from upstream. The first assumption is likely valid since local bedrock topography follows surface topography fairly well (Curragh Resources Inc., 1988). The observed lack of local surface and subsurface runoff along the pit wall to the south-west which faces downslope and does not command any significant catchment area (Figure 4-10) provides some assurance that surface topography does indeed control local runoff patterns. A detailed field investigation would be required to evaluate the second assumption. It is conceivable that the main pit receives some additional runoff, in particular from upstream of the Faro Creek Diversion. This would, of course, imply that estimated contributions of Faro Creek diversion leakage to total pit inflows would be smaller accordingly.

During excavation of the open pit and the Faro underground workings all inflows were continuously pumped from two sumps, the A30 sump on the "4030 bench" below Faro Valley and from sump A8 at the lowest point in the pit. In other words, natural outflows from the pit were negligible. The dewatering of the open pit was discontinued in July of 1992, when tailings discharge into the main pit commenced. The history of tailings discharge into the pit and projected capacity of the pit to hold tailings in the future are described in section 4.4. It should be noted here, however, that all natural inflows to the main pit are small in comparison to the discharge of tailings slurry into the pit. For example, Anvil Range Mining Corporation is currently discharging tailings slurry into the Faro pit at a rate of approximately 35,000 m^3/day (405 l/s), i.e. at a rate ten times higher than that of natural inflows. The rate of tailings discharge will be reduced to about 12,000 m^3/day when recycle pumping to the mill begins.

At present there are no appreciable outflows from the Faro main pit. However, the potential for seepage from the open pit increases with an increase in pit water levels. It is planned to flood the main pit to a maximum flood elevation of 1173.5 m (3850 ft) MASL (see section 4.4) At this pit water level there may be some seepage from the open pit. Two main regions of potential seepage leaving the pit have been identified (e.g. SRK, 1991a):

- the south wall (in vicinity of old Faro Creek channel)
- the southeast corner of the Faro main pit adjacent to the Zone 2 pit

The south wall of the pit near the Faro Creek valley has not been investigated in any detail. The sediments in the lower Faro Creek channel can be expected to be similar to those in the Faro Creek Valley north of the pit. It is not clear if any of these sediments were removed during pit development. A map of the bedrock topography shown in the 1988 Abandonment Plan (Figure 5 in Curragh, 1988) indicates that the lowest bedrock elevation in this area of the pit wall is approximately 1158 m (3800 ft) AMSL, with a maximum 50 feet thickness of porous sediments overlying bedrock. Flooding of the main pit to the 1173.5 m (3850 ft) elevation would be expected to result in some seepage through the sediments which would lead to base flow in Faro Creek. Total seepage losses into the lower Faro Creek valley are estimated to range from 0.1 to 1 l/s, assuming a cross-sectional area of 10m deep by 120 wide, a hydraulic gradient of 0.083, and a permeability ranging from $1x10^{-3}$ to $1x10^{-4}$ cm/sec.

The second area of potential seepage from the main pit would be along the east and southeast corner of the main pit to the adjacent Zone 2 pit and/or to the North Fork of Rose Creek. Seepage losses would be expected to occur along fractures and/or fault zones, in particular those associated with the Big Indian fault system which intersects the east and south-east wall of the Faro main pit striking predominantly south-ward (see Figure 4-6). Special attention was given in this report to evaluating seepage from the south-east corner of the Faro main pit given the potential for contaminant migration from the main pit to the Zone 2 pit and/or North Fork of Rose Creek along this pathway.

Two groundwater investigations have been conducted in this part of the pit providing insight into the rock conditions and local hydrogeology. In 1986, Piteau & Associates drilled several boreholes in the east and south-east wall of the pit to assess the potential for dewatering in the vicinity of the fault zones for pressure relief (Piteau & Associates, 1986). As described above, this study indicated that primary permeability in bedrock is minimal with secondary permeability depending on the ability of the rock to sustain open fractures. The only borehole drilled in the south-east corner of the pit (in close proximity to the haul road connecting the main pit and Zone 2 pit; see Figure 4-9) encountered the non-competent phyllite and was dry at the time of drilling.

In 1994, Steffen, Robertson and Kirsten conducted a detailed field investigation to assess the potential of seepage from the main pit and Zone 2 pit into North Fork of Rose Creek (SRK, 1994). Several boreholes were drilled in bedrock downstream of the south-east corner of Faro main pit, packer tests performed, and multilevel piezometers installed for groundwater quality monitoring (see Figure 4-9). There were a number of varying rock types encountered during drilling, but these

can be separated into three general rock types and their altered facies; a chloritic biotite schist, a hornblende biotite diorite, and a quartz diorite.

The quartz diorite was encountered in boreholes to the north-east of the Zone 2 pit. The quartz diorite contained well defined joints and not many fractures or drillbreaks. However, most if not all joints were infilled with a good clay gouge and crushed rock. The calculated hydraulic conductivities for the quartz diorite ranged from $1x10^{-6}$ to $1.5x10^{-6}$ cm/sec.

The chloritic biotite schist was the dominant rock formations in all boreholes drilled directly to the north and south of the Zone 2 pit. To the north of the Zone 2 pit the schist extends from the surface to a depth of several hundred feet below ground. Here, most joints were infilled with crushed core and clay gouge. The majority of fractures and drillbreaks occurred parallel to the well developed biotite cleavage or penetrative foliation and were clean and very tight (SRK, 1994). Hydraulic permeabilities calculated from packer tests in the chloritic biotite schist ranged from $9x10^{-7}$ to $6x10^{-6}$ cm/s. To the south of the Zone 2 pit the chloritic biotite schist is intruded and overlain by some 50-80 feet of hornblende biotite diorite. As a result of these intrusions, hydraulic permeabilities of the schist in this area are generally higher than in vicinity of the main pit ranging from $2x10^{-5}$ to $2x10^{-4}$ cm/s.

At depth, the intrusive hornblende biotite diorite appeared to have a lower hydraulic permeability ($5.6x10^{-6}$ cm/s) than the surrounding host rock. However, near surface the diorite was extremely weathered and oxidized. Often the core was so highly broken up by drilling and fractures that distinct joint sets were unrecognizable. No packer tests were performed here but the hydraulic conductivity of this rock type can be expected to be similar to that of a medium dense gravel (SRK, 1994).

Based on the field data described above a cross-sectional flow model was developed to estimate the potential for seepage from the south-east corner of the Faro main pit to the Zone 2 pit and/or North Fork of Rose Creek. Three hypothetical cross-sections were modeled:

- Case 1: "Intact bedrock" with a permeability of $1x10^{-5}$ cm/sec extending from the main pit passed the Zone 2 pit down into the North Fork valley of Rose Creek;
- Case 2: A "limited fault zone" with an average permeability of $1x10^{-4}$ cm/sec extending from the Faro main pit into the Zone 2 pit; the section from the Zone 2 pit into the North Fork valley of Rose Creek consists of intact bedrock w/ $K=1x10^{-5}$ cm/s;
- Case 3: An "extensive fault zone" with an average permeability of $1x10^{-5}$ cm/sec extending from the main pit passed the Zone 2 pit down into the North Fork valley of Rose Creek.

In all three cases it was assumed that permeabilities of all units decrease by one order of magnitude several hundred feet below ground (i.e. below an elevation of 1030 m (3380 ft) AMSL). The model approach taken and the various model assumption used in these simulations are discussed in more detail in Appendix J. A summary of the results of these flow simulations is given below. A more complete description of the simulation results is also given in Appendix J.

Table 4.5 summarizes the simulated seepage losses from the Faro main pit. Shown are the simulated unit fluxes (m^3/day per meter cross-section) for the three hypothetical cross-sections and the estimated total seepage from the Faro main pit along the south-east wall. Note that

values shown in brackets indicate seepage into the Faro main pit (influx) whereas all other values indicate seepage out of the main pit (outflux).

It is important to note that all simulations summarized in Table 4.5 assume that the water level in the Zone 2 pit is maintained at an elevation of 1110 m (3642 ft) AMSL to simulate the present water level control in the pit by pumping from the Zone 2 well (see section 4.5.5 for details). The pumping of groundwater from the Zone 2 well forces most of the seepage from the main pit (and other areas surrounding the Zone 2 pit) to flow to the well rather than flowing down-gradient towards the North Fork Rose Creek. The capture zone of this well is illustrated in Figure 4-11 which shows simulation results for the case of intact bedrock and a limited fault zone at maximum flood level in the main pit. The modeling results suggest that under current operating conditions the Zone 2 well is capturing the large majority of flow from the Faro main pit (see below).

At a flood level of 1110 m (3642 ft) AMSL in the Faro main pit there is no seepage out of the pit in all three modeled cross-sections (Table 4.5). With equal heads in the Faro main pit and the Zone 2 pit a groundwater mound develops between the two pits as a result of recharge from infiltrating precipitation resulting in a groundwater divide with some infiltrating precipitation flowing to the Faro main pit and some flowing towards the Zone 2 pit. At this stage there is no seepage from the Faro main pit to the Zone 2 pit and/or the North Fork of Rose Creek. Once the water level in the Faro main pit rises above this groundwater mound all infiltrating precipitation will flow towards the Zone 2 pit and additional seepage of pit water from the main pit towards the Zone 2 pit will occur. Simulation results suggests that under high recharge conditions (MAR=240 mm/yr.) seepage from the Faro main pit through "intact bedrock" would commence at a flood level of about 1150 m (3773 ft) AMSL. Seepage from the main pit along a fault zone would commence earlier (i.e. at a flood level some 25m (75 ft) lower) since the higher permeability of the fault zone material can not sustain as high a water table mound as the intact bedrock.

At a high water level in the pit of 1170.4 (3840 ft) the simulated unit flux leaving the Faro main pit per meter width of "intact bedrock" is 0.115 m^3/day. The simulated unit outflux per meter width of "limited fault zone". i.e. a fault zone extending from the Faro main pit to about the center of the Zone 2 pit, is about fourteen times greater (1.16 m^3/day). The simulation results shown in Table 4.5 further indicate that there is no appreciable increase in seepage leaving Faro pit if this fault zone would extent further downstream into the valley of the North Fork of Rose Creek. The amount of seepage leaving the Faro main pit is almost entirely controlled by the permeability of the reach between the main pit and Zone 2 pit since there is no appreciable flow of deep groundwater by-passing the capture zone of the pumping well in the Zone 2 pit (Figure 4-11).

Table 4.5

Summary of Simulated Seepage Losses from the Main Pit

Water level in Faro Main Pit (elevation AMSL)	Simulated Unit fluxes from Faro main pit[1,2] (m^3/day per unit width cross-section)			Estimated Total Seepage from Faro Main pit[3]	
	Case 1: "Intact bedrock"	Case 2: "Limited Fault Zone"	Case 3: "Extensive Fault Zone"	(m^3/year)	(l/s)
1110 m (3642 ft)	(0.149)	(0.153)	(0.153)		
1140 m (3740 ft)	(0.072)	0.422	0.424	9,100	(0.28)
1158.2 m (3800 ft)	0.046	1.16	1.17	11,000	0.35
1170.4 m (3840 ft)	0.115	1.66	1.663	23,000	0.73

Notes:

1. all simulations assume steady-state flow conditions and high recharge (R=240mm/yr)
2. all simulations assume water level in Zone 2 pit is maintained at 1110m (3642 ft) by means of pumping.
3. assuming 400m wide "intact bedrock" zone and 10m wide "limited fault zone"

The simulated seepage fluxes were used to estimate steady-state seepage losses from the south-east section of the Faro main pit (Table 4.5). Assuming a 400 m wide cross-section of "intact bedrock" and an additional 10 m wide cross-section of "limited fault zone" the maximum seepage would be in the order of 23,000 m^3/year or 0.73 l/s assuming a pit water level of 1170.4 (3840 ft). The vast majority of this seepage would flow into the Zone 2 pit (assuming the pumping well is maintaining a water level of 1110 m AMSL in the Zone 2 pit). The amount of seepage from Faro main pit which would by-pass the Zone 2 well and reach the North Fork of Rose Creek is estimated to be less than 350 m^3/year (see Appendix J).

It should be noted that all flow simulations and estimated seepage rates assume steady-state conditions which may not develop for many years after flooding of the pit. This is particularly true in the "intact bedrock" where flow velocities are very slow. In addition, it should be noted that the permeability values chosen for the various hydrogeological units are considered conservative resulting in conservative (i.e. high) estimates of seepage losses from the Faro main pit after flooding.

Concerns have been expressed in the past regarding the presence of extensive major faults between the main pit and the North Fork valley of Rose Creek (Morin, 1992). Certainly, the Big Indian fault system consists of a series of significant faults. However, these have been found to be mostly healed with either mineral deposits or fault gouge. This is an agreement with findings from the Piteau study which found no appreciable water flows in several major fault zones upstream of the east wall except for one fault zone in massive sulphides. Furthermore, the southern extension of the Big Indian fault zone system strikes towards the Zone 2 pit (see Fig. 4-9). Any significant flow would be expected to flow through the Zone 2 area and would be captured by the Zone 2 well (see Figure 4-11).

To put the potential for seepage from the Faro main pit along a hypothetical and unrecognized "major" fault zone into the North Fork of Rose Creek into perspective, a "worst-case scenario" was modeled using the geometry of case 3 ("extensive fault zone") but assuming that this fault zone lies outside the influence of the Zone 2 well. The simulation results for such an "isolated fault zone" are summarized in Table 4.6. Note that the simulated unit fluxes in this "isolated fault zone" are generally lower than in the "extensive fault zone" (case 3). Pumping in the Zone 2 pit generally induces higher gradients in the reach between the main pit and Zone 2 pit forcing more seepage out of the Faro main pit. In an "isolated fault zone" these high gradients induced by pumping do not develop resulting in smaller seepage rates. The total seepage from Faro main pit that could reach the North Fork of Rose Creek via an "isolated fault zone" would be not more than 2,800 m^3/day or 0.07 l/s, assuming a fault zone width of 10 m (Table 4.6). To put this seepage rate into perspective the mid-winter baseflow in North Fork of Rose Creek is in the order of 100 to 200 l/s. In other words, any contaminants moving from the Faro pit towards North Fork of Rose Creek via an "isolated fault zone" would be diluted by a factor of 1500 to 3000 under the worst of conditions. Note, that it would take at least 50-200 years before any contamination would reach the stream, assuming the contaminants do not interact with the fracture surfaces. It should be stressed that the possibility of an "isolated fault zone" generating seepage which by-passes the Zone 2 pit and the influence of the Zone 2 well is considered very unlikely given all geological data available.

In summary, flooding of the Faro main pit to the 3840 ft level may result in some seepage from the south-east corner of the main pit along limited fault zones and intact bedrock towards the Zone 2 pit. The total seepage losses from the main pit along this route are estimated to be in the order of 23,000 m^3/year (or 0.73 l/s). Flow simulations suggest that maintaining a water level of approximately 1110 m (3642 ft) AMSL in the Zone 2 pit is probably sufficient to capture most, if not all of this seepage before it can reach the North Fork of Rose Creek.

Table 4.6

Summary of Simulated Seepage Losses from the Main Pit Hypothetical Isolated Fault Zone

Water level in Faro Main Pit (elevation AMSL)	Simulated Unit fluxes from Faro main pit in "isolated fault zone"[1,2] (m^3/day per unit width cross-section)		Estimated Total Seepage from Faro Main pit in "isolated fault zone"[3]	
	no recharge	R = 110m	(m^3/year)	(l/s)
1112.5 m (3650 ft)	0.11	0	400	0.01
1127.8 m (3700 ft)	0.22	0.06	800	0.03
1143.0m (3750 ft)	0.36	0.21	1,300	0.04
1158.2m (3800 ft)	0.57	0.43	2,100	0.07

[1] all simulations assume steady-state flow conditions; boundary geometry as in case 3: "extensive fault zone"; 240mm/yr)

[2] all simulations assume that this hypothetical fault zone lies outside the influences of the Zone 2 well (i.e. no pumping simulated)

[3] assuming a 10 m wide "isolated fault zone" with no recharge form precipitation.

4.1.3.5 Geochemistry

There are two primary influences on the water chemistry of an open pit which fills with water after mining ceases; runoff from pit walls and groundwater inflow. It is recongized that the primary source of contaminant loading from the pit walls is from the loose rock which remains on benches, rather than from the wall rock because of the much higher surface area of the loose rock. In the case of the Faro Main Pit, there are three other influences on water chemistry that are much more significant than the above:

- the contaminant loading from sulphidic waste rock deposited in the pit during the latter years of operation;
- the neutralization potential and possible metal loading potential from tailings deposited in the pit after mining ceased; and,
- the drainage from the waste dumps, and particularly the old Faro Creek channel which flows through the Faro Valley Dump (discussed in section 4.2.12.1)

The geology of the wall rock is shown in Figure 4-7. From the pit walls, the only major source of contaminant loading would be the sulphides exposed in the pit wall, above the water table, particularly in the area of the ramp.

The contribution of loading from the waste dumps and stockpiles located within the catchment of the Main Pit. The geochemistry of the drainage from the waste dumps is discussed in detail in the following subsections.

A significant source of loadings to the pit water in the short term would be the rock deposited in the pit. Approximately 12 million tonnes of sulphide waste rock was placed in the Main Pit during the early 1990's. This waste rock is largely under water and for the most part will be covered with tailings. However this material is net acid generating and thus, in the short term, may contribute a significant stored load of soluble contaminants to the pit water. Once covered by tailings, the flushing of this stored load would be reduced in proportion to the reduction of the exchange of water through the waste pile.

The material averages approximately 18% total sulphur with approximately 1% as sulphate sulphur. Zinc contents average about 1.6%Zn. The estimated stored soluble load of this waste rock is provided in Table 4-7 below. Estimates are based on testing of sulphides from waste dumps and pit walls, and of similar oxidized fines and sulphides from the Vangorda site.

The geochemical controls on pit water chemistry are discussed in more detail in section 6.2

Table 4-7

Estimated Stored Soluble Load of Sulphide Wastes in Faro Pit

Parameter	SO4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	11440	9386	6	554	777	70	4	4713

4.1.4 Zone 2 Pit

4.1.4.1 General Description

The Zone 2 pit was excavated into the hillside sloping towards the valley of the North Fork Rose Creek immediately south of the Faro main pit (Figure 4-1). Excavation of the Zone 2 pit was completed in the early 1980's with a total of ~6.8 million m^3 of in-situ bedrock and overburden material being removed. The ultimate outline of the Zone 2 pit measures a circumference of 1.9 km and covers a surface area of ~0.27 km^2. The deepest point in the Zone 2 pit lies to the west at an elevation of 1094.5 m (3591 ft) AMSL, i.e. approximately 100 m below original ground surface (Figure 4-9). The natural discharge point of the Zone 2 pit is located in the south-eastern corner of the pit perimeter at an elevation of ~1128.4 m (3703 ft) (Curragh Resources Inc., 1987).

After final excavation, the Zone 2 pit was backfilled with broken rock from stripping operations. Approximately 32 million tonnes of unclassified mine rock were dumped prior to 1987 covering the northern and western sections of the Zone 2 pit (Curragh Resources Inc., 1987). The remaining south-eastern section of the Zone 2 pit was backilled with ~8 million tonnes of non-acid generating waste rock from 1988 to 1989, according to the development plan submitted to the Yukon Territory Water Board in 1987 (Curragh Resources Inc., 1987). The dumping of mine rock over the Zone 2 pit has significantly altered the surface topography in this area and the Zone 2 pit

The Zone 2 pit has been known to contain water of poor quality and contaminated pit water overflowed the pit perimeter in at least two occasions prior to complete backfilling (Curragh Resources, 1987). The most serious incidence occurred in late October 1983 when the partially backfilled pit overflowed carrying contaminated pit water into North Fork Rose Creek. Zinc levels in the North Fork Rose Creek increased tenfold during this period but declined within days after pumping of the pit water began. Two provisions were made in the plans submitted for complete backfilling of the Zone 2 pit to control any seepage from the pit after backfilling. First, an internal rock drain was constructed prior to complete backfilling to allow collection of water from the pit below the level at which it would naturally overflow. The design plan for the rock drain called for a 4m thick calc-silicate rip-rap layer placed at an elevation of 1125m (3690 ft) delivering pit water to an 16" ∅ H.D.P.E. pipe equipped with an endvalve which in turn feeds into a sump (see Fig. 4 in Curragh, 1987). A compacted till dyke with a crest height of 1130 m (3708 ft) was to be placed at the low point in the pit perimeter to allow for some additional storage of pit water. The maximum water level would be controlled by an overflow pipe with an invert elevation of 1128 m (3700 ft) (see Fig. 4 in Curragh, 1987). It is not known whether the rock drain was constructed to these design specifications. No as-built plans are available and the rock drain including the sump are now completely covered with mine rock. In addition to the rock drain, a 74.7 m deep borehole was drilled through backfilled waste rock into bedrock in 1989. A 100mm (4") ID PVC standpipe was installed to monitor the water level in the Zone 2 pit. This monitoring well was used only until 1991 and is now destroyed.

In late 1990, it was noted that the water level in the Zone 2 pit had reached an elevation several feet above the "drain elevation" of 1125m (3690 ft) resulting in seepage to the North Fork Rose Creek. At that time, dewatering of the Zone 2 pit by means of pumping was selected as a short-

term solution to reduce seepage discharge from the Zone 2 pit. In early 1991, a borehole was drilled through backfilled mine rock into the deepest part of the Zone 2 pit, i.e. from the south-east access ramp to the main pit and equipped with a submersible pump. This pumping well is still in use today to control the water level in the Zone 2 pit (see section 4.5.5).

The Zone 2 pit has a significant capacity to store water that is accumulating in the pit. Figure 4-12 shows the height-capacity curve for the Zone 2 pit before and after backfilling. The total pit volume at the drain invert elevation of 1128 m (3700 ft) is approximately 1.6 million m^3. Assuming an average porosity of 30% for the backfilled mine rock, the maximum storage capacity available for water collecting in the pit would be approximately 480,000 m^3.

4.1.4.2 Geology

The Zone 2 pit mined a small faulted extension of the Faro ore body. This was one of the few areas where the Faro deposit actually cropped out. Thus there was a weathered mantle over part of the deposit. The Zone 2 ore body was at the edge of the overall Faro deposit and was rich in ribbonbanded quartzite ore, massive sulphide was limited. overall, ore grade was low compared to the main pit. The geology of the Zone 2 pit is shown in Figure 4-13. It is noted that Intrusives were found along the northeast side of the orebody.

The fault zone separating the Zone 2 orebody from the main orebody runs along the west wall of the pit and is a likely connection for water flow.

The lower lip of the pit is highly fractured and weathered rock.

4.1.4.3 Water Balance

Four potential sources of inflow to the Zone 2 pit have been identified (Curragh, 1987; 1988; SRK, 1994):

- direct recharge from precipitation infiltrating through the backfilled Zone 2 pit;
- surface and shallow subsurface runoff from the area north-northeast of the pit;
- deep groundwater Inflows from the area north-northeast of the pit; and
- groundwater seepage from the Faro main pit after flooding.

Direct recharge from precipitation infiltrating through the backfilled Zone 2 pit may account for a substantial portion of the inflows to the Zone 2 pit. The precipitation falling directly onto the backfilled Zone 2 pit (either as snow or rain) will either runoff as overland flow, or Infiltrate into and percolate vertically through the unsaturated mine rock. The rate of infiltration through the mine rock depends on the permeability of the mine rock, i.e. its grain size distribution and degree of compaction. Unfortunately, no data are available on the backfill material itself and the degree of compaction. Visual observations at the present surface suggest that the backfill material is very heterogeneous consisting of boulder sized rock fragments intermixed with finer soil-like material. Apparently the backfill material was not compacted except for the haul and access roads used in the past and/or at present. Smith et al. (1995) discuss several minesites where rock dumps have been instrumented to study infiltration and recharge. It is clear from their discussion that the

infiltration ratio (expressed as percent of annual precipitation) may vary considerably from site to site. Assuming an infiltration ratio of 70% for this coarse material the direct recharge to the Zone 2 pit (surface area ~0.27 km^2) would be in the order of 75,000 m^3/yr (or ~2.4 l/s). Note, that some of this infiltrating water will be held by suction by the finer backfill material.

In the past, surface and shallow subsurface runoff from the area north-northeast represented the largest inflow to the Zone 2 pit (Curragh, 1987). Figure 4-10 shows the Zone 2 pit and rock dumps outline superimposed on the pre-mining topography. Based on this pre-mining topography, the majority of the Zone 2 sub-catchment lies to the north-east extending as far as the Faro Creek Diversion. A seep survey conducted by Curragh in the fall of 1987 indicated that at total of 13 L/s seeped into the Zone 2 pit originating from this area via lined diversion ditches (9 L/s) and as buried, visible seeps in the north-east wall. Curragh estimated that 30% of this seepage (~4 l/s) originated from leakage of the Faro Creek Diversion which were collected in lined ditches and diverted into the Zone 2 pit prior to final backfill. The remainder originated from the area below the North-East Rock Dump where formerly a small lake and creek were located (see Fig. 4-10). This creek channel was apparently ditched in the early 1970's to act as a diversion during mining of the Zone 2 pit (Curragh, 1987). This channel emerges in the area of seepage observed during the 1987 survey. Flow from this part of the drainage basin north and east of the Zone 2 pit would collect in this channel and pond in a flat area partially covered in mine rock. Curragh argued that the source of much of this seepage must be groundwater since total flows far exceeded runoff expected from the sub-catchment of the lake and stream itself (Curragh, 1987). It is also possible that some of this seepage originated from the Faro Creek Diversion, recharging the local groundwater and re-surfacing in the vicinity of the buried lake and stream. In order to minimize all inflows from the north-east into the Zone 2 pit all seepage collected in the north-eastern part of the Zone 2 sub-catchment were redirected to the Main pit using a new diversion, the so-called Zone 2 ditch. Curragh estimated that this Zone 2 ditch may eliminate as much as 50% of the recharge originating from the north-eastern portion of the Zone 2 sub-catchment (Curragh, 1988). However, some shallow groundwater and potentially deep groundwater within the bedrock may by-pass this diversion ditch and still recharge the Zone 2 pit from the north-east (see below).

There is a potential that deep groundwater recharges the Zone 2 pit from the area north-northeast of the pit as well. As outlined in section 4.1.3.4 the movement of deep groundwater is believed to be structurally controlled and closely related to faults, joint sets and foliations. In this area, the foliations dip 20 to 30 degrees towards the southwest (Curragh, 1988). In addition, the Big Indian Fault and associated splays are oriented along a south-north direction (see Fig. 4-9). As a result of these structural features it is expected that the bedrock in this area is strongly anisotropic with preferential flow of deep groundwater in a southerly direction. Hence it is likely that some deep groundwater flows into the Zone 2 pit originating from an area to the north-east beyond the limits of the Zone 2 sub-catchment. However, these inflows to the Zone 2 pit are probably very small in comparison to inflows from surface and shallow subsurface runoff.

The potential for seepage from the Main pit to the Zone 2 pit after flooding of the Main pit has been discussed already in section 4.1.3.4. As outlined in this earlier section, the Zone 2 pit is isolated from the Faro main pit by approximately 90 m of in-situ bedrock. The bedrock-overburden contact in the south-east corner is believed to be above the flood level of 3850 ft and would

therefore not act as groundwater conduit between the main pit and the Zone 2 pit. Recent drilling directly north of the Zone 2 pit suggests that the in-situ bedrock between the main pit and the Zone 2 pit consists predominantly of chloritic biotite schist and graphitic phyllite with comparatively low permeabilities (K ranging from $9x10^{-7}$cm/sec to $6x10^{-6}$cm/s; see Piteau, 1986 and SRK, 1994). The results of the cross-sectional flow models suggest that total seepage from a flooded main pit through this type of "intact bedrock" would be no greater than 17,000 m^3/yr (or 0.54 l/s, see Appendix J). The most likely pathway for seepage from the flooded main pit to the Zone 2 pit is along interconnected fractures and fissures in zones of disturbed bedrock. Although unit seepage fluxes are expected to be at least one order of magnitude greater than in the intact bedrock (see Table 4.5) the total seepage flowing through these fault zones would not be greater than ~6050 m^3/yr (or 0.19 l/s), since the fault zones are relatively narrow (probably less than 10m in total width).

Modeling results suggest that virtually all of this seepage would flow into the Zone 2 pit, provided the Zone 2 well is maintaining a water level of 1110 m (3642 ft) in the Zone 2 pit. Figure 4-11 illustrates this point for the case of "intact bedrock" (upper) and a "limited fault zone" (lower) assuming steady-state recharge of 110 mm/yr. This boundary line of the capture zone of the Zone 2 well is shown superimposed on the hydraulic head contours. This line separates the area discharging to the Zone 2 well from the area discharging into the North Fork Rose Creek. The plots indicate that the Zone 2 well capture zone extends to a depth several hundred feet below the bottom of the Zone 2 pit capturing virtually all of the seepage originating from the Faro main pit. Based on these simulation results total inflows into the Zone 2 pit originating from the Faro main pit after flooding to an elevation of 3850 ft AMSL is estimated to be no greater than ~23,000 m^3/yr (0.72 l/s), Note that these seepage inflows are significantly smaller than the inflows from direct precipitation onto the backfilled pit and potential inflows of surface/shallow subsurface flows originating from the north-east of the pit (see above).

Anvil Range Mining Corporation estimated that approximately 140,000 m^3 were pumped from the Zone 2 pit based on spot measurements. In the first six months of 1996 a total of ~25,000 m^3 were pumped from the Zone 2 pit. Both estimates are within the range of values expected based on seep surveys and groundwater modeling. The latter estimate may perhaps be more reliable since it is based on recorded hours of pumping rather than spot measurements. However, 1995 has also been a below average dry year. The continued monitoring of the volumes pumped from the Zone 2 pit will provide a better understanding of the sources and volumes of inflows to the Zone 2 pit.

Two potential sources of outflows from the Zone 2 pit have been identified (Curragh. 1987; 1988; SRK, 1994):

- groundwater seepage through the southern pit walls; and
- surface flow through the overflow rock drain discharge.

Prior to pumping in the Zone 2 pit seepage losses from the pit via groundwater had been reported (Curragh, 1987). The bedrock conditions along the southern perimeter of the Zone 2 pit are not well known. Recent drilling to the south of the Zone 2 pit indicated that in this area the chloritic

biotite schist is intruded and overlain by a hornblende biotite diorite. Near surface the diorite is extremely weathered and oxidized and hydraulic conductivities are probably comparable to that of a medium dense gravel (SRK, 1994). Packer tests at depth returned a permeability value of $5.6x10^{-6}$ cm/sec. Permeability values for the chloritic biotite schist at depth just downstream of the Zone 2 pit ranged from $1.6x10^{-5}$ cm/sec to $4.5x10^{-5}$ cm/sec. These observations suggest that - prior to pumping - the majority of groundwater seepage occurred in the upper layers of the weathered hornblende biotite diorite and in the overlying thin veneer of overburden soils.

At present, the water level in the pit is controlled to be near the 1110 m elevation by means of pumping from the Zone 2 pump. Modeling results suggest that this water level is sufficiently low to not cause any significant seepage from the Zone 2 pit downslope and into the North Fork Rose Creek. The simulated capture zone of this pumping well covers virtually the entire south-facing side of the Zone 2 pit (see Figure 4-11). In other words, all direct precipitation infiltrating through the backfill material would flow to the Zone 2 well. Note that the flow simulations shown here are assuming steady-state flow conditions. In reality, the capture zone of the pumping well would vary over time depending on the transient recharge conditions. Transient groundwater flow simulations would be required to evaluate the potential for groundwater seepage through the south face of the Zone 2 pit during periods of high recharge such as spring runoff.

Surface flow through the rock drain would only occur if the water level in the Zone 2 pit would rise above the drain invert elevation of 1128 m (3700 ft) AMSL. At present, the water level controlled Zone 2 pump does not allow the water level to rise to that drain elevation.

4.1.4.4 Geochemistry

The geochemistry of the Zone 2 pit is dominated by the chemistry of the pit water which has been determined by the flushing of waste rock in the pit, and by the drainage into the pit. The geochemistry of the waste dumps is discussed in Section 4.2.8.4. Zone 2 pit water chemistry is discussed in Section 4.1.5.3.

4.1.5 Open Pit Water Chemistry

The water quality in the Main Zone pit is affected by:

- flow through the old Faro Creek channel, as measured at A30;
- leakage from the Faro Creek Diversion and flow through the northeast pit wall;
- groundwater infiltration through all pit walls;
- precipitation and runoff on exposed pit walls;
- leaching and or attenuation by waste rock and tailings stored in pit; and,
- water pumped back from the Zone 2 pit.

Water quality in the Zone 2 pit is affected by:

- leakage from the Faro Creek Diversion and flow through the northeast pit wall and dumps as measured as SP5/6;

- groundwater infiltration through pit walls;
- leaching and or attenuation by waste rock stored in pit; and,
- water from the Main Zone pit.

A detailed evaluation of pit wall runoff and seepage water chemistry was reported in (SRK, 1989, Report 60612), and will be used for the predictions of pit water chemistry.

The four monitoring stations that will be used to evaluate pit water chemistry are:

- seepage from the Faro Creek Channel (at A30 discussed in Section 4.2.12.1);
- water pumped from the Main Zone Pit (X22, X22A and X22B);
- seepage through the upper pit wall and Northeast dump (SP5/6); and,
- water pumped from the Zone 2 pit (X26).

4.1.5.1 Main Zone Pit Water

Station X22 is the licence specified monitoring station for the water pumped from the Main Zone Pit during operation. Station X22A was used to monitor water pumped from the A30 sump. After mining ceased and tailings deposition began into the pit, the monitoring station X22B was used.

The data are summarized for the three stations X22, X22A, and X22B in Tables 4.8, 4.9a, and 4.9b. Detailed data as well as summary statistics are provided in Appendix D.

Table 4.8

Data Summary for Station X22: Faro Pit dewatering (from bottom of pit)

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	pH unit	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
JAN 1988 - OCT 1996							
Mean	6.90	167.9	668.2	0.06	0.058	5.52	57.056
Minimum Value	5.80	13.0	21.0	0.05	0.001	0.78	0.061
Maximum Value	7.81	1325.0	1400.0	0.07	0.761	43.60	287.000
Median	6.88	164.0	696.0	0.06	0.014	2.81	49.600
Num. Values	112.00	93.0	108.0	11.00	115.000	38.00	112.000
JAN 1988 - DEC 1991							
Mean	6.91	170.8	662.0	0.06	0.053	5.34	54.978
Minimum Value	5.80	13.0	21.0	0.05	0.001	0.78	0.061
Maximum Value	7.81	1325.0	1350.o	0.07	0.761	43.60	196.000
Median	6.88	165.0	696.0	0.06	0.014	2.73	49.500
Num. Values	110.00	91.0	106.0	9.00	113.000	36.00	110.000
JAN 1995 - OCT 1996							
Mean							
Minimum Value							
Maximum Value							
Median							
Num. Values							

Table 4.9a

Data Summary for Station X22A: Faro Pit dewatering (from Sump A30)

Date	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
	pH unit	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
JAN 1988 - OCT 1996							
Mean	7.24	61.6	278.0	0.06	0.147	3.12	13.733
Minimum Value	3.30	1.0	169.0	0.05	0.002	0.08	0.330
Maximum Value	8.10	297.0	1000.0	0.06	3.350	60.50	86.500
Median	7.43	60.0	226.0	0.06	0.024	0.34	9.250
Num. Values	90.00	85.0	90.0	3.00	92.000	41.00	89.000
JAN 1988 - DEC 1991							
Mean	7.24	61.6	278.0	0.06	0.147	3.12	13.733
Minimum Value	3.30	1.0	169.0	0.05	0.002	0.08	0.330
Maximum Value	8.10	297.0	1000.0	0.06	3.350	60.50	86.500
Median	7.43	60.0	226.0	0.06	0.024	0.34	9.250
Num. Values	90.00	85.0	90.0	3.00	92.000	41.00	89.000
JAN 1995 - OCT 1996							
Mean							
Minimum Value							
Maximum Value							
Median							
Num. Values							

Table 4.9b

Data Summary for Station X22B: Faro Pit Water while Filling with Tailings

Date	LAB pH pH unit	TOTAL ALKALINITY mg/L	TOTAL SULPHATE mg/L	TOTAL ARSENIC mg/L	TOTAL COPPER mg/L	TOTAL IRON mg/L	TOTAL ZINC mg/L
JAN 1988 - OCT 1996							
Mean	7.64		698.0		0.358		4.721
Minimum Value	6.92		678.0		0.003		0.251
Maximum Value	8.07		731.0		0.660		8.940
Median	7.72		686.0		0.347		4.080
Num. Values	9.00		3.0		6.000		9.000
JAN 1988 - DEC 1991							
Mean							
Minimum Value							
Maximum Value							
Median							
Num. Values							
JAN 1995 - OCT 1996							
Mean	7.60		698.0				6.810
Minimum Value	7.30		678.0				4.080
Maximum Value	7.78		731.0				8.560
Median	7.72		686.0				7.800
Num. Values	3.00		3.0				3.000

Sample pH values from the two pit dewatering stations are quite variable, ranging from pH 3.3 to 8.1 over the period of record. Overall, there is a trend to decreasing pH, particularly since the start of deposition in sulphide waste into the open pit. Within each year, the pH's are variable, with the lowest pH values typically recorded in early spring. As shown in Figure 4-14, the most variable pH values are seen for the A30 sump reflecting, it is believed, the seasonal pattern of flushing of the Faro Valley Dump.

Once the pit began filling with tailings, the alkalinity contributed by the tailings dominated the pit water pH (Figure 4-14 for station X22B). Pit water pH values are slightly lower than tailings water pH however, indicating that there is consumption of alkalinity by acidity within the pit water. There is some natural reduction in pH from the tailings pH due to equilibration with the atmosphere. This mechanism will be more dominant though during periods where there is not deposition of fresh alkaline tailings. Alkalinity shows a similar pattern of variability, Figure 4-15.

Sulphate concentrations also show the effect of increased release of oxidation products in the pit, probably as a result of the placement of the sulphide waste rock. Sulphate concentrations steadily increased to 1992 in the two pit dewatering stations (Figure 4-16), but insufficient data are available for comparison with the pit water over the history of filling. Currently, the concentrations in the pit water in 1996 (X22B) are similar to the lower range of concentrations measured at X22 and X22A. Since the pit water is comprised of tailings water, groundwater influx, and the continued seepage from A30 (X22A) this suggests that there is a source of sulphate to the pit water other than the seep; probably the sulphide waste rock.

Total zinc concentrations increased over the period of record for both of the pit dewatering stations, particularly the dewatering station at the bottom of the pit (X22), as shown in Figure 4-17. This increase in zinc concentration is further evidence of the significant effect on pit water quality of the sulphide waste rock backfilled into the pit in the late 1980's.

The addition of the tailings to pit resulted in a marked decrease in zinc concentrations due probably to a combination of both dilution and chemical precipitation, due to the alkalinity in the tailings water. Total zinc concentrations in 1996 are similar to those measured in 1993, at an average value of 4.6 mg/L. Dissolved zinc concentration data are limited, and are not available for the pit water after filling with tailings.

Other metals of note in the pit water include nickel at an average concentration in 1996 of 0.15 mg/L, lead at 0.15 mg/L, and copper at 0.02 mg/L. As shown in Appendix D, these values are within the lower concentration range of previously measured values for the pit dewatering stations.

4.1.5.2 Northeast Dump (SP5/6)

The seep surveys include sampling of the drainage from the Northeast Waste Dump, at SP5/6. Table 4.10 summarizes the data for this station including seep surveys to 1991, and again in 1995 and 1996. The water chemistry at this station has been affected over time by the development of the dump and the ditch, which necessitated moving the station (refer to SRK, 1989).

Table 4.10

Data Summary for Station SP 5/6: Zone 2 Area Seep Survey Interceptor Ditch

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	pH unit	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
JAN 1988 - OCT 1996							
Mean	6.78	180.0	383.0	0.2	0.010	0.92	9.541
Minimum Value	2.99	18.0	207.0	0.2	0.001	0.04	1.870
Maximum Value	*8.45*	*321.0*	*894.0*	*0.2*	*0.015*	*3.62*	*61.500*
Median	7.41	174.0	272.0	0.2	0.010	0.34	2.765
Num. Values	10.00	8.0	10.0	2.0	8.000	8.00	10.000
JAN 1988 - DEC 1991							
Mean	6.54	158.0	337.0		0.010	1.20	10.982
Minimum Value	2.99	18.0	207.0		0.001	0.04	1.870
Maximum Value	8.45	229.0	691.0		0.015	3.62	61.500
Median	7.31	172.0	270.0		0.011	0.83	2.765
Num. Values	8.00	6.0	8.0		6.000	6.00	8.000
JAN 1995 - OCT 1996							
Mean	7.71	248.0	566.0	0.2	0.01	0.057	3.770
Minimum Value	7.68	174.0	239.0	0.2	0.01	0.044	2.300
Maximum Value	7.75	321.0	894.0	0.2	0.01	0.070	5.250
Median	7.71	248.0	566.0	0.2	0.01	0.057	3.770
Num. Values	2.00	2.0	2.0	2.0	2.00	2.000	2.000

The drainage pH is generally in the neutral to alkaline range although one acidic sample was noted. Alkalinity values range over the period of record from 18 to 321 mg/L $CaCO_3$ eq. with higher values occurring in recent surveys. Sulphate concentrations are elevated, with the highest value of 894 mg/L measured in the 1996 survey.

Zinc and total iron concentrations are elevated at this station. Essentially all of the zinc occurs in the dissolved form, with the highest concentrations of 5.1 mg/L dissolved zinc detected in the 1996 survey. Total zinc concentrations of 61.5 mg/L have been measured in the past. Calcium and, particularly, magnesium concentrations indicate that there is considerable dissolution of alkali minerals at this station - comparable to values measured at A30.

4.1.5.3 Zone 2 (X26)

Water from the Zone 2 pit is periodically sampled at Station X26. These data are summarized in Table 4.11. Detailed data as well as summary statistics are provided in Appendix D.

Table 4.11

Data Summary for Station X26: Zone 2 Pit water (from production well)

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	pH unit	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
JAN 1988 - OCT 1996							
Mean	6.55	526.0	1477.0	0.13	0.186	24.25	52.427
Minimum Value	6.07	447.0	1166.0	0.06	0.001	18.80	4.630
Maximum Value	7.26	607.0	1780.0	0.21	1..100	29.70	73.900
Median	6.50	524.0	1486.0	0.13	0.004	24.25	61.400
Num. Values	7.00	3.0	3.0	2.00	6.000	2.00	6.000
JAN 1988 - DEC 1991							
Mean	6.66	607.0			0.001		54.300
Minimum Value	6.66	607.0			0.001		54.300
Maximum Value	6.66	607.0			0.001		54.300
Median	6.66	607.0			0.001		54.300
Num. Values	1.00	1.0			1.000		1.000
JAN 1995 - OCT 1996							
Mean	6.40		1166.0	0.21	0.006	29.70	43.530
Minimum Value	6.40		1166.0	0.21	0.006	29.70	43.530
Maximum Value	6.40		1166.0	0.21	0.006	29.70	43.530
Median	6.40		1166.0	0.21	0.006	29.70	43.530
Num. Values	1.00		1.0	1.00	1.000	1.00	1.000

While the pit water samples are neutral in pH (averaging pH 6.6) with high alkalinity (averaging 527 mg/L), the zinc concentrations are consistently high. The median concentration since 1991 is 61 mg/L for total zinc, with a maximum value of 74 mg/L. Calcium and magnesium concentrations are also high and, in comparing these concentrations to the alkalinity, it appears that there continues to be neutralization of acidity occurring within the pit water.

Sulphate concentrations are elevated but appear from the limited data set to be decreasing, with the most recent measurement showing 1166 mg/L. This would suggest that the oxidation reactions are limited by the flooding of the waste rock in the pit, but that there is continued flushing and/or diffusion of oxidation products into the pit water.

Lead, cobalt, copper, and manganese also appear to be elevated although the database is limited.

4.1.6 Underground Workings

Curragh mined 1.7 million tonnes of ore from a room and pillar underground mine developed through a portal into the southwest wall of the Faro Pit (see Figure 4-6 for outline of underground mine area). All openings into this mine were internal to the Faro pit and are now flooded. Before flooding, Curragh prepared the underground workings for possible use in biological water treatment (sulphate reduction).

The potential for sulphate reduction using the undergound workings as a reactor site are discussed in SRK Report 60643 "Sulphate Reduction as a Water Treatment Alternative at the Faro Mine" (SRK, 1991b). The most likely flow path in the underground working would be from the injection point in a ventilation shaft which accesses the northern section of the underground mine to a cased borehole to be drilled in the most extreme point of the southern section of the workings (SRK, 1991b). It was estimated that the total void of the room and pillar development, accessible to contaminated water injected for treatment would be in the order of 100,00 to 250,000 m^3 depending on the (unknown) conditions of the underground workings (SRK, 1991b).

4.2 Waste Rock Dumps

4.2.1 History of Development

The waste dumps were developed over the sequence of the mining of the Faro pits, with waste rock deposition occurring at several dumps over the same time period. The history of waste rock deposition in the different dumps is summarized in Table 4.12. Figure 4-18 shows graphically the approximate years of construction for each of the dumps.

Generally, the Faro Valley and the Northwest Dumps were the first to be developed, from 1968 to the early 1970's, receiving waste from the early stripping and mining of the Faro Zone 1 pit. The other rock piles developed during this period were marginal ore or low grade stockpiles.

In the 1970's, the Northeast Dumps were built, primarily with waste from the Zone 1 and Zone 2 pits. The third section of the Northwest Dump, the Lower Northwest Dump as shown in Figure 4-5, was also built from about 1970 to 1971. The two largest dumps on the Faro site, the Main and the Intermediate Dumps, were also started during the 1970's. These dumps continued to be used

until 1990 when mining at Faro was almost finished. In the mid 1970's the "Parking Lot Dumps", as shown in Figure 4-5, were built.

Dump construction in the early 1980's was primarily in the Zone 2 East dump. In the later 1980's several smaller dumps were built (<10,000 tonnes) as shown in Table 4.12. The majority of the waste was deposited in the Outer Haul Road West dump, with continued deposition on the Main and Intermediate dumps.

In the 1990's, deposition continued on the Main and Intermediate Dumps, and on the low-grade stockpiles. In addition, waste was also placed on some of the smaller dumps which were started in the late 1980's.

Table 4.12

Period of Construction of Faro Waste Rock Dumps

Symbol	Name	Age of Dump	
		start	end
NWU	Upper Northwest Dump	1968	1969
NWM	Middle Northwest Dump	1969	1970
NWL	Lower Northwest Dump	1970	1971
UPL	Upper Parking Lot Dump	1975	1976
LPL	Lower Parking Lot Dump	1975	1976
FVN	Faro Valley North	1968	1970
FVS	Faro Valley South	1968	1975
MDW	Main Dump West	1974	1990
MDE	Main Dump East	1972	1990
ID	Intermediate Dump	1979	1990
NEU	Upper Northeast Dump	1974	1977
NEL	Lower Northeast Dump	1975	1979
NEO	Outer Northeast Dump	1975	1980
ZIIW	Zone 2 West	1987	1990
ZIIE	Zone 2 East	1980	1985
RZD	Ramp Zone Dump	1989	1990
RD	Ranch Dump	1989	1990
SWPWD	Southwest Pit Wall Dump	1990	1991
LGSPA	Low Grade Stockpile A	1987	1990
LGSPC	Low Grade Stockpile C	1987	1990
FTW	Fuel Tank Dump W	1969	1971
FTE	Fuel Tank Dump E	1969	1971
MMW	Mt. Mungly West	1969	1970
MME	Mt. Mungly East	1969	1970
SPB	Stock Piles Base	1969	1975
OXSP	Oxide Fines Stockpile	1969	1974
MGSP	Medium Grade Stockpile		active
CHSP	Crusher Stockpile		active
OHRW	Outer Haul Road West	1987	1989
OHRE	Outer Haul Road East	1983	1989
NFRD	North Fork Rock Drain	1988	1988

4.2.2 Dump Composition

The estimation of the dump composition is based on the understanding of the pit geology, the pit development and on observations of the dumps in 1966 (Pigage, 1988). Earlier reported estimates of rock tonnages and composition have been slightly revised in this evaluation, as the dumps are characterized more detail, that is, the dumps are considered in smaller sections. For example, the Faro Valley Dump is considered in two sections; the Faro Valley South and North, rather than as the Faro Valley as a whole.

Three tables are provided to show the composition of each of the dumps:

- the tonnage, height and surface area of each of the dumps are shown in Table 4.13;
- the characterization of the different rock types in each of the dumps, and smaller dump sections, is shown in Table 4.14 expressed as a percentage; and,
- the characterization of the different rock types in each of the dumps, and smaller dump sections, is shown in Table 4.15 expressed in terms of tonnage.

Table 4.13

Estimated Size of Faro Waste Rock Dumps

Symbol	Name	Area (sqm)	Max. height (m)	Average height (m)	Volume (cu.m.)	Tonnage (tonnes)
NWU	Upper Northwest Dump	128,833	15	10	1,332,833	2,665,666
NWM	Middle Northwest Dump	158,069	30	18	2,861,748	5,723,496
NWL	Lower Northwest Dump	105,653	37	31	3,279,066	6,558,131
UPL	Upper Parking Lot Dump	53,716	27	21	1,111,427	2,222,855
LPL	Lower Parking Lot Dump	32,724	12	10	336,540	677,080
FVN	Faro Valley North	135,869	23	13	1,757,025	3,514,051
FVS	Faro Valley South	32,605	18	9	303,583	607,166
MDW	Main Dump West	220,861	76	57	12,566,943	25,133,886
MDE	Main Dump East	436,065	85	78	33,834,525	67,669,051
ID	Intermediate Dump	421,463	82	62	26,161,236	52,322,473
NEU	Upper Northeast Dump	254,309	67	31	7,892,780	15,785,561
NEL	Lower Northeast Dump	290,351	61	39	11,264,246	22,528,492
NEO	Outer Northeast Dump	12,787	9	8	99,211	198,423
ZIIW	Zone 2 West	89,315	67	34	3,003,004	6,006,008
ZIIE	Zone 2 East	126,084	137	65	8,152,422	16,304,843
RZD	Ramp Zone Dump	60,265	18	18	1,091,072	2,182,144
RD	Ranch Dump	42,305	8	6	262,597	525,195
SWPWD	Southwest Pit Wall Dump	78,294	15	10	809,981	1,619,962
LGSPA	Low Grade Stockpile A	29,353	18	16	455,502	911,003
LGSPC	Low Grade Stockpile C	34,537	11	11	393,034	786,069
FTW	Fuel Tank Dump W	8,372	6	5	43,308	86,615
FTE	Fuel Tank Dump E	95,879	21	13	1,239,888	2,479,775
MMW	Mt. Mungly West	20,287	8	6	125,927	251,853
MME	Mt. Mungly East	34,130	34	13	441,364	882,728
SPB	Stock Piles Base	91,250	21	16	1,416,028	2,832,056
OXSP	Oxide Fines Stockpile	20,793	9	8	161,335	322,670
MGSP	Medium Grade Stockpile	33,899	-	-	-	-
CHSP	Crusher Stockpile	22,917	-	-	-	-
OHRW	Outer Haul Road West	186,942	46	34	6,285,461	12,570,923
OHRE	Outer Haul Road East	86,644	49	26	2,240,913	4,481,826
NFRD	North Fork Rock Drain				-	-
	Total	**3,344,570**			**128,925,000**	**257,850,000**

Table 4.14

Rock Type Composition, Percentage of Dumps

Symbol	Name	Tonnage (tonnes)	Proportion of rock types					
			sulphides	schist	calc-sil	intru	OB	Total
NWU	Upper Northwest Dump	2,665,666	7%	15%	8%	65%	5%	100%
NWM	Middle Northwest Dump	5,723,496	10%	30%	40%	15%	5%	100%
NWL	Lower Northwest Dump	6,558,131	8%	37%	30%	15%	10%	100%
UPL	Upper Parking Lot Dump	2,222,855	5%	25%	70%	0%	0%	100%
LPL	Lower Parking Lot Dump	677,080	10%	5%	85%	0%	0%	100%
FVN	Faro Valley North	3,514,051	15%	50%	10%	15%	10%	100%
FVS	Faro Valley South	607,166	0%	65%	0%	30%	5%	100%
MDW	Main Dump West	25,133,886	10%	75%	10%	0%	5%	100%
MDE	Main Dump East	67,669,051	15%	40%	35%	5%	5%	100%
ID	Intermediate Dump	52,322,473	20%	20%	54%	1%	5%	100%
NEU	Upper Northeast Dump	15,785,561	5%	25%	30%	10%	30%	100%
NEL	Lower Northeast Dump	22,528,492	5%	30%	30%	10%	25%	100%
NEO	Outer Northeast Dump	198,423	0%	40%	40%	10%	10%	100%
ZIIW	Zone 2 West	6,006,008	10%	50%	20%	10%	10%	100%
ZIIE	Zone 2 East	16,304,843	0%	75%	20%	5%	0%	100%
RZD	Ramp Zone Dump	2,182,144	2%	30%	68%	0%	0%	100%
RD	Ranch Dump	525,195	5%	85%	0%	10%	0%	100%
SWPWD	Southwest Pit Wall Dump	1,619,962	45%	50%	0%	5%	0%	100%
LGSPA	Low Grade Stockpile A	911,003	100%	0%	0%	0%	0%	100%
LGSPC	Low Grade Stockpile C	786,069	100%	0%	0%	0%	0%	100%
FTW	Fuel Tank Dump W	86,615	100%	0%	0%	0%	0%	100%
FTE	Fuel Tank Dump E	2,479,775	2%	60%	35%	3%	0%	100%
MMW	Mt. Mungly West	251,853	50%	10%	20%	0%	20%	100%
MME	Mt. Mungly East	882,728	30%	40%	20%	10%	0%	100%
SPB	Stock Piles Base	2,832,056	0%	70%	10%	0%	20%	100%
OXSP	Oxide Fines Stockpile	322,670	100%	0%	0%	0%	0%	100%
MGSP	Medium Grade Stockpile	-	100%	0%	0%	0%	0%	100%
CHSP	Crusher Stockpile	-	100%	0%	0%	0%	0%	100%
OHRW	Outer Haul Road West	12,570,923	2%	78%	10%	10%	0%	100%
OHRE	Outer Haul Road East	4,481,826	10%	70%	0%	5%	15%	100%
NFRD	North Fork Rock Drain	-	0%	0%	100%	0%	0%	100%
	Total	**257,850,000**	**13%**	**42%**	**31%**	**6%**	**8%**	

Table 4.15

Rock Type Composition of Dumps (tonnes)

Symbol	Name	Tonnage	Tonnage of Rock Types				
		(tonnes)	sulphides	schist	calc-sil	intru	OB
NWU	Upper Northwest Dump	2,665,666	186597	399850	213253	1732683	133283
NWM	Middle Northwest Dump	5,723,496	572350	1717049	2289398	858524	286175
NWL	Lower Northwest Dump	6,558,131	524650	1967439	2426509	983720	655813
UPL	Upper Parking Lot Dump	2,222,855	111143	555714	1555998	0	0
LPL	Lower Parking Lot Dump	677,080	67708	33854	575518	0	0
FVN	Faro Valley North	3,514,051	527108	1757025	351405	527108	351405
FVS	Faro Valley South	607,166	0	394658	0	182150	30358
MDW	Main Dump West	25,133,886	2513389	18850414	2513389	0	1256694
MDE	Main Dump East	67,669,051	10150358	27067620	23684168	3383453	3383453
ID	Intermediate Dump	52,322,473	10464495	10464495	28254135	523225	2616124
NEU	Upper Northeast Dump	15,785,561	789278	3946390	4735668	1578556	4735668
NEL	Lower Northeast Dump	22,528,492	1126425	6758548	6758548	2252849	5632123
NEO	Outer Northeast Dump	198,423	0	79369	79369	19842	19842
ZIIW	Zone 2 West	6,006,008	600601	3003004	1201202	600601	600601
ZIIE	Zone 2 East	16,304,843	0	12228632	3260969	815242	0
RZD	Ramp Zone Dump	2,182,144	43643	654643	1483858	0	0
RD	Ranch Dump	525,195	26260	446415	0	52519	0
SWPWD	Southwest Pit Wall Dump	1,619,962	728983	809981	0	80998	0
LGSPA	Low Grade Stockpile A	911,003	911003	0	0	0	0
LGSPC	Low Grade Stockpile C	786,069	786069	0	0	0	0
FTW	Fuel Tank Dump W	86,615	86615	0	0	0	0
FTE	Fuel Tank Dump E	2,479,775	49596	1487865	867921	74393	0
MMW	Mt. Mungly West	251,853	125927	25185	50371	0	50371
MME	Mt. Mungly East	882,728	264819	353091	176546	88273	0
SPB	Stock Piles Base	2,832,056	0	1982439	283206	0	566411
OXSP	Oxide Fines Stockpile	322,670	322670	0	0	0	0
MGSP	Medium Grade Stockpile	-	0	0	0	0	0
CHSP	Crusher Stockpile	-	0	0	0	0	0
OHRW	Outer Haul Road West	12,570,923	251418	9805320	1257092	1257092	0
OHRE	Outer Haul Road East	4,481,826	448183	3137278	0	224091	672274
NFRD	North Fork Rock Drain	-	0	0	0	0	0
	Total	**257,850,000**	**32,790,672**	**107,640,106**	**81,193,308**	**15,235,319**	**20,990,595**

The size of each of the waste rock dumps is shown graphically in terms of total estimated tonnage of rock in Figure 4-19.

The geochemistry of each of the dumps is discussed with reference to the size and composition of the dumps discussed above in this section. Results of the geochemical testing were used to estimate the range of metal content within the dump and the range of acid generation and acid neutralization potential. These were calculated as weighted parameters based on the tonnage of each rock type in each, and on the calculated statistics for the ABA and metal analyses of the solids discussed in Appendix G.

The second set of comparisons that are provided in this chapter are the estimated contained "soluble load" for each dump. These were estimated from the results of the extraction tests, for each rock type, and for both "old" and "young" dumps. Classification of age of dumps is based somewhat arbitrarily at 1980 chosen to represent approximately half way through development and a point at which several dumps were finished and deposition in new areas began (as illustrated in Figure 4-18). The calculations are described in detail in Appendix G. Average soluble loads were calculated for each rock type and age, providing a production rate in terms of mg/kg. These figures were corrected for surface area and then multiplied by the tonnage of each rock type in each dump. The values for each rock type were then summed to calculated the total contained soluble load within each dump. These calculated values are discussed in the following subsections. For illustration, Figures 4-20 and 4-21 show the calculated total soluble loads for each dump, in terms of sulphate and zinc respectively.

These values are intended to be used as a basis for comparison of the different dumps and a basis for evaluating the comparative environmental concerns associated with each. They are not intended to represent an absolute value in terms of either metal content or acid generation potential, as it is recognized that there are limitations associated with any such estimate for which there is considerable variability and scaling of results required. The following tables and statistics do however provide a reasonable basis for comparison of the different dumps as the calculations and implicit assumptions are common for all dumps.

4.2.3 Dump Hydrology and Drainage

Considerable effort has been made over the past ten years to detect and monitor water quality from the dumps. Seep surveys are conducted annually around the site, with emphasis on the waste dumps to identify and sample any drainage from the dumps. Test pits were dug in all areas immediately downstream of the dump toes in 1990 and water samples taken from shallow piezometers which were placed in the backfilled pits (SRK, 60648). Several additional monitoring piezometers were installed south of the sulphide cells (Intermediate Dump) in 1989 and southeast of the Zone 2 pit first in 1990 and again in 1994 (SRK, A114101). Three additional monitoring wells were drilled as part of this study in the fall of 1996 (RGC, 1996). A brief description of these monitoring piezometers as well as a discussion of observed groundwater quality and seepage estimates is provided in section 4.2.13.

To evaluate the effect of the dump drainage on the receiving waters, the original ground topography was reviewed to identify surface water divides. Figure 4-10 shows the watershed

divides in the immediate area of the Faro Rock Dumps. The original topography is shown for reference. The watershed divides shown here are based on the original topography (with the exception of those divides created by pit excavation and diversion ditches). Superimposed on the original topography is the current configuration of the dumps. The placement of these rock dumps has significantly changed the surface topography in this area. However, it is assumed that the placement of the dump material did not significantly change the drainage pattern.

There are seven primary "receiving" waters identified for the dumps, including:

- the Main pit;
- Zone 2 pit;
- North Fork Rose Creek (considered both upstream and downstream of the Rock Drain);
- Rose Creek Valley;
- X23;
- X2; and,
- Guardhouse Creek.

The drainage patterns for each dump are discussed in the following sections, in which each of the dumps is addressed.

4.2.4 Faro Northwest Dumps

4.2.4.1 Description

The Northwest Dumps are located to the northwest of the main pit, and north of the plant site area. The dumps were constructed primarily by end-dumping. There are three major lifts to the dump, referred to as the Upper, Middle and Lower Northwest Dumps.

These dumps cover a total area of about 393,000 m^2, with an average height of 21 m. The total tonnage of waste rock is estimated at about 15 million tonnes.

There are two other dumps located immediately to the north of the mill site and south of Northwest Dumps which are described as the "Lower Parking Lot Dump" and the "Upper Parking Lot Dump". These dumps were constructed between 1975 and 1976. The two dumps contain about 2.9 million tonnes of rock and cover an area of about 0.1 km^2.

4.2.4.2 Physical Stability

These dumps were constructed at their angle of repose on moderately sloping well drained terrain. The dumps have been stable since construction, over 30 years ago, and there are no signs of instability. There is no significant upstream water source that could cause elevated pore pressures in the dumps.

Over time, as the surficial rock weathers, some shallow slope creep and slumping of the surficial layers on the angle of repose dump faces may be anticipated. Such slumping is typical of natural slope forming processes in the area. Very little water flows from the dumps and there is no significant erosion from surface water flows.

4.2.4.3 Hydrology and Drainage

The Northwest Dumps drain to both Guardhouse Creek and to X23. The Upper and the Lower Northwest Dumps drain equally to the two. However the sulphides in the dumps are located in the X23 drainage and do not contribute to loadings to Guardhouse Creek. The Middle Dump is located such that about 60% of the drainage reports to Guardhouse Creek, however the sulphides are also located in the portion of the dump which is in the X23 drainage.

The Lower Parking Lot Dump, which is the smaller of the two dumps at 0.7 million tonnes, drains to Guardhouse Creek. The larger Upper Parking Lot Dump, which is located to the east of the Lower Dump, drains to both the Zone 2 pit and to the North Fork Rose Creek.

4.2.4.4 Geochemistry

The Northwest Dumps comprise primarily intrusives (diorite) and calc. silicates. The Upper Northwest Dump is primarily intrusives, with some schist. Free dumped sulphides are evident in one area of the Upper Dump, as shown on Figure 4-5.

The Middle and Lower Dumps are primarily schist and calc. silicates, with some free dumped sulphides evident on the Middle Northwest Dump. Sulphides are also evident in some areas of the dump slopes where a truckload was end dumped from the existing crest at the time. The composition of each of the three Northwest Dumps has been summarized in Table 4.14, and is shown graphically in Figure 4-22.

The Northwest Dumps are not considered to be net acid generating, and acidic seepage has not been detected from these dumps. Elevated sulphates and iron, and evidence of iron staining from the Upper Northwest Dump indicate there is some oxidation occurring, in localized areas (probably of sulphide deposition). These observations reflects the estimated geochemical composition of the dump shown below for metals and ABA (Table 4.16 and Table 4.17) and are dominated by the higher tonnage of sulphides in the Middle and Lower Northwest Dumps. The indicator ratio of NP/AP is dominated in these dumps by the presence of the calc. silicate, particularly in the Middle and Lower Northwest Dumps.

Only the Upper Dump would appear to have a net acid generation potential. Such ratios have to be used carefully however, and consider the physical circumstances in the field. In the Upper Dump, the sulphides tend to be concentrated in an area on the highest lift and as a unit could generate acidic drainage as seen in the extraction tests on these samples. If the water percolates through the dump, or combines with runoff from areas of alkaline drainage, neutralization reactions would modify the pH and enhance precipitation of iron and metals within the solubility controls - although some zinc and sulphate would tend to remain in solution.

The influence of the tonnage of sulphide, and to a lesser extent the schists, is also seen in the comparison of the contained soluble loads shown in Table 4.18. These dumps represent a moderate stored soluble load compared to the other dumps on site, as shown in Figures 4-20 and 4-21.

Table 4.16
Calculated Grades in Northwest Dumps

Upper Northwest Dump									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	49	0.9	20	4.6	0.5	393	24	2304	1714
min	1	0.4	7	1.6	0.3	123	12	185	135
max	209	1.9	56	7.2	0.7	787	35	5379	6147

Middle Northwest Dump									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	89	1.3	28	5.7	0.9	388	36	3616	2202
min	1	1.0	1.5	3.2	0.7	232	27	295	202
max	356	1.9	84	8.1	1.2	724	46	7872	8423

Lower Northwest Dump									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	79	1.1	25	5.2	0.8	374	34	2833	1743
min	1	0.9	14	3.0	0.7	220	26	244	178
max	349	1.7	68	7.2	1.1	686	42	6422	655

Table 4.17

Calculated Acid Generation Potential in Northwest Dumps

Upper Northwest Dump							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	1.7	0.3	1.4	42	14.8	-27.6	1.1
min	0.4	0.0	0.3	8	3.4	-113.7	0.2
max	4.5	1.0	3.5	120	45.1	28.4	3.4

Middle Northwest Dump							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	2.7	0.4	2.3	73	29.1	-43.4	2.0
min	0.7	0.1	0.6	14	24.1	-178.9	1.6
max	6.9	0.9	6.0	208	43.3	21.3	2.7

Lower Northwest Dump							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	2.1	0.3	1.8	57	26.0	-31.0	1.8
min	0.6	0.1	0.5	12	20.6	-134.9	1.4
max	5.4	0.7	4.7	161	41.1	19.9	2.5

Table 4.18

Calculated Stored Soluble Loads in Northwest Dumps

Upper Northwest Dump								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	1087	444	0	54	13	0	0	76

Middle Northwest Dump								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	1607	673	1	65	40	1	1	212

Lower Northwest Dump								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	1752	718	1	78	43	1	1	199

The Parking Lot Dumps were constructed later than the Northwest Dumps, although located within the same general area. Both dumps are comprised primarily of calc. silicates with some schist and minor sulphides.

The composition of these dumps is shown graphically in Figure 4-23

Table 4.19 and 4.20 summarize the solids geochemistry of the Parking Lot Dumps. Table 4.10 shows the calculated total soluble loads contained in these two dumps. Both of the dumps are primarily calc. silicate, resulting in the comparatively low estimated metal concentrations and overall predicted net acid consuming potential for both dumps. As shown in Figures 4-20 and 4-21, and Table 4.21, there is very little stored soluble load.

Table 4.19

Calculated Grades in Parking Lot Dumps

Upper Parking Lot Dumps									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	49	0.9	20	4.6	0.5	393	24	2304	1714
min	1	0.4	7	1.6	0.3	123	12	185	135
max	209	1.9	56	7.2	0.7	787	35	5379	6147

Middle Parking Lot Dumps									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	37	2.3	23	4.6	1.6	457	40	1784	1157
min	1	2.1	17	3.6	1.5	401	34	215	218
max	122	2.6	50	5.5	1.7	588	47	3586	4090

Table 4.20
Calculated Acid Generation Potential in Parking Lot Dumps

Upper Northwest Dump							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	1.0	0.1	0.9	29	51.3	22.0	3.5
min	0.5	0.0	0.5	13	48.1	-13.1	3.3
max	2.2	0.2	2.0	66	59.3	38.8	3.6

Middle Northwest Dump							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	1.6	0.2	1.4	44	60.3	16.1	4.1
min	0.6	0.0	0.6	16	58.4	-49.4	4.0
max	3.6	0.4	3.2	110	63.5	45.5	4.1

Table 4.21
Calculated Stored Soluble Loads in Parking Lot

Upper Parking Lot Dumps								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	295	105	0	13	8	0	0	26

Lower Parking Lot Dumps								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	66	25	0	2	2	0	0	11

4.2.5 Faro Valley Dumps

4.2.5.1 Description

The Faro Valley Dump was constructed during the same period as the Northwest Dumps, from the early development of the Faro Main Pit. This dump is located to the north of the open pit, in the original channel of the Faro Creek. Faro Creek was diverted around the pit to the northeast to allow mining of the pit. The dump fills the original creek channel and is, in part, draped over the

edge of the pit resulting in a variable dump height, with a maximum value of 23 m and an average value of 11 m.

The Faro Valley Dump is described in two sections; the larger Faro Valley North Dump covering an area of approximately 136,000 m^2 and smaller Faro Valley South Dump with an area of about 32,600 m^2. The two dumps contain a total of about 4.1 million tonnes of waste rock.

4.2.5.2 Physical Stability

These dumps are located on the deep Faro valley alluvium and immediately adjacent to the Faro pit north slope. They currently impound a shallow water pool on the upstream side. Stability of the southern slopes of this dump is dependent on the stability of the north wall of Faro pit in the Faro valley alluvium. The valley alluvium is an aquifer and has a relatively high water table which is drawn down as seepage occurs into the Faro pit. The alluvium has, over time, slumped and ravelled into the pit and this may be expected to progress with time. Large discharges, due to a breach or excessive seepage of the Faro Creek diversion, would result in more water being dammed behind the Faro Valley dump and/or greater seepage in the valley alluvium. Such increased seepage would result in additional slumping of the alluvial slopes and the Faro Valley dumps above them. Should the Faro Creek diversion breach at a time when flows are sufficiently large to cause overtopping of the Faro Valley dump then rapid breakback and erosion of the alluvial valley fill and Faro Valley dump material may be anticipated.

4.2.5.3 Hydrology and Drainage

The Faro Valley Dumps drain entirely to the Main Pit. The drainage from these dumps arises from three main sources:

- precipitation and infiltration for both dumps;
- "surface flow" from the old Faro Creek through the lower lift, foundation and alluvium of the old Faro Creek channel; and
- groundwater flow in the deep alluvial sediments of the upper Faro Creek valley

As discussed in more detail in section 4.1.3.4, seepage in the Faro Valley overburden soils represents a major inflow to the main pit. Seep surveys and pumping records from sumps which collect this seepage suggest that seepage rates are in the order of ~22.7 l/s. Most of this seepage apparently occurs in permeable alluvial sediments of the Faro Creek Valley perched on less permeable bedrock (see section 4.1.3.4).

4.2.5.4 Geochemistry

The composition of the Faro Valley Dump is similar to the Upper Northwest Dump and was constructed over the same period. Overall, the dominant rock type is schist lesser diorite and some sulphides as groupings of truck loads throughout the entire dump area. Immediately adjacent to the Faro pit, the dump is built on undisturbed gravel till; the overburden was not stripped off before the dump was constructed.

The Faro Valley North dump comprises about 50% schist from the early development of the Faro pit, with lesser schist, sulphides, calc. silicate and overburden. The Faro Valley South dump is much smaller but was built over a longer time period. This dump is primarily composed of schist and does not contain either calc. silicates or sulphides. The composition of these two dumps, as described in Tables 4.14 and 4.15, is shown graphically in Figure 4-24.

As noted in the static testing data, the older dumps tend to show lower sulphide and, in some cases, metal content. Tables 4.22 and 4.23 summarize the estimated metal, sulphide and alkali contents of each dump. The Faro Valley North Dump tends to have higher metal and sulphide content than does the Faro Valley South Dump, due in large part to the presence of sulphides in the North Dump.

Estimates for acid generation potential support field observations that both dumps contain net acid generating material, although much more so in the North Dump which contains sulphides. This dump also contains calc. silicates, which provides some neutralization potential in these weighted average calculations.

Comparison of the extraction tests on samples from the different ages of dumps indicates that, over time, there is oxidation and depletion of the soluble metals. Older sulphides, such as those from the Faro Valley Dumps, appear to have been depleted in soluble metals over time. However, the presence of sulphides within this dump do result in a moderate predicted soluble load from the Faro Valley North Dump in particular (Table 4.24).

It is recognized that for this, and other older dumps, the total soluble load may be somewhat underestimated by using near surface samples for this comparison as these samples would tend to be more completely flushed than samples at depth. This would be an issue to consider in terms of total treatment requirements if the dump were to be moved (for any drainage not reporting to the pit). However, it does reflect the conditions within the active flushing area of the dump - the area on which water quality predictions for the surface drainage are based.

Table 4.22

Calculated Grades in Faro Valley Dumps

Faro Valley North									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	100	0.7	26	5.6	0.6	364	34	2953	1765
min	1	0.4	13	3.2	0.4	174	26	245	166
max	494	1.3	71	7.8	1.0	747	43	7139	6662

Faro Valley South									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	71	0.6	19	4.7	0.5	414	32	742	564
min	1	0.3	11	3.0	0.3	177	24	109	119
max	479	1.2	28	6.0	0.9	779	40	3153	1569

Table 4.23

Calculated Acid Generation Potential in Faro Valley Dumps

Faro Valley North							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	2.1	0.3	1.8	57	13.5	-43.3	0.9
min	0.5	0.1	0.4	10	6.9	-148.8	0.5
max	5.5	0.8	4.8	165	33.5	8.9	1.7

Faro Valley South							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	0.5	0.1	0.4	13	9.9	-3.1	0.7
min	0.1	0.0	0.1	5	0.0	-18.5	0.0
max	1.3	0.4	0.9	30	38.8	14.1	2.1

Table 4.24

Calculated Stored Soluble Loads in Faro Valley Dumps

Faro Valley North								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	1162	473	1	56	31	1	0	119

Faro Valley South								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	218	84	0	13	5	0	0	10

4.2.6 Faro Main and Intermediate Dumps

4.2.6.1 Description

The Main and Intermediate Dumps are the largest waste rock dumps, as shown in Figure 4-19, and were used for waste rock disposal over a period of about 18 years. The Main Dump East was the first to be constructed, beginning in 1972. The Main Dump West was initiated in 1974. Deposition of waste rock in the Intermediate Dump began in 1979.

The Main and Intermediate Dumps are located to the south and southwest of the open pit, covering a total area of about 1.1 km^2. The tonnage of rock in each of the dumps is shown in Figure 4-19 and Table 4.13. The two dumps represent over half to the total waste rock on site at a combined total of 145 million tonnes.

4.2.6.2 Physical Stability

These dumps were constructed at their angle of repose on moderately sloping well drained terrain. The outer slopes of these dumps have been stable since construction (at least 6 to 10 years), and there are no signs of instability. There is no significant upstream water source that could cause elevated pore pressures in the dumps. Flooding of the Faro pit would not result in significant seepage to the base of these dumps.

Over time, as the surficial rock weathers, some shallow slope creep and slumping of the surficial layers on the angle of repose dump faces may be anticipated. Such slumping is typical of natural slope forming processes in the area. Very little water flows from the dumps and there is no significant erosion from surface water flows.

4.2.6.3 Hydrology and Drainage

The drainage reports to two different areas, that is, to X23 and to North Fork Rose Creek.

The Main Dump West reports entirely to X23. Drainage from the Main Dump East reports primarily to X23, although an estimated 30% reports to the North Fork of Rose Creek, based on original surface topography and bedrock contours. Drainage from the Intermediate Dump reports to the North Fork of Rose Creek, primarily via an old drainage channel that enters the North Fork downstream of the Rock Drain (Figure 4-10). Based on the original contours however, it appears that up to 30% of the drainage from the Intermediate Dam could enter the North Fork of Rose Creek upstream of the Rock Drain as shallow and deeper groundwater flow. For the purposes of the water balance, however, it has been assumed that the drainage all enters downstream since the chemistry does not change significantly between the two, and the drain will be removed at closure.

4.2.6.4 Geochemistry

The Faro Main Dump East is comprised primarily of calc. silicate and schist, with about 15% of sulphides and minor intrusives and overburden. The Faro Main Dump West is somewhat different in that it is about 75% schist, and about 10% each of sulphides and calc. silicate. The Intermediate Dump is about half calc. silicate, with 20% of the rock estimated to be massive and disseminated sulphides. The composition of each of the dumps is shown graphically in Figure 4-25.

The Main and the Intermediate Dumps were constructed with "sulphide cells" in the upper lifts. The design intention was to isolate the sulphides within the center of the dump and between layers of the alkaline calc. silicate or schist material to control drainage water chemistry. Calc. silicate material would also be draped over the outside face of the sulphide lifts. Deposition in the sulphide cells continued into 1991.

Observation of the Main and Intermediate Dumps shows that there are areas where sulphides are exposed on the face the dump, in both the lower and upper lifts, and have not been encapsulated within the calc. silicate or schist material. There are also more recently placed free-dumped piles of sulphides and schists, particularly on the Main Dump.

Thus there are two areas where sulphide deposition was focused as shown on Figure 4-5. In addition, Pigage (1992) notes that sulphide area evident in scattered areas of the uppermost dump slopes adjacent to the North Fork of Rose Creek. He also comments that the exposed portion of the lower lift of these dumps contains extensive amounts of schist and sulphide which suggests that there may be significant amounts of sulphide in the lower lifts of the central and southern portion of the Main and Intermediate Dumps.

Estimates of the metal content of the Main and Intermediate Dumps are shown in Table 4.25. The maximum values are strongly influenced by the sulphide content in each dump, although the average values are comparable to other dumps containing sulphides. The estimated range of acid generation potential within each of the dumps is summarized in Table 4.26. While the maximum sulphur values again show the influence of the sulphides, these sulphides represent a

relatively small proportion of the total dumps. Therefore only the Main Dump West is clearly acid generating. It is recognized however that the sulphides are located in localized areas within the dumps, that there are net acidic conditions within these localized areas, and that drainage with elevated sulphates, zinc and potentially other metals can be expected. Where calc. silicates are present along the flowpath, some mitigation of the drainage can be expected.

The presence of relatively fresh sulphides within these large waste dumps leads to a significant predicted total soluble load for zinc, sulphate, and acidity as well as other metals within the dumps. These are shown in Figures 4-20 and 4-21, and Table 4.27.

Table 4.25

Calculated Grades in Main and Intermediate Dumps

Main Dump East									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	89	1.2	26	5.6	1.0	399	38	2829	1680
max	1	1.0	16	3.6	0.8	256	31	254	193
min	420	1.7	70	7.4	1.3	712	47	6567	6311

Main Dump West									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	111	0.6	26	5.8	0.7	398	40	2197	1224
min	1	0.4	16	4.0	0.4	215	33	212	166
max	630	1.0	58	7.4	1.1	777	48	6090	4465

Intermediate Dump									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	83	1.6	28	5.6	1.2	398	39	3485	2076
min	1	1.4	17	3.6	1.0	294	32	300	220
max	317	2.0	83	7.7	1.4	667	48	7382	7961

Table 4.26

Calculated Acid Generation Potential in Main and Intermediate Dumps

Main Dump East							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	2.1	0.3	1.9	58	28.8	-29.1	2.0
min	0.6	0.1	0.5	12	24.1	-130.7	1.6
max	5.4	0.7	4.7	162	42.6	18.6	2.4

Main Dump West							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	1.6	0.2	1.4	43	13.2	-29.6	0.9
min	0.4	0.0	0.4	9	6.9	-100.5	0.5
max	4.0	0.5	3.5	121	33.6	4.3	1.3

Intermediate Dump							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	2.7	0.3	2.3	73	39.9	-32.9	2.7
min	0.8	0.1	0.7	16	37.1	-163.7	2.5
max	6.7	0.8	5.9	205	47.5	25.9	2.9

Table 4.27

Calculated Stored Soluble Loads in Main and Intermediate Dumps

Main Dump East								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	17397	7039	18	778	505	12	7	2057

Main Dump West								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	8001	3136	12	428	259	8	2	704

Intermediate Dump								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	10712	4496	7	364	318	6	7	1773

4.2.7 Faro Northeast Dumps

4.2.7.1 Description

The Northeast Waste Dumps are considered in three areas as shown in Figure 4-5; the Outer Northeast Dump, the Upper Northeast Dump, and the Lower Northeast Dump. These dumps are located to the northeast of the Zone 2 dumps, but are to the southeast of the Main Pit. The western portion of the Upper and Lower Dumps infill the Zone 2 pit.

The Upper and Lower Northeast dumps are relatively large, containing a total of 38.3 million tonnes of waste rock. Since these dumps are located within the pit, the dumps are high, averaging 31 and 39 m respectively. They also cover an appreciable area - a total of about 0.5 km^2.

The Outer Northeast Dump is small by comparison, containing about 0.2 million tonnes of rock, with an average dump height of 8 m and an area of 0.01 km^2.

4.2.7.2 Physical Stability

Comments regarding the physical stability of these dumps are similar to those for the Main dumps.

4.2.7.3 Hydrology and Drainage

The three components of the Northeast Dumps drain to both the pits and to the North Fork of Rose Creek. The Upper Northeast Dump drains almost equally to the Zone 2 pit and to the North Fork of Rose Creek, with about 5% of the drainage reporting to the Main Zone pit.

About 60% of the drainage from the Lower Northeast Dump reports to the Zone 2 pit, with the remainder reporting to the North Fork of Rose Creek. Although there is very little sulphide in this dump, that which is present is located in the portion of the dump which drains to the Zone 2 pit.

Although the Outer Northeast Dump lies not within the Zone 2 pit perimeter, it is located within the subcatchment of the Zone 2 pit and drainage from this dump reports entirely to the Zone 2 pit.

4.2.7.4 Geochemistry

The Upper, Lower and Outer Northeast Dumps contain almost equal amounts of calc. silicate and schist, which represents the majority of the waste rock in the dumps. The Outer Northeast Dump does not contain sulphides. The Upper and Lower Dumps may each contain up to 5% sulphides, occurring as sulphides mixed with schist and overburden and dumped in temporary berms (Pigage, 1992) or free dumped piles. Massive sulphides are visible mixed with calc. silicates in the ramp on the dump.

The composition of each of the three dumps is shown graphically in Figure 4-26.

The Upper and Lower Northeast Dumps are similar in terms of metal content (Table 4.28) and acid generation potential (Table 4.29) as they are of similar composition in terms of rock types.

The Outer Dump is much smaller and has generally lower metal content, lower sulphide content and higher NP. With relatively low sulphide content, and these sulphides generally in the upper lifts, these dumps are not predicted to be acid generating.

The Upper and Lower Northeast Dumps are quite large however, and therefore there is a moderate total soluble sulphate, and to a lesser extent zinc, load estimated for these dumps (Table 4.30).

Table 4.28
Calculated Grades in Northeast Dumps

Lower Northeast Dump									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	48	1.0	17	3.7	0.8	331	28	1165	757
min	1	0.9	11	2.6	0.7	215	23	136	132
max	263	1.4	34	4.7	1.0	537	34	3042	2581

Outer Northeast Dump									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	44	1.3	17	4.0	1.0	413	34	441	352
min	1	1.1	13	3.2	0.9	283	28	109	145
max	293	1.7	22	4.6	1.2	617	40	1804	795

Upper Northeast Dump									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	43	1.0	16	3.5	0.8	309	26	1128	739
min	1	0.8	10	2.4	0.6	204	21	130	125
max	227	1.4	32	4.4	0.9	496	32	2855	2539

Table 4.29

Calculated Acid Generation Potential in Northeast Dumps

Lower Northeast Dump							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	0.9	0.1	0.8	25	25.1	-0.3	1.7
min	0.3	0.0	0.3	8	20.6	-38.5	1.4
max	2.2	0.3	1.9	65	37.7	20.5	2.2

Outer Northeast Dump							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	0.5	0.0	0.4	13	33.0	20.1	2.2
min	0.3	0.0	0.3	8	27.5	12.9	1.9
max	0.9	0.2	0.7	22	48.0	27.7	2.8

Upper Northeast Dump							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	0.9	0.1	0.8	25	24.7	0.0	1.7
min	2.2	0.3	1.8	63	36.0	20.4	2.2
max	0.3	0.0	0.3	8	20.6	-37.8	1.4

Table 4.30

Calculated Stored Soluble Loads in Northeast Dumps

Lower Northeast Dump								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	4174	1607	4	211	103	3	1	324

Outer Northeast Dump								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	38	14	0	2	1	0	0	2

Upper Northeast Dump								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	2654	1025	3	131	63	2	1	213

4.2.8 Zone 2 Dumps

4.2.8.1 Description

The Zone 2 dumps are located mostly within the old Zone 2 pit, to the southeast of the Main Pit. The dumps were built as the pit was mined, with the Zone 2 East Dump built first in the early 1980's, and the Zone 2 West Dump built in the late 1980's.

In total, the two dumps comprise about 2.3 million tonnes of waste rock. The Zone 2 East Dump is the larger of the two in terms of tonnage and covers an area of about 0.1 km^2. The Zone 2 West Dump cover a slightly smaller area, at about 0.09 km^2. The difference in the two dumps is the height of the dump, as a result of the configuration of the area of the pit and surrounding topography into which each was dumped. The Zone 2 East Dump has a maximum height of 137 m and average height of 65 m, compared to values 67m and 34 m respectively for the Zone 2 West Dump.

4.2.8.2 Physical Stability

Except for the potential issues associated with a potential discharge of water from the Faro pit the same comments apply as for the Faro Main dump.

Should the Faro pit flood to the extent that discharge occurs through the slot formed by the east haul road entry ramp, then the discharge water would flow first into the Zone 2 pit. Once this is filled it would decant from the low point in the west lip of the pit rim and from there through the outer part of the Zone 2 dumps to the North Fork of Rose Creek. Under these remote circumstances there is some potential that the such flows would cause erosion and slumping of the Zone 2 dump at the seepage exit point.

4.2.8.3 Water Balance

The Zone 2 waste dumps are located in, and therefore drain primarily to the Zone 2 pit. There is a portion of the Zone 2 East Dump, which drains to the North Fork of Rose Creek, representing about 10% of the drainage from this dump.

4.2.8.4 Geochemistry

The composition of the Zone 2 Dumps is shown graphically in Figure 4-27. As shown in the Figure, the majority of the waste rock is schist and calc. silicate. Only the Zone 2 West dump contains sulphides, with the sulphides representing about 10% of the total waste rock in the dump.

The composition of the waste dumps is reflected in the metal content and the acid generation characteristics shown in Tables 4.31 and 4.32. Metal contents are dominated by the abundant schist in these dumps, and are reasonably typical of other waste dumps on site. On average, the dumps are not predicted to be acid generating and there are no major localized pockets of sulphides in the dumps.

Soluble loads are estimated to be moderate for the Zone 2 West Dump, and low for the Zone 2 East Dump (Table 4.33).

Table 4.31
Calculated Grades in Zone 2 Dumps

Zone 2 West									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	62	0.9	22	4.4	0.7	536	32	1249	2647
min	1	0.3	5	1.6	0.2	110	12	69	181
max	219	2.2	42	7.3	1.8	1203	64	4456	10578

Zone 2 East									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	27	1.0	19	4.1	0.8	603	34	530	1795
min	1	0.3	5	1.4	0.2	116	11	47	139
max	118	2.7	30	5.6	2.3	1242	71	1996	6932

Table 4.32
Calculated Acid Generation Potential in Zone 2 Dumps

Zone 2 West							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	2.8	0.1	2.7	84	19.6	-64.3	1.3
min	0.1	0.0	0.1	3	2.4	-178.9	0.3
max	6.1	0.6	5.5	187	59.7	46.8	4.1

Zone 2 East							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	1.2	0.1	1.1	33	22.3	-10.7	1.4
min	0.1	0.0	0.1	3	2.4	-72.3	0.3
max	2.8	0.5	2.3	81	73.1	58.9	5.1

Table 4.33

Calculated Stored Soluble Loads in Zone 2 Dumps

Zone 2 West								
Parameter	*SO_4*	*Acidity*	*As*	*Fe*	*Mn*	*Ni*	*Pb*	*Zn*
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	2225	1729	2	85	47	1	0	255

Zone 2 East								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	1020	225	8	277	139	5	0	14

4.2.9 Near Pit Dumps

4.2.9.1 Description

The "near pit" dumps are considered to include the Ramp Zone Dump, the Ranch Dump, and the Southwest Pit Wall Dump. The other dumps in the vicinity which can be seen on Figure 4-5 are discussed under "Low Grade Stockpiles", Section 4.2.10.

These dumps are located immediately to the south and southwest of the pit, and just north of the Main and Intermediate Dumps. The three were constructed between 1989 and 1991 and are relatively small dumps comprising a total of about 4.3 million tonnes of rock. Since the dumps are located at the edge of the pit and on the ramp, the dumps are quite high with a maximum height of 60 m. The total area therefore of the three dumps is comparatively low at about 0.2 km^2.

4.2.9.2 Physical Stability

The near pit dumps were developed on well drained terrain sloping away from the pit. The outer slopes of these dumps have been stable since construction (at least 6 to 10 years), and there are no signs of instability. There is no significant upstream water source that could cause elevated pore pressures in the dumps. Flooding of the Faro pit would not result in significant seepage to the base of these dumps. Instability of Faro pit to the extent that failures could affect the stability of these dumps is not anticipated.

Over time, as the surficial rock weathers, some shallow slope creep and slumping of the surficial layers on the angle of repose dump faces may be anticipated. Such slumping is typical of natural slope forming processes in the area. Very little water flows from the dumps and there is no significant erosion from surface water flows.

4.2.9.3 Hydrology and Drainage

The Ranch Zone Dump and the Ramp Dump drain directly to the Zone 2 pit. The Southwest Pit Wall Dump, which is located in the Main pit, drains entirely to the Main Zone Pit.

4.2.9.4 Geochemistry

The three dumps were developed over short periods of time and are quite different in composition, as shown in Figure 4-28. The Ranch Dump is about 85% schist, with lesser intrusives and sulphides. The Ramp Zone Dump comprises primarily calc. silicate with schist and up to 2% sulphides. The Southwest Pit Wall Dump is comprised of almost equal amounts of schist and sulphides.

Table 4.34 shows the overall metal content of the three dumps. Table 4.35 shows the range of acid generation and acid neutralization characteristics for the dumps. Total soluble loads with the dumps are estimated in Table 4.36.

The influence of the sulphides on the maximum zinc and lead grades is evident in the table, with predicted range of 3.1 to 0.01% Zn, and 1.5 to 0.04% Pb in the three dumps; the highest metal concentrations correspond to the highest total sulphur concentrations. Calcium concentrations tend to the opposite with the highest concentrations occurring with the lowest sulphide content. This is reflected to a degree in the comparatively low NP values in the Southwest Pit Wall Dump. Both the Southwest Pit Wall Dump and the Ranch Dump would be predicted to be net acid generating based on static tests.

The Ranch Zone Dump and the Ramp Dump are relatively small dumps for this site, and the total predicted soluble load is relatively low. The Southwest Pit Wall Dump however shows moderate soluble loads, primarily as a result of the sulphide content of this dump.

Table 4.34

Calculated Grades in Near Pit Dumps

Ranch Dump									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	52	0.7	22	4.5	0.6	583	32	949	2612
min	1	0.2	2	1.1	0.0	69	6	45	137
max	204	2.2	40	7.0	2.1	1257	63	3594	10469

Ramp Zone Dump									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	20	1.8	19	4.1	1.3	643	41	614	1339
min	1	0.7	11	2.5	0.7	227	25	87	219
max	76	3.4	27	5.5	2.4	1329	91	1942	4783

Southwest Pit Wall Dump									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	215	0.5	47	8.2	0.4	619	40	4175	7396
min	1	0.1	6	2.5	0.0	40	13	146	358
max	710	1.6	106	16.6	1.6	1789	67	14940	30674

Table 4.35

Calculated Acid Generation Potential in Near Pit DumpsTable

Ranch Dump							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	2.3	0.2	2.1	66	11.8	-53.8	0.3
min	0.1	0.0	0.1	2	0.0	-148.1	0.0
max	5.1	0.6	4.5	155	53.2	39.0	3.7

Ramp Zone Dump							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	1.1	0.1	1.0	32	45.7	13.9	3.9
min	0.2	0.0	0.2	5	8.1	-53.3	0.9
max	2.3	0.4	1.9	67	110.7	96.2	7.7

Southwest Pit Wall Dump							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	9.7	0.3	9.4	294	8.1	-285.8	0.2
min	0.4	0.1	0.3	3	0.0	-636.4	0.0
max	20.8	1.6	19.2	640	36.3	20.8	2.3

Table 4.36
Calculated Stored Soluble Loads in Near Pit Dumps

Ranch Dump								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	123	82	0	11	6	0	0	12

Ramp Zone Dump								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	197	122	0	14	8	0	0	18

Southwest Pit Wall Dump								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	2237	1962	1	28	23	0	0	298

4.2.10 Low Grade Stockpiles

4.2.10.1 Description

The main low grade stockpiles are shown on Figure 4-5, labeled as Stockpile "A" and "C" (SPA and SPC). These stockpiles were built from 1987 to 1990 with low grade ore from the Zone 3 pit. Some of the material originally placed in these stockpiles has been recovered to the mill for processing, and the stockpiles currently contain about 1.7 million tonnes. These two stockpiles are located to the southwest of the Main Zone pit, as shown in Figure 4-5.

There are eight other stockpiles or small dumps in the same vicinity;

- the two "Fuel Tank" Dumps (waste dumps);
- Mt. Mungly East and West (waste dumps);
- the Stockpiles Base;
- the Oxide Fines Stockpile;
- the Medium Grade Stockpile (currently active); and,
- the Crusher Stockpile (currently active).

4.2.10.2 Physical Stability

With regard to geotechnical properties the low grade ore is not significantly different from the mine rock. Thus the comments applicable to the Faro Main dumps apply also to the low grade stockpiles.

4.2.10.3 Hydrology and Drainage

Low-grade stockpiles A and C are located such that they drain primarily to X23, with about 30% of the drainage reporting to the Main Zone Pit. The other eight stockpiles or dumps all drain entirely to X23.

4.2.10.4 Geochemistry

In general, the stockpiles are primarily composed of sulphides as the intention of a stockpile is to process it through the mill. The Low Grade Stockpiles A and C, the Medium Grade, Oxide, and Crusher Stockpiles, and the Fuel Tank Dump West are composed entirely of sulphides. The other dumps contain a mixture of rock types, as shown in Tables 4.14 and 4.15.

The stockpiles show the highest metal concentrations and most significant acid generation characteristics of all of the rock piles on site. These data are summarized in Tables 4.37 and 4.38. The solids geochemistry is dominated by the abundance of sulphides in the majority of these stockpiles, and some of the dumps. Predicted soluble metal loads for each of these dumps are summarized in Table 4.39 and shown graphically in Figures 4-20 (sulphate) and 4-21 (zinc). The largest estimated loads are seen in the Low Grade Stockpiles A and C - the most recently placed of the stockpiles, and from the Fuel Tank East Dump.

Table 4.37

Calculated Grades in Stockpiles

Mt. Mungly East									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	135	0.9	35	7.1	0.8	380	39	5288	3094
min	1	0.6	17	3.7	0.5	192	30	395	230
max	549	1.6	118	10.5	1.2	838	52	11551	12137

Mt. Mungly West									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	164	0.8	41	7.7	0.6	249	33	8230	4738
min	1	0.5	16	3.1	0.4	115	25	555	261
max	499	1.4	173	12.3	1.0	725	47	16708	19143

Fuel Tank Dump E									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	71	1.2	22	4.9	1.0	450	40	847	520
min	1	1.0	16	4.0	0.8	298	34	143	165
max	454	1.6	32	5.8	1.3	720	47	3031	1472

Fuel Tank Dump W									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	305	0.4	71	12.8	0.3	215	42	16253	9305
min	1	0.0	22	3.8	0.0	1	28	1035	414
max	853	1.6	332	21.9	1.0	1079	67	32600	38000

Table 4.37 con't

Calculated Grades in Stockpiles

Oxide Fines Stockpile									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	305	0.4	71	12.8	0.3	215	42	16253	9305
min	1	0.0	22	3.8	0.0	1	28	1035	414
max	853	1.6	332	21.9	1.0	1079	67	32600	38000

Stock Piles Base									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	75	0.5	18	4.2	0.6	355	33	535	275
min	1	0.4	13	3.4	0.4	203	28	102	117
max	508	0.8	24	4.9	1.0	628	39	2642	623

Low Grade Stockpile A									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	437	0.2	80	13.2	0.2	664	51	8603	13936
min	1	0.0	11	4.4	0.0	1	23	285	661
max	1403	0.8	197	29.7	1.0	2507	73	30500	58300

Low Grade Stockpile C									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	437	0.2	80	13.2	0.2	664	51	8603	13936
min	1	0.0	11	4.4	0.0	1	23	285	661
max	1403	0.8	197	29.7	1.0	2507	73	30500	58300

Medium Grade Stockpile									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	437	0.2	80	13.2	0.2	664	51	8603	13936
min	1	0.0	11	4.4	0.0	1	23	285	661
max	1403	0.8	197	29.7	1.0	2507	73	30500	58300

Table 4.38

Calculated Acid Generation Potential in Stockpiles

Mt. Mungly East							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	3.8	0.5	3.3	103	19.1	-83.6	1.3
min	0.9	0.1	0.8	16	13.8	-282.7	0.9
max	10.0	1.3	8.7	304	35.5	8.1	1.9

Mt. Mungly West							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	5.9	0.8	5.1	159	15.2	-143.8	1.0
min	1.3	0.2	1.1	20	13.8	-465.1	0.9
max	15.6	1.9	13.7	481	21.0	-2.2	1.2

Fuel Tank Dump E							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	0.7	0.1	0.6	20	30.0	9.9	2.0
min	0.3	0.0	0.3	9	24.1	-9.7	1.6
max	1.5	0.2	1.3	45	47.6	21.7	2.5

Fuel Tank Dump W							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	11.6	1.7	9.9	309	0.6	-308.6	0.0
max	30.8	3.7	27.1	949	6.3	-27.5	0.2
min	2.4	0.4	2.0	34	0.0	-949.1	0.0

Oxide Fines Stockpile							
	% S total	% S SO4	%S sulphide	AP	NP	NNP	NP/AP
ave	11.6	1.7	9.9	309	0.6	-308.6	0.0
min	2.4	0.4	2.0	34	0.0	-949.1	0.0
max	30.8	3.7	27.1	949	6.3	-27.5	0.2

Table 4.38 con't

Calculated Acid Generation Potential in Stockpiles

Stock Piles Base							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	0.4	0.0	0.4	11	12.8	1.5	0.9
min	0.2	0.0	0.2	5	6.9	-4.9	0.5
max	0.9	0.1	0.8	24	31.2	6.9	1.2

Low Grade Stockpile A							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	19.9	0.4	19.4	607	3.0	-604.4	0.0
max	42.2	2.8	39.4	1306	12.1	-5.0	0.3

Low Grade Stockpile C							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	19.9	0.4	19.4	607	3.0	-604.4	0.0
min	0.8	0.2	0.6	5	0.0	-1306.1	0.0
max	42.2	2.8	39.4	1306	12.1	-5.0	0.3

Medium Grade Stockpile							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	19.9	0.4	19.4	607	3.0	-604.4	0.0
min	0.8	0.2	0.6	5	0.0	-1306.1	0.0
max	42.2	2.8	39.4	1306	12.1	-5.0	0.3

Table 4.39

Calculated Stored Soluble Loads in Stockpiles

Mt. Mungly East								
Parameter	SO4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	322	138	0	13	9	0	0	48

Mt. Mungly West								
Parameter	SO4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	84	40	0	2	3	0	0	19

Fuel Tank Dump E								
Parameter	SO4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	594	219	1	34	18	1	0	34

Fuel Tank Dump W								
Parameter	SO4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	51	25	0	1	2	0	0	13

Oxide Fines Stockpile								
Parameter	SO4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	191	93	0	4	6	0	0	49

Stock Piles Base								
Parameter	SO4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	686	253	1	42	22	1	0	34

Table 4.39 con't
Calculated Stored Soluble Loads in Stockpiles

Low Grade Stockpile A								
Parameter	SO4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	2703	2427	0	11	17	0	1	371

Low Grade Stockpile C								
Parameter	SO4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	2332	2094	0	10	15	0	1	320

Medium Grade Stockpile								
Parameter	SO4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	0	0	0	0	0	0	0	0

4.2.11 Haul Roads and Rock Drain

4.2.11.1 Description

There are three main structures which should also be considered within the discussion of waste dumps in terms of site water chemistry, despite the fact that their use is for infrastructure. These include:

- the North Fork Rock Drain, located on the North Fork Rose Creek to the south of the pit and the Intermediate Dump;
- the Outer Haul Road East which is located between the Intermediate Dump and the North Fork Rock Drain; and,
- the Outer Haul Road West which forms the haul road around the south of the Intermediate and Main Dumps.

The North Fork Rock Drain was built in 1988 and forms part of the haul road between the Faro mine and the Vangorda Plateau. The two Haul Road Dumps were built between 1983 and 1989, with the East Dump the first to be constructed.

4.2.11.2 Physical Stability

Since the haul road is constructed from mine rock it has similar stability characteristics to small rock dumps. No stability problems have been experienced on the haul road since construction.

The physical stability of the rock drain will depend on the long term maintenance of permeability through the drain. The drain was formed by end dumping coarse durable mine rock from the top of the haul road embankment as it was advanced over the North Fork of Rose Creek. As rock rolled down the dump slope the coarse boulders segregated from the finer material and rolled to the base of the slope. This has resulted in the accumulation of the largest boulders (typically from 0.5 to 1.5 m in diameter) at the base with the fines remaining higher up the slope. The permeability of the drain is demonstrated by its ability to pass the full flow of the North Fork of Rose Creek with only a modest rise in the water level on the inlet side.

Concern does however exist that, with time, there will be accumulations against the upstream drain face of debris and detritus carried by the North Fork steam flow, or that over the very long term there will be weathering of the drain rock resulting in a decrease in drain permeability. Under these circumstances there may be a slow but progressive rise in the height of water retained behind the rock drain embankment. The behavior of such a dam is uncertain as it may, over the very long term, be metastable with unpredictable increases and decreases in water level, as flow channels develop and plug, and concerns regarding the stability of the downstream discharge zone.

4.2.11.3 Hydrology and Drainage

The flows in the North Fork of Rose Creek are described in section 3.7. The resistance to flow through the rock drain, plus the accumulation of debris on the upsteam face of the rock drain, has resulted in a pond on the upstream side of a few meters in depth. During periods of high flow this pond level rises marginally indicating that there is ample flow capacity in the drain.

The Outer Haul Road East Dump is located on North Fork Rose Creek and drains entirely to the creek, upstream of the rock drain. Drainage from the Outer Haul West is somewhat more complicated. It is estimated that about half of the drainage reports to the Rose Creek Valley and to Rose Creek. About 30% of the drainage report to the North Fork Rose Creek, just downstream of the rock drain. The remainder reports to X23.

4.2.11.4 Geochemistry

The North Fork Rock Drain and the haul road were constructed entirely of calc. silicate material.

The Outer Haul Road Dumps are predominantly composed of schist, with the West Dump containing minimal sulphides (<2%) and the East Dump containing up to 10% sulphides. The composition of these two dumps is shown in Figure 4-29.

Tables 4.40 and 4.41 show the range of metal concentrations and acid generation and neutralization potentials for these three rock piles. Since the Haul Road dumps are comprised primarily of schists, the solids geochemistry reflects the schist analyses with a wide range of zinc and other metal contents, and an average content of about 0.25% Zn.

The Outer Haul Road West contains only 2% sulphides in terms of rock type composition but, with the relatively low NP of the schist, would be predicted to be net acid generating. However the total sulphide content averages only 1.6%. The Outer Haul Road East contains more sulphides and therefore tends to show higher metal and sulphide contents, and lower neutralization potentials. It would also be predicted to be acid generating based on solids analyses. However, the dominant material in this dump is schist, which does show an acid generation potential, rarely shows acidic pH values in the field. The potential for net acid generation from this, and other dumps dominated by schist therefore may not be strictly accurate in terms of kinetic behaviour (as discussed with the static testing). These Haul Road dumps show a moderate soluble load, as shown in Table 4.42 and Figures 4-20 and 4-21.

The North Fork Rock Drain, being composed of calc. silicates, is low in metals and soluble content, with an overall acid consuming potential. The monitoring stations located immediately upstream and downstream of the drain indicate no detectable increase in metals loads.

Table 4.40

Calculated Grades in Haul Road Dumps

North Fork Rock Drain									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	1	2.4	17	3.7	1.6	660	44	388	580
min	1	0.9	15	3.1	1.0	308	34	106	252
max	1	3.9	19	4.6	2.5	1321	103	897	1423

Outer Haul Road East									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	69	0.5	22	4.4	0.5	507	29	1283	2962
min	1	0.1	3	1.1	0.0	51	6	53	149
max	250	1.9	44	7.3	1.8	1165	55	4747	12036

Outer Haul Road West									
	As (ppm)	Ca (%)	Co (ppm)	Fe (%)	Mg (%)	Mn (ppm)	Ni (ppm)	Pb (ppm)	Zn (ppm)
ave	37	0.8	20	4.2	0.7	587	33	688	2097
min	1	0.3	3	1.2	0.1	96	8	45	134
max	151	2.4	34	6.1	2.1	1225	66	2601	8244

Table 4.41

Calculated Acid Generation Potential in Haul Road Dumps

North Fork Rock Drain							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	0.4	0.0	0.4	11	61.4	50.3	5.5
min	0.2	0.0	0.2	7	11.9	2.8	1.3
max	0.6	0.2	0.4	14	135.6	121.3	9.4

Outer Haul Road East							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	3.0	0.2	2.9	89	9.7	-79.8	0.3
min	0.1	0.0	0.1	2	0.0	-198.7	0.0
max	6.7	0.7	6.0	204	44.2	31.8	3.0

Outer Haul Road West							
	% S total	% S SO_4	%S sulphide	AP	NP	NNP	NP/AP
ave	1.6	0.1	1.5	46	16.9	-28.7	0.9
min	0.1	0.0	0.1	2	1.2	-101.8	0.2
max	3.7	0.5	3.1	110	62.1	48.0	4.3

Table 4.42

Calculated Stored Soluble Loads in Haul Road Dumps

North Fork Rock Drain								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load								

Outer Haul Road East								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	1577	1254	2	77	44	1	0	186

Outer Haul Road West								
Parameter	SO_4	Acidity	As	Fe	Mn	Ni	Pb	Zn
	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes
Total load	1804	959	6	240	118	4	0	124

4.2.12 Surface Water Chemistry

At the Faro site, it has long been perceived that the sources of contaminants are the waste dumps, the tailings and, particularly after closure, the seepage or discharge from the open pits. The objectives therefore in this assessment of the database are to evaluate the trends and mechanisms in contaminant release from each of these sources, compare the relative contributions to the total loading for the site, and from this identify priorities for closure and remediation. Trends are evaluated graphically in the following discussion. Although the entire database is evaluated in the appendices, the summary tables generally look at two time periods for which detailed and comparable data are available; the period at which the last detailed evaluation of water chemistry was done (1988 to 1991) for the tailings facility closure plan, and the most recent data set (1995 and 1996). This is also done as the data are somewhat limited for the period of 1993 and 1994, particularly during the shutdown.

The mechanisms by which contaminants can be released from the rock include:

- physical transport of particulates;
- leaching of more soluble metals, particularly from the more oxidized (over geologic time) portions of the deposit; and,
- the dominant mechanism of on-going sulphide oxidation and associated metal leaching after mining of the rock.

This review of the database also was used for the calculations of contaminant loading from the site, both for during operation (which was used as the base case for calibration of the loading spreadsheet) and then for the post closure scenarios.

4.2.12.1 Faro Creek Drainage

The diversion of Faro Creek around the open pit is described in Section 4.5.1. The diversion is sampled at five stations as shown in Figure 3-19:

- upstream of the diversion (FC Site #25);
- original Faro Creek Channel, upstream of the Faro Valley Dam (FCO);
- within the diversion channel, but upstream of diversion culvert (FDU);
- within the diversion channel, downstream of the culvert (FDL).

The upstream, or background Faro Creek station was discussed in Section 3.9.4.1 above.

4.2.12.1.1 Faro Creek Diversion (FDU, FCD or FDL)

These two stations are not licence stations, but are sampled as part of the periodic site monitoring for the seep surveys.

Data are summarized in Tables 4.43 and 4.44 for these two stations along the Faro Creek Diversion. The water chemistry at the two stations would be expected to be similar, as the only

potential influence on water chemistry would be from the natural materials used to construct the diversion. The water chemistry at these two stations is compared to the upstream chemistry for several parameters:

- pH (Figure 4-30)
- sulphate (Figure 4-31)
- total zinc (Figure 4-32)
- dissolved zinc (Figure 4-33).

TABLE 4.43

Data Summary for Station FDU: Faro Creek Diversion Upstream

Date	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
	pH unit	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
JAN 1988 - OCT 1996							
Mean	7.41	13.3	3.1	0.2	0.005	0.07	0.021
Minimum Value	6.51	7.0	2.1	0.2	0.001	0.01	0.008
Maximum Value	8.15	17.0	5.0	0.2	0.010	0.20	0.033
Median	7.49	14.6	2.7	0.2	0.005	0.04	0.021
Num. Values	4.00	4.0	4.0	2.0	4.000	4.00	4.000
JAN 1988 - DEC 1991							
Mean	7.83	16.0	4.0		0.001	0.03	0.021
Minimum Value	7.52	14.0	3.0		0.001	0.01	0.008
Maximum Value	8.15	17.0	5.0		0.001	0.05	0.033
Median	7.83	16.0	4.0		0.001	0.03	0.021
Num. Values	2.00	2.0	2.0		2.000	2.00	2.000
JAN 1995 - OCT 1996							
Mean	6.99	11.1	2.2	0.2	0.01	0.12	0.021
Minimum Value	6.51	7.0	2.1	0.2	0.01	0.03	0.009
Maximum Value	7.46	15.2	2.4	0.2	0.01	0.2	0.033
Median	6.99	11.1	2.2	0.2	0.01	0.12	0.021
Num. Values	2.00	2.0	2.0	2.0	2.00	2.0	2.000

TABLE 4.44

Data Summary for Station FCD: Faro Creek Diversion adjacent NE Waste Dumps

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	pH unit	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
JAN 1988 - OCT 1996							
Mean	7.47	13.4	1.7	0.2	0.003	0.12	0.018
Minimum Value	6.50	1.0	0.9	0.2	0.001	0.04	0.003
Maximum Value	8.16	21.0	4.0	0.2	0.010	0.21	0.075
Median	7.5.0	14.0	1.0	0.2	0.002	0.10	0.015
Num. Values	23.00	23.0	13.0	2.0	10.000	8.00	19.000
JAN 1988 - DEC 1991							
Mean	7.52	13.6	1.6		0.002	0.12	0.017
Minimum Value	6.50	1.0	0.9		0.001	0.06	0.003
Maximum Value	8.16	21.0	4.0		0.002	0.21	0.075
Median	7.66	14.0	1.0		0.002	0.10	0.015
Num. Values	21.00	21.0	11.0		8.000	6.00	17.000
JAN 1995 - OCT 1996							
Mean	6.97	11.3	2.3	0.2	0.01	0.123	0.027
Minimum Value	6.59	7.8	2.2	0.2	0.01	0.036	0.017
Maximum Value	7.35	14.9	2.4	0.2	0.01	0.21	0.037
Median	6.97	11.3	2.3	0.2	0.01	0.123	0.027
Num. Values	2	2	2	2	2	2	2

These figures and the detailed data show that the water quality along the diversion is similar, although there is a sight increase in zinc, calcium, and magnesium concentrations in the diversion compared to the upstream station. There is no evidence though of oxidation (based on sulphate values) or net acid generation along the diversion. Calculated loadings are slightly lower at the lower sampling station, FCD, due to leakage from the diversion to the open pit.

4.2.12.1.2 Old Faro Creek Channel (FCO Site #29 and/or 30) and Station A30

While the majority of the Faro Creek water is diverted around the open pit by the Faro Creek Diversion, there is a component of the drainage which leaks from the Diversion, and returns to the original Faro Creek channel. In addition, there is a small area of catchment within the original Faro Creek Valley (see Figure 4-10).

The seepage from the diversion and the precipitation within the valley drain to the old Faro Creek valley channel and are sampled at Station FCO.

Until 1990, this water was collected and pumped back to the Faro Creek Diversion, rather than allowing it to flow through, and under, the Faro Valley Dump and into the Main pit. Due to elevated metals in this water however it was not acceptable for discharge (to North Fork Rose Creek via the diversion) and, since 1990, the water has been allowed to flow into the Main Zone Pit.

During operation, the water flowed into the A30 sump and was pumped from the pit via X22A. Water samples were collected at both A30 and X22A. Station X22A also includes seepage from other areas of the pit wall and therefore does not directly represent the chemistry of the water from the old Faro Creek channel.

Currently, the drainage through the old Faro Creek channel reports directly to the pit and is sampled, if safe access is possible, at seep A30 in the pit wall.

The flow and water chemistry data for these stations are limited to the seep survey data, and are summarized in Tables 4.45 and 4.46 for stations FCO and A30 respectively. Selected data are shown graphically in Figures 4-34 through 4-38. A statistical review by year and/or season are provided in Appendix D.

TABLE 4.45

Data Summary for Station FCO: Old Faro Creek Site 29/30

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	**pH unit**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**
JAN 1988 - OCT 1996							
Mean	6.45	11.9	57.0	0.2	0.183	1.21	4.170
Minimum Value	3.87	1.9	31.0	0.2	0.045	0.06	1.780
Maximum Value	8.87	19.0	114.0	0.2	0.472	3.77	8.190
Median	6.77	14.0	50.0	0.2	0.129	1.00	3.150
Num. Values	13.00	13.0	13.0	2.0	9.000	9.00	13.000
JAN 1988 - DEC 1991							
Mean	6.49	12.7	58.0		0.221	1.38	4.473
Minimum Value	3.87	2.0	31.0		0.063	0.06	1.780
Maximum Value	8.87	19.0	114.0		0.472	3.77	8.190
Median	7.08	14.0	50.0		0.176	1.00	3.310
Num. Values	11.00	11.0	11.0		7.000	7.00	11.000
JAN 1995 - OCT 1996							
Mean	6.23	7.3	51.1	0.2	0.053	0.62	2.51
Minimum Value	5.69	1.9	46.8	0.2	0.045	0.21	1.86
Maximum Value	6.77	12.6	55.5	0.2	0.060	1.03	3.15
Median	6.23	7.3	51.1	0.2	0.053	0.62	2.51
Num. Values	2.00	2.0	2.0	2.0	2.000	2.00	2

TABLE 4.46

Data Summary for Station A30: Upper Pit Wall Zone MPA6 Sump

Date	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
	pH unit	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
JAN 1988 - OCT 1996							
Mean	7.11	48.8	237.1	0.2	0.10	2.429	16.647
Minimum Value	5.27	4.3	0.2	0.2	0.01	0.051	5.260
Maximum Value	7.96	79.0	403.0	0.2	0.28	12.900	42.700
Median	7.40	51.5	224.0	0.2	0.07	0.546	14.200
Num. Values	13.00	13.0	13.0	1.0	8.00	8.000	13.000
JAN 1988 - DEC 1991							
Mean	7.27	52.5	223.3		0.072	2.77	14.476
Minimum Value	6.30	33.0	0.2		0.013	0.14	5.260
Maximum Value	7.96	79.0	403.0		0.154	12.90	27.100
Median	7.43	51.8	223.5		0.047	0.61	13.150
Num. Values	12.00	12.0	12.0		7.000	7.00	12.000
JAN 1995 - OCT 1996							
Mean	5.27	4.3	403.0	0.2	0.280	0.05	42.700
Minimum Value	5.27	4.3	403.0	0.2	0.280	0.05	42.700
Maximum Value	5.27	4.3	403.0	0.2	0.280	0.05	42.700
Median	5.27	4.3	403.0	0.2	0.280	0.05	42.700
Num. Values	1.00	1.0	1.0	1.0	1.000	1.00	1.000

The 1995 and 1996 seep survey data for Station FCO indicate that sulphide oxidation and metal leaching are affecting the quality of the water. Solution pH values are neutral to slightly acidic, with elevated sulphates (up to 50 mg/L compared to the background of <5 mg/L). Calcium and magnesium values are also elevated compared to background for this area, indicting that there is some acid generation and neutralization occurring along the flowpath. These values are consistent with those measured in 1987 and 1988. For comparison, the average sulphate concentration in the five surveys in 1987/88 was 48 mg/L.

In terms of metal concentrations, zinc is the most elevated compared to background values at up to 3.2 mg/L with at least 90% of the zinc in the dissolved form. Iron, copper, lead and manganese values are also slightly elevated compared to background values. These values are consistent with values measured in 1987 and 1988, and do not indicate any increase in metal release.

The source of this loading is recognized to be some sulphide waste rock that was used in either construction or repair of a section of the Faro Creek Diversion embankment, on the downstream (i.e. Faro Creek Valley) side.

Sump and seep A30 represent the water that flows from Station FCO, through the old Faro Creek channel and into the pit. The change in water chemistry between the two stations would be caused primarily by drainage from the Faro Valley Dump.

Because of the pit filling and sloughing of the pit walls it was not possible to sample the A30 seep in 1996. 1995 data however show that the seep was slightly acidic at pH 5.4, with essentially no alkalinity, and acidity values of 85 mg/L $CaCO_3$ eq. These data are compared for the two stations in Figures 4-34 and 4-35.

Calcium and magnesium values are elevated above those at station FCO, indicating that there is still neutralization occurring along the flowpath. Further evidence of sulphide oxidation is seen in the sulphate concentrations, at 400 mg/L compared to 85 at FCO (Figure 4-36).

The most significant difference between the chemistry at A30 and FCO is the zinc concentration, at 42.7 mg/L total zinc and 42 mg/L dissolved zinc. Manganese, lead and aluminum were also elevated, although iron concentrations were relatively low at 0.05 mg/L. The combination of slightly acidic pH and low iron and only moderate sulphate may suggest that the dominant mechanism for contaminant release is leaching of stored products, with either local areas or a limited rate of sulphide oxidation.

Comparison of the statistics calculated for the 1987 and 1988 data (SRK, 1992, SRK Report 60648) with the 1995 seep survey suggests that there had been an increase in oxidation and in metal release from the Faro Valley Dump. Although it is recognized that there are seasonal peaks to the chemistry of this seep, the 1995 exceed the maximum values previously reported. The seep is more acidic, with sulphate and zinc concentrations almost double the previous maximum values, and no alkalinity. However, copper concentrations are essentially the same, and iron concentrations are much lower than the previous minimum values recorded.

4.2.12.2 Guardhouse Creek Drainage

Guardhouse Creek is located to the west of the Faro pit and waste dumps. There are no licence specified monitoring stations on the creek, but it is sampled periodically by the mine at four locations:

- W10 Upper Guardhouse Creek, upstream of the Northwest Dump;
- W8/9 Upper Guardhouse Creek, downstream of the Northwest Dump (at the toe);
- GHSCR1 Guardhouse Creek opposite truck wash area; and,
- GHSCR2 Guardhouse Creek upstream of the final culvert before entering the tailings pond.

4.2.12.3 Upper Guardhouse Creek at toe of dump (W8)

This station was monitored from 1987 to 1990, and then again in the seep surveys of 1995 and 1996. Upper Guardhouse Creek passes under a small portion of the Northwest Dump, as shown in Figure 3-19.

Table 4.47 summarizes the results of the sampling data at this station for the period of record. Results are shown graphically in Figures 4-39 to 4-42. A statistical review by year and/or season is provided in Appendix D.

TABLE 4.47

Data Summary for Station W8: Upper Guardhouse Creek d/s of NW Dump

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	**pH unit**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**
JAN 1988 - OCT 1996							
Mean	7.65	54.6	6.8	0.2	0.004	0.052	0.069
Minimum Value	6.70	30.7	2.0	0.2	0.001	0.029	0.011
Maximum Value	9.38	68.5	16.0	0.2	0.01	0.093	0.577
Median	7.63	55.0	5.8	0.2	0.003	0.056	0.024
Num. Values	15.0	15.0	15.0	3.0	11.000	11.000	15.000
JAN 1988 - DEC 1991							
Mean	7.68	56.7	6.2		0.002	0.055	0.082
Minimum Value	6.70	41.0	2.0		0.001	0.029	0.011
Maximum Value	9.38	68.5	16.0		0.004	0.093	0.577
Median	7.57	55.8	5.4		0.001	0.057	0.026
Num. Values	12.00	12.0	12.0		8.000	8.000	12.000
JAN 1995 - OCT 1996							
Mean	7.53	46.2	8.9	0.2	0.010	0.04	0.015
Minimum Value	7.34	30.7	5.2	0.2	0.010	0.03	0.013
Maximum Value	7.63	54.3	10.8	0.2	0.010	0.07	0.016
Median	7.63	53.5	10.8	0.2	0.010	0.03	0.015
Num. Values	3.00	3.0	3.0	3.0	3.000	3.00	3.000

The flow at this station is quite variable, ranging from 0.2 L/s to 19.9 L/s. The chemistry however is relatively consistent, with alkaline pH values, alkalinity averaging 54.6 mg/L, and very low sulphate concentrations. Seep survey results from 1995 and 1996 are consistent with these values. Figure 4-39 compares the pH values at the upstream station, W10, with this station, W8. Figures 4-40 and 4-41 show similar comparisons for sulphate and alkalinity.

Metal concentrations are also low at this station, similar to the upstream values (Figure 4-42). However at this, as at some other stations with low metal concentrations, the total zinc concentration sometimes exceeds the dissolved value as has happened even in the more recent samplings. This is noted only for zinc. Total zinc concentrations in 1995 and 1996 were lower than the average values in the earlier samplings, and slightly higher than measured upstream (0.013 and 0.015 mg/L at W8 compared to <0.005 mg/L upstream). Barium, magnesium and calcium are also slightly higher in the downstream station, compared to W10.

Based on these results, it appears that drainage from the area of the Northwest Dump under which the creek flows does not contribute significantly to metal loadings in this creek.

4.2.12.3.1 North Wall Interceptor (NWI)

Guardhouse Creek flows past the mill area, and is the diverted into the North Wall Interceptor, upstream of the tailings area. Natural (i.e. from undisturbed areas) and some runoff from the mill area also flow into the North Wall Interceptor Ditch.

The data for NWI are limited to three samples, as shown in Table 4.48.

TABLE 4.48

Data Summary for Station NWI: North Wall Interceptor

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	**pH unit**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**
JAN 1988 - OCT 1996							
Mean	8.00	119.6	22.7	0.2	0.007	0.05	0.015
Minimum Value	7.85	91.9	13.0	0.2	0.001	0.01	0.006
Maximum Value	8.30	134.0	31.0	0.2	0.010	0.11	0.039
Median	7.93	133.0	23.5	0.2	0.010	0.04	0.008
Num. Values	4.00	3.0	4.0	2.0	3.000	3.00	4.000
JAN 1988 - DEC 1991							
Mean	8.09	133.0	28.9		0.001	0.005	0.025
Minimum Value	7.88	133.0	26.7		0.001	0.005	0.010
Maximum Value	8.30	133.0	31.0		0.001	0.005	0.039
Median	8.09	133.0	28.9		0.001	0.005	0.025
Num. Values	2	1.0	2.0		1.000	1.000	2.000
JAN 1995 - OCT 1996							
Mean	7.92	113.0	16.6	0.2	0.010	0.08	0.006
Minimum Value	7.85	91.9	13.0	0.2	0.010	0.04	0.006
Maximum Value	7.99	134.0	20.2	0.2	0.010	0.11	0.006
Median	7.92	113.0	16.6	0.2	0.010	0.08	0.006
Num. Values	2.00	2.0	2.0	2.0	2.000	2.00	2.000

Although the pH at NWI remains alkaline, there is evidence of contaminant loading to this water course. Sulphate concentrations and conductivity tend to be slightly higher than at W8 (averaging 17 mg/L in 1995/96), as does alkalinity. In recent years, the zinc concentrations at NWI are lower than at W8; due probably to dilution.

4.2.12.4 North Fork of Rose Creek

The North Fork of Rose Creek originates upstream of the mine and to the east of the pit and dumps. Water from the Faro Creek diversion flows into the North Fork to the northeast of the waste rock piles. The North Fork of Rose Creek then flows past the Zone II pit and dumps, and through a rock drain under the Vangorda Haul Road, adjacent to the Intermediate Dump.

The water chemistry in the North Fork of Rose Creek is represented by the following stations, as shown in Figure 3-19:

- R7 which is on the North Fork of Rose Creek, upstream of the mine;
- FDL on the Faro Creek Diversion which joins Rose Creek downstream of R7;
- NF1 which is located on Rose Creek just upstream of the Vangorda Haul Road rock drain, and downstream of the Zone 2 pit and waste dumps;
- NF2 which is located on Rose Creek just downstream of the Vangorda Haul Road rock drain; and,
- X2 located at the mine access road, just prior to the confluence with the South Fork of Rose Creek.

Station R7 is discussed in the section on background water chemistry (section 3.9.4.2).

4.2.12.4.1 North Fork of Rose Creek at the Rock Drain (NF1 or NFU and NF2 or NFL)

The influence of the Zone 2 Pit, and the Zone 2 Dumps and the Northeast Waste Dumps can be evaluated by comparing the upstream water chemistry (R7 plus FDL) to that at station NF1. Although seeps have occasionally been detected from the Northeast Dumps to Rose Creek, it is anticipated that the majority of the drainage from the dumps flows in the subsurface through the underlying colluvium.

Stations NF1 and NF2 are not licence stations, but are monitored within the seep survey program. Data are available from monitoring from 1988 to 1990, and the seep surveys in 1995 and 1996. Comparisons have shown that the rock drain appears to have little influence on the water chemistry in Rose Creek, with values at station NF1 similar to those at NF2.

Tables 4.49 and 4.50 summarize the results of the sampling data at these stations for the period of record. Results are shown graphically in Figures 4-43 to 4-50, which show the measured values over time for both the NF1 and NF2 stations, for each parameter. A statistical review by year and/or season is provided in Appendix D.

TABLE 4.49

Data Summary for Station NF1: North Fork Rose Creek Site 1 u/s of Haul Road

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	pH unit	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
JAN 1988 - OCT 1996							
Mean	7.71	67.7	4.6		0.001	0.23	0.029
Minimum Value	7.23	48.0	1.0		0.001	0.12	0.006
Maximum Value	8.18	90.0	7.5		0.002	0.38	0.109
Median	7.65	63.0	4.0		0.001	0.25	0.010
Num. Values	7.00	8.0	9.0		7.000	7.00	9.000
JAN 1988 - DEC 1991							
Mean	7.71	67.7	4.6		0.001	0.23	0.029
Minimum Value	7.23	48.0	1.0		0.001	0.12	0.006
Maximum Value	8.18	90.0	7.5		0.002	0.38	0.109
Median	7.65	63.0	4.0		0.001	0.25	0.010
Num. Values	7.00	8.0	9.0		7.000	7.00	9.000
JAN 1995 - OCT 1996							
Mean							
Minimum Value							
Maximum Value							
Median							
Num. Values							

TABLE 4.50

Data Summary for Station NF2: North Fork Rose Creek Site 2 d/s of Haul Road

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	pH unit	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
JAN 1988 - OCT 1996							
Mean	7.61	74.6	7.8	0.2	0.003	0.15	0.019
Minimum Value	6.30	39.3	4.0	0.2	0.001	0.06	0.005
Maximum Value	9.16	132.0	16.0	0.2	0.010	0.30	0.051
Median	7.64	76.0	8.0	0.2	0.001	0.12	0.018
Num. Values	11.00	11.0	11.0	2.0	9.000	9.00	11.000
JAN 1988 - DEC 1991							
Mean	7.64	79.4	8.0		0.001	0.15	0.021
Minimum Value	6.30	53.0	4.0		0.001	0.08	0.010
Maximum Value	9.16	132.0	16.0		0.002	0.30	0.051
Median	7.64	78.0	8.0		0.001	0.12	0.020
Num. Values	9.00	9.0	9.0		7.000	7.00	9.000
JAN 1995 - OCT 1996							
Mean	7.46	52.9	7.2	0.2	0.010	0.17	0.009
Minimum Value	7.23	39.3	4.8	0.2	0.010	0.06	0.005
Maximum Value	7.69	66.5	9.7	0.2	0.010	0.29	0.013
Median	7.46	52.9	7.2	0.2	0.010	0.17	0.009
Num. Values	2.00	2.0	2.0	2.0	2.000	2.00	2.000

Solution pH values are neutral to alkaline at both stations, and have been over the period of record (Figure 4-43). Alkalinity values range from 49 to 132 mg/L, with the recent measurements similar those from the early 1990's (Figure 4-44). It appears that alkalinity may increase between stations NF1 and NF2, which is possible as the haul road (and rock drain) are composed primarily of calc. silicate.

Sulphate concentrations are consistently low (Figure 4-45), with the average value for NF1 of 4.5 mg/L and for NF2 of 5 mg/L.

Metal concentrations for both total and dissolved metals are generally low, as shown in Tables 4.49 and 4.50, and Figures 4-46, 4-47 and 4-48 for iron, manganese and zinc concentrations respectively. Although variable within each year, the values have remained consistent over the period of record. Figure 4-49 shows the zinc loadings for these stations and FCL for comparison.

4.2.12.4.2 North Fork at Confluence with South Fork (X2)

Station X2 is located on North Fork Rose Creek downstream of the rock drain and the haul road and upstream of the mine access road. This station includes loading from primarily the Zone 2 waste dump as well as the Intermediate Waste Dump and any upstream loadings in North Fork Rose Creek.

This station is a licence station and thus there is a reasonably extensive database for this station. Table 4.51 summarizes the results of the sampling data at X2 for the period of record. Results are shown graphically in Figures 4-51 to 4-57 which show the measured values over time. Appendix D provides a statistical review by year and/or season.

TABLE 4.51

Data Summary for Station X2: North Fork of Rose Creek u/s of mine access road

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	**pH unit**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**
JAN 1988 - OCT 1996							
Mean	7.55	95.9	16.5	0.04	0.005	0.30	0.0356
Minimum Value	6.20	24.0	2.0	0.02	0.001	0.09	0.000
Maximum Value	8.86	765.0	374.0	0.06	0.074	1.15	0.170
Median	7.53	86.0	11.0	0.04	0.002	0.19	0.025
Num. Values	119.00	72.0	91.0	42.00	122.000	69.00	122.000
JAN 1988 - DEC 1991							
Mean	7.48	94.8	16.9	0.05	0.004	0.27	0.034
Minimum Value	6.60	24.0	2.0	0.05	0.001	0.09	0.002
Maximum Value	8.25	765.0	374.0	0.06	0.049	1.15	0.150
Median	7.50	75.8	8.1	0.05	0.003	0.15	0.024
Num. Values	55.00	52.0	50.0	11.00	55.000	26.00	55.000
JAN 1995 - OCT 1996							
Mean	7.58	103.0	17.0	0.02	0.005	0.36	0.027
Minimum Value	7.20	33.0	4.0	0.02	0.002	0.12	0.000
Maximum Value	8.20	163.0	32.0	0.04	0.021	0.92	0.100
Median	7.50	129.0	19.0	0.02	0.002	0.22	0.018
Num. Values	21.0	7.0	17.0	18.00	23	18.00	23.000

The water quality data in the figures show a distinct seasonal trend. This is most evident for sulphate, zinc, manganese and alkalinity with peak concentrations occurring during the winter low flow period. Iron follows the opposite trend with peak concentrations occurring during the peak flows during spring freshet. This may be due to the different mechanisms of transport, with the flushing of particulate iron (precipitates) during the high flows of freshet, and relatively low production and flushing of dissolved metals. There are insufficient dissolved metals data over time, however, to evaluate this mechanism.

Alkalinity and pH have remained relatively constant since 1987 with average values of 96 mg/L and 7.56 respectively. Sulphate concentrations are above background and show a slight increase from 12 mg/L in 1987 to 16 in 1993. This increase can also be seen in the trend of the sulphate/(Ca+Mg) ratio. From 1993 on, annual average sulphate concentrations have remained constant.

Zinc, iron and manganese concentrations also remain relatively constant, other than the obvious seasonal variations, with average concentrations of 0.0372 mg/L, 0.304 mg/L and 0.0562 mg/L respectively. These values are above background. The concentrations of other contaminants such as copper, cobalt, lead and arsenic are consistently near or at detection limits and show no increasing or decreasing trend.

4.2.12.5 Waste Dump Drainage

4.2.12.5.1 Station X23

The evaluation of waste dump water chemistry is one of the most difficult tasks of the closure plan, in part because of the limited number of points at which surface flow is evident from the waste dumps. Station X23 collects drainage from most of the Main Dump, a portion of the Northwest Dumps, and the Low Grade Stockpiles and provides the most complete database of dump drainage. Seepage is sampled at X23 in the former Faro Creek Channel below the main waste dump. This station is also the largest regularly monitored input to the tailings facility. It is a licence specified station.

There are other influences on the drainage water chemistry at X23, notably, flow from the X22A discharge. During mining of the Faro pit however, the pit water discharge from X22A was pumped through a diversion ditch to the original Faro Creek Channel, upstream of X7. However, ditch leakage was high, estimated at about 50%, and some of the X22A flow leaked from the ditch to report to X23 via an old stream bed under the dump.

The data from X23 are summarized in Table 4.52. Annual statistics for the database are provided in Appendix D. Selected data are presented graphically in Figures 4-58 through 4-64.

TABLE 4.52

Data Summary for Station X23: Old Faro Creek channel at toe of waste dumps

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	pH unit	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
JAN 1988 - OCT 1996							
Mean	7.16	379.3	1649.0	0.08	0.017	0.589	35.303
Minimum Value	5.77	450.0	1.6	0.02	0.001	0.005	0.080
Maximum Value	8.08	630.0	3930.0	0.33	0.490	16.500	141.000
Median	7.10	434.5	1560.0	0.06	0.009	0.160	27.800
Num. Values	185.00	128.0	161.0	38.00	189.000	91.000	189.000
JAN 1988 - DEC 1991							
Mean	7.17	367.1	1306.6	0.06	0.016	0.901	25.165
Minimum Value	5.77	56.0	1.6	0.05	0.001	0.005	0.080
Maximum Value	8.08	630.0	2600.0	0.06	0.490	16.500	120.000
Median	7.18	427.5	1480.0	0.06	0.009	0.148	22.500
Num. Values	126.00	108.0	120.0	7.00	128.000	48.000	128.000
JAN 1995 - OCT 1996							
Mean	7.10	521.0	2709.0	0.13	0.038	0.193	36.269
Minimum Value	6.80	455.0	2536.0	0.02	0.002	0.070	23.220
Maximum Value	7.80	607.0	3200.0	0.33	0.450	0.390	54.100
Median	7.00	505.0	2650.0	0.14	0.015	0.160	35.200
Num. Values	22	6.0	17.0	17.00	22.000	17.000	22.000

The water quality at this station is typical of buffered ARD with elevated metals and sulphate at neutral pH.

The pH at X23 has been generally near neutral ranging from 5.8 to 8.1, although there can be considerable variability in each year, as shown in Figure 4-58 and 4-59. There are no clear seasonal trends in pH at this station.

Alkalinity values are also quite variable, although generally alkaline, with an apparent trend to increasing alkalinity.

Sulphate concentrations increased to a peak in 1994, with an apparent leveling off in recent years. Figures 4-60 and 4-61 show the trend in sulphate concentrations over the period of record and the seasonal patterns in sulphate concentrations.

Arsenic concentrations show a dramatic increase in 1995 from near detection limit of 0.05 mg/L to an average of 0.14 mg/L in 1995 (Figure 4-62). Lead (Figure 4-63) and copper concentrations also increased during 1995, although not as drastically, to 0.063 mg/L and 0.039 mg/L respectively. Iron concentrations have remained relatively stable since mid 1993 with a very slight increasing trend in 1995. The average iron concentration in 1995 was 0.19 mg/L.

There has been an appreciable decrease in total zinc and manganese concentrations since 1993/94 following an increasing trend from 1988. In 1995, average total zinc concentration was 36.4 mg/L. The trends in zinc concentration over time, and over the seasons are shown in Figures 4-64 and 4-65. There tends to be a seasonal peak in concentrations during freshet, as seen to a lesser extent for sulphate, particularly during the higher flow years. Nickel and cobalt exhibit the same behaviour over time with concentrations in 1995 of 0.324 mg/L and 0.131 mg/L respectively.

There are a number of possible explanations or contributing factors for these observations. It has been postulated that:

- the leakage from the X22A drain significantly influenced the chemistry at X23. It is also possible that the decrease is associated with the fact that in October, 1992, dewatering of the main pit stopped resulting in a decreased loading to X23. This appears to be supported to some extent by considering the trends shown in Figures 4-61 and 4-62 for sulphate and zinc respectively at X22, X22A, and X23. There is a slight lag seen between the peaks in the pit water compared to X23;
- since the end of pumping from the pit there has been a period of flushing of the pit water which leaked towards X23 from the flow channel. This is contributing to the apparent decrease in metals from the waste dumps when, in fact, the influence of the X22 water has dominated the dump drainage chemistry;
- the hydrologic cycle is controlling dump drainage quality. 1989 was a very dry year, in which there could have been an accumulation of oxidation products within the dump which are subsequently flushed over the very wet years of 1991 through 1993. Since that flushing, in the comparatively drier years of 1994 and 1995, the concentrations appear to be the same or decreasing because to the stored oxidation products have

been removed, at least along the flowpath channels during average to lower precipitation periods;

- there is a relatively slow movement of water through the dumps and the drainage to 1994 represents the first transport of oxidation products from the dumps - probably from the more readily oxidized disseminated sulphides. It is also possible that more rapid oxidation of the disseminated sulphides is complete and the slower oxidation rate of the massive sulphide oxidation is continuing or increasing.

Calcium and sulphate also peaked in 1993/94 with sulphate concentrations apparently declining to about 2700 mg/L. As a result the ratio of sulphate to Mg+Ca has been increasing slightly since 1993, initially due to increasing sulphate levels and more recently due also to the decreasing calcium concentrations. This suggests that initially oxidation and sulphate production dominated the chemistry and, more recently, depletion of alkalinity with continued sulphate production is dominating the drainage chemistry.

4.2.12.5.2 Station X7

During operation, the drainage from X23 combined with the drainage from the pit (X22), in the old Faro Creek Channel and the combined stream was monitored at X7, a licence specified station.

There is relatively little data available for this station since 1990; the data are summarized in Table 4.53. A statistical review of the data is provided in Appendix D.

TABLE 4.53

Data Summary for Station X7: Old Faro Creek Channel at mine access road

Date	LAB pH pH unit	TOTAL ALKALINITY mg/L	TOTAL SULPHATE mg/L	TOTAL ARSENIC mg/L	TOTAL COPPER mg/L	TOTAL IRON mg/L	TOTAL ZINC mg/L
JAN 1988 - OCT 1996							
Mean	7.13	150.1	1044.0	0.11	0.759	441.89	43.820
Minimum Value	5.83	13.0	439.0	0.07	0.002	0.10	9.160
Maximum Value	7.74	319.0	4057.0	0.14	12.800	4000.00	145.000
Median	7.19	124.0	690.0	0.13	0.037	17.50	25.700
Num. Values	21.00	19.0	24.0	3.00	24.000	13.00	24.000
JAN 1988 - DEC 1991							
Mean	7.13	150.1	770.0	0.07	0.819	388.12	38.359
Minimum Value	5.83	13.0	439.0	0.07	0.002	0.10	9.160
Maximum Value	7.74	319.0	1650.0	0.07	12.800	4000.00	145.000
Median	7.19	124.0	611.0	0.07	0.034	13.10	23.200
Num. Values	21.00	19.0	22.0	1.00	22.000	11.00	22.000
JAN 1995 - OCT 1996							
Mean			4051.0	0.13	0.097	737.60	103.895
Minimum Value			4045.0	0.13	0.092	701.10	97.090
Maximum Value			4057.0	0.14	0.101	774.10	110.700
Median			4051.0	0.13	0.097	737.60	103.895
Num. Values			2.0	2.00	2.000	2.00	2.000

The water chemistry at this station is dominated by the flow from X23, with some contribution from reactions along the flowpath between X23 and X7. Spills from the mill have periodically been dumped into this area. While the drainage is generally near neutral, zinc concentrations are elevated - and often are higher than measured at X23 as shown in Figure 4-66. There appears to be a continuing source of zinc between the two stations. In contrast, sulphate concentrations were typically lower at X7 than at X23 during the years when there was flow at X22 which provided dilution which is no longer available (Figure 4-67).

4.2.13 Groundwater Chemistry and Seepage Estimates

A considerable volume of groundwater data has been collected downstream of the Faro Rock Dumps from piezometers, wells and test pits. The Summary Report of the Schedule D Water License #IN-89-001 (SRK Report No. 60648) reviewed most groundwater data collected prior to 1992. Additional data have been collected since 1992 as part of routine groundwater monitoring programs and special groundwater investigations (i.e. SRK, 1994; RGC, 1996). The groundwater quality data are reviewed here in order to assess contaminant migration via this subsurface pathway. All available groundwater quality data are summarized in tables compiled in Appendix K.

The magnitude of subsurface seepage originating from the Faro rock dumps was estimated as well using Darcy's Law and observed and/or estimated permeabilities and hydraulic gradients. Seepage estimates were derived for seven reaches downstream of the Faro Rock Dumps as indicated in Figure 4-10. These seepage estimates are summarized in Table 4.54 and are discussed further in the context of groundwater quality observed downstream of the various rock dumps (see below).

Subsurface seepage from the Faro rock dumps can be expected in three distinct areas downstream of the dumps:(after SRK, 60648):

- west of the North Fork of Rose Creek, north of the Vangorda Haulroad;
- south of the Main, Sulphide and Intermediate Dumps;
- east of Upper Guardhouse Creek, north of the mill.

The installations and groundwater data for each of these three areas is described and reviewed sequentially below.

Table 4.54

Faro Rock Dumps, Source Areas and Estimates of Subsurface Seepage

Dump Subcatchment		Downstream Reach for estimating seepage						
Source and receiving water	Area	Reach ID (see Fig. 4-10)	Geological Material	Range of K values	Hydraulic gradient	Section Height	Section Length	Seepage Estimate
	ha			(cm/s)		(m)	(m)	m^3/yr
Northeast Dump draining into North Fork Rose Creek	32	1	gravelly sand over weathered schist and quartz diorite	2.20E-05	0.12	4	1100	3,700
				5.40E-04	0.12	4	1100	90,000
Zone 2 pit & dump draining into North Fork Rose Creek		2	weathered pyllitic bedrock	1.60E-06	0.15	18	280	384
				4.50E-05	0.15	18	280	10,800
Intermediate Dump draining into NFRC (north of Haul Road)	20	3	sandy gravel confined at depth	5.00E-04	0.08	4	600	30,000
				5.00E-03	0.08	4	600	300,00
Intermediate Dump (incl. Sulphide Dump) draining southwest towards NFRC (south of Haul Road)	43	4	silty/sandy till over weathered bedrock	1.00E-05	0.08	6	450	680
				5.00E-04	0.08	6	450	10,200
Southern perimeter of Intermediate Dump draining south towards Rose Creek valley	15	5	silty/sandy till over weathered bedrock	1.00E-05	0.1	2	1400	890
				5.00E-04	0.1	2	1400	13,300
Main & Northwest Dump draining towards lower Faro Creek Valley	132	6	alluvial sands and gravel	1.00E-03	0.02	5	250	7,800
				1.00E-02	0.02	5	250	78,000
Northwest Dump draining into (Upper) Guardhouse Creek	35	7	silty/sandy till over weathered bedrock	1.00E-05	0.08	2	1040	580
				5.00E-04	0.08	2		30,200

4.2.13.1 West of North Fork

The area west of the North Fork and north of the Haul Road rock drain receives seepage from the Intermediate Dump, the Zone 2 Pit and Dump, and the Northeast Dump. A number of boreholes were drilled in this area during separate drilling programs to characterize groundwater conditions and monitor future contaminant migration (see Fig. 4-68 for borehole locations and Appendix C for borehole logs):

- "BH-90 series" (BH 1-4): Four shallow boreholes were drilled into the overburden soils of the North Fork valley to the south-east of the Zone 2 pit. Water quality in these boreholes is being monitored since 1990. The piezometer in BH-3 was inadvertently destroyed in December 1991 and is not sampled any longer.
- "BH-94 series" (BH 94-5 to 16): A total of 12 boreholes were drilled in the area west of the North Fork as part of a detailed groundwater investigation in 1994 (SRK, 1994). The drilling program included three shallow boreholes drilled through the valley sediments of the North Fork valley floor south-east of the Zone 2 pit/dumps and the adjacent Intermediate Dump (BH 94-5 to 7), four deep boreholes (BH 94-8 to 11) drilled through the Zone 2 mine rock into the underlying weathered bedrock immediately downslope of the Zone 2 pit, and three shallow boreholes (BH 94-12 to 14) drilled through the overburden soils at the toe of the Northeast Dump and into weathered bedrock. Most of these boreholes were equipped with multi-level piezometers to allow groundwater sampling at discrete depths. These piezometers are being monitored routinely since 1994.
- An additional monitoring well (BH 96-6) was drilled in the fall of 1996 into deep overburden soils near the toe of the Intermediate Dump up-slope of the rock drain pond (RGC, 1996). A single piezometer was installed screening a water-bearing gravel layer at a depth of ~18m below ground surface (Appendix C). Groundwater from this piezometer has been sampled once in September 1996.

In addition to these boreholes, a total of 19 test pits were dug and logged in the summer of 1990. Standpipe piezometers were installed (WE series) and the test pits backfilled (SRK, 1992). Nine water samples were collected from the piezometers in the test pits and analysed in the laboratory. There are no records of additional sampling of these piezometers subsequent to the initial sampling campaign.

The area west of North Fork is influenced by seepage from the Zone 2 pit and dumps, Northeast rock dumps and Intermediate Dump (see Fig. 4-68). The extent of subsurface seepage and contaminant migration originating from these three sources is discussed sequentially below.

Seepage from Zone 2 pit and dumps

Seepage from the Zone 2 pit to the North Fork Rose Creek is a distinct possibility, in particular, if runoff from the surrounding rock dumps is allowed to accumulate in the pit. In order to minimize seepage from the Zone 2 pit, a pumping well was installed in the deepest section of the pit in 1991.

Zone 2 pit water is routinely pumped from this well to control the water level in the pit (see section 4.5.5.). The quality of Zone 2 Pit water is monitored at the pumping well with samples being collected from pump discharge or from the well bore when the pump is removed for maintenance. Monitoring at the well is a permit requirement, and the point is designated X26.

Water quality data of the Zone 2 pit pumping well have been already been described briefly in section 4.1.5.3. A summary table of water quality data for X26 is shown in Table 4.11. The pH is consistently marginally acidic, and total zinc concentrations fluctuate between 48 - 70 mg/L. This is approximately half the concentration reported from an old monitoring piezometer prior to well construction. Values are highest in winter and lowest in summer. The lower values may reflect the effects of dilution from spring thaw recharge. Sulphate values do not appear to follow this seasonal pattern, typically ranging from 1000 - 1500 mg/L. The well is screened in the lower section of the pit profile only (a drop-set was released from a solid drill casing). Therefore, the water quality indicated is relevant to this part of the profile only. The quality of water higher in the profile is not known. It is also unknown to what extent the water quality monitored in the pumping well is representative of pit water quality in other parts of the Zone 2 pit.

In order to monitor subsurface seepage and potential contaminant migration from the Zone 2 pit to North Fork several boreholes have been installed west of North Fork immediately downslope of the Zone 2 pit and dumps (see inset in Fig. 4-68). Sample results of the four boreholes comprising the old "BH series" are summarized in Table K2 (all tables labelled "K" are shown in Appendix K). All four boreholes show signs of contamination with Zn levels significantly higher than the near-by creek. BH1 data indicate that water quality showed an improvement in the period from 1990 to 1992 (see Figure 4-69). During this period Zinc concentrations reduced from 4.32 to 0.89 mg/L and sulphate concentrations from 421 to 34 mg/L. The pH of the water has been marginally acidic throughout this period. Water quality has not changed significantly since 1992.

Water quality trends in BH2 show significant seasonal variations and less overall improvement since observation began in 1990 (Figure 4-69). In 1991 and 1992, when samples were taken frequently, peak concentrations of Zinc and sulphate were observed in May and June, i.e. during periods of high runoff, and lowest concentrations in the winter during low flow periods. A few measurements taken in 1994-1996 suggest that Zinc and sulphate concentrations in this monitoring well remain above concentration levels observed in BH-1 (Figure 4-69).

Data up to December 1991 indicated that the water quality of BH3 has fluctuated markedly and has not shown any clear improvement. Zinc concentrations were as high as 67.6 mg/L, but remained above 30 mg/L in 1991. Sulphate values have fluctuated in the 400-1200 mg/L range, while pH values varied from 4-6. The piezometer was inadvertently destroyed after December 1991.

The water quality in BH4 is the worst of the 4 wells with Zn concentrations ranging from over 125 to 48.7 mg/L and sulphate concentrations ranging from over 9000 to 158 mg/L (Figure 4-70). The pH is generally less than 4 and has remained so. Copper, iron and manganese concentrations are also appreciable, and all have shown improvements, copper from over 11 mg/L to 0.5 mg/L. Monthly sampling in 1992 also showed a strong seasonal trend with highest concentrations of Zn, sulphate and other contaminants during spring freshet and lowest concentrations during fall and winter (Figure 4-70).

It is not clear what the source(s) of the contamination is in the BH series wells. If either the Zone 2 Pit or also rock dumps are sources of contamination, it would be expected that boreholes BH1 and 2 would show the greatest contamination as they are closer to the sources. However, it is possible that the contaminant release is sufficiently old that the contaminant front has passed these boreholes. It is possible that highly contaminated water spilled from the pit prior to Curragh's operations commencing in 1986. The spill would have seeped into the shallow permeable sediments occupying the valley. This might explain why groundwater quality was generally improving following pit water-level control, and on-going groundwater recharge from creek leakage and the infiltration of precipitation.

An alternative source of contamination could be metal-rich outwash material present in the alluvial plain where the boreholes are located. Early during mining operation, Faro Creek was diverted into a small creek channel running over the Zone 2 orebody down into the alluvial plain. Apparently this diversion resulted in substantial erosion of overburden material and possibly mineralized bedrock as well. Such a local source of metal-rich outwash material could explain the very heterogeneous spatial distribution of contaminants in the valley sediments.

In 1994, several additional boreholes were drilled in the alluvial sediments south-west of the Zone 2 pit. Table K3 summarizes the water quality observed in these monitoring wells since 1994. BH 5 and 6 show Zn concentrations around 3-4 mg/L and sulphate concentrations ranging from 110-240 mg/L. Both wells are slightly acidic with pH values ranging from 5.3 to 6.05. Much lower contaminant levels were observed in piezometers BH 7A/B with Zn concentrations ranging from 0.47 to 0.73 mg/L and sulphate concentrations ranging from 25 to 43 mg/L. In both piezometers of BH 7 the pH was only slightly acidic (pH ~ 6.5). The water quality of these additional boreholes is similar to that observed in BH 1 and 2. The data strongly suggest that the much higher contamination observed in BH 4 is not representative of current seepage water quality originating from the Zone 2 pit. The significant variations in Zn levels observed in near-by wells in the alluvial sediments further suggest that groundwater quality is indeed influenced by acid-generating outwash material present in the alluvial sediments.

During the 1994 groundwater investigation several boreholes were drilled upslope of the outwash plain immediately below the Zone 2 pit into the weathered bedrock. The water quality observed in these deep boreholes is summarized in Table K4. Similar to the situation in the alluvial flood plain, there are large differences in the degree of contamination between these boreholes completed in weathered bedrock. BH 8 is heavily contaminated with Zn concentrations up to 158 mg/L and sulphate concentrations ~3500 mg/L. The pH in this borehole ranges from 5.1 to 5.7. In contrast, BH 9, 10 A/B and 11 show much lower levels of contamination with Zn concentrations ranging from 0.04 to 4.3 mg/L, sulphate concentrations ranging from 16 to 245 and pH values ranging from 5.3 to 7.3. Unfortunately, several of these monitoring piezometers (i.e. BH 9 and 11) were sampled only once or twice after installation and are now either frozen or malfunctioning.

The limited data available so far suggest that groundwater quality worsens during spring runoff when oxidation products are flushed from the rock dumps and possibly the Zone 2 pit and move down-gradient towards North Fork Rose Creek (see Figures 4-69 and 4-70 and time trends of BH 10A/B in Table K4).

The very high contamination levels in BH 8 require further commenting. BH 8 is located directly below the culvert under the access ramp, in the outwash channel. The borehole was augered through dry, alluvial outwash material and rust orange brown gravelly sands into highly weathered bedrock (SRK, 1994). Current seepage from the Zone 2 pit can not explain the contamination at this location since the observed concentrations of zinc and sulphate in this borehole are significantly higher than presently observed in the Zone 2 pit. Instead, it is likely that the elevated contaminant concentrations observed in BH 8 are a result of a historic release of the previously more contaminated Zone 2 pit water, prior to Curragh's start of operations in 1986. The observed poor recharge to the monitoring well suggests that the on-going pit water level control does not allow significant seepage from the Zone 2 pit into the outwash channel where BH 8 is located. This could explain why the contaminant levels in this borehole have decreased little over time and are still much higher than in the Zone 2 pit today. This hypothesis is supported by the fact that boreholes immediately down-gradient (e.g. BH 7A/B, BH 1) do not show comparable levels of contamination. It can also not be ruled out, that the high metal concentrations observed at this location are a result of oxidation of local outwash material originally derived from the Zone 2 orebody.

A cross-sectional flow model was developed to estimate the amount of subsurface seepage entering the North Fork Rose Creek along the reach below the Zone 2 pit and dumps (see Appendix J). In the model it was assumed that current pumping would control the water level in the Zone 2 pit at an elevation of 1110 MASL. The total volume of groundwater discharging into Rose Creek was estimated to be approximately 14,400 to 25,600 m^3/yr (see Appendix J), depending on the amount of recharge from precipitation. The model indicated that only a very small fraction of this groundwater flow would originate from the Zone 2 pit itself (assuming pumping is continued). The majority of subsurface seepage would originate from precipitation infiltrating into the Zone 2 dumps lying just down-slope of the Zone 2 pit and onto the flood plain itself (see Appendix J).

Using observed hydraulic gradients and average permeabilities observed in the monitoring wells the subsurface seepage from the Zone 2 pit and dumps to the North Fork was estimated to be in the order of ~400 to 11,000 m^3/yr (Table 4.54). These seepage estimates are within the same order of magnitude as those obtained by modelling and those provided in SRK Report A114101. Although total seepage from the Zone 2 pit and dumps to North Fork Rose Creek is relatively small this source may contribute significantly to the load balance due to the comparatively poor water quality.

Seepage from the Northeast Dump

Three boreholes were drilled at or beneath the toe of the Northeast Dump around the east-draining perimeter as part of the 1994 groundwater investigation (see Figure 4-68). The drilling indicated shallow overburden soils ranging in depth from 2.8 - 8 m overlying weathered bedrock. The water level was typically encountered at shallow depths some 2-4 meters below ground near the overburden-weathered bedrock interface. Several of the installed monitoring piezometers are frozen seasonally or permanently suggesting that this area is affected by permafrost.

The results of the water quality samples taken from these boreholes are summarized in Table K5. In general, the samples are near neutral with pH values between 6 and 7 and alkalinity between 200 and 400 mg/L. Sulphate concentrations range from 400 mg/L (BH 12) to about 1000 mg/L (BH 14).

Dissolved zinc concentrations range from <0.002 mg/L at BH 14 to 0.06 mg/L at BH 12. These values are comparable with results of shallow groundwater sampling from standpipes installed in test pits in 1992 (see WE 7 to WE 13 in Table K6). They do not indicate any significant loading of metals from the Northeast rock pile to the shallow groundwater.

The total area of the Northeast Dump draining towards the North Fork is in the order of 32 ha. Assuming an infiltration rate of 200 mm the total runoff from these dumps would be approximately 64,000 m^3/yr. The total runoff from the Northeast dump was previously estimated by SRK (1994). They assumed a surface area of 48 ha and an infiltration rate of 240 mm resulting in a somewhat higher runoff estimate (115,000 m^3/yr). Some of this runoff would occur as surface runoff and some would infiltrate into the permeable overburden soils and drain towards Rose Creek as shallow groundwater. Preliminary estimates of shallow subsurface flows from the Northeast Dump towards North Fork have been made using Darcy's Law and groundwater data from BH 12 to 14 (Table 4.54). It was assumed that the majority of groundwater seepage occurs in the overburden soil and weathered bedrock. Using the range of permeabilities measured in-situ in BH 12 and 14 (K of the weathered bedrock ranging from $5.4x10^{-4}$ cm/sec to $2.2x10^{-5}$ cm/sec) the total seepage would range from 3,700 to 90,000 m^3/yr (Table 4.54). These seepage estimates are within the range of estimates for total runoff from the Northeast Dump discussed above.

Seepage from the Intermediate Dump (north of Haul Road)

The area west of North Fork and just north of the Haul Road rock drain receives seepage from part of the Intermediate Dump. The amount of groundwater data for this area is limited. In 1990, several test pits were dug in this area and sampled for water quality (see Fig. 4-68). The pit logs indicated that bedrock was not encountered during excavation varying in depth from 1.8 to 4.3 m. The majority of the profiles logged are comprised of well-compacted sandy and silty till with some boulders and pebbles and a thin cover of organic material. Ice lenses were evident in all test pits near the creek. Two test pits near the North Fork stream channel were equipped with a standpipe piezometer and backfilled (WE 2 and WE 4). Shallow groundwater samples taken from these piezometers was slightly acidic (pH ~6.4) with alkalinities ranging from 188 to 212 mg/L CaCO3. Zn and sulphate concentrations ranged from 0.04 to 0.16 mg/L Zn and 60 to 378 mg/L SO4, respectively (Table K6). Subsequent sampling of these shallow piezometers in 1995 revealed even lower concentrations of Zn and sulphate. The data suggest, that any contaminated seepage migrating from the Intermediate Dump towards Rose Creek must be immobilized and/or diluted in the Rose Creek valley sediments prior to reaching the stream.

In 1996, a deep borehole (BH 96-6) was drilled upslope of the Rose Creek valley below the toe of the Intermediate Dump (see Figure 4-68). A water-bearing layer of sandy gravel was intersected at a depth of approximately 18 m and screened with a piezometer RGC, 1996). This gravel layer appears to be confined by several meters of predominantly silty till overlying the gravel zone at depth. No water was found in several meters of sandy till near the ground surface The water quality data obtained from this piezometer are shown in Table 4.55.

Table 4.55

Summary of water quality data for BH-96 series near Faro Rock Dumps.

Site	Date	pH (lab)	Acidity (mg/L)		Alkalinity (mg/L CaCO3)	Sulphate (mg/L)	Calcium (mg/L)	Cu (mg/L)	Fe (mg/L)	Zn (mg/L)
			pH 4.5	pH 8.3						
BH 96-6	26-Sep-96	8.00	<1	114	228	106	67.4	0.008	3.53	0.54
BH 96-7	27-Sep-96	7.66	<1	15	160	882	256.5	<0.002	1.86	<0.01
BH96-8A (shallow)	26-Sep-96	6.31	<1	102	484	2278	458.6	0.012	0.08	16.1
BH96-8B (deep)	26-Sep-96	6.30	<1	173	702	2576	632,2	0.026	0.40	2.37

The groundwater in this gravel layer down-gradient of the Intermediate Dump is slightly alkaline (pH = 8.00) and has an alkalinity of 228 mg/L. This groundwater is relatively high in Zn and iron (3.53 mg/L Fe and 0.54 mg/L Zn). Presumably, the elevated metal concentrations observed in BH 96-6 are a result of oxidation of mine rock from the Intermediate Dump. It should be noted, that the concentration of sulphate is relatively low (106 mg/L SO4) suggesting only limited neutralisation.

The total area of the rock dumps draining towards the North Fork upstream of the Haul Road is approximately 20 ha. Assuming an infiltration rate of 200 mm the total runoff from these dumps would be in the order of 40,000 m^3/yr. In this reach of the North Fork the overburden soils are deep and all runoff is expected to flow as subsurface seepage towards North Fork. Seepage estimates for this area based on Darcy's Law using groundwater data from BH 96-6 range from 30,000 to 300,000 m^3/yr (Table 4.54). These preliminary calculations suggest that essentially all runoff from this part of the Intermediate Dump could drain in a confined gravel layer similar to that encountered at BH 96-6 assuming such a permeable confined gravel layer is continuous in this region and extends at least along part of this reach.

The water quality data discussed above suggest that the current contaminant loading to North Fork Rose Creek as a result of seepage from the Intermediate Dump is small. Although there is potential for significant volumes of groundwater seepage at depth this seepage appears to be significantly diluted in the alluvial soils by creek leakage and/or infiltration of precipitation in the stream valley.

4.2.13.2 South of Main, Sulphide and Intermediate Dump

This area lies west of the North Fork of Rose Creek, east of the old Faro Creek Channel, and immediately south of several large rock piles including the "sulphide waste dump". Several monitoring boreholes were drilled in this area to monitor the quality of any seepage migrating south

TABLE 4.60

Data Summary for Station X5: Cross Valley Dam decant

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	pH unit	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
JAN 1988 - OCT 1996							
Mean	7.78	99.7	387.0	0.04	0.034	0.42	0.252
Minimum Value	6.28	14.0	54.0	0.02	0.001	0.04	0.001
Maximum Value	10.00	308.0	1300.0	0.10	0.318	4.03	1.240
Median	7.74	98.6	367.0	0.02	0.011	0.25	0.180
Num. Values	350.00	228.0	197.0	66.00	353.000	107.00	353.000
JAN 1988 - DEC 1991							
Mean	7.663	95.5	366.0	0.06	0.027	0.301	0.234
Minimum Value	6.280	29.0	263.0	0.05	0.001	0.044	0.001
Maximum Value	8.500	308.0	1300.0	0.09	0.262	1.110	1.050
Median	7.695	97.0	353.0	0.05	0.011	0.241	0.161
Num. Values	232.000	203.0	150.0	14.00	234.000	55.000	234.000
JAN 1995 - OCT 1996							
Mean	8.32	176.0	483.0	0.03	0.007	0.54	0.267
Minimum Value	7.00	14.0	236.0	0.02	0.002	0.05	0.010
Maximum Value	10.00	244.0	640.0	0.10	0.029	4.03	0.750
Median	8.30	202.0	495.0	0.02	0.006	0.29	0.300
Num. Values	37.00	6.0	28.0	32.00	41.000	32.00	41.000

Intermediate Dam Tailings Impoundments. Tailings still remain, however, in pockets at various locations within the Rose Creek flood plain downstream of the Cross Valley Dam.

4.3.3 Geochemical Characterization

The geochemistry of the tailings deposited in the Down Valley Tailings Impoundments has been analysed in some detail in the Down Valley Tailings Decommissioning Plan, SRK Report 60635 (SRK, 1991c). A re-evaluation of the tailings geochemistry was not warranted since the preferred closure option for the tailings is relocation and/or flooding of the tailings and since the methods of geochemical characterization have not changed significantly since that time. Instead, the water chemsitry in the surface waters draining from the impoundments and in the aquifer underlying the tailings deposits is reviewed in more detail (see sections 4.3.4 and 4.3.6)

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4.3.4 Surface Water Chemistry

The South Fork of Rose Creek is joined by the North Fork of Rose Creek upstream of the tailings impoundment, and is then diverted around the tailings facility in the Rose Creek Diversion. Flows in Rose Creek Diversion are directed into the old Rose Creek stream channel a few hundred metres downstream of the Cross Valley Dam. Additional flows entering the Rose Creek stream channel at this point are runoff from the North Wall Interceptor, decant water from the Cross Valley Pond, and seepage at the toe of Cross Valley Dam. These surface flows are sampled routinely for water quality (see Figure 3-19 for sampling locations). In the following water quality data for the following surface water sampling stations is reviewed:

- decant from the Cross Valley Pond (X5);
- seepage from the Cross Valley Dam (combined seepage at X13); and
- Rose Creek (X 14) downstream of the confluence of above mentioned flows from Rose Creek Diversion.

4.3.4.1 Pond Decant (X5)

X5 is a licence station, representing the treated water decanted from the tailings facility. The water chemistry at this station is dependent on the operation of the treatment and controls on decant, rather than directly on the geochemistry of the contributing sources.

The data for this station are summarized in Table 4.60, and statistical summaries are provided in Appendix D.

Tailings were deposited in the Intermediate Dam Impoundment from a single discharge at the northeast corner of the impoundment (near the north abutment of the Second Tailings Dam). This resulted in a sloped tailings surface, with the apex at the discharge point and the low point at the Intermediate Dam. Baffles were constructed across the tailings surface in 1990 and 1991 to steepen the tailings surface, but these were later inundated with tailings.

4.3.2.4 Cross Valley Dam

The second major embankment dam in the Down Valley scheme is the Cross Valley Dam, located approximately 500 m downstream of the Intermediate Dam (Figure 4-74). The reservoir formed between the two dams has been used as a polishing pond for excess water decanted from the tailings impoundments. Water from the Intermediate Dam Impoundment decants via siphon pipes and/or the emergency spillway into the polishing pond to allow suspended solids to settle. The water then exits the polishing pond via both siphon and spillway overflow at the north abutment of the dam.

The Cross Valley Dam was constructed in 1980-81 as a zoned earthfill embankment using borrow materials from within the Rose Creek valley. It is founded on permeable valley bottom sands and gravels, and incorporates both a low-permeability core and an upstream blanket of glacial till to control seepage. It has a 6 m crest width, and upstream and downstream slopes of 2 horizontal to 1 vertical. With a crest elevation of approximately 1033.4 m a.m.s.l. (1065.7 m Down Valley Project datum), the overall height of the dam is approximately 19 m at its maximum section. The stratigraphy of the soils underlying the Cross Valley Dam is shown on Figure 4-84.

The dam was originally provided with a seepage collection ditch at its downstream toe. This ditch was replaced by a granular toe drain in 1991.

The Cross Valley Dam is equipped with a riprap-lined emergency spillway on the north abutment of similar dimensions (and capacity) as the Intermediate Dam Spillway. The invert elevation of the spillway is approximately 1031.2 m a.m.s.l. (1063.5 m Down Valley project datum).

4.3.2.5 1975 Tailings Spill

On March 19, 1975, a breach occurred in the Original Tailings Dam releasing approximately 600 acre feet (740,000 m^3) of tailings water into Rose Creek (B.C. Research, 1975). This led to the breach of a second, temporary, dam that had been constructed across Rose Creek (see Figure 4-76 to provide emergency tailings storage during the winter of 1974/1975. Signs of the resulting flood were observed up to 10 miles downstream of the impoundment (1 mile upstream of the confluence with Anvil Creek). The creek had been frozen over at the time, and the wave of water filled the flood plain and froze in place. The Department of Indian Affairs and Northern Development completed a survey of the area on April 3, 1975, and estimated that approximately 200 acre feet (247,000 m^3) of frozen slurry, containing approximately 10 acre feet (12,300 m^3) of tailings solids, had been deposited between the tailings impoundment and the mouth of Rose Creek. The most significant deposits were located within the first 2 miles downstream of the impoundment. The area immediately below the breach was later incorporated in the Second and

Resources, 1987). Following October, 1986, tailings were placed in the Intermediate Dam Impoundment, with only occasional (emergency) discharge into the Second Impoundment.

4.3.2.3 Intermediate Dam Impoundment

In 1981, a new tailings area, termed the Down Valley Scheme (Golder Associates, 1980), was developed to the west of the Second Impoundment. The development involved the construction of two embankment dams across the Rose Creek valley, and extension of the Rose Creek Diversion, as shown on Figure 4-74. The first of the two dams is the Intermediate Dam, located approximately 2 km downstream of the Second Tailings Impoundment. It retains tailings deposited in the Intermediate Dam Impoundment, and is described below. The second dam, the Cross Valley Dam, is located approximately 500 m downstream of the Intermediate Dam. The Cross Valley Dam forms a polishing pond retaining surface water decanted from the tailings impoundments. The Cross Valley Dam is described in section 4.3.2.4. The extension to the Rose Creek Diversion is described in section 4.5.3.

Both the Intermediate and Cross Valley Dams were constructed as zoned earthfill dams, using materials from borrow source within the Rose Creek valley. The Intermediate Dam is shown in plan on Figure 4-81 and profile on Figure 4-82 (note that elevations shown on the figure are relative to the local project datum; subtract 32.3 m to convert to mean sea level datum). The first phase of the Intermediate Dam was constructed to a crest elevation of 1035.7 m a.m.s.l. (1068 m Down Valley Project datum), with provision for it to be raised to an ultimate crest elevation of 1050.7 m a.m.s.l. (1083 m Down Valley Project datum). It was raised (by downstream construction methods) in 1988, 1989 and 1991. The current crest elevation is 1049.4 m a.m.s.l. (1081.7 m Down Valley Project datum), giving it a maximum height of 34.4 m.

A representative section through the dam is shown on Figure 4-83. As shown on the section, seepage control is provided by a low-permeability core extending into a key trench excavated into the underlying alluvium. The core is provided with granular filter zones on either side of it, and a drainage blanket extends under the entire downstream toe. Upstream and downstream slopes were constructed at 2 horizontal to 1 vertical. The downstream slope also includes a 20 m wide bench at the 1031.7 m (a.m.s.l.) elevation (1064 m Down Valley Project datum), to give it an overall slope of 2.1 horizontal to 1 vertical at its maximum section.

The dam is founded on alluvial deposits. The northern two thirds of the dam rests on a thick deposit of cobbly sand and gravel (referred to as the "upper level terrace gravels", Golder Associates, 1980). The soils underlying the southern third, in the proximity of the Rose Creek channel, are less uniform in stratigraphy, and include layers of organic silt and silty sands typical of meandering stream channel environments. As most of the foundation drilling was completed using air rotary equipment, Standard Penetration Test "N" values were not generally recorded, so there are few data available for assessing the liquefaction resistance of the foundation soils.

The Intermediate Dam is equipped with a riprap-lined spillway channel on the right (north) abutment. The spillway has a bottom width of 30 m, a depth of 1.5 m (to top of riprap), an invert elevation 1047.7 m a.m.s.l. (1080.0 m Down Valley Project datum), and a maximum flow capacity (with no freeboard) of 88 m^3/s (Golder Associates, 1980b, Appendix IX).

Table 4.59

Second Tailings Dam, Construction History

Date	Crest Elevation (minimum)			Comments
	ft., Down Valley Project Datum	m., Down Valley Project Datum	m., mean sea level datum	
1974-75	3556	1083.9	1051.6	starter dam; diversion dam built to 3576 ft. elev.
1978	3566	1086.9	1054.6	West Dam raise
1979	3571	1088.4	1056.1	first significant construction of East Dam
1980	3586	1093.0	1060.7	

The West Dam was initially constructed as a zoned compacted earthfill dam, incorporating a low-permeability core, filter zone and drainage blanket. Subsequent raises were completed by centreline construction using both alluvial sands and gravels, and compacted tailings. The West Dam has a maximum height of approximately 90 ft. (27.4 m), and a downstream slope of 2 horizontal to 1 vertical. The available records indicate that it was designed and constructed according to sound engineering practices, and is expected to provide adequate stability under static loading conditions. Seismic stability is less assured, however. Foundation conditions under the West Dam are shown on Figure 4-80 (after Golder Associates). As shown on the figure, a portion of the West Dam (at its maximum section) is underlain by loose silt and fine sand up to 40 ft. (12 m) thick. Standard Penetration Test "N" values (uncorrected) range between 5 and 9, suggesting that these loose silts and fine sands would be susceptible to liquefaction during severe earthquake events.

The East Dam was constructed of compacted tailings sands, using downstream construction methods (see Figure 4-79). The final lift was capped with a protective layer of sand and gravel fill, and provided with a toe drain. The typical height of the East Dam (from downstream toe to crest) is 14 ft. (4.3 m). with a downstream slope of 2 horizontal to 1 vertical.

In the winter months tailings were deposited in the new facility from a single point discharge originating from various locations along the Original Tailings Dam. Excess surface water was decanted via a surface decant spillway located at the right abutment of the West Dam. In summer months tailings were spigotted from multipoint discharges along the crest of the new (Second) tailings dam, until 1978. Multipoint discharge was discontinued in 1978, and from 1978 to 1982 all tailings were deposited into the new impoundment from the hillside to the north of the impoundment, or from the Original Tailings Dam (Cornish, 1985). Tailings deposition was suspended in June of 1982, when the mine halted operations (Cyprus Anvil, 1983), and resumed in June of 1986, when the mine re-opened (Curragh Resources, 1987). From June, 1986, to October, 1986, all tailings were deposited in the Second Tailings Impoundment (Curragh

Table 4.58
Original Tailings Dam, Construction History

Date	Crest Elevation (average)			Comments
	ft., Down Valley Project Datum	m., Down Valley Project Datum	m., mean sea level datum	
1969	3540	1079.0	1046.7	starter dyke, mine waste rock
1970	3550	1082.0	1049.7	upstream raise, tailings
1971	3560	1085.1	1052.8	upstream raise, tailings
1972	3570	1088.1	1055.8	upstream raise, tailings and mine rock
1973	3585	1092.7	1060.4	upstream raise, gravel and tailings
1974	3600	1097.3	1065.0	upstream raise, tailings

Today the Original Tailings Dam is almost completely inundated by tailings deposited in the Second Tailings Impoundment (see Figure 4-77).

4.3.2.2 Second Tailings Impoundment

In 1973 studies began for the design of an expanded tailings storage area extending across the Rose Creek Valley. The layout of the resulting impoundment is shown on Figure 4-76. The new facility required diversion of Rose Creek along a terrace at the foot of the south valley wall, and construction of a new tailings retention dyke. Construction began in 1974 and was completed in 1975.

Details of the Rose Creek Diversion are provided in Section 4.5.3. The channel itself was completed in 1974, and was brought into service in April, 1975, at which time the diversion dyke at the upstream end of the impoundment was finally closed (Sigma, 1975).

Initial construction of the new tailings dyke began in 1974 and was completed in 1975. Subsequent raises were completed in 1978, 1979 and 1980. The construction history is summarized in Table 4.59. The new tailings dyke is divided into two sections, the East Dam and the West Dam, as shown on Figure 4-76. Typical sections are shown on Figures 4-78 and 4-79.

4.3.2 Description of Facilities

4.3.2.1 Original Tailings Impoundment

Tailings were initially deposited in the Original Impoundment, located on the north side of Rose Creek at the mouth of the old Faro Creek Channel. The impoundment is confined by the Original Tailings Dam, a pear-shaped embankment aligned approximately parallel to the original Rose Creek channel, as shown in Figure 4-76. Sections through the Original Tailings Dam are provided in Figure 4-77.

The impoundment was developed initially by constructing a 25 to 30 ft. high waste rock starter dyke extending along the southwest side of the pond and taking advantage of a natural esker at its east end. Construction details for the initial starter dyke are not available, although it was likely constructed by end-dumping with minor compaction from the haul trucks and dozers. The initial decant system consisted of a vertical riser leading to a 4 ft. diameter prestressed concrete pipe culvert placed in the base of the starter dyke (Golder Associates, 1973). Cornish (1985) reports that the initial starter dyke was raised in the winter of 1969 using uncompacted pit run waste rock with no impervious core.

Subsequent raises were constructed primarily of tailings using the upstream method of construction. Unclassified tailings were used in 1970, with cyclone separation introduced in the summer of 1971 (Cornish, 1985). During the summers, tailings were spigotted from a line on the crest of the dyke to build large fans extending into the ponds. The deposited tailings were then retrieved by draglines and shaped with crawler tractors to construct a continuous dyke on the pond side of the existing crest. In winter months tailings were deposited from a single point discharge at the north end of the pond. The initial decant tower was blocked and backfilled in the summer of 1971 and replaced by a surface decant.

In 1972, the eastern portion of the dyke was raised using mine waste rock, while the western portion was raised using tailings as described above (Cornish, 1985). In 1973, the dyke was raised using a combination of placed gravel and spigotted tailings. Dyke raising around the Original Tailings Impoundment continued each summer until 1975, when the Second Tailings Impoundment was ready for use. The construction history is summarized in Table 4.58.

Table 4.57

References Pertaining to the Development of the Rose Creek Tailings Facility

Date	Author	Title
Oct., 1973	Golder Brawner Assoc.	Report to Anvil Mining Corporation Limited on theTailings Containment Facility, Anvil Mine, Faro, Yukon.
June, 1975?	Golder Associates	Figures 3, 5, 6, 7, 8, 9 & 10 from report describing foundation conditions and material properties for the Second Tailings Dam (Project Number V74011), Text not available.
Sept., 1975	Golder Associates	Drawing No. V74011, Cyprus Anvil Mining Corporation, Tailings Containment Construction, Construction Carried Out to End of 1975.
Aug, 1977	Golder Associates	A Report Providing Geotechnical Recommendations Concerning Proposed Sloping Tailings Disposal Method for Cyprus Anvil Mining Corporation, Faro Mine, Faro, Yukon Territory
Apr., 1978	Golder Associates	An Update Study Concerning Design and Construction of Tailings Retention Structures at Cyprus Anvil Mines, Faro, Y.T.
Apr., 1979	Golder Associates	Report to Cyprus Anvil Mining Corporation Limited Summarizing 1978 Dyke Raising for the Tailings Retention Facility, Faro, Y.T.
May, 1980	Golder Associates	A Report to Cyprus Anvil Mining Corporation Summarizing Geotechnical Considerations, Tailings Containment Construction, 1980 Embankment Raising, Faro, Y.T.
June, 1980	Golder Associates	A Report to Cyprus Anvil Mining Providing Final Design Recommendations for the Down Valley Tailings Disposal Project, Faro, Y.T. (Vol. I, 2 & 3).
July, 1981	Golder Associates	Construction Report, Cyprus Anvil Mining Corporation, 1980 Raising of the Tailings Dam, Faro, Y.T. (Vol. 1, 2, &3)
Feb., 1982	Golder Associates	A Report to Cyprus Anvil Mining Summarizing the Down Valley Tailings Containment Project, 1980-81 Construction, Faro, Yukon (Vol. I, 2 & 3).
Nov., 1985	N. Cornish	History of CAMC Tailings Ponds (Dome Petroleum internal memorandum)
Apr., 1990	Golder Associates	Report to Curragh Resources Inc. Summarizing the Down Valley Tailings 1988 and 1989 Construction Projects, Faro, Yukon (Vol. 1 & 2).
July, 1991	Golder Associates	Curragh Resources, Intermediate Dam 1991 Raising and Cross Valley Dam Toe Drain Construction (construction drawings).

4.3 Rose Creek Tailings Facility

4.3.1 General

From the start of operations in September, 1969, until the start of in-pit deposition at the end of July, 1992, tailings from the Faro Mill were deposited in surface impoundments in the Rose Creek Valley. The resulting Rose Creek Tailings Facility is shown in plan in Figure 4-74, and is comprised of three impoundments: the Original Impoundment, the Second Impoundment and the Intermediate Dam Impoundment. Table 4.56 lists periods of use, surface areas and stored tailings volumes for each of the three impoundments. Tailings thickness isopachs (as of September 17, 1990) are shown on Figure 4-75. Details of the impoundment development are available from a number of references, listed in Table 4.57. The following description of the facilities has been derived from the references listed in the table.

Table 4.56

Rose Creek Tailings Facility, Tailings Volumes and Impoundment Surface Areas

Impoundment	Period(s) of Tailings Deposition	Surface Area (ha)		Tailings Volume (m^3)	
		As of Sept. 1990	Estimated Current	As of Sept. 1990	Estimated Current
Original Impoundment	Sept. 1969- June 1982	41.7	41.7	6,300,000	6,300,000
Second Impoundment	mid 1975 - June 1982, June 1986 - Oct. 1986	54.5	54.5	10,400,000	10,400,000
Intermediate Dam Impoundment	Oct. 1986 - July 1992	88	99	7,600,000	11,900,000[1]
Total		**184.3**	**195.7**	**24,300,000**	**28,600,000**

Notes: 1. Assuming 4,300,000 m^3 of tailings were deposited in Intermediate Dam Impoundment between September 17, 1990, and July 31, 1992 (350,000 tonnes per day at a settled density of 1.75 t/m^3).

constant (WW 9). For At WW-11, Zn and sulphate concentrations were 0.55 mg/L and 566 mg/L, respectively. Overburden depth appears to be very limited in this area and much of this seepage discharges into a swamp further downstream. Much lower concentrations of zinc and sulphate were observed at WW 9 (0.05 mg/L Zn and 19 mg/L sulphate, respectively).

The relatively thin till cover (2-4 m) over bedrock and generally fair groundwater quality presently precludes this area as a significant source of contaminated groundwater. The total area of the Northwest dump draining south-west towards Upper Guardhouse Creek and Guardhouse Creek proper is estimated to equal ~35 ha. Assuming an infiltration rate of 200 mm the total runoff from these dumps would be in the order of 70 dam^3/yr. Preliminary estimates of subsurface seepage suggests that only a portion of this runoff can enter the shallow overburden soils. Assuming typical permeabilities for the till material in this area ranging from $5x10^{-4}$ cm/sec to $1x10^{-5}$ cm/sec the total flow of subsurface seepage towards Guardhouse Creek is estimated to range from 580 to 30,200 m^3/yr (Table 4.54).

The water quality data and seepage estimates suggest that the load contributions from this source would be relatively small.

~6.3) but alkalinity values are relatively high (484-702 mg/L). There is a notable vertical gradient in Zn concentrations with high concentrations in the upper sediments (16.1 mg/L) and lower concentrations near the sediment-bedrock interface (2.37 mg/L). The water quality of shallow groundwater in the upper valley sediments is very similar to that of the near-by stream (e.g. ~23 mg/L Zn measured at X23) suggesting that there is a hydraulic connection between the stream and the groundwater. The water quality data suggest that the creek might be recharging the valley sediments.

The total area of the rock dumps draining south into the lower Faro Creek valley is approximately 132 ha. Assuming an infiltration rate of 200 mm the total runoff from these dumps would be in the order of 264,000 m^3/yr. The majority of this runoff enters the stream and discharges as surface runoff. Spot measurements of stream flow at X23 suggest that the annual surface runoff is in the order of 126 to 284 dam^3/yr. The remainder would flow as subsurface seepage in the valley sediments. Preliminary estimates of subsurface seepage along the lower Faro Creek valley are shown in Table 4.54. Assuming typical permeabilities for the permeable valley sediments ranging from $1x10^{-2}$ cm/sec to $1x10^{-3}$ cm/sec total seepage along the stream valley is estimated to range from 7,800 to 78,000 m^3/yr.

The water quality data and seepage estimates clearly indicate that subsurface seepage in the Faro Creek valley represents a significant component of the contaminant load leaving the Faro Rock dumps.

4.2.13.3 East of Upper Guardhouse Creek

This area is bounded by Upper Guardhouse Creek to the west, waste dumps to the east, and the mill site (formerly the Guardhouse Creek channel) to the south. No boreholes have been installed in this area, although a test pit program involving 14 pits was conducted in 1990 (SRK, 1992). Water samples were collected from 8 shallow piezometers (WW Series) (see Fig. 4-73).

The test pits varied in depth from 1.0 - 4.0 m. As in other areas, till overlies predominantly phyllite bedrock. Bedrock was encountered in 3 pits (12,13 and 14) in the southern part of the area. Weathering and fracturing were noted. Permafrost and ice was mostly absent in the pits.

The water quality data collected from these standpipe piezometers is shown in Table K6. Shallow groundwater in this area is generally slightly alkaline (pH typically ranging from 7.0 to 7.70). Zinc concentrations were generally below 0.07 mg/L, except for one sample (WW4 - 0.56 mg/L). Sulphates were similarly low, with some indication of neutralised drainage (WW3 - 141 mg/L, WW4 - 280 mg/L). One piezometer, WW11, located in close proximity of the Northwest dump showed high sulphate values of 1570 mg/L but very low zinc, indicating neutralization of acid rock drainage. The sample had a circumneutral pH (~7.3) and an alkalinity of 174 mg/L.

This area was revisited in the summer of 1995 and the shallow piezometers inspected. Shallow groundwater was encountered in only two piezometers, WW-11 and WW-9, both in close proximity of the dumps (see Figure 4-73). Zn concentrations were markedly higher at both locations compared to 1990 levels whereas sulphate concentrations either decreased (WW 11) or remained

In order to monitor seepage water quality directly south of the dumps a new borehole (BH 96-7) was drilled at the toe of the Intermediate Dump upslope near the drainage divide between North Fork and Faro Creek (see Fig. 4-71). This monitoring well was first sampled in September 1996. The results of this water quality sampling are shown in Table 4.55. The shallow groundwater is slightly alkaline (pH ~7.6) and shows an alkalinity of 160 mg/L. Concentrations of sulphate and iron are elevated (882 mg/L and 1.86 mg/L, respectively) suggesting some influence of acid rock drainage. However, metal concentrations of Zn and Cu were below the detection limit suggesting that these metals are either not released from the dump areas draining towards the tailings facility or are immobilized in the soil profile.

The total area of the rock dumps draining south towards the Rose Creek tailings facility is approximately 15 ha. Assuming an infiltration rate of 200 mm the total runoff from these dumps would be in the order of 30 dam^3/yr. Some of this runoff would occur as surface runoff and some would infiltrate into the permeable overburden soils and drain towards Rose Creek as shallow groundwater. Preliminary estimates of shallow subsurface flows draining from the southern perimeter of the Intermediate Dump south towards Rose Creek Valley are shown in Table 4.54. The majority of groundwater seepage occurs in the overburden soil and weathered bedrock. Assuming typical permeabilities for these units ranging from $5x10^{-4}$ cm/sec to $1x10^{-5}$ cm/sec estimated total seepage would range from 890 to 13,300 m^3/yr.

The water quality data and seepage estimates suggest that this area does not contribute significantly to the load balance of Rose Creek.

Seepage from Main Dumps and North-West Dumps draining south to Faro Creek channel

The main discharge point for seepage from the Faro rock dumps is the lower Faro Creek valley. Significant portions of the Main Dumps and the Northwest Dumps drain towards this valley. There is sufficient accumulation of dump seepage in this valley to maintain a small, perennial stream which emerges just below the Main Dump near the sampling station X23 (see Fig. 4-71). This stream shows consistently high levels of Zn, sulphate and other oxidation products.

In 1990 two shallow test pits were dug to the east of the stream channel and equipped with standpipe piezometers (see Fig. 4-71). At WS 10, in proximity to the stream the groundwater quality was significantly contaminated showing 3 mg/L zinc and 470 mg/L sulphate (Table K6). Some hundred meters further east in WS 11, the water quality in the shallow groundwater had markedly improved, showing only 0.016 mg/L zinc and 264 mg/L sulphate.

A new monitoring well (BH 96-8) was drilled in the lower Faro Creek valley next to the sampling station X23 as part of the 1996 drilling program (see Fig. 4-71). The alluvial sediments are eight meters thick at this location and consist of permeable sands and gravel overlying phyllite bedrock. One shallow piezometer (96-8A) was completed near the water table and a deep piezometer was completed just above the sediment-bedrock interface.

This monitoring well was first sampled in September 1996. The results of this water quality sampling are shown in Table 4.55. As expected the groundwater in the Faro Creek valley is impacted by acid rock drainage showing high sulphate and elevated Zn concentrations. The pH is slightly acidic (pH

mg/L in August 1989 to 937 mg/L in May of 1996 whereas Zn concentrations increased from 0.02 to 0.05 mg/L during the same period (Figure 4-72). The alkalinitiy in this shallow piezometer decreased from 324 mg/L to 184 mg/L. The pH of the groundwater at this location has also decreased from circum-neutral to slightly acidic (pH ~6.5).

At the shallow piezometer 2B, both Zn and sulphate concentrations peaked in 1990-1991 and have since levelled off again (Figure 4-72). For example, Zn concentrations increased from 0.056 mg/L Zn in August 1989 to 0.51 mg/L in August 1990 and have since stabilized at about 0.1 mg/L. Sulphate concentrations increased from 79 mg/L in September 1989 to a maximum of 1240 mg/L in September 1991 and have since levelled off to around 700 mg/L (Figure 4-72). However, there is significant seasonal variation in the data and more data are needed to verify this improvement in water quality at this location.

The water quality of groundwater flowing in the weathered bedrock zone (see "deep piezometers" 1A and 2A) shows similar general water quality trends than observed in the overburden soils (1B and 2B, see Figure 4-72). The groundwater in the weathered bedrock zone shows similar signs of the early stages of acid rock drainage, i.e. an increase in metals and sulphate and a decrease in alkalinity and pH.

The total area of the rock dumps draining towards the North Fork south of the Haul Road is approximately 43 ha. Assuming an infiltration rate of 200 mm the total runoff from these dumps would be in the order of 86,000 m^3/yr. Some of this runoff would occur as surface runoff and some would infiltrate into the permeable overburden soils and weathered bedrock and drain towards Rose Creek as shallow groundwater. Preliminary estimates of shallow subsurface flows in this area have been made using Darcy's Law and available groundwater data (see Table 4.54). The majority of groundwater seepage occurs in the overburden soil and weathered bedrock. Assuming typical permeabilities for these units ranging from $5x10^{-4}$ cm/sec to $1x10^{-5}$ cm/sec estimated flows of subsurface seepage would range from 680 to 10,200 m^3/yr.

The water quality data suggest that this area is significantly impacted by contaminated seepage from the rock dumps. Hence, this area may contribute significantly to the load balance for the North Fork despite the comparatively low seepage estimates.

Seepage from Intermediate Dump and Main Dump draining south towards Rose Creek Valley

The area south of the Main and Intermediate Dump was first investigated in 1990 when a series of shallow pits (WS series) were dug and logged (SRK, 1992). Pit depths varied from 1.0 - 4.3 m. In most cases permafrost and ice lenses were encountered at a depth of around 1.0-1.5 m. The profile typically consists of a thin till cover over weathered phyllite bedrock. Till thickness is generally 1 m, but thickens to over 4 m in proximity to the North Fork and the Faro Creek valley (near X23). Shallow groundwater was encountered in only 4 locations (see Fig. 4-71). With the exception of one location, shallow groundwater south of the dumps showed little signs of contamination (Table K6). Sulphate concentrations ranged from 53 - 89 mg/L and Zinc concentrations were near the detection limit (0.009 to 0.028 mg/L). Poor water quality was only encountered in WS 10 in proximity to the Faro Creek valley (see below).

towards the Rose Creek valley (see Fig. 4-71 for location of boreholes and Appendix C for borehole logs):

- "Intermediate Dump boreholes" (previously called "sulphide cell series"): Three monitoring boreholes were drilled in 1989 immediately south-east of the Intermediate Dump and the so-called "sulphide cells". Water quality in these boreholes is being monitored routinely since 1989.
- "BH 96 series" (BH 96-7 & 8): Two monitoring boreholes were drilled in 1996 south of the rock dumps as part of the 1996 drilling program. BH 96-7 is located immediately south of the sulphide waste dump and BH 96-8 is located in the Faro Creek channel south of the Main Dump.

In addition to these boreholes, test pits were dug throughout the area in the summer of 1990 investigate shallow groundwater conditions. Eighteen test pits were excavated below the Main Dumps and shallow piezometers (WS Series) were installed for water sampling where shallow groundwater was encountered.

The area south of the rock dumps is influenced by seepage from the Main and Intermediate Dumps and the so-called Sulphide Cells. This area can be subdivided into three sub-catchments draining into different directions (see Fig. 4-71). Most of the seepage from the Intermediate Dump and the Sulphide Dumps runs into a small gully now almost completely covered by the Intermediate Dump and the Haul Road. This sub-catchment drains into the North Fork of Rose Creek downstream of the Haul Road rock drain. Most of the seepage from the Main Dump and some seepage from the sulphide dumps drains towards the old Faro Creek channel where it either surfaces as toe seepage (station X23) or flows in the alluvial sediments. Some peripheral areas of the Main Dump and Intermediate Dump including parts of the Haul Road drain directly south towards the Rose Creek Tailings Facility.

The extent of subsurface seepage and associated water quality within these three sub-catchments is discussed sequentially below.

Seepage from Intermediate Dump and Sulphide Cells draining south-east to North Fork Rose Creek

Three boreholes were drilled in 1989 south-east of the Intermediate Dump and the Sulphide Dumps near the North Fork Rose Creek. Phyllite bedrock was encountered at 6-7 m depth in the boreholes, and was typically weathered. Several meters of till overly phyllite bedrock The bedrock was found to be weathered in the top few meters. The profile became saturated near the bedrock contact in each borehole. Piezometers were installed in each borehole, and in the weathered bedrock in two of them (boreholes 1 and 2). Water quality results are presented in Table K7 and discussed below.

The water quality data in the three boreholes indicate a trend of increasing contamination over the seven years of sampling (Table K7). During the observation period from 1989 to 1996, sulphate concentrations have increased almost tenfold in all piezometers while alkalinities have decreased markedly. Zn concentrations have also increased albeit at a much slower rate than sulphate concentrations. For example in piezometer 1B, sulphate concentrations have increased from 108

- upstream of the Rose Creek Tailings Facility
- downstream of Original and Second Tailings Impoundment
- at Intermediate Dam (downstream of Intermediate Impoundment)
- at Cross Valley Dam (downstream of Cross Valley Pond)
- downstream of Cross Valley Dam

The groundwater quality data for these five reaches are discussed sequentially below. All groundwater quality data for the boreholes discussed here are compiled in table form in Appendix K.

4.3.6.1 Upstream of Rose Creek Tailings Facility

Groundwater quality data for the reach upstream of the Rose Creek Tailings Facility is relatively sparse. The borehole P81-09 located near the pumphouse pond (Figure 4-97) was monitored from 1987 to 1990. In 1987, the concentrations of total Zinc and sulphate were at or close to background levels (0.03 mg/L and 11 mg/L sulphate, respectively). However, the water quality at this location deteriorated significantly over the next three years as a result of pumping groundwater from PW 4 and PW 5 into the pumphouse pond (see Fig. 4-97). The pumped groundwater had high concentrations of zinc (>1 mg/L) due to recharge from the Original Tailings Impoundment. Discharge of this contaminated water into the pumphouse pond and subsequent leakage from there towards P81-09 caused an increase in total zinc. By 1990, the zinc concentrations were as high as 0.19 mg/L (Table K8). Sulphate concentrations did not rise significantly above background levels (~20 mg/L) since the contaminated groundwater pumped into the pumphouse pond was also low in sulphates (~ 30 mg/L).

The practice of pumping contaminated groundwater to the pumphouse pond was discontinued in 1990 and presumably the groundwater quality in P81-09 has returned to 1987 levels. P81-09 is not sampled any longer.

4.3.6.2 Downstream of Original and Second Impoundment

The boreholes located immediately downstream of the Original and Second Impoundments show clear signs of contamination caused by recharge from tailings pore water infiltrating into the valley aquifer. For example, at X21 near the northern abutment of the 1974 tailings berm, sulphate and zinc concentrations are elevated in all three piezometers (Table K9). However, pH values are still circum-neutral suggesting that ARD is well-buffered in the aquifer. The time trends of sulphate at X21 are shown in Figure 4-99. The figure shows that sulphate increased from background levels in 1981 (30-46 mg/L) to a maximum of ~1200 mg/L in 1987. Unfortunately there are no records in the period from 1981 to 1987 when groundwater contamination levels peaked. Sulphate levels a this location improved slightly until the end of observation in 1989 (Figure 4-99).

Total Zinc concentrations increased from 0.03-0.05 mg/L in 1981 to concentrations as high as 1.55 mg/L in 1987. The total Zinc concentrations have decreased since 1987 but remain above background (0.06-0.096 mg/L in October 1989). A new borehole (BH 96-5) was drilled in 1996 just

the Original and Second Impoundments were estimated to range from 3450 to 13,800 m^3/day depending on local permeability conditions.

The Original and Second Impoundments are not used any longer for tailings disposal The Intermediate Dam Impoundment has generally finer and less permeable tailings, and a small pond. Current seepage is estimated to be less than 1200 m^3/day, comprised of approximately equal flows from the Intermediate Dam Impoundment and from the Original and Second tailings impoundments (see Table 4.63).

The flow regime in operation at the toe of the Down Valley Tailings impoundment is illustrated in Figure 4-98. The regime shown is based on a number of piezometer water-level readings. Two particular flow paths are shown:

- a near surface flow path comprising flow into and out of impoundment structures and ponds;
- a deeper flow path completely contained within the aquifer.

Near surface flow below the Intermediate Dam recharges the polishing pond, this being the flow path of least resistance. The polishing pond seeps to the groundwater at a rate of approximately 7000-8000 m^3/day (SRK, 1991). This is based on the flows measured at a number of weirs in the toe channel below the Cross Valley Dam. The majority of pond seepage comes immediately to surface in the toe channel, while a smaller component remains in the groundwater and comes to surface downstream as baseflow in Rose Creek. A portion of the deep flow path carries contaminants from seepage into the aquifer from the tailings impoundments up-gradient (see discussion on groundwater quality, section 4.3.6).

On decommissioning, the polishing pond will be closed, and the current seepage condition will disappear. The much smaller seepage component from the Intermediate Dam Impoundment will remain.

4.3.5.3 Rose Creek Diversion

The Rose Creek Diversion Canal and the Valley aquifer are not well connected hydraulically. Construction records indicate that the Canal was built on till or bedrock and was lined with till over permeable sections. Seepage from the canal has been estimated to be approximately 2900 m^3/day (Steffen Robertson and Kirsten, 1991c). However, this seepage volume is expected to enter the Intermediate Dam Impoundment and Cross Valley Pond, and not the aquifer. Higher than expected intermediate pond water-levels, and the presence of surface seeps below the canal, indicate that canal seepage flows towards the ponds rather than directly into the aquifer.

4.3.6 Groundwater Chemistry

The Rose Creek Tailings Facility has been built in several stages resulting in different "loading histories" in different reaches of the valley aquifer. For this and related reasons the network of groundwater monitoring stations is clustered in five areas (see Figure 4-97 for location of boreholes and Appendix C for piezometer installation details):

Hydraulic conductivities have been calculated from piezometer tests conducted in sand and gravel, silty sand and gravel, and till sections of the aquifer, and in bedrock and till adjacent to the valley. The hydraulic conductivities obtained from piezometer tests in the sand and gravel portion of the aquifer are in the range of $6x10^{-4}$ to $8x10^{-2}$ cm/sec (Table 4.64). Till and bedrock have hydraulic conductivities 2 to 4 orders of magnitude lower than the sand and gravel portion of the aquifer, on average.

The pump tests reflect the hydraulic conductivities of the more permeable portions of the aquifer as pump tests were only conducted where drilling indicated a possible large groundwater source for mill use. Groundwater flow and contaminant migration will occur predominantly in these portions of the aquifer, and therefore these values are the most appropriate for use in modelling. The pump tests were performed mainly in the aquifer upstream from the confluence of Faro and Rose Creeks. Values of hydraulic conductivity obtained varied from $1.2x10^{-2}$ to over 4 cm/sec (Table 4.65).

Groundwater Flow Estimates

The valley aquifer consists of coarse sands and gravels in distinct upper and lower layers, partially separated by a discontinuous silty horizon. Pump tests and piezometric readings indicate that these sand and gravel deposits act as an unconfined contiguous aquifer, with no clear separation between the upper and lower layers in terms of flow. Flow in the aquifer is predominantly horizontal flowing in a north-westerly direction along the principal direction of the valley. Hydraulic gradients along the valley profile range from 0.01 to 0.005 depending on local permeabilities and rate of recharge. The vertical flow component is very minor as indicated by very similar piezometric head values in different layers of the aquifer.

Information is sparse to define the exact nature of the hydraulic interaction between the tailings and native soils. Modelling was performed to assess water-level elevations in the aquifer for various tailings seepage rates (Steffen Robertson and Kirsten, 1991c). Model results indicated that, below the Original and Second Tailings Impoundments at least, a narrow unsaturated zone probably exists, and flows in the tailings and native soils are independent. While it cannot be directly verified, a similar profile was expected below the Intermediate Impoundment (Robertson and Kirsten, 1991c). The profile below the polishing pond is likely to be completely saturated due to the more permeable sediments in this pond.

Results of recent drilling carried out in the aquifer below the tailings of the Intermediate Impoundment suggest that flows in the aquifer and in the tailings may not be independent (RGC, 1996). The piezometric data from this borehole indicate that the piezometric heads in the sand and gravel layers below the tailings are higher than the aquifer-tailings interface. This would imply that the aquifer is now semi-confined by the tailings deposits, in particular in the reach of the Intermediate Impoundment where the tailings are significantly less permeable than the underlying sands and gravels.

Prior to tailings impoundment development, the aquifer received recharge from run-off, precipitation and Rose Creek. Average flow rates in the aquifer at this time were estimated to be in the order of 4800 m^3/day just below the confluence of Rose Creek and Faro Creek (International Water Supply, 1967). In 1981, Klohn Leonoff estimated that the Original and Second Impoundments were contributing 4000 m^3/day of seepage to the aquifer. Flow rates in the aquifer outside the influence of

Table 4.65

Hydraulic Conductivities of aquifer material form Pump Tests (Steffen Robertson and Kirsten, 1991c)

Pumping Well	Observation Well	K(cm/s)	S	Reported in*
TW1/67	A1	$3.3x10^{-1}$	$2.3x10^{-2}$	IWS, 1987
	A2	$3.3x10^{-1}$	$1.7x10^{-4}$	IWS, 1987
	A3	$3.3x10^{-1}$	$9.9x10^{-6}$	IWS, 1987
TW21	TH7	$4.4x10^{-2}$	$5.5x10^{-4}$	IWS, 1987
	85-1	$1.2x10^{-2}$	$1.2x10^{-3}$	IWS, 1987
TW28	TH6	$3.4x10^{-1}$	$5.5x10^{-3}$	IWS, 1987
	TH24	$3.4x10^{-1}$	$1.1x10^{-2}$	IWS, 1987
	TH26	$3.4x10^{-1}$	$5.9x10^{-3}$	IWS, 1987
	TH2	1.14	$9.2x10^{-3}$	IWS, 1987
	TH3	1.14	$3.0x10^{-3}$	IWS, 1986
	TH2	$5.9x10^{-1}$	$4.6x10^{-3}$	IWS, 1986
	TH4	$7.8x10^{-1}$	$7.4x10^{-3}$	IWS, 1986
	TH20	$6.7x10^{-1}$	$4.2x10^{-3}$	IWS, 1986
	TH29	1.16	$3.4x10^{-4}$	IWS, 1986
	TH31	$8.0x10^{-1}$	$6.5x10^{-3}$	IWS, 1986
	TH40	1.08	$7.7x10^{-2}$	IWS, 1986
	TH41	$8.0x10^{-1}$	$8.2x10^{-3}$	IWS, 1986
PW4	TH13	$2.1x10^{-1}$	$1.2x10^{-2}$	IWS, 1986
	TH14	$9.9x10^{-1}$	$7.8x10^{-4}$	IWS, 1986
	TH15	$9.9x10^{-1}$	$1.8 x10^{-3}$	IWS, 1986
	TH19	1.70	$1.4x10^{-4}$	IWS, 1986
	TH22	$8.0x10^{-1}$	$1.5x10^{-2}$	IWS, 1986
	TH24	$6.9x10^{-1}$	$1.1x10^{-5}$	IWS, 1986
	TH26	$6.9x10^{-1}$	$2.9x10^{-6}$	IWS, 1986
	TH27	$7.4x10^{-1}$	$1.2x10^{-2}$	IWS, 1986
	TH28	$7.8x10^{-1}$	$9.9x10^{-4}$	IWS, 1986
	TH34	$8.9x10^{-1}$	$5.0x10^{-4}$	IWS, 1986
	TH35	$6.9x10^{-1}$	$1.2x10^{-5}$	IWS, 1986
	P81-09	1.00	$1.0x10^{-3}$	IWS, 1986
PW5	TH14	$6.4x10^{-1}$	$4.7x10^{-2}$	IWS, 1986
	TH22	$5.3x10^{-1}$	$1.4x10^{-2}$	IWS, 1986
	TH24	$5.4x10^{-1}$	$8.1x10^{-6}$	IWS, 1986
	TH30	1.39	$5.8x10^{-2}$	IWS, 1986
	TH34	1.21	$7.7x10^{-4}$	IWS, 1986
	TH35	$5.1x10^{-1}$	$6.5x10^{-4}$	IWS, 1986
	TH39	$5.0x10^{-1}$	$5.1x10^{-3}$	IWS, 1986
PW6	PW3	$6.5x10^{-1}$	$3.3x10^{-3}$	IWS, 1986
	TH5	4.35	$1.7x10^{-1}$	IWS, 1986
	TH31	$4.4x10^{-1}$	$2.5x10^{-3}$	IWS, 1986
	TH40	2.15	$1.1x10^{-1}$	IWS, 1986
	TH41	$3.5x10^{-1}$	$1.2x10^{-3}$	IWS, 1986
TW2/67		$2x10^{-2}$		IWS, 1987
TW3/67		$3x10^{-2}$		IWS, 1987
TW6/67		$3x10^{-1}$		IWS, 1987

Data obtained from:
IWS 1987 = International Water Supply Ltd., Water Well Development Report #1, 1987, Book #1
IWS 1986 = International Water Supply Ltd., Report #2, Book #5, "24 Hour Tests Run on Wells PW3, PW4, PW5 and PW6 during October - November, 1986."

Table 4.64.

Hydraulic Conductivities of aquifer material from Piezometer Tests (Steffen Robertson and Kirsten, 1991c).

BOREHOLE/ PIEZOMETER	MATERIAL[1]	HYDRAULIC CONDUCTIVITY (cm/s)			REFERENCE
		FALLING HEAD	RISING HEAD	PUMP TEST (STEADY STATE/ TRANSIENT)	
81-K1/P1	SG			$2x10^{-2}/4x10^{-2}$	KL
81-K2/P1	SG			$5x10^{-2}/8x10^{-2}$	KL
81-K2/P2	SG	$>1.5x10^{-2}$	$>1.8x10^{-2}$	$2x10^{-2}/5x10^{-2}$	KL
81-K3/P1	B	$3.8x10^{-4}$	$1.3x10^{-4}$		KL
81-K3/P2	SSG	$2.2x10^{-2}$	$2.2x10^{-2}$		KL
81-K4/P1	SG	$>1.0x10^{-2}$	$>1.0x10^{-2}$		KL
81-K5/P1	SSG	$5.7x10^{-3}$	$>5.7x10^{-3}$		KL
81-K6/P1	SG	$>1.4x10^{-2}$	$>1.4x10^{-2}$		KL
81-K6/P2	SG	$>1.1x10^{-2}$	$>1.1x10^{-2}$		KL
81-K6/P3	SG	$>5.1x10^{-3}$	$>5.1x10^{-3}$		KL
81-K7/P1	SG	$6.7x10^{-3}$	$1.3x10^{-2}$	$3x10^{-2}/5x10^{-2}$	KL
81-K7/P2	SG	$1.4x10^{-3}$	$5.7x10^{-4}$		KL
81-K8/P1	SSG	$>5.0x10^{-3}$	$>5.0x10^{-3}$		KL
79-6	TILL	$2.3x10^{-6}$ to $4.5x10^{-7}$			GA
79-7	TILL	$1.7x10^{-5}$ to $3.2x10^{-5}$			GA
79-16	TILL/B	$5.9x10^{-6}$			GA
79-19	TILL/B	$8.1x10^{-5}$			GA
79-20	B	$<3.6x10^{-6}$			GA
79-21	TILL/B	$6.3x10^{-6}$			GA
		FALLING AND RISING HEAD TESTS (BY VAN DER KAMP AND HVORSLEV METHODS)			
		MIN	MAX		RGC
96-3A	SG	$6x10^{-2}$	$2x10^{-1}$		RGC
96-3B	S	$7x10^{-2}$	$2x10^{-1}$		RGC
96-4C	S	$1x10^{-2}$	$1x10^{-1}$		RGC
96-5B	SG	$4x10^{-2}$	$1x10^{-1}$		RGC
96-5C	SG	$3x10^{-4}$	$4x10^{-4}$		RGC

KL = Klohn Leonoff Ltd., Faro Mine Tailings Abandonment Plan, Sept. 1, 1981

GA = Golder Associates, Final Design Recommendations for the Down Valley Tailings Disposal Project, June, 1980

RGC = Robertson GeoConsultants Inc.Anvil Range Mining Complex - 1996 Piezometer Installation Program, December 1996

necessary to define the properties of the materials along the flowpath. Physical and hydraulic characteristics of the valley aquifer have been compiled in Steffen Robertson and Kirsten (1991c) and are summarized below.

Physical and Hydraulic Characterization

Figures 4-95 and 4-96 show idealized sections through the Rose Creek Valley deposits at several locations along the valley. The locations of these sections are shown on Figure 4-97. These sections illustrate that Rose Creek Valley is underlain by a complex of fluvial and glaciofluvial sands and gravels with irregular and apparently discontinuous fine grained deposits. The fine grained deposits are either lacustrine, silty till-like soils or fine grained layers in flood plain deposits. Both the sand and gravel and the fine grained portions of the valley fill have been dissected by fluvial and glacial action. A till layer has been encountered in most places between bedrock and the aquifer complex and in many areas a thin till-like layer has been reported to overlie the aquifer on the original ground surface.

Hydraulic conductivity values for the sand and gravel aquifer have been obtained from piezometer tests and from the reduction of pump test data. Storativities have also been determined from pump tests. Values are listed in Table 4.64 and 4.65, respectively.

- approximately 66 ha of the Intermediate Impoundment (i.e. ~2/3 of total area in the slimes zone) is ponded, i.e. covered with decant water;
- the average permissibility of the fine tailings underlying the ponded area ranges from $1x10^{-5}$ cm/sec to $1x10^{-6}$ cm/sec; and
- the hydraulic gradient below the ponded area approximates unity.

The above assumptions are considered conservative. The resulting seepage estimates for the Original, Second and Intermediate Impoundments compare well with those given in Steffen Robertson and Kirsten (1991c). A groundwater flow model has been developed to model the influence of the tailings impoundment on the valley aquifer for past, current and future (closure conditions (see Appendix L). The seepage estimates given above agree fairly well with preliminary results obtained from this flow model (Appendix L).

Table 4.63

Estimate of current seepage rates from Tailings Impoundments to Rose Creek Valley aquifer.

Impoundment	Surface Area	Seepage Rate	
		Average	Range
	(ha)	(m^3/day)	(m^3/day)
Original Impoundment	42	173	115 - 230
Second Impoundment	55	226	150 - 301
Intermediate Impoundment			
exposed area	33	136	90 - 181
ponded area	66	2800	509 - 5090
TOTAL	196	3335	864 - 5800

The potential for lateral seepage from the tailings deposits to the Rose Creek Diversion is limited to the upstream sections of the Original and Second Impoundments where under high recharge conditions (spring runoff) hydraulic heads in the tailings may rise temporarily above stream levels. No significant lateral discharge from the tailings to the Rose Creek diversion is expected along the reach of the Intermediate Impoundment due to relatively higher water levels in the creek compared with water-levels in adjacent piezometers, and in the tailings.

4.3.5.2 Rose Creek Valley Aquifer

Seepage from the tailings will enter the sand and gravel aquifer underlying the deposit. In order to predict the potential migration of contaminants discharging from the tailings in the aquifer, it is

properties. The approximate extent and total thickness of tailings in the three impoundments at present are shown as isopachs in Figure 4-75. The dykes separating the three impoundments have not been included. The tailings surfaces slope gently to the south away from the original discharge points.

A coarser tailings fraction is found on the northern sides of the Original and Second Impoundments. The tailings become progressively finer to the south. Intermediate Dam Impoundment tailings are generally finer on average, but also exhibit a coarse-fine separation from east to west as tailings were discharged in the east.

Steffen Robertson and Kirsten (1991c) reviewed all available hydraulic conductivity data for the Rose Creek tailings, including laboratory permeability tests and in-situ tests in saturated tailings. A wide range of hydraulic conductivities was observed for the tailings ranging from $8x10^{-3}$ cm/sec to $1x10^{-7}$ cm/sec (Steffen Robertson and Kirsten, 1991c). The data suggested that unsegregated tailings vary from an average of $1x\ 10^{-5}$ - $5x10^{-5}$ cm/sec, increasing to $1x10^{-3}$ cm/sec at surface where coarse tailings have perhaps been leached and subjected to frost action. Tailings slimes have an average hydraulic conductivity of approximately $1x10^{-6}$ cm/sec, although this may be an order lower at high levels of consolidation, such as at the bases of the tailings deposit (Steffen Robertson and Kirsten, 1991c).

Past drilling has shown substantial zones of frozen tailings in the original tailings impoundment up to 18m from surface, close to the original winter discharge points. Frozen zones appear to be greatly reduced in volume in the Second Impoundment and with distance from winter discharge points in the Original Impoundment (Steffen Robertson and Kirsten, 1988). No information is available on the distribution of frozen tailings in the Intermediate Impoundment. Frozen zones within the tailings would have effectively a zero hydraulic conductivity.

Seepage Estimates

Water-levels are found to be relatively high in the profile in fine grained tailings, and lower in coarse tailings (Steffen Robertson and Kirsten, 1991c). The lower permeability of the finer tailings limits the drainage of infiltration, and this combined with the enhanced moisture retention capacity, leads to higher or perched water-levels. At the base of the tailings, hydraulic heads tend to be dissipated rapidly due to the highly permeable sands and gravels in the aquifer tend to rapidly dissipate hydraulic head (Steffen Robertson and Kirsten, 1991c). As a result a strong hydraulic head gradient is maintained (often near unity), and there is a strong downward flow component from the tailings into the aquifer. The downward flow is locally modified by the horizontally layered nature of the tailings deposit, and the occurrence of frozen zones. Lateral flow may result locally within the tailings.

Estimates of current seepage rates from the tailings impoundments to the aquifer (excluding the polishing pond) have been made (Table 4.63). The following assumptions apply:

- infiltration in the exposed areas of the tailings impoundments ranges from 100 to 200 mm/yr; this range brackets the infiltration rate of 146 mm/yr. obtained by Steffen Robertson and Kirsten based on modelling of the infiltration process (Steffen Robertson and Kirsten, 1991c);

occupying Rose Creek Valley. These were approximately 3150 m3/day above the confluence of Faro and Rose Creeks, and 4600 m3/day just below the confluence.

Prior to tailings deposition, Rose Creek was flowing in a meandering stream channel in the valley floor. Rose Creek was fed by several smaller streams and creeks (including Faro Creek and Guardhouse Creek) which drain the hillsides of Rose Creek Valley. The water table of the underlying valley aquifer was very close to the stream level indicating a good hydraulic connection between the stream and the groundwater flowing in the permeable valley sediments. Depending on local conditions groundwater was either discharging into Rose Creek or was being recharged by the stream.

The development of the Rose Creek Tailings facility had a significant, albeit local, influence on the groundwater flow in the valley aquifer. Those areas of the tailings impoundments covered with tailings decant water act as a constant source of recharge through the tailings to the aquifer. The amount of groundwater recharge in the ponded areas depends primarily on the permeability of the deposited tailings. Highest recharge rates occurred during the operation of the Original and Second Impoundments which comprise zones of more permeable (sandy) tailings. It was estimated that in 1981 the Original and Second Impoundments were contributing in the order of 4300 m^3/day of seepage to the aquifer (Klohn Leonoff, 1981). However, the influence of the tailings pond on groundwater flows was probably not far-reaching. Significant volumes of groundwater, presumably the recharge water, was observed discharging in borrow pits immediately downstream of the Second Embankment (Klohn Leonoff, 1981).

In 1981, the Cross Valley Scheme was constructed and tailings were deposited into the Intermediate Impoundment. Water level readings in piezometers below the Intermediate Dam indicate that the water table has risen in the order of ten meters as a result of tailings discharge. SRK (1991c) estimated that recharge from the Intermediate Impoundment was in the order of 2000 m^3/day. Some of this recharge water would discharge into the Cross Valley pond. The downstream section of the Cross Valley Dam also provides a source of recharge to the aquifer. The increased groundwater flows due to pond recharge, however, are confined to a local area at or immediately downstream of the pond. The recharge of groundwater via seepage from the overlying tailings is of critical importance in estimating the chemical loading of oxidation products introduced to the underlying aquifer (see also discussion on groundwater chemistry, section 4.3.6). The rate of recharge from the tailings into the aquifer depends on the hydraulic properties of the tailings as well as the hydraulic properties of the underlying aquifer. The properties of the tailings and the valley sediments differ greatly and are reviewed separately below. Most of the discussion given below has been extracted from the SRK Report 60635 (Down Valley Tailings Impoundment Decommissioning Plan) submitted in April 1991 to the Yukon Territorial Government.

4.3.5.1 Tailings Deposits

Hydraulic Properties

The area variation in hydraulic properties within the tailings is largely a function of distance from the original point of discharge. This controlled the prevailing grain size, which in turn controls tailings

The water quality data for this station shows a distinct seasonal trend as shown, for example, for sulphate in Figure 4-91. Seasonal trends are most obvious for sulphate, zinc, manganese and calcium with peak concentrations occurring during the winter low flow period. Peak concentrations in iron are seen at the onset of spring freshet. This is most likely due to the physical flushing of iron precipitates during the high flows of freshet.

The pH at X14 is consistently alkaline with a value of 7.76 in 1995. The influence of the mine discharge on the Rose Creek water is evident through elevated concentrations of sulphate, iron, zinc and manganese. Figures 4-92 through 4-94 show the data for pH, iron and zinc. There are no apparent trends, other than seasonal, in concentrations of zinc, iron, and sulphate with 1995 concentrations of 0.0516 mg/L, 0.703 mg/L, and 163 mg/L respectively. This is probably due to the available dilution, which would mask any increases in these contaminants seen at the other upstream stations. It should be noted that both iron and zinc levels are above guideline values set out by CREM.

4.3.5 Hydrogeology

Groundwater flow in the Rose Creek Valley is largely controlled by the presence of very permeable valley fill materials consisting predominantly of fluvial, and glaciofluvial sands and gravels. Within the sand and gravel are irregular deposits of silts, sands and gravels which may have been deposited in lacustrine or flood plain environments. These silty deposits appear to be discontinuous which may be a result of post deposition glacial or fluvial action. Maximum thicknesses of the sand, gravel and silt deposits are 35 to 40 m. Prior to mine development, Rose Creek had incised this deposit by up to 10 m in the region of the tailings impoundments.

This valley aquifer is confined below by phyllite and schist bedrock which is overlain in most places by very dense glacial till. The permeability of the bedrock and dense till are orders of magnitude lower than that of the granular aquifer material.

In general, the valley sides act as recharge areas and the valley floor as a groundwater discharge area to Rose Creek. Recharge from Rose Creek to the aquifer does occur locally, a reflection of the complex nature of the overburden materials in the valley.

This valley aquifer was first studied in 1967 by International Water Supply Ltd. for the purpose of defining a reliable water supply for the planned mining and processing operation (IWS, 1967). Twelve tests holes were drilled in the Rose Creek Valley aquifer above the confluence with Faro Creek, and pump tests were performed near the confluence of Faro Creek and Rose Creek to define aquifer properties. These data are the only pre-mining data available for this aquifer.

Water levels recorded for boreholes in the Rose Creek valley in 1967 varied between 1.5 and 3 m depth from surface. The hydraulic gradient in this part of the aquifer was approximately 0.011 which is in close agreement with the creek gradient at the time. The pump tests suggested highly permeable formations with hydraulic conductivity values ranging from 5x10-2 and 1x10-1 cm/sec. Using data obtained in their program, IWS made estimates of groundwater flow in the aquifer

TABLE 4.62

Data Summary for Station X14: Rose Creek downstream of the diversion channel

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	**pH unit**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**
JAN 1988 - OCT 1996							
Mean	7.72	115.9	184.8	0.04	0.014	0.61	0.091
Minimum Value	5.90	43.0	23.0	0.02	0.001	0.17	0.000
Maximum Value	8.98	247.0	492.0	0.07	0.201	4.62	0.729
Median	7.74	114.0	140.0	0.04	0.006	0.43	0.058
Num. Values	121.00	93.0	85.0	52.00	124.000	65.00	125.000
JAN 1988 - DEC 1991							
Mean	7.69	107.3	214.3	0.05	0.011	0.5	0.099
Minimum Value	7.19	43.0	43.0	0.05	0.001	0.17	0.020
Maximum Value	8.98	179.4	453.0	0.07	0.059	1.59	0.729
Median	7.70	111.0	193.5	0.05	0.006	0.43	0.062
Num. Values	67.0	63.0	48.0	13.00	68.000	26.00	69.000
JAN 1995 - OCT 1996							
Mean	7.77	156.0	188.0	0.03	0.007	0.754	0.056
Minimum Value	7.30	49.0	40.0	0.02	0.002	0.360	0.000
Maximum Value	8.20	247.0	492.0	0.06	0.028	1.470	0.170
Median	7.80	157.0	114.0	0.02	0.003	0.537	0.045
Num. Values	15.00	6.0	13.0	13.00	15.000	13.000	15.000

The pH values are generally neutral to alkaline, as shown in Figure 4-86. Variations in pH frequently paralleled the pH values in the Cross Valley Pond, particularly during the time of active tailings deposition. While the pH values remain reasonably consistent, alkalinity values are increasing over time, particularly in the seepage from the north toe of the dam, as shown in Figure 4-87.

Sulphate concentrations elevated, again more so in the seepage from the north toe of the dam. Concentrations appear to be increasing slightly; the average concentration over the period of record at X13 is 500 mg/L. Figure 4-88 shows the ratio of sulphate to calcium and magnesium concentrations. This ratio provides an indication of the comparative rates of oxidation and neutralization. The data suggest that the geochemical reactions at the north and south toes of the dam are proceeding at similar relative rates at both sides of the tailings impoundment, particularly for the latter years.

The iron data exhibit an increasing trend since 1988 from 1.039 mg/L to 3.073 in 1994 and an apparent drop to a concentration of 2.792 mg/L in 1995, as shown in Figure 4-89. The peak in iron concentration from the north toe seepage was evident in mid-1993, with the peak at the south toe lagging slightly into 1994.

The zinc data, Figure 4-90,show no apparent trend with an average concentration of 0.0168 in 1995. Manganese concentrations have been increasing since 1989 from 4 mg/L to 8 mg/L in 1993 at which point the concentrations level off. Arsenic concentrations are at or near detection level except for a few elevated concentrations of 0.125 in 1995. The nickel data also exhibits this behaviour with isolated peak concentrations of 0.03 mg/L occurring at the same time. Copper, cobalt and lead are at or near detection limits.

4.3.4.3 Rose Creek (X14)

Downstream of the tailings impoundment area on Rose Creek is licence station X14. The flow at this station consists of inputs from the Northwall Interceptor Ditch (NWI), the decant from the Cross Valley Dam (X5), the combined seepage from the toe of the dam (X13) and the discharge from the Rose Creek Diversion (X10).

The water quality data for this station are summarized in Table 4.62. Detailed data as well as summary statistics are provided in Appendix D. Figures 4-91 through 4-94 show selected data for this station.

TABLE 4.61

Data Summary for Station X13: Combined seepage from Cross Valley Dam

	LAB pH	TOTAL ALKALINITY	TOTAL SULPHATE	TOTAL ARSENIC	TOTAL COPPER	TOTAL IRON	TOTAL ZINC
Date	pH unit	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
JAN 1988 - OCT 1996							
Mean	7.52	205.9	445.5	0.04	0.007	2.08	0.022
Minimum Value	6.64	8.0	152.0	0.02	0.001	0.47	0.000
Maximum Value	9.90	461.0	1220.0	0.14	1.110	8.98	0.439
Median	7.50	203.0	430.5	0.04	0.002	2.03	0.010
Num. Values	362.00	225.0	202.0	63.00	368.000	111.00	368.000
JAN 1988 - DEC 1991							
Mean	7.50	202.0	412.1	0.05	0.009	1.25	0.022
Minimum Value	6.64	86.0	152.0	0.05	0.001	0.47	0.001
Maximum Value	8.40	461.0	786.0	0.06	1.110	2.09	0.439
Median	7.48	202.0	419.0	0.05	0.002	1.22	0.009
Num. Values	227.00	203.0	149.0	12.00	230.000	53.00	230..000
JAN 1995 - OCT 1996							
Mean	7.47	222.0	564.0	0.04	0.006	2.79	0.017
Minimum Value	7.00	8.0	350.0	0.02	0.002	1.92	0.000
Maximum Value	9.90	309.0	1220.0	0.14	0.038	8.98	0.100
Median	7.36	246.0	547.0	0.04	0.005	2.43	0.010
Num. Values	44.00	8.0	33.0	37.00	48.000	37.00	48.000

The variations on water chemistry at this station are controlled by the water treatment schedule, and the retention time in the pond for settling suspended solids. The key point with respect to downstream water chemistry is the variability in zinc concentrations, which can be seen in Figure 4-85. During active operation of the mine, particularly in more recent years, the zinc concentrations are reasonably stable within the required decant specifications.

4.3.4.2 Tailings Dam Seepage (X13)

Seepage is collected from the tailings dam at the north and south ends of the dam, at Stations X11 and X12. These two seepage streams combine together and the combined stream sampled at X13. All three stations are licence stations.

The data for Station X13 are summarized in Table 4.61. Detailed data as well as statistical summaries for the three tailings seepage monitoring stations are in Appendix D. Figures 4-86 through 4-90 show selected data for these three stations.

below the 1974 tailings berm from the tailings surface into the valley aquifer to replace the X21 monitoring well which was destroyed in 1989 (see Figure 4-97). A shallow piezometer (96-5A) was screened in the lower section of the tailings just above the natural ground surface. The middle piezometer 96-5B was placed in sands and gravel a few meters below the tailings surface and the deep piezometer 96-6C was placed ~20 meters below original ground surface. These piezometers were first sampled in September 1996 (see Table 4.66).

The upper piezometer screened in the tailings above the original ground surface was dry. No sample was taken. The groundwater immediately below the tailings in natural aquifer material shows a water quality typical of buffered acid rock drainage (ARD) (Table 4.66). The pH is near-neutral, sulphate concentrations are high (767 mg/L) and dissolved metal concentrations are elevated (0.13 mg/L Zinc and 9.89 mg/L Fe). Note, however, that current sulphate concentrations in 96-5 are actually lower than they were during the 1980s in X21 (see Figure 4-99). The dissolved Zinc concentrations observed today are slightly higher than total Zinc concentrations observed in X21 in 1989.

The deep piezometer (96-5C) was completed in a gravel layer confined above by a low permeability silt. The very low concentrations of both Zinc and sulphate in this layer may be a result of limited recharge from the upper aquifer. More likely, however, they are a result of reducing conditions in this layer causing a reduction of sulphate to sulfide and subsequent precipitation of Zinc and other metals The grey water colour and the fact that pyrite-rich fines were pumped from this piezometer during initial well development (RGC, 1996) support the latter hypothesis. Although the data from X21 and BH 96-5 may not be directly comparable considering the heterogeneity of the alluvial aquifer they do suggest that a significant increase in metal levels in the aquifer has not taken place.

Model predictions described in Nicholson, Scharer and Kwong (1996) which suggest that oxidation of the tailings in the Old and Second impoundment may accelerate resulting in progressively higher loadings of zinc and sulphate to the aquifer should be reconciled with these field observations. The comparatively low levels of contamination in the aquifer just a few meters below the tailings may at least in part be due to the significant acid neutralizing capacity of the groundwater (~200 mg/L alkalinity) here and in other reaches of the valley aquifer (Table 4.66).

4.3.6.3 Intermediate Dam Area

The water quality of groundwater at the Intermediate Dam has been monitored at two licenced locations from 1987 to 1989: X24 near the North Abutment of the Dam and X25 near the south abutment (see Figure 4-97). During the period of observation (1987-1989) groundwater at these locations also showed signs of buffered ARD. The pH of the groundwater remained slightly alkaline over the period of observation (pH ~7.3 to 7.8) whereas sulphate concentrations were typically high (ranging from 753 to less than 100 mg/L, see Table K11). Zinc and sulphate concentrations do not show clear trends in X24 and X25 over the observation period from 1987 to 1989 (see Figures 4-100 and 4-101).

In 1996, two new monitoring wells were installed at the Intermediate Dam to replace X24 and X25 which had been lost during dam raising in 1990. A comparison of the water quality data collected

from these piezometers in 1996 to those collected in 1989 at X24 and X25 suggest that there has been little change in water quality over the last seven years (see Figures 4-100 and 4-101). The only possible exception is an increase in sulphate concentrations near the North abutment which was observed throughout the profile in 96-4 (see Figure 4-100). In 1996 the sulphate concentrations were near ~1000 mg/L in all four piezometers up from ~400 mg/L in X21 in 1989 suggesting an increased influx of sulphate. However, this increase in sulphate is accompanied by an increase in alkalinity (now ranging from 315-394 mg/L in the different aquifer layers, see Table 4.66). PH values still remain well above 7 and dissolved Zinc concentrations remain very low (<0.04 mg/L). Such an increase in sulphate concentrations was not observed in 96-3 located closer to the center of the Intermediate Dam (Figure 4-101).

Table 4.66

Groundwater quality in BH-96 boreholes in tailings area.

BOREHOLE	DATE	LAB pH	Fe (mg/L) dissolved	Zn (mg/L) dissolved	Ca (mg/L)	SO4 (mg/L)	ALKALINITY (mg/L)	ACIDITY (mg/L) pH 8.3	ACIDITY (mg/L) pH4.5
BH-96-4A	26-Sept-96	7.7	0.04	0.02	256.3	855	315	618	<1
BH-96-4B	26-Sept-96	7.68	9.19	0.04	256.8	758	332		<1
BH-96-4C	26-Sept-96	7.55	0.06	0.02	311.9	1005	363	855	<1
BH-96-4D	26-Sept-96	7.33	<.01	0.01	352.9	1048	394	1102	<1
BH-96-3A	26-Sept-96	8.05	0.05	0.03	100.7	260	236	336	<1
BH-96-3B	26-Sept-96	7.89	<.01	0.01	151.8	488	277	478	
BH-96-5B	26-Sept-96	7.16	9.89	0.13	178.3	767	202	35	<1
BH-96-5C	26-Sept-96	7.54	0.05	<.01	46.7	12	178	7	<1

4.3.6.4 Cross Valley Dam

Several monitoring piezometers located on the crest and toe of the Cross Valley Dam were routinely sampled from 1987 to 1990. Table K11 in Appendix K shows groundwater quality data for two representative sites CVDC 4 in the northern section of Cross Valley Dam and CVDC 7 in the southern section of Cross Valley Dam). During the period of observation the groundwater quality at the Cross Valley Dam (i.e. immediately downstream of the polishing pond) was very similar to that observed at the Intermediate Dam (see Table K10 and K11). These data suggest that recharge of limed water from the polishing pond had no significant effect on groundwater quality.

The monitoring wells at the Cross Valley Dam are currently not sampled for water quality.

4.3.6.5 Downstream of Cross Valley Dam

The most complete record of groundwater quality has been collected for three monitoring wells (X16, X17 and X18) located at various distances downstream of the Cross Valley Dam. The water quality data for these three locations are summarized in Table K12. Time trends for sulphate are shown in Figures 4-102 to 4-104. It is likely, that water quality data from September 1981, i.e. prior to construction of the Cross Valley Scheme, represent groundwater quality in this area unaffected by acid rock drainage. At this time, total Zinc concentrations in this area of the valley aquifer ranged from 0.009 to 0.024 mg/L and sulphate concentrations ranged from 19 to 60 mg/L.

Over the 15 years of observation there has been a significant increase in sulphate (from 40 mg/L to 400 mg/L) at monitoring well X18 located closest to the Cross Valley Dam (Figure 4-102). In contrast, sulphate concentrations in X16, several hundred meters downstream, have not increased significantly fluctuating only within a narrow range (25 to 50 mg/L, see Figure 4-104). Elevated sulphate concentrations have only recently been observed in X17 (Figure 4-103) which is located between X 18 and X16. These data suggest that sulphate, an indicator of buffered ARD, has moved beyond the Cross valley Dam but not yet as far as the confluence of the Rose Creek Diversion with the old Rose Creek channel.

Note that the increase in sulphate in X18 and X17 is not accompanied by an increase in metals or a decrease in pH (Tables L12). Similar observations have been made in the other wells further upstream (see above). Again, the acid neutralizing capacity of the groundwater appears sufficient to buffer any inputs of acidity typically associated with the oxidation of tailings and subsequent release of sulphates.

Figure 4-105 shows average concentrations of dissolved zinc and sulphate for 1995-1996 for several boreholes which are located on a transect running parallel to the valley. The concentrations are plotted as a function of the distance from the Original Tailings Impoundment, i.e. the oldest source of contamination. The figure illustrates two important points. First, the highest sulphate concentrations are not observed closest to the source. Instead, the highest sulphate concentrations in 1995/96 are observed at the Intermediate Dam some 2000 meter downstream. Approximately 9 years ago, the highest sulphate concentrations were observed in X21 (near the Second Impoundment (see Fig. 4-99). In other words, the data suggest that a "sulphate pulse" has moved downstream in the aquifer at an average velocity of approximately 1400m / 9 years = ~150 m/yr. Using Darcy's Law, the average permeability can be back-calculated from this transport velocity. Assuming a hydraulic gradient of ~0.013 and a porosity ~0.3, an average permeability of 1.1×10^{-2} cm/sec is calculated. Such a permeability is consistent with results from in-situ permeability tests performed in this area (compare section 4.3.5.2). The important point to be made here is that the release of sulphate into the groundwater does not appear to have increased as one might expect due to increased oxidation with time. In fact, it appears as if the recharge of ARD to the aquifer (at least from the Original and Second Impoundment) may even have slowed down.

Second, the peak in sulphate concentrations is not matched by a similar peak in dissolved zinc. The highest zinc concentrations are still observed near the source (at 96-5) suggesting that Zinc

is much less mobile than sulphate. As discussed above the valley aquifer provides sufficient alkalinity to buffer ARD and immobilize zinc very close to its source. Continued monitoring of contamination levels in these monitoring wells is required to confirm temporal trends of ARD in the Rose Creek Valley aquifer.

4.4 Faro Pit Tailings Facility

From August 1992 to April 1993, Curragh Resources Inc. (CRI) discharged tailings from the Grum and Vangorda ore deposits into Faro pit. No tailings were discharged into Faro pit during the shutdown period of the mine. The discharge of tailings from the Grum and Vangorda orebodies re-commenced in August 1996 when the mine went into full production again under the current owner, Anvil Range Mining Corporation (ARMC). ARMC plans to discharge the tailings of the remaining Grum and Vangorda ore into Faro pit over the next five years.

Besides the Grum/Vangorda tailings, several other sources of tailings are currently being considered for in-pit disposal by Anvil Range Mining Corporation:

- tailings from Grum Underground
- tailings from Grizzly deposit
- old tailings from the Original, Second and Intermediate Impoundment

Table 4.67 lists the tonnage of ore from the various orebodies and the volume of tailings that are expected to be generated from these sources. In calculating these volumes it was assumed that processing of the ore would extract about 20% of the material whereas re-processing of the tailings from the Rose Creek Tailings Facility would extract about 5% of the material as concentrate. It was further assumed that the tailings would have a dry density of ~1.7 t/m^3 after initial settling and early consolidation. This value is considered representative for recent deposition of the tailings, i.e. before final consolidation has occurred. At full consolidation the tailings may have a dry density of 2.1 t/m^3 or higher (e.g. see Golder Associates, 1991). However, it is currently not known how much time it will take before full consolidation is reached. The tailings have a significant fines content with very low permeabilities and high pore pressures are expected to develop in the tailings during deposition in the Faro Pit (see Appendix M). Tailings will be deposited in the Faro Pit to thicknesses of up to 200 m and it may take tens of years to dissipate pore pressures and achieve consolidation under those circumstances. Hence, a low dry density typical of unconsolidated tailings is used here for estimating tailings volumes to be placed into the pit.

Table 4.67

Summary of Tailings considered for in-pit placement.

Source	Total tonnage	Tailings Volume[3]
	(million tonnes)	(million m^3)
Grum[1]	27.0	12.7
Vangorda[1]	6.5	3.0
Grum underground[1]	5.0	2.4
Grizzly[1]	15.0	7.0
Relocated Tailings[2] (all tailings)	54.34	30.4
Relocated Tailings[2] (to 1042 m level only)	40.9	22.9

[1] assuming 20% removed as concentrate

[2] assuming 5% removed as concentrate

[3] assuming a dry density of 1.7 t/m^3

Figure 4-106 illustrates the potential for placing tailings into Faro pit. The total storage capacity of Faro pit is in the order of 44 million m^3 without a plug dam (3800 ft sill level) which is insufficient to store all tailings considered for in-pit disposal (55.5 million m^3). The storage capacity of Faro pit can be increased to approximately 55 million m^3 with the construction of a plug dam which has a spillway invert at 3850 ft (i.e. the maximum water level). However, not all of this capacity is available for storage of tailings. The following three water storage requirements need to be considered:

- a water cover of at least 1 m depth (~0.75 million m^3 at the 3850 ft level) to minimize resuspension of tailings due to wave action and/or direct exposure of the tailings to the atmosphere.
- life storage to account for seasonal and year-to year fluctuations in inflows to the pit. An approximate volume of 1.9 million m^3 of required life storage was estimated assuming pit water is treated seasonally (6 months/year) at a rate of 400,000 m^3 per month. A higher intake rate to the treatment plant and/or a continuous operation would reduce the volume required for life storage accordingly.
- a flood contingency to allow for extra storage during a flood event. Two "worst-case" scenarios were considered to estimate the required flood contingency. First, the North-east wall of Faro pit may fail potentially requiring excavation of a new diversion channel for Faro Creek further up the hill side. Assuming it takes two months to excavate the new diversion channel and further assuming that construction occurs during the two wettest months of a 1 in 10 wet year, the required storage would be

approximately 4 million m^3. Second, Faro Creek Diversion may breach during very high flows. Assuming it takes one week to repair the diversion and further assuming the largest possible amount of rainfall occurs during that week (i.e. the PMP of one week duration), the required storage would be 4.5 million m^3. These flood contingencies are considered conservative and would only apply if the Faro Creek diversion is maintained (see chapter 7). No flood contingency would be required if Faro Creek was allowed to flow through Faro pit.

In summary, a maximum of 7.1 million m^3 has to be reserved for water storage and is not available for tailings disposal.

Assuming all tailings from Grum/Vangorda open pits, Grum underground and Grizzly deposit are deposited in Faro pit, an incremental storage capacity of approximately 22.9 million m^3 remains. This volume is sufficient to deposit those old tailings from the Original, Second, and Intermediate Impoundment which have been cut from above an elevation of 1042 m AMSL (22.9 million m^3). As outlined in chapter 7, a lowering of the tailings surface to the 1042 m elevation would provide sufficient storage room for a 3m permanent water cover without having to raise the Intermediate Dam.

4.5 Water Management Structures

Throughout the course of mine development, several water management structures have been constructed at the Faro minesite. These include surface water diversions in Faro Creek, Rose Creek and the north wall of the Rose Creek Valley, water supply reservoirs (the Fresh Water Supply Reservoir and Pumphouse Reservoir), a retention pond below the Intermediate Tailings Impoundment (Cross Valley Pond), a flow-though rock drain across the North Fork of Rose Creek, spillways and decants associated with the tailings impoundments and reservoirs, and assorted surface water ditches and culverts. The pumphouse reservoir is described in section 4.5.7.2. The other water management structures are described below.

4.5.1 Faro Creek Diversion

The original channel of Faro Creek passed through the center of the Faro Main Pit, past the mill site, and joined Rose Creek at what is now the toe of the Original Tailings Embankment (see Figure 4-8. As part of mine development Faro Creek was diverted towards the east, emptying into the North Fork of Rose Creek.

Aerial photographs of the site during the early stages of development [check with Greg Jilson for date of photo] show that, initially, a temporary diversion was constructed which emptied into the North Fork of Rose Creek immediately below the Zone 2 Pit. Aerial mapping of this area done in the early 1970's suggests that stream flow originating from this diversion had eroded some overburden material on this hillside forming a small stream gully. The sand and gravel delta formed where this temporary diversion entered the North Fork of Rose Creek is still visible today. A comparison of aerial mapping done prior to mining and in 1990 suggest that erosional deposits originating from this stream gully caused the North Fork Rose Creek to carve a new stream bed

several tens of meters to the south of this delta. This temporary diversion was replaced shortly afterwards by the current Faro Creek Diversion.

The current Faro Creek Diversion is shown in plan in Figure 4-1. The diversion starts approximately 4500 ft. (1370 m) upstream of the Faro Main Pit, follows along the eastern side of the Faro Creek valley, skirts along the northern crest of the pit past the Northeast Waste Dumps, and empties into the North Fork of Rose Creek approximately 7000 ft. (2100 m) upstream of the Vangorda haulroad. The total length of the diversion, from its start to its confluence with the North Fork, is approximately 11,000 ft. (3350 m).

No records are available of the design or construction of the Faro Creek Diversion. However, from field observations and available topographic plans it appears to have an average bottom width of 12 ft. (3.7 m.), and an average gradient (up to the point where it passes the Northeast Waste Dumps) of approximately 0.5 per cent. For the upper 3500 ft. of the channel (from its origin to the Faro Valley Waste Dump), the right bank is formed by a dyke constructed of waste rock fill placed at an angle of approximately 1.5 horizontal to 1 vertical, and the left bank by shallow excavation into native soil cut to an angle generally flatter than 2 horizontal to 1 vertical. Starting at the Faro Valley Waste Dump, the depth of cut increases sharply, reaching a maximum depth of approximately 25 ft. where the ditch rounds the corner to follow the crest of the pit (approximately 4000 ft. from the start of the ditch) and is founded in bedrock. It continues to follow the bedrock surface as it passes along the northern edge of the pit. Side slopes are typically excavated at 1 horizontal to 1 vertical in rock, and 2 horizontal to 1 vertical in soil. Beyond the Northeast Waste Rock Dumps, the gradient increases sharply (to as steep as 35 per cent) as it plunges into the valley of the North Fork of Rose Creek. The final 2000 ft. of the diversion (within the North Fork valley floor) flows at a gradient of approximately 14 per cent.

The existing Faro Creek Diversion is unlined and is known to leak, both into the Faro Creek Valley (through the mine rock fill or permeable alluvium) and into the pit (through fractured bedrock). Some of the mine rock used in construction of the containment dyke along the upper reaches of the diversion shows considerable iron oxide staining, and is suspected to be the source of some of the zinc measured in surface water samples in the Faro Creek Valley above the Faro Valley Waste Dump. The capacity of the existing diversion is not known but is considered inadequate for long term closure.

4.5.2 Faro Valley Interceptor Ditch

Runoff from the hillsides on the northwest side of the Faro Valley Waste Dump are intercepted by the Faro Valley Interceptor Ditch. This interceptor ditch, shown on Figure 4-1 diverts this runoff into the Faro Creek Diversion, which, in turn, conveys it to the North Fork of Rose Creek. No records are available of the design, construction or as-built details of the Faro Valley Interceptor Ditch. Its flow capacity is not known.

4.5.3 Rose Creek Diversion

The Rose Creek Diversion, which conveys Rose Creek around the south side of the tailings impoundment, was developed in two stages, referred to as the Upper and Lower Diversions on

Figure 4-74. The Upper Diversion was constructed in 1974 in conjunction with the development of the Second Tailings Impoundment. The Lower Diversion is an extension of the upper diversion. It was constructed in 1980-81 in conjunction with the development of the Intermediate Dam Impoundment (the "Down Valley Tailings Facility"). The tailings facilities are described in Section 4.3, above.

4.5.3.1 Upper Diversion

The Upper Rose Creek Diversion was constructed along the southern edge of the Rose Creek Valley to by-pass the Second Tailings Impoundment, as shown in Figure 4-76. The design capacity was 1700 cfs, or 48 m^3/s (Sigma, 1975). The channel was excavated with a bottom width of 50 ft. (15 m) (Golder, Brawner and Associates, 1975) and side slopes of 2 horizontal to 1 vertical, and lined with riprap for erosion protection. The channel has an initial gradient of 0.23 per cent from station 0+00' to station 20+00' (see Figure 4-76 for station locations). At station 20+00' the gradient increases to 2 per cent, and the channel includes a number of drop weirs in addition to riprap for erosion protection (Sigma, 1975). At station 43+70 the gradient increased to 5 per cent before it rejoined the original channel of Rose Creek below the toe of the Second Tailings Embankment (see Figure 4-76). This last section was equipped with a series of drop weirs in close succession. The channel for the Upper Rose Creek Diversion was completed in 1974, along with most of the diversion dam which diverts the North and South Forks of Rose Creek into the diversion channel. Final closure of the diversion dam was completed in April 1975 (Sigma, 1975). The initial 2600 feet of the Upper Diversion remains in use, but the remainder of the channel was abandoned with the development of Lower Diversion.

4.5.3.2 Lower Diversion

One of the key components of the Down Valley Tailings Facility was an extension of the Rose Creek Diversion along the south valley wall to beyond the Intermediate and Cross Valley Dams, thereby diverting inflow from Rose Creek into the new impoundment (see Fig 4-74). Details of the design and construction of the Lower Rose Creek Diversion are provided in Golder Associates 1980a, 1980b and 1982. The diversion is shown in plan on Figure 4-81. Typical sections (after Golder Associates 1980b) are provided in Figure 4-82. Some significant aspects are listed below.

- The Lower Diversion has a total length of 3.8 km (2.4 mi.), starting at approximately Station 26+00 ft. of the Upper Rose Creek Diversion (at an elevation of approximately 1054 m AMSL) and rejoining the original Rose Creek channel approximately 1 km (0.6 mi.) downstream of the Cross Valley Dam (at an elevation of 1012 m AMSL).
- Most of the channel has a gradient of 0.19 per cent, with two drop weir sections at a 5 per cent gradient: one from 2820 m to 3400 m, and from 3660 m to 3840 m.
- The channel has a bottom width of 12.2 m (40 ft.), with side slopes of 2 horizontal to 1 vertical in soil, and 0.5 horizontal to 1 vertical in rock..

- The low-gradient (0.19 per cent) sections of the channel are provided with a pilot channel 3.65 m (12 ft) wide by 0.6 m (2 ft.) deep to control glaciation during low flows.
- The channel has a capacity of 160 m^3/s at a depth of flow of 3.4 m (with no freeboard), and 88 m^3/s at a depth of flow of 2.4 m (with 1 m freeboard).
- The crest of diversion dam which diverts the flow from the Upper Rose Creek Diversion into the Lower Faro Creek Diversion was constructed approximately 1 m lower than the crest of the adjacent diversion canal dyke, and armoured with riprap. This is to ensure that any flows in excess of the design flow overtop the diversion dam and are redirected into the Intermediate Dam Impoundment.

Construction of the Lower Rose Creek Diversion began in October 1990, and it was substantially completed by the end of May 1991.

4.5.3.3 Stability Issues

Both the Upper and Lower Rose Creek Diversions are inspected annually by the design engineers (Golder Associates). Stability issues that have been identified include:

- erosion damage to the thermal blanket and rip rap on the canal backslope at local drainages;
- erosion and sediment accumulation from tributary streams;
- bedload accumulation in the Upper Diversion;
- ice accumulation during winter months;
- localized erosion and redeposition of the riprap lining, particularly in the drop weir sections; and,
- deformation of the canal dyke (believed to be related to melting of excess ice in areas of frozen ground).

Periodic maintenance and remedial measures have been required to address these issues, will continue to be required to ensure that the diversion functions as designed.

4.5.4 North Valley Wall Interceptor Ditch

The North Valley Wall Interceptor Ditch (Figure 4-74) was also constructed in conjunction with the Down Valley Tailings Facility. It intercepts runoff from north side of the Rose Creek valley, and diverts it around the north (right) abutment of the Cross Valley Dam.

The diversion consists of three segments. The first, referred to as the "mine leg" begins just north of the guardhouse within the drainage of Upper Guardhouse Creek (the stream which passes below the toe of the North west Waste Dumps), and diverts flow from that drainage into the next adjacent drainage to the west. From this natural drainage the second leg of the Interceptor, referred to as the "Borrow Area F leg", conveys the flow to yet another natural drainage channel.

The final leg of the interceptor conveys the flow around the north abutment of the Cross Valley Dam.

The North Valley Wall Interceptor Ditch is excavated in a variety of materials ranging from silty sand and gravel till to coarse sand and gravel alluvium, and bedrock. The design capacity (with 0.5 m of freeboard) is 8.9 m^3/s for the mine leg, 12.4 m^3/s for the Borrow Area F segment (Golder Associates, 1983), and 17.7 m^3/s at the north abutment of the Cross Valley Dam (SRK, 1988).

Erosion and sedimentation have caused partial blocking of this ditch at times, and periodic maintenance and repairs have been required.

4.5.5 Zone 2 Well

In early 1991, a monitoring well was installed in the deepest section of the Zone 2 pit to control the water level in the pit. The borehole was drilled from the haul road just south of the Faro main pit (see Fig. 4-9). The borehole was probably drilled using an air-rotary with an ODEX drill bit which allows a steel casing to follow the bit and the bit to be retracted through the casing following borehole completion (SRK Letter Report, 1990). No drill logs of the borehole were available. However, a schematic of the pumping well and pump issued by E. Soprovich in April 1991 indicates that the borehole was drilled using 8" Ø steel casing with bedrock encountered approximately 74 m below ground surface, i.e. at an elevation of 1093.8 m (3585 ft) AMSL. The steel casing was retracted to allow for placement of a 15 ft long well screen and a 2.5 ft packer above the screen. A submersible pump was set some 20 ft above the well screen to minimize sediment passage through the pump and to effect maximum cooling. The submersible pump was equipped with automatic water level control switches. A large-diameter pipeline was mounted to the well collar which runs some 100 m down the access ramp towards the Main pit. Currently, the pipeline is discharging the pumped water freely onto the access ramp from where it flows via surface flow into the Faro Main pit.

The pump has been repaired several times (and possibly replaced) in the last 5 years. At present, a 40 hp Pleuger submersible pump (Model QN 64) is in use. The pump was last pulled and serviced in March 1996 at which time the settings were changed. Now, the automatic water level switch activates the pump when the water level in the pumping well rises to an elevation of 1110.9 m (3653 ft) and turns off the pump when the water level drops below an elevation of 1109.2 m (3639 ft) (Anvil Range Mining Corporation, pers. com.).

The hours of pumping are also being recorded continuously since the fall of 1995. Flow spot measurements at the discharge point of the pipeline suggest that the pump is now pumping at approximately 250 US gpm (16 l/s) (Anvil Range Mining Corporation, pers. comm.). The continuous recording of pumping hours provides the opportunity to accurately estimate the volumes of water flowing into the Zone 2 pit. This information is critical in improving the estimates of the water balance for the Zone 2 pit (see section 4.1.4.3), in particular for monitoring potential seepage of groundwater into the Zone 2 pit originating from the Faro Main pit in response to the current flooding of the Main pit. It is recommend that a circular orifice weir be installed at the end of the discharge pipeline in order to facilitate the measurements of pit water volumes pumped from the Zone 2 pit.

Since the installation of the Zone 2 pumping well in early 1991, it has also been used to measure water levels in the Zone 2 pit (replacing the old monitoring well which is believed to be lost to backfill today). However, this practice was discontinued in the fall of 1995 due to difficulties in lowering and recovering the water level tape. For closure, at least one new monitoring well will be required to monitor water levels in the Zone 2 pit (see section 7.1.2). It is recommended that this new monitoring well be installed already during operation of the mine, preferably near the natural discharge point of the Zone 2 pit. This new monitoring well would provide the opportunity to measure the water level drawdown within the Zone 2 pit, but some distance from the pumping well, This information would help to finalize the pumping requirements for the Zone 2 pit after closure (see section 7.1.2). In addition, the water level inspection tube in the pumping well should be refurbished or replaced to allow direct water level readings to be taken in the pumping well.

4.5.6 Cross Valley Pond

The Cross Valley Retention and polishing Pond is discussed in section 4.3.

4.5.7 Freshwater Reservoir and Supply Facilities

Anvil Range Mining Corporation extracts fresh water from Rose Creek primarily to supply the requirements of the metallurgical plant within the mill, the principal water uses are for grinding (33%) flotation (42%), and dewatering (25%).

Anvil Range's fresh water supply system consists of:

- a freshwater reservoir
- a pumphouse pond with pumphouse
- the North Fork Rose Creek Diversion; and
- a supply line from the pumphouse to the mill.

At present, the main source of fresh water is the water supply reservoir which supplies water to the pumphouse year round. In addition, the reservoir supply capacity is supplemented by the North Fork of Rose Creek during the winter months. During Curragh's operation, freshwater supply during the winter months was further supplemented by pumping groundwater from four near-by groundwater wells to the pumphouse reservoir. However, these pumping wells have not been used since 1991.

The location of the various components of the Freshwater Supply Facility are shown in Figure 4-1. A brief description of the components is given below.

4.5.7.1 Freshwater Reservoir

The freshwater reservoir is located 4 km south of the Faro property plant site (Figure 4-1). It was constructed in 1969 and is designed to hold 5,800,000 m^3 of water. The reservoir is positioned just west of this minesite access road and is 1400 m long by 400 m wide (Figure 4-1).

In August of 1984, the Frontier Engineering Group prepared a preliminary engineering plan and cost estimate for the raising of the reservoir dam in order to increase freshwater storage capacity, and allow a greater milling rate. By the summer of 1985, pursuant to the sale of property, it was deemed

necessary to establish the feasibility of raising the Rose Creek Dam some 20 feet. Subsequent investigations by Acres determined that the raising of the dam was technically impractical.

4.5.7.2 Pumphouse Pond and North Fork Rose Creek Diversion

A pumphouse pond was constructed in the Rose Creek channel by building a dam just downstream of the confluence of North Fork and South Fork of Rose Creek. The pumphouse and associated dam constituted part of initial mine construction in 1969, and were rebuilt in about 1974. As a consequence of construction of the first tailings storage expansion in 1974, the tailwater elevation at the Pumphouse dam was raised to within 2 m of the crest of the Pumphouse dam. This raising required the diversion of North Fork of Rose Creek from its natural channel into the excavated 1974 diversion channel. The diversion channel headpond is now filled with bedload material delivered by North Fork flow.

4.5.7.3 Water Supply Pipeline

The freshwater is pumped from the pumphouse pond to the mill site in an insulated steel pipe. The location of the pipeline is shown in Figure 4-1. The total length of the pipeline is approximately 2000 m.

4.5.8 Flow-Through Rock Drain

Refer to section 4.2.11.

4.5.9 Ditches and Culverts

Many of the access roads on the Faro minesite are equipped with drainage ditches and culverts. Where these roads are required for long-term monitoring and maintenance, these ditches and culverts will require regular inspection and periodic cleaning. In roads that are no longer required the culverts should be removed and natural drainage patterns restored.

4.6 Mill and Pipelines

4.6.1 Buildings and Equipment

The following facilities are located at the Faro mill site (see Figure 4-107 for location):

- Crusher coarse ore storage, Mill and Loadout
- Offices and Warehouse
- Heavy Duty Equipment Repair Shops
- Guardhouse and Administration Building; and
- Tire shop and Light Vehicle repair shops

Other buildings and facilities not located directly at the mill site include the Explosives Plant just upstream of the Original Tailings Impoundment west of the mine access road, a lube station, a

series of core shacks, and an explosives magazine, all located near the Faro open pit. As well there are several areas on the rock dumps, in the pit and near mill areas in which metal salvage or scrap is stored (Figure 4-107).

These facilities are described in Curragh's 1988 "Other Facilities Abandonment Plan". The following sections have been extracted from this report. It is noted that Anvil Range Mining Corporation is currently using essentially all facilities previously operated by Curragh Resources.

Crusher, Mill and Loadout

The Faro Mill complex is described using the path of run-of-mine (ROM) ore from initial receipt to loading of the final product.

ROM ore Is initially received at the Crusher (Figure 4-107), where it is broken into sizes the mill can accommodate. ore is then delivered to Coarse Ore Storage or to the Mill; for further grinding and size-classifying. Ground ore In transferred to the flotation circuit: the fine ore particles are mixed with water as a carrier medium and chemical reagents added In this beneficiation process to liberate concentrate lead & zinc product from waste. Fine waste is routed to tailings. Lead and zinc concentrate (pulp) Is transferred to thickeners, filters and then driers where moisture is driven out. The concentrate product Is finally delivered to the Loadout and loaded directly Into trailer-mounted containers for transport to Skagway.

Offices and Warehouse

Immediately adjacent to the Mill complex Is an Office & Warehouse facility. This complex is utilized by Anvil's Operations, Technical and Administrative staff. The majority of floor space is assigned for materials storage as well as light- and medium-duty fabrication and repair.

The warehouse and office complex is constructed mainly from structural steel with lesser amounts of dimension lumber and other building materials. Reinforced concrete is used for foundation footings and basement walls & floors. The warehouse has a floor space of approximately 18,000 ft^2 with ~4,000 ft^2 of second floor office space.

Equipment Repair Shops

A heavy equipment shop,, used for repairing haul trucks and other heavy-duty equipment, is semi-attached to the Office/Warehouse facility. A second equipment repair shop, utilized for lighter-duty trucks and construction equipment, is located near the Office & Warehouse building to the south.

The repair shop consists of 10 bays for mobile equipment including two lubrication bays. A general shop located in a 13,400 ft^2 housing includes an electric shop, a welding bay, a carpenter shop, and a machine shop. The Wabco Repair Shop consists of 6 bays on 10,000 ft^2. South-west of the heavy duty equipment repair shops is the tire shops, a steal framed, two storage metal clad building with a concrete slab. The building is approximately 1200m^3.

Guardhouse and Administration Buildings

The Guardhouse is located at the entrance to Faro Mine's main operational area. This facility is utilized by Security, Personnel, and Health & Safety personnel.

Explosives Plant

This facility, owned by Bulk Explosives Limited (a subsidiary of CIL), consists of one large and two smaller metal pre-fabricated buildings which house chemicals and machinery utilized for explosives manufacture and delivery. A small, lined extraction pit is also used by Bulk Explosives Limited, and is adjacent to this facility.

Table 4.68 summarizes major equipment items present in the facilities described above.

Table 4.68

List of Major Equipment: FARO CONCENTRATOR

Primary Crusher Plant	
Primary Crusher, Allis Chalmers 54 X 74 gyrator	
Apron Feeder	
Oil lubrication system for above	
2 Double Deck Vibrating Screens	
Coarse Ore Storage	
Syntron feeders	
Coarse ore storage 16,000 tons live capacity	
Mill Complex	
a) Fine Crushing	
1	Standard 7 ft Symons Cone Crusher (Secondary
1	Single Deck Vibrating Screen
2	Short Head 7 ft symons Cone Crushers (Tertiary
2	Single Deck Vibrating Screens
b) Grinding Bay	
3	Hardinge 9 ft X 12 ft Rod Mills (450 Hp each)
3	Hardinge 9 ft X 12 ft Ball Mills (450 Hp each)
6	Krebs Cyclone Clusters
1	Allis Chalmers 13.5 ft X 22 ft Ball Mill (2500 Hp)
NOTE: There is a lubrication system for each Rod and Ball Mill including Regrind Mills.	
c) Floatation Circuit	
15	Denver Pb Roughers and Scavengers
3	Zn Conditioning Tanks
30	Denver Zn Roughers and Scavengers
30	Hardinge 9 ft X 12 ft Pb Regrind Mill (450 Hp)
26	Ist Pb Cleaners
13	Pb Retreatment
12	2nd Pb Cleaners
10	rd Pb Cleaners
1	Hardinge 9 ft X 12 ft Ist Zn Regrind (450 Hp)
26	Ist Zn Cleaners
1	Hardinge 9 ft X 12 ft 2nd Zn Regrind (450 Hp)
1	Retreatment Conditioning Tanks
13	Zn Retreatment
22	2nd Zn Cleaners

12	3rd Zn Cleaners
4	Zn Cleaners
d)	**Thickening/Clarifying/Tanks**
	75 ft dia. Centre Rake Drive Eimoo Thickener (Pb)
1	90 ft dia. Eimco Thickener (Zn
1	65 ft Zn Clarifier
1	Pb Clarifier
	Stock Tanks (1 Pb, 1 Zn)
e)	**Filtering**
6	Peterson Disk Filters (2 Pb, 2 Zn, 1 Bulk, 1 Extra)
f)	Drying
1	Standard Steel Oil Fired Rotary Kiln (Pb)
4	Coal Fired Rotary Kilns (2 Zn, 1 Bulk, 1 Extra)
g)	Miscellaneous
3 -	Hot Water Boilers
3 -	Air Compressors
	Coal Handling Facility
1	Coal Breaker
	Mincentrate Storage
1	Shed, Capacity 7000 tons
	Warehouse and Shop Facilities
1	Repair Shop with 10 bays for mobile equipment (including 2 lubrication bays)
1	General Shop 13,400 sq ft housing:
1	electric shop
1	welding bay
1	carpenter shop
1	machine shop
1	Warehouse Facilities 18,000 sq ft (plus 4,000 sq ft of second floor office space)
1 -	Waboo Repair Shop with 6 bays 10,000
	Major Overhead Cranes (located at)
a)	Primary Crusher Plant
b)	Secondary/Tertiary Crushing
c)	Grinding Bay
d)	Mobile Equipment Repair Shops

4.6.2 Stockpiles Near Mill

There are several stockpiles of various material near the Mill. Mostly these are various ore types awaiting processing, however, some materials may not be amenable to processing. The crusher stockpile is nearest the crusher and is essentially a transfer point for pit run ore brought from the transfer pad at Grum. This stockpile is always active and as production rates drop in the pits the stockpile is rapidly depleted. In the normal course of events, at the completion of the mine plan the entire contents of this stockpile will be processed.

Three hundred metres east of the Crusher stockpile is the medium grade stockpile. This stockpile is essentially depleted already as it contains a small tonnage of Vangorda carbonaceous ore only. The base of the stockpile was not investigated but probably requires some clean-up.

About 400 m. northeast of the Crusher Stockpile in the west Mt. Mungly Dump is material brought from the bulk terminal in Skagway during a cleanup of that site. The material was delivered by Curragh and characterized as "concentrate contaminated with soil returned for re-processing". The material appears to consist of sand, gravel and cobbles but also contains lead and zinc concentrates and plastic sheet remnants. It is unclear what Curragh intended to do with this material but in the context of this mine area the material poses a negligible additional environmental risk.

Between the Crusher and Medium Grade Stockpiles was a large stockpile of oxidized ore from the sub-crop of the Faro Deposit. This material was mined by Cyprus Anvil and stockpiles here. Curragh recognized that the effects of oxidation were mainly confined to the fine fraction and that the coarse fraction could be processed. As a result of this finding the pile was run through a screening plant and the coarse fraction was processed. The fines remain in several piles on the site of the original stockpile and a small amount is across the main haul road in the west Fuel Tank Dump.

Two large stockpiles have been developed near the main haul road entrance to the Faro pit. These stockpiles, low grade "A" and "C", are between the lube shack and the Ranch Dump and behind the lube shack respectively. The Dumps contain ore mined in the mid 1980's by Curragh as fresh pit low grade. The material is now oxidized and may not be suitable for direct processing. The screening approach used for the oxidized ore noted above may be effective for this material. If the low grade is screened then the remaining fines should be placed in the Faro pit with the tailings.

4.6.3 Tailings Pipelines/Drop Boxes

Most of the remnants of the old tailings pipelines to the intermediate impoundment have been removed and used elsewhere. A unused drop box remains near the mill beside the highway.

A 20 inch OD Sclar pipeline extends from the mill up to the Faro pir on the east side of the main pit haul road entrance. This is the current tailings pipeline and will remain in use through the milling of all reserves and re-processing of tailings.

4.7 Infrastructure

The Faro mine area includes several facilities that supported mining but are not themselves pits or dumps or other mine workings. These are outlined below.

4.7.1 Roads

There are several light vehicle and heavy haul roads around the Faro site. Few of these are maintained any longer but all are used to some extent either for site cleanup, characterization studies such completed for this plan, inspections and exploration. For the foreseeable future the roads will remain in use.

4.7.2 Borrow Areas and other disturbed land

There are many areas around the Faro site that have been disturbed in addition to the mine and mill site itself. These are mostly former borrow areas from which surficial material was extracted for a variety of purposes including tailings dam construction, road and highway construction, camp sites. Most of these sites are in sands and gravels or glacial till and are gradually naturally re-vegetating. Included area are listed in Table 4.69 below.

Table 4.69

Disturbed Areas other than Faro Mine and Mill Site - Rose Creek Valley

Site	Purpose	Area (ha.)
Next Creek mouth	borrow for tailings dam construction	150
Lower South Fork Rose Creek Valley	borrow for highway or dam construction?	4 areas=200
North of Down Valley Tailings	borrow for tailings dam construction	150
Across Guardhouse Ck. from mine entrance	borrow for tailings dam construction	125
At mine entrance	construction campsite	100
Across highway from BXL plant	borrow for highway or dam construction?	50
Between X5 and X14	development of effluent canal	65
Lower North Fork Rose Creek	Recharge areas, dam and road building?	75
Below Fresh water dam	Dam construction	70
Mid Haul Road (W of S. Fk. Rose Ck.)	Haul Road Construction	175
Total		**1,160**

The above areas are approximate and there are many, much smaller, additional areas as well as the power line right of way and haul road footprint

4.7.3 Buildings and Equipment

There are a few buildings outside of the mill area, most of these are wood frame construction and small, included are the two explosives magazines on the upper Northwest Dump, several core shacks on the Mt. Mungly Dump, the lube shack near the main pit haul road entrance, the nearby lunchroom and assorted wooden structures in the Ranch area.

Equipment scattered around the mine site is mostly derelict and is dealt with under scrap yards

4.7.4 Scrap Yards and Refuse Areas

There are many scrap yards on the tops of the various dumps around the Faro site. Included are materials left over from old mill expansions, old mobile equipment (shovels, trucks), old light vehicles, tires etc.. The major sites are shown on Figure 4-107 and include: the east Main Dump, the north end of the west Main Dump (possibly a long term parking area), the east Tank Farm Dump, the upper and lower Parking Lot Dumps, the middle and upper Northwest Dumps and the Ranch Dump area. This material has not been inventoried for this study. We understand that Anvil Range is gradually cleaning up the areas and discarding material of no value in the Faro Pit tailings area. Some of the material may be worth backhauling to southern Canada in times of high steel prices but this has not happened to any significant degree during the mine life despite occasional attempts.

4.7.5 Fuel oil storage facilities and Lube Shack

There are several above ground storage tanks in the 10,000 gallon range on the minesite, these are used to store diesel and gasoline. Most of these tanks are relatively inactive now that the focus of mining has moved to the Vangorda plateau.

4.7.6 Power

Power is supplied to the Faro site by a power line connected to the Whitehorse Aishihik Faro grid in Faro. There are transformers at the Faro mill to step this power down to the distribution voltage on site and a local distribution system around the site. Most of this system will remain in place in case it is required for mitigation purposes. To the best of out knowledge temporary lines and transformers have been moved to the Vangorda Plateau site for continued use.

A standby diesel generator at the mill can supply basic power needs in case of a emergency, but it is not adequate to operate the mill.

4.8 References

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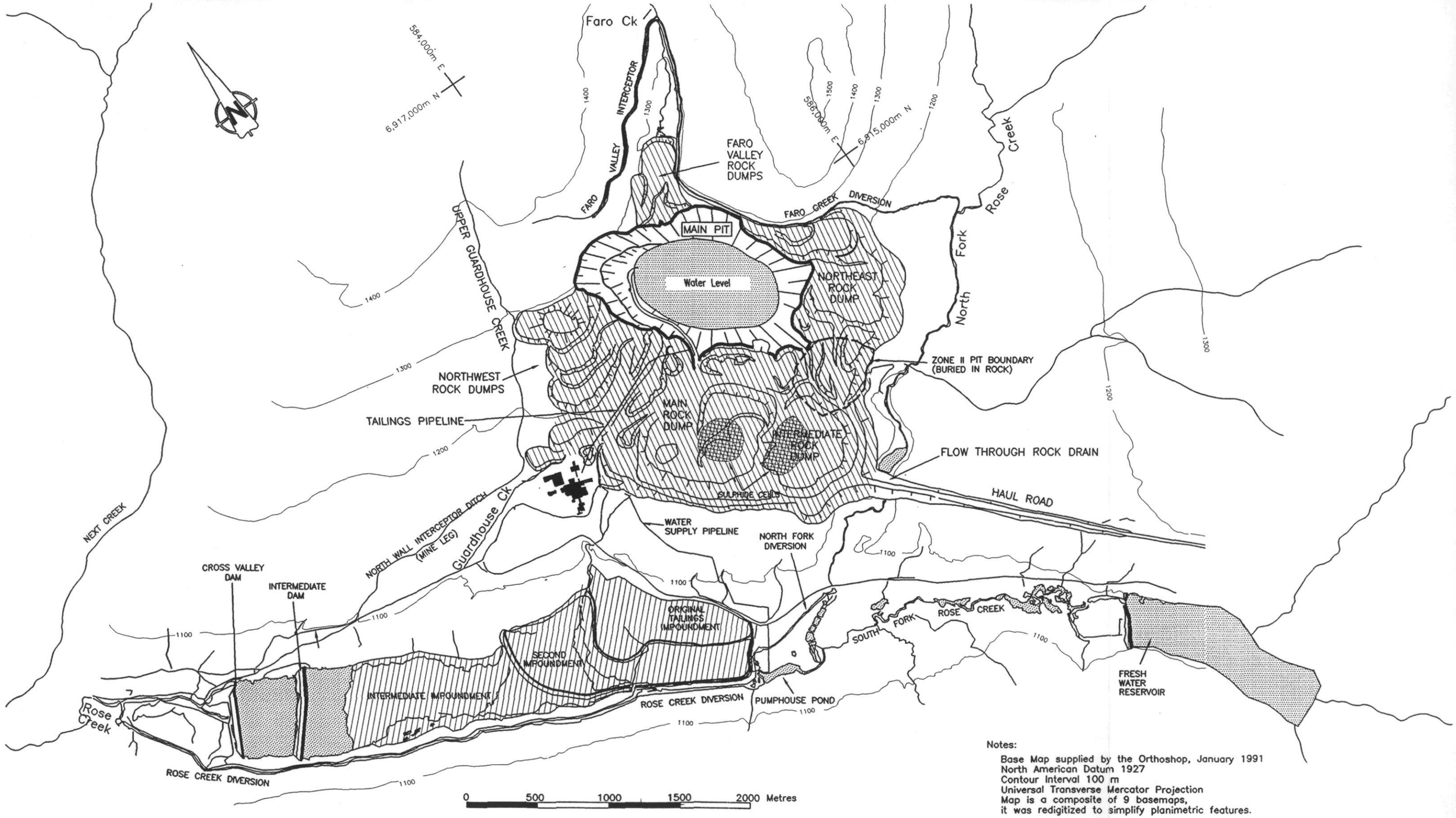
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FIGURES



Scale 1:25 000

Contour Intervals at 100m

Notes:

Base Map supplied by the Orthoshop, January 1991
North American Datum 1927
Contour Interval 100 m
Universal Transverse Mercator Projection
Map is a composite of 9 basemaps,
it was redigitized to simplify planimetric features.

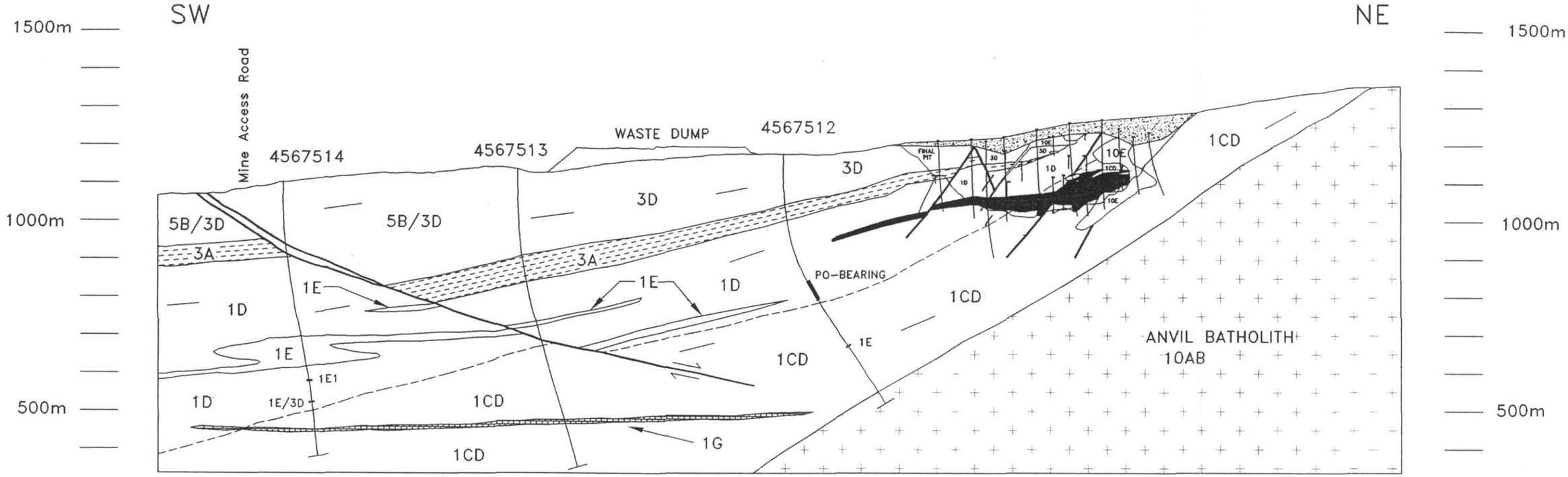
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Anvil Complex Closure Plan

Overview of Faro Mine Site

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-1



ANVIL PLUTONIC SUITE
- 10AB Biotite–muscovite granite

ANVIL DYKE SUITE
- 10E Hornblende–biotite quartz diorite

VANGORDA FORMATION
- 5B/3D Calcareous schist with calc–silicate
- 3D Calc–silicate
- 3A Carbonaceous schist with greenstones

MOUNT MYE FORMATION
- 1D Noncalcareous andalusite schist
- 1CD Biotite–muscovite–andalusite–staurolite schist
- 1E Carbonaceous schist
- 1G Marble

MINERALIZED HORIZON
- Massive and disseminated sulphides of the Faro Deposit

Fault

S2 foliation

metamorphic transition

200 0 200 400 600
metres

VERTICAL CROSS SECTION
FARO 118 + 000
REGIONAL

Figure: 8	Dwg: AN-AN-92-005
Rpt: 96-01	File: REGF118.DWG
Drawn by: LCP	Date: 28/01/1996

Source: Access Mining Consultants, 1996

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Anvil Complex Closure Plan

Faro Mine Area
Geological Cross Section

PROJECT NO.	DATE	APPROVED	FIGURE
033001	August 1996		4-2

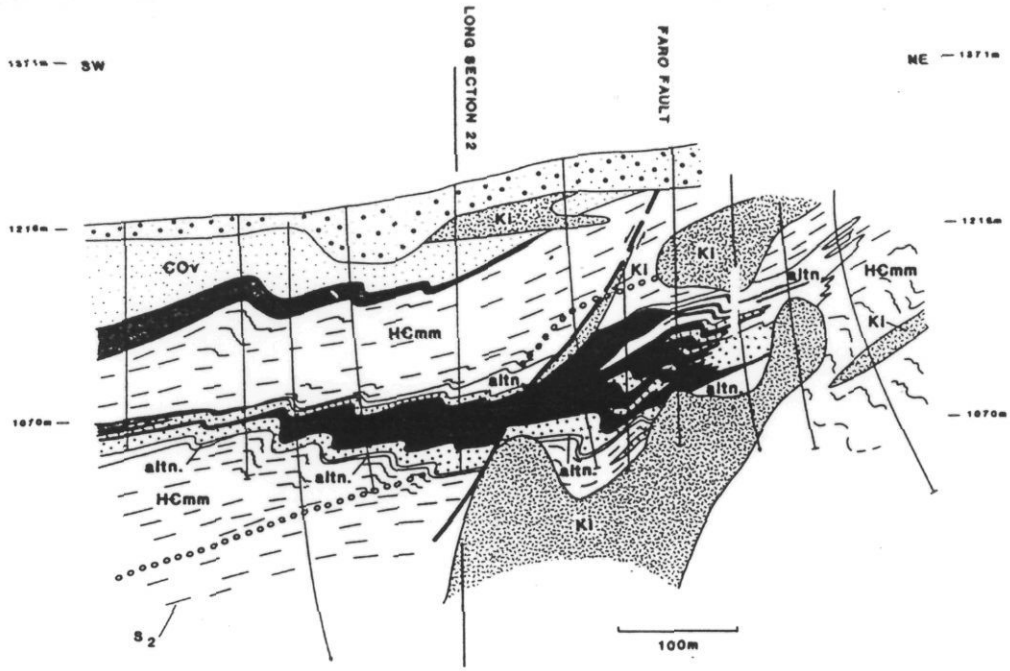


FIGURE Vertical cross section 118 of the Faro deposit, looking northwest, no vertical exaggeration. The present pit outline is not shown. Black is massive sulphides; coarse stipple is quartzose disseminated sulphides; grey is carbonaceous phyllite; altn. = white mica envelope (bleached altered phyllites). Northeast of Faro Fault most massive sulphides are low grade and transitional into quartzose sulphides with high-grade bands at the top of the deposit and scattered through the remainder. Southwest of the Faro Fault the massive sulphides are mostly high grade, the upper half is variably baritic. On this and all other sections of the Faro deposit pyrrhotitic sulphides are well developed in the southwest part of the deposit where the massive sulphides thin rapidly. The basal and peripheral quartzose unit is largely ribbon banded graphitic quartzite. The folds in S_2 and the deposit outline are probably D_4 folds. Unit ϵO_v on this section is all calc-silicate. The dotted line approximates the first appearance of staurolite in Mt. Mye schists.

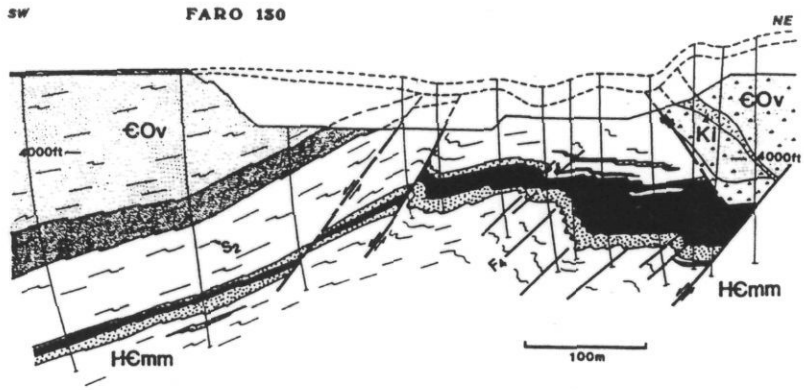


FIGURE Vertical cross section 130 of the Faro deposit, looking northwest, no vertical exaggeration. The pit outline shown is as of the June 1982 interruption of mining. Units are the same as those of Figure 30. The triangles indicate the breccia cap (Fig. 29 and text). In this portion of the deposit, a thin upper horizon is separate from the remainder of the deposit. The fault at the northeast end of the section is part of the system that separated zones 3 and 2 (Fig. 29). The distribution of sulphide facies is essentially the same as that of Figure 30 with barren pyrite common northeast of the large D_4 fold in the base of the deposit and pyrrhotitic sulphides abundant for about 150 m southwest of the fold.

Note: Information Taken from: Jennings, D.S. and Jilson, G.A. (1986), Geology and Sulphide Deposits of Anvil Range, Yukon. In: J.A. Morin (ed), Mineral Deposits of Northern Cordillera, Canadian Inst. Mining and Metallurgy, Special Volume 37, pp. 319–361.

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Anvil Complex Closure Plan

FARO DEPOSIT CROSS SECTIONS

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-3

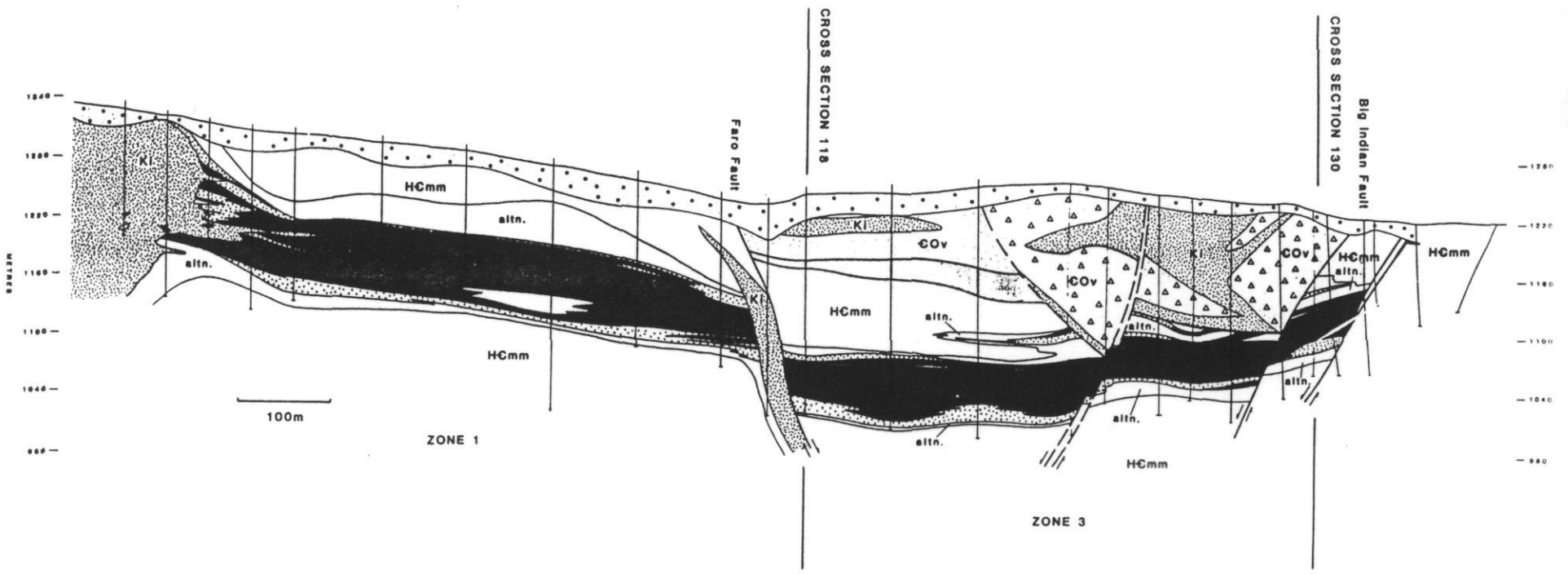


FIGURE **Vertical longitudinal section 22 of the Faro deposit, looking northeast, no vertical exaggeration. Units are same as Figure 4-3 Zones 1 and 3 are indicated, the largely eroded northeast extension of zone 2 is just beneath the overburden southeast of the Big Indian Fault. Within about 150 m of the diorite dyke at the northwest end of the section, the massive sulphides are altered to pyrrhotite rich assemblages containing local large magnetite octahedra and partly reverted to pyrite. Adjacent to the diorite dyke along the Faro Fault massive magnetite is developed in the sulphides.**

Note: Information Taken from: Jennings, D.S. and Jilson, G.A. (1986), Geology and Sulphide Deposits of Anvil Range, Yukon. In: J.A. Morin (ed), Mineral Deposits of Northern Cordillera, Canadian Inst. Mining and Metallurgy, Special Volume 37, pp. 319–361.

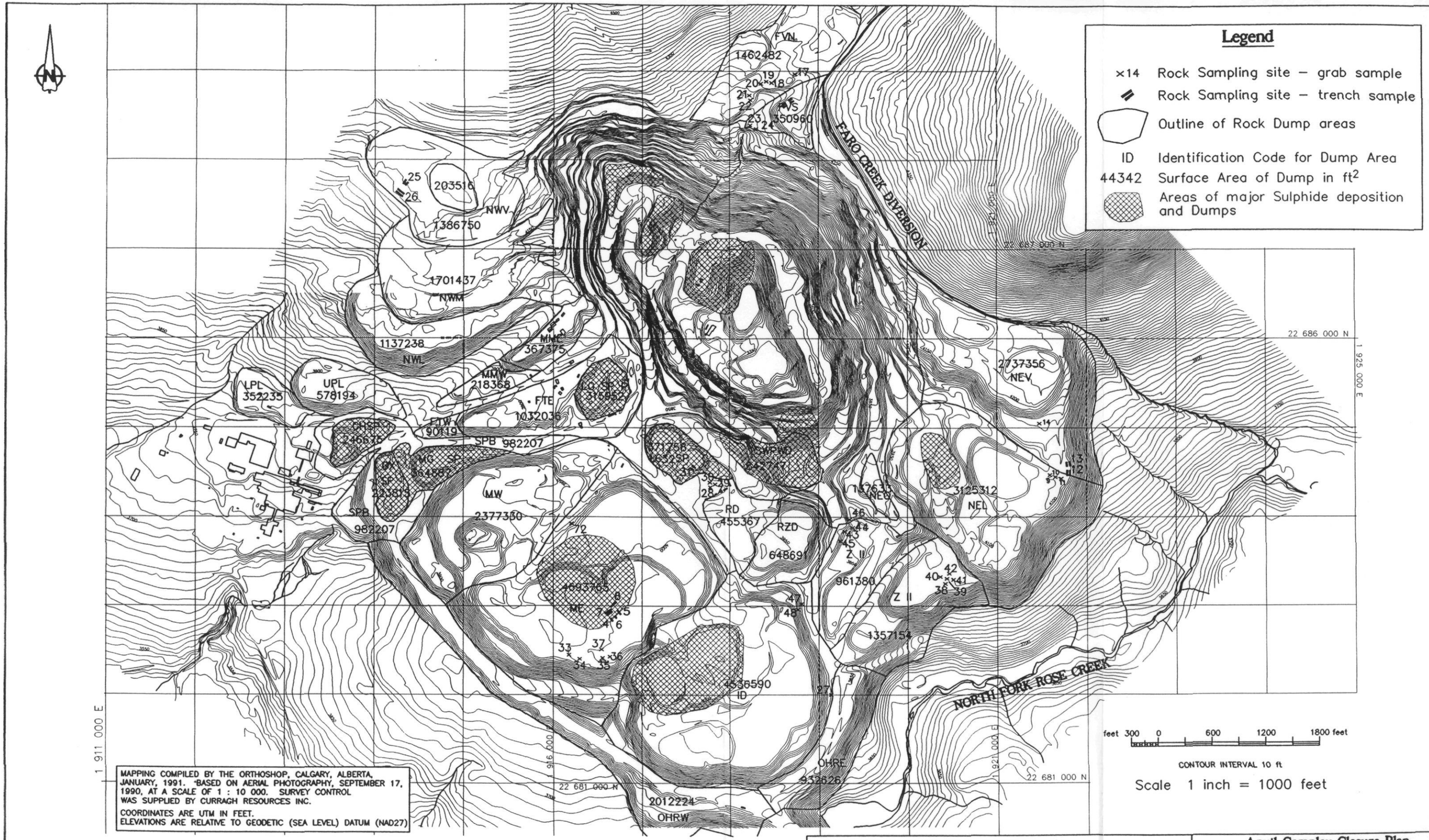
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FARO DEPOSIT VERTICAL LONGITUDINAL SECTION 22

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-4



MAPPING COMPILED BY THE ORTHOSHOP, CALGARY, ALBERTA, JANUARY, 1991. BASED ON AERIAL PHOTOGRAPHY, SEPTEMBER 17, 1990, AT A SCALE OF 1 : 10 000. SURVEY CONTROL WAS SUPPLIED BY CURRAGH RESOURCES INC.

COORDINATES ARE UTM IN FEET.
ELEVATIONS ARE RELATIVE TO GEODETIC (SEA LEVEL) DATUM (NAD27)

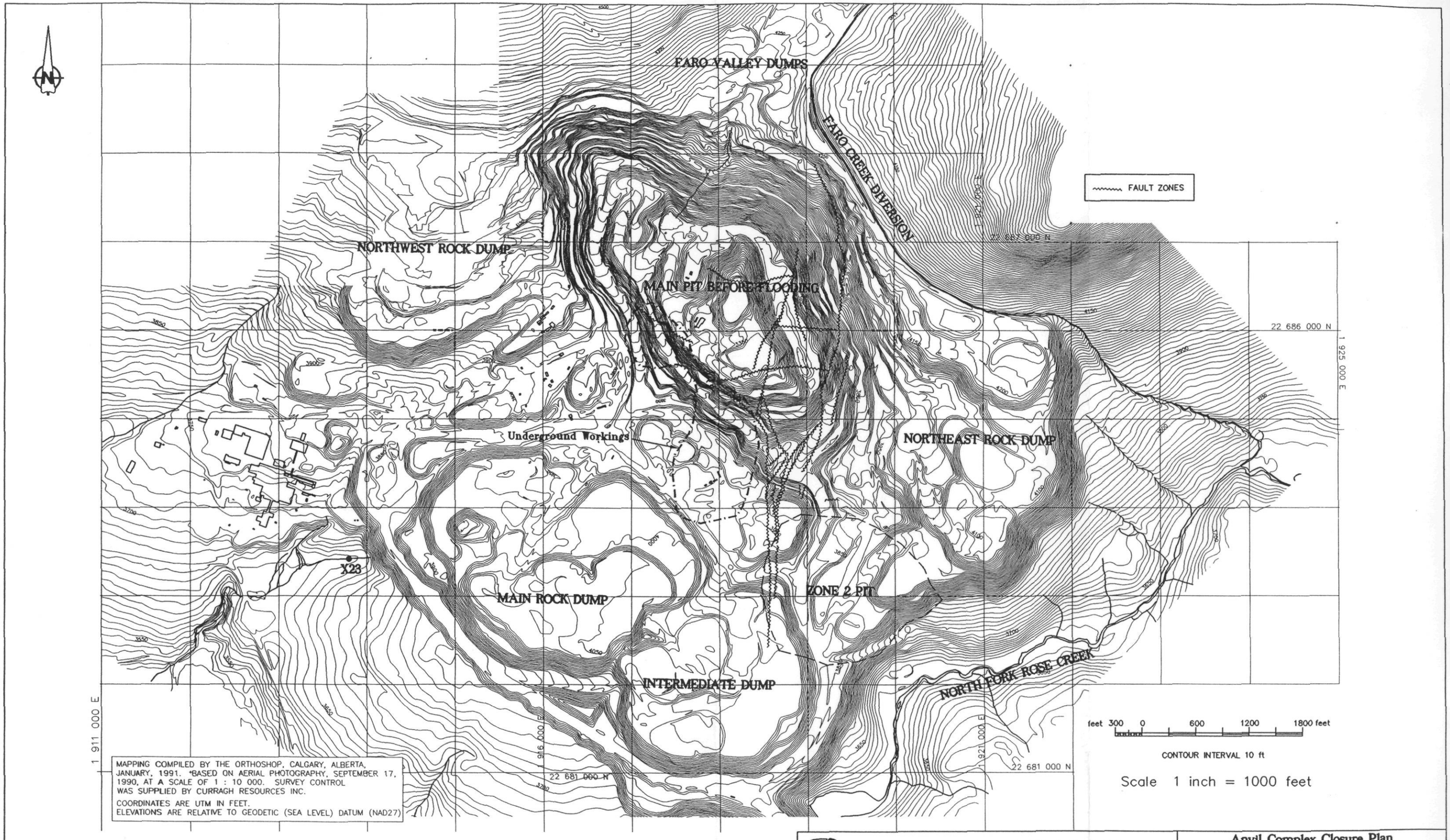
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Faro Dumps Sampling Sites and Dump Classification

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Oct. 1996		4-5



MAPPING COMPILED BY THE ORTHOSHOP, CALGARY, ALBERTA, JANUARY, 1991. *BASED ON AERIAL PHOTOGRAPHY, SEPTEMBER 17, 1990, AT A SCALE OF 1 : 10 000. SURVEY CONTROL WAS SUPPLIED BY CURRAGH RESOURCES INC.

COORDINATES ARE UTM IN FEET.
ELEVATIONS ARE RELATIVE TO GEODETIC (SEA LEVEL) DATUM (NAD27)

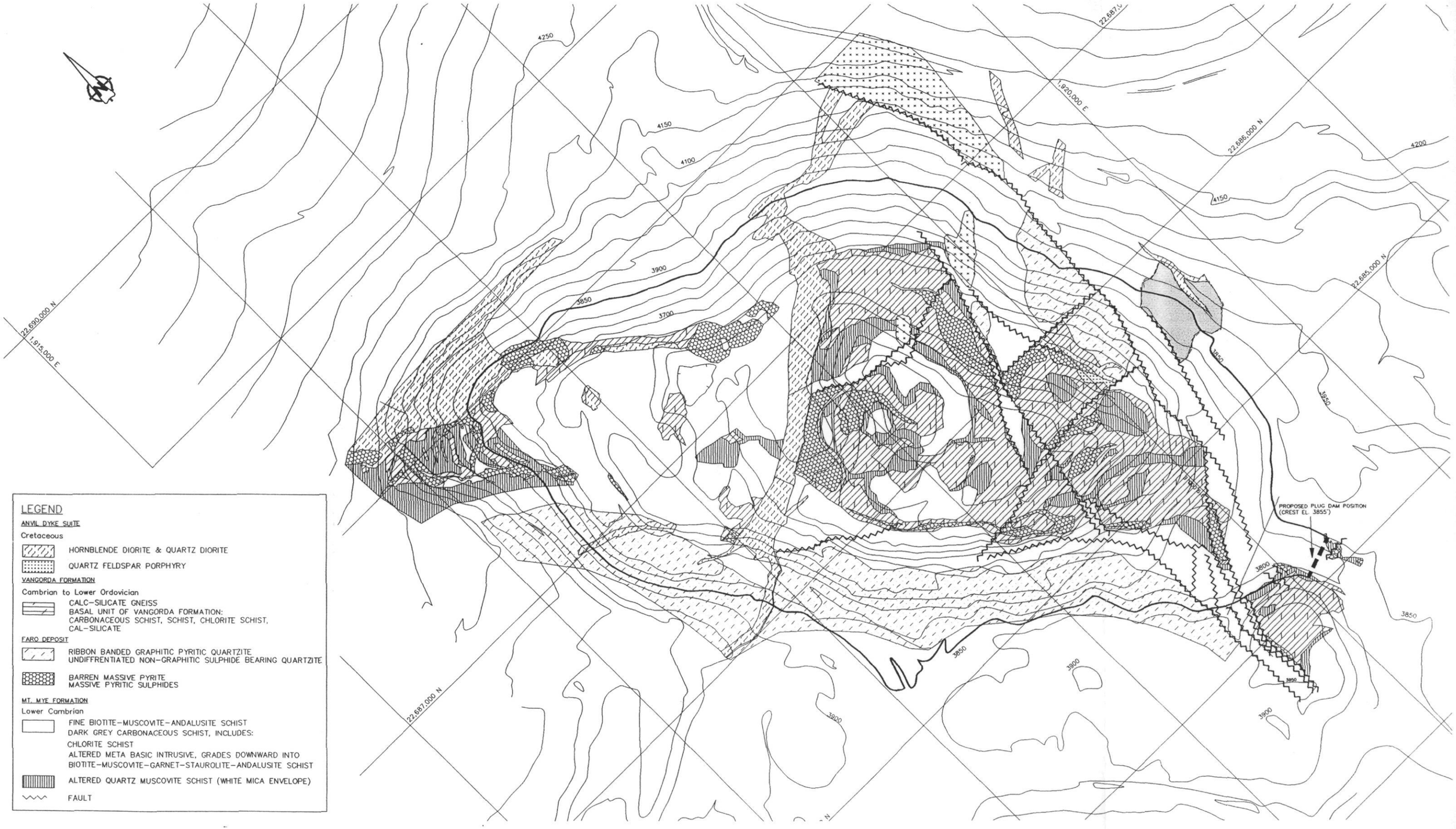
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Anvil Complex Closure Plan

Faro Pit and Dumps 1990 Topography

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Oct. 1996		4-6



LEGEND

ANVIL DYKE SUITE

Cretaceous

- HORNBLENDE DIORITE & QUARTZ DIORITE
- QUARTZ FELDSPAR PORPHYRY

VANGORDA FORMATION

Cambrian to Lower Ordovician

- CALC-SILICATE GNEISS
  BASAL UNIT OF VANGORDA FORMATION:
  CARBONACEOUS SCHIST, SCHIST, CHLORITE SCHIST,
  CAL-SILICATE

FARO DEPOSIT

- RIBBON BANDED GRAPHITIC PYRITIC QUARTZITE
  UNDIFFRENTIATED NON-GRAPHITIC SULPHIDE BEARING QUARTZITE
- BARREN MASSIVE PYRITE
  MASSIVE PYRITIC SULPHIDES

MT. MYE FORMATION

Lower Cambrian

- FINE BIOTITE-MUSCOVITE-ANDALUSITE SCHIST
  DARK GREY CARBONACEOUS SCHIST, INCLUDES:
  CHLORITE SCHIST
  ALTERED META BASIC INTRUSIVE, GRADES DOWNWARD INTO
  BIOTITE-MUSCOVITE-GARNET-STAUROLITE-ANDALUSITE SCHIST
- ALTERED QUARTZ MUSCOVITE SCHIST (WHITE MICA ENVELOPE)
- FAULT

SCALE: 1"=500'

Note:
All information obtained from Curragh Resources Inc. Fig. 10, "Faro Mine – Plan View, Ultimate Pit Geology". March 1988.

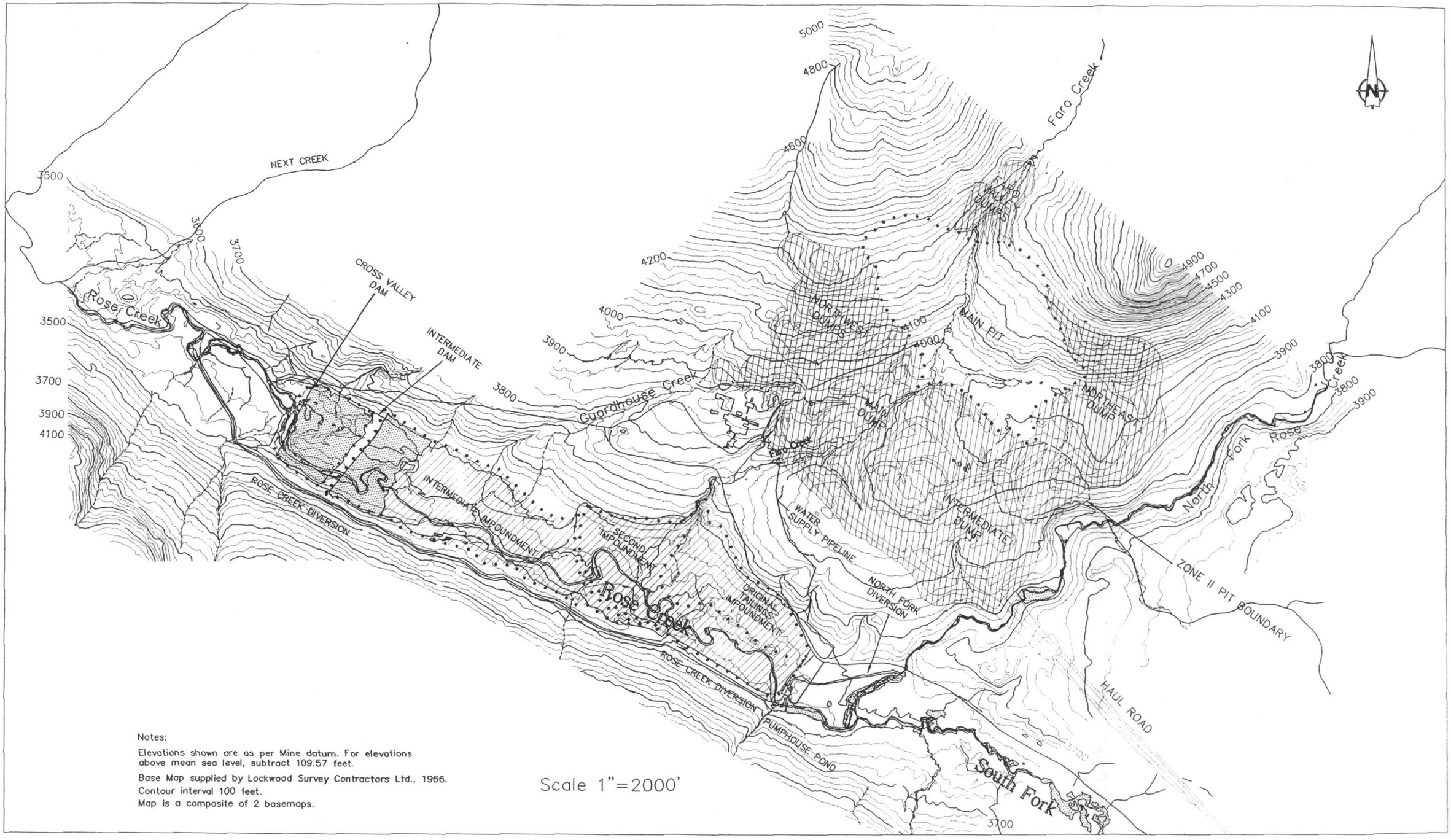
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Faro Pit Geology

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-7



Notes:

Elevations shown are as per Mine datum. For elevations above mean sea level, subtract 109.57 feet.

Base Map supplied by Lockwood Survey Contractors Ltd., 1966.

Contour interval 100 feet.

Map is a composite of 2 basemaps.

Scale 1"=2000'

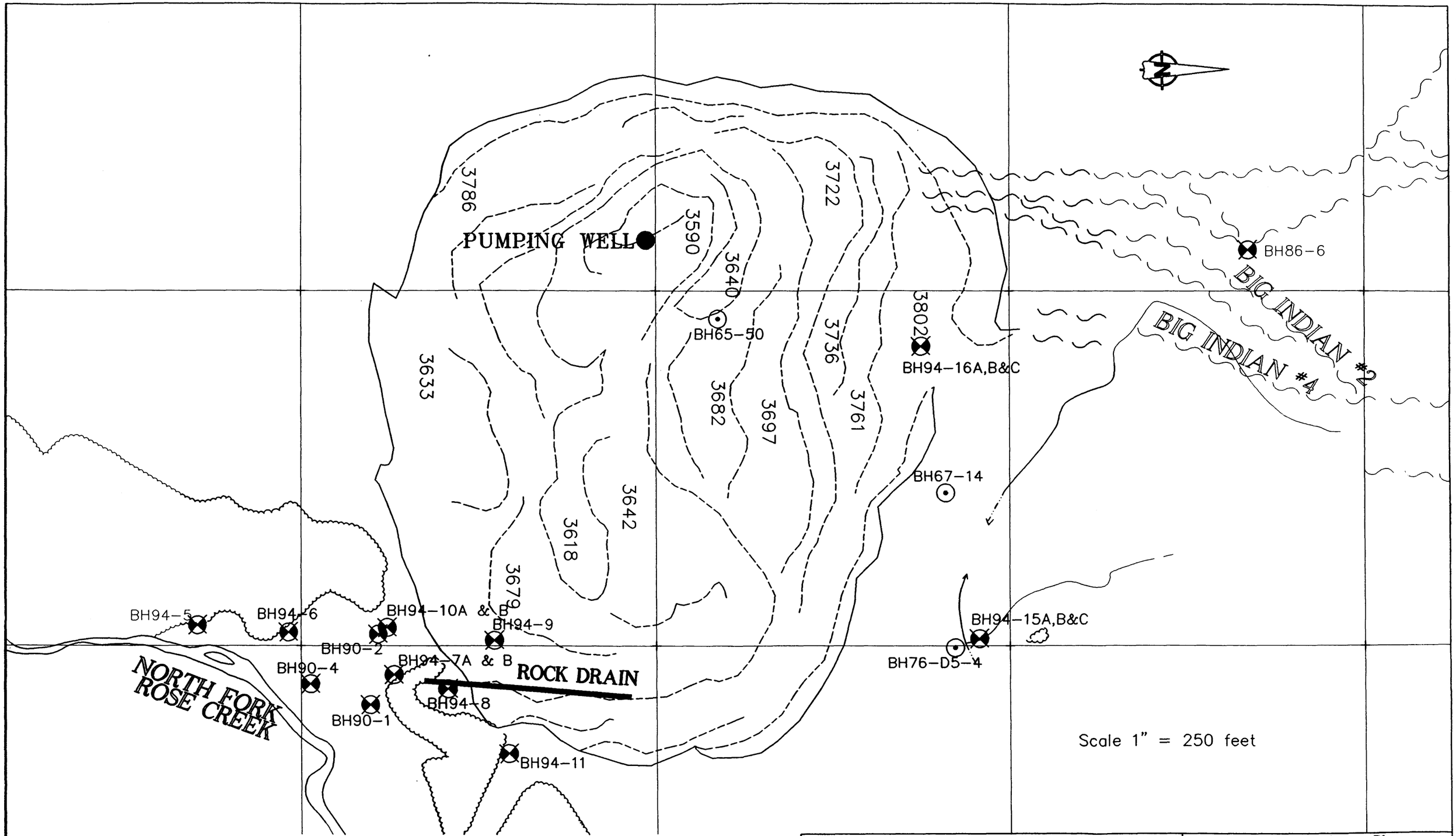
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Faro Pre-Mining Topography with Pit and Tailings location

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-8



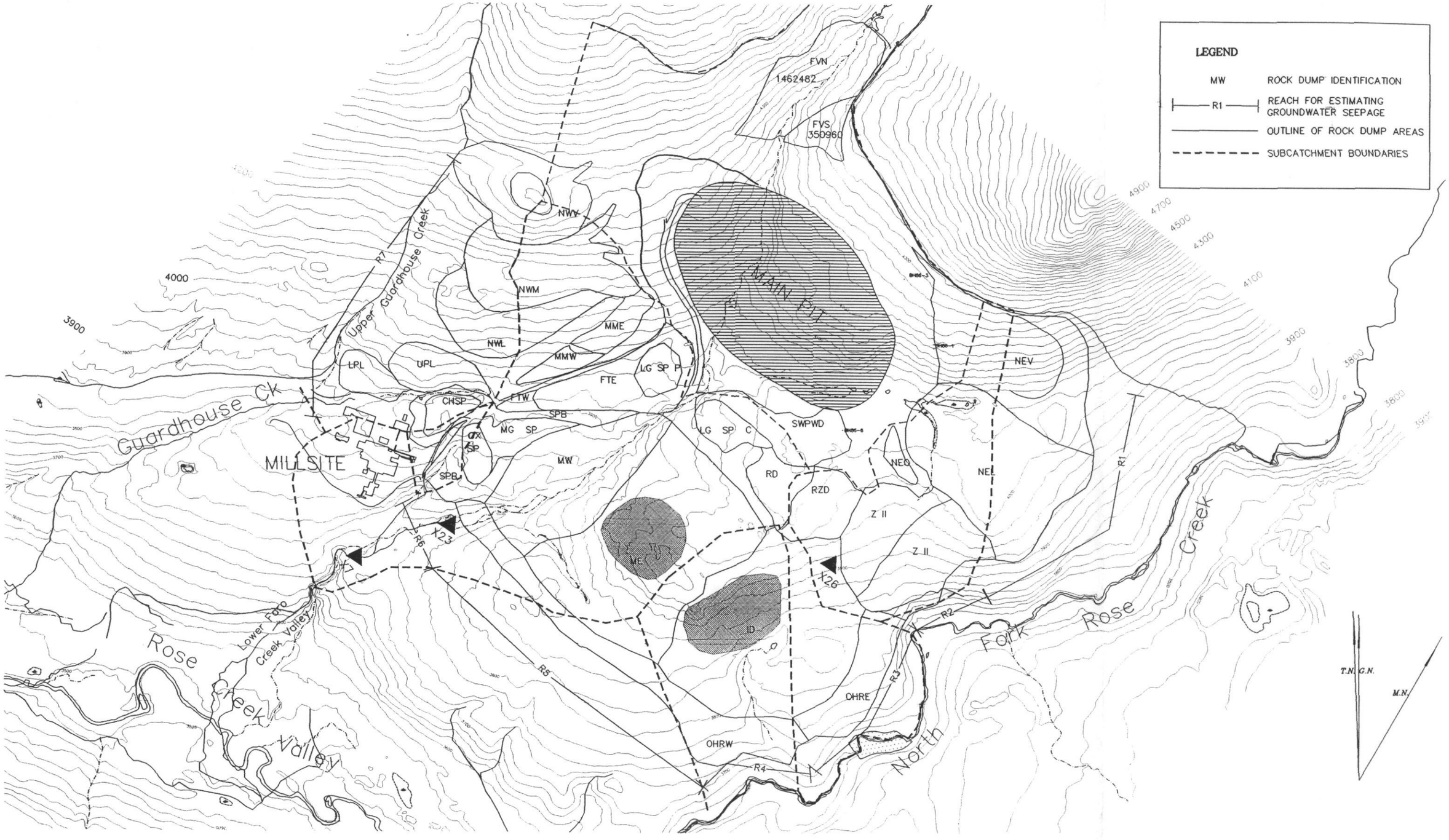
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Zone 2 Pit Bench Plan

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-9



Scale 1"=1200'

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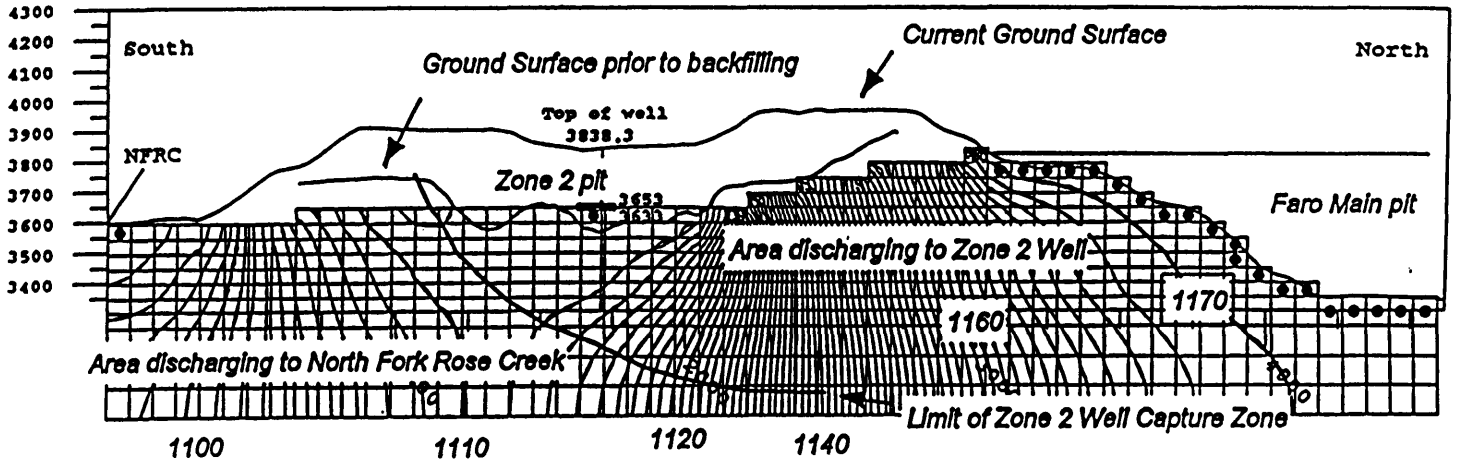
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Anvil Complex Closure Plan

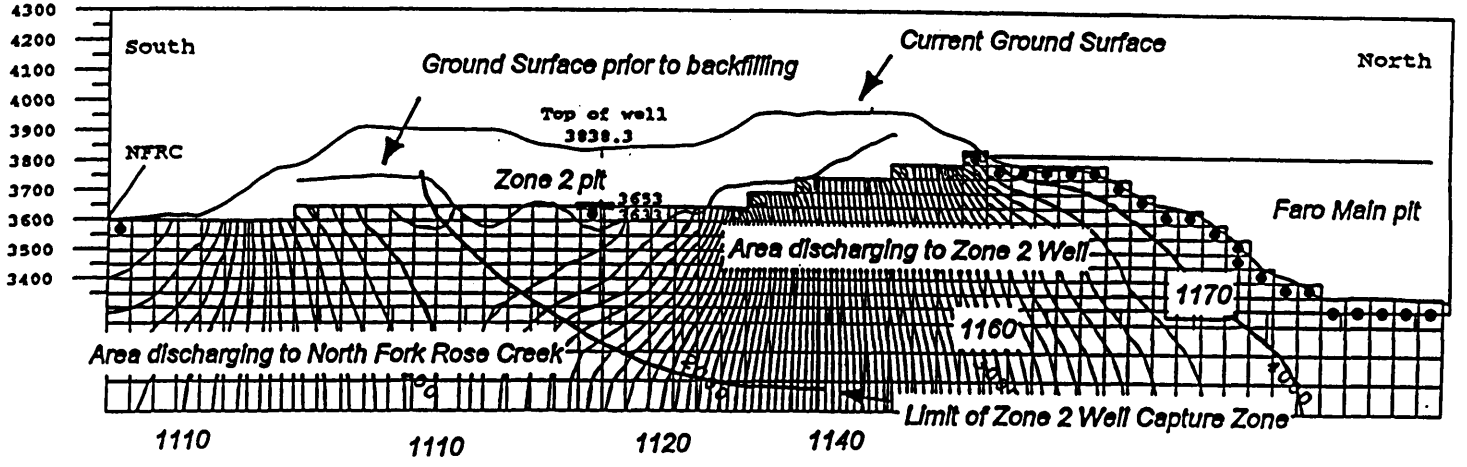
Subcatchments of Faro Pit and Dumps

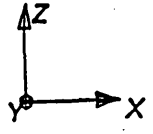
PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-10

Capture Zone of Zone 2 Well - Intact Bedrock



Capture Zone of Zone 2 Well - Limited Fault Zone





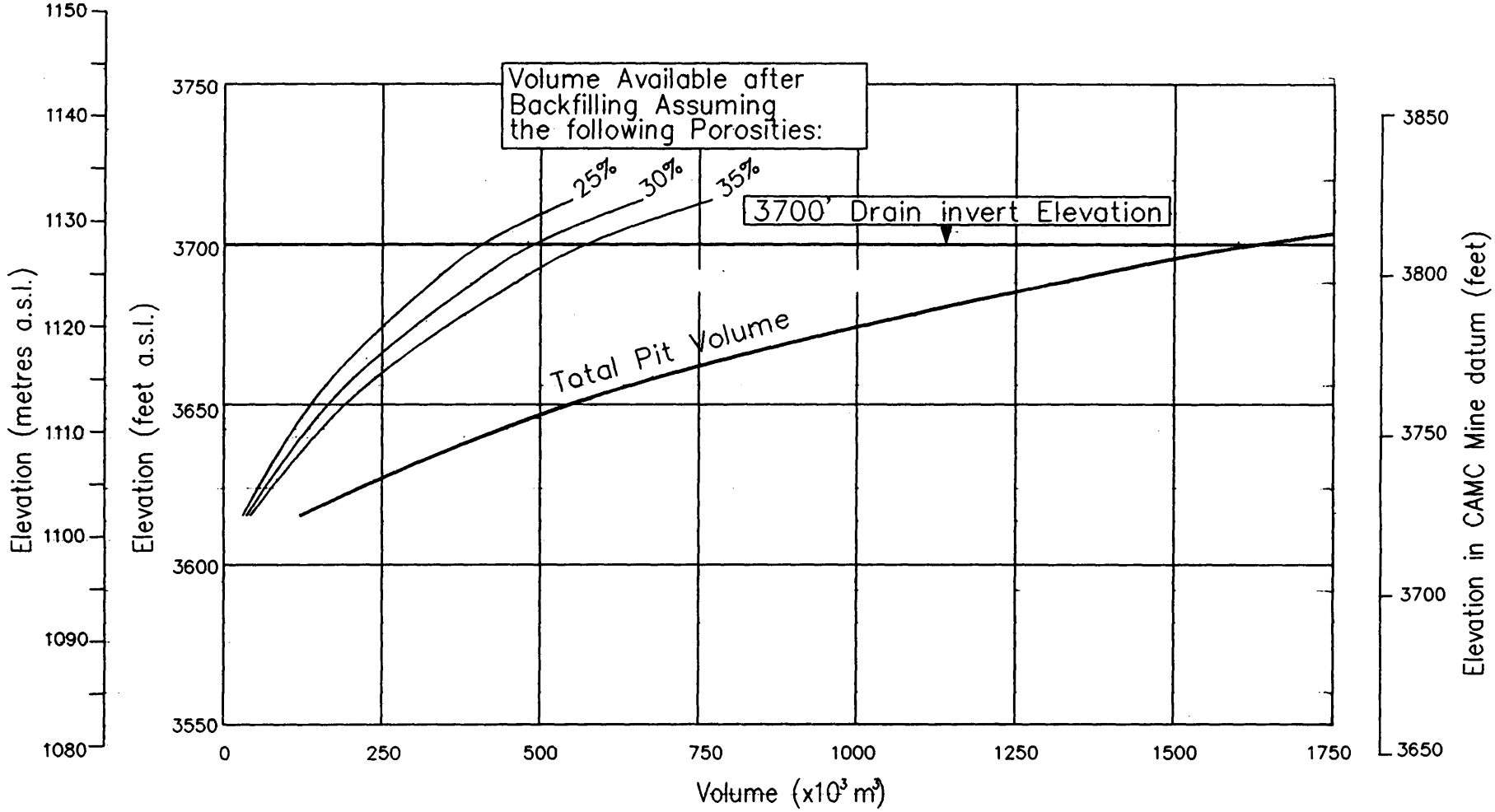
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Anvil Complex Closure Plan

SIMULATION CAPTURE ZONE OF ZONE II WELL

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-11



Note:
1. From SRK report – Project No. 60643 for Curragh Resources Inc.
2. Mine datum is UTM + 109.57'

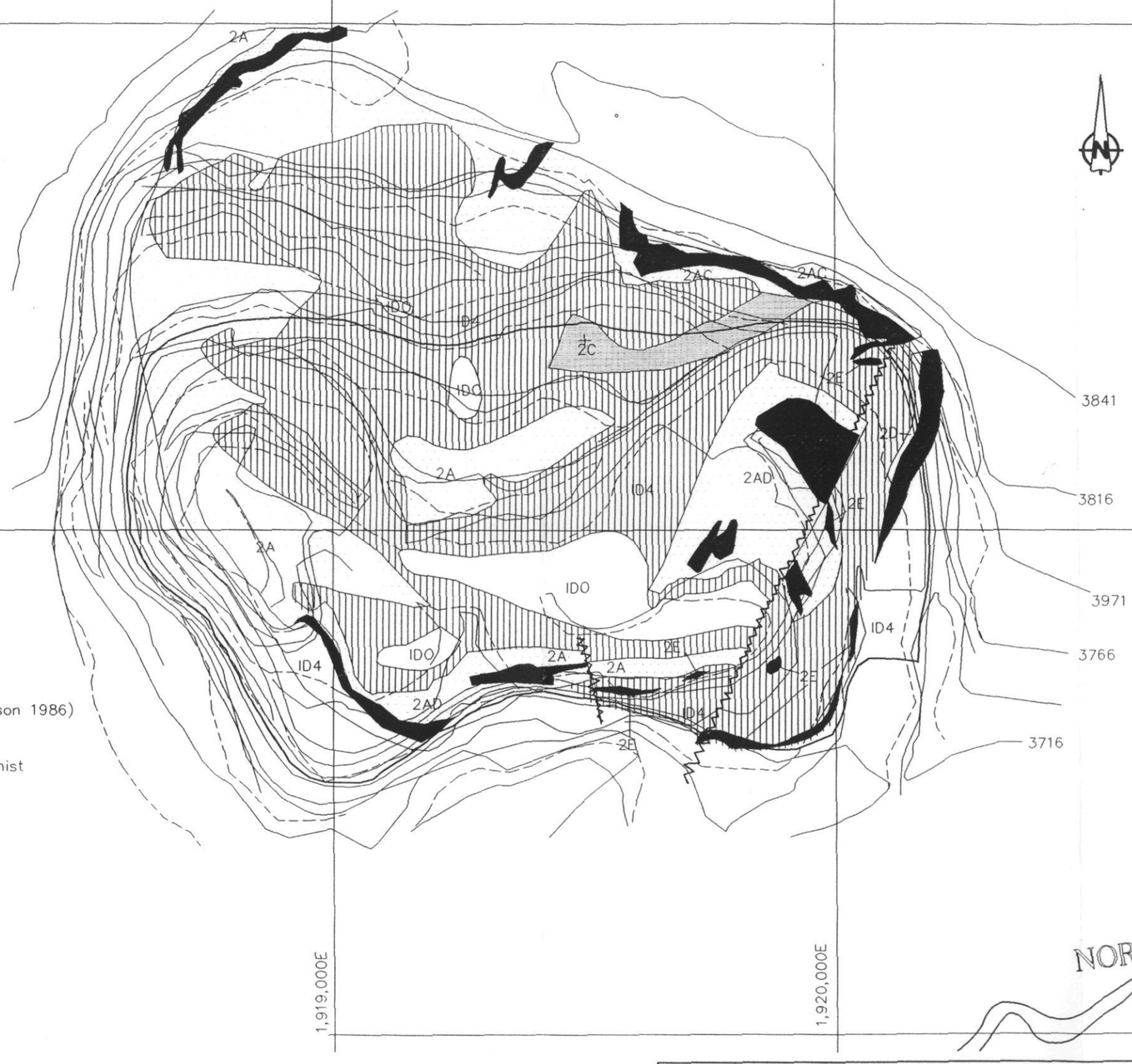
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Anvil Complex Closure Plan

Height Capacity Curve Zone II Pit

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-12



LEGEND

ROCK TYPES (described in Jennings and Jilson 1986)

- Biotite muscovite andalusite schist
- White mica envelope: muscovite schist
- Ribbon–banded graphitic quartzite
- Semi–massive pyritic sulphide
- Massive pyritic sulphide
- Fault
- 3703' Water Elevation

Note: All information obtained from Curragh Resources Inc., Figure 10 in "Development of the Zone 2 Waste Dump", December, 1987
Coordinates are in Feet as per UTM datum
Elevations are in Feet above mean sea level (NAD27)

Scale 1"=250'

ROBERTSON GEOCONSULTANTS INC.
Consulting Geotechnical and Environmental Engineers

Anvil Range Mining Corporation

Anvil Complex Closure Plan

ZONE II PIT GEOLOGY

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4–13

Figure 4-14
pH time trend, Faro Pit dewatering and Faro Pit water while filling with tailings

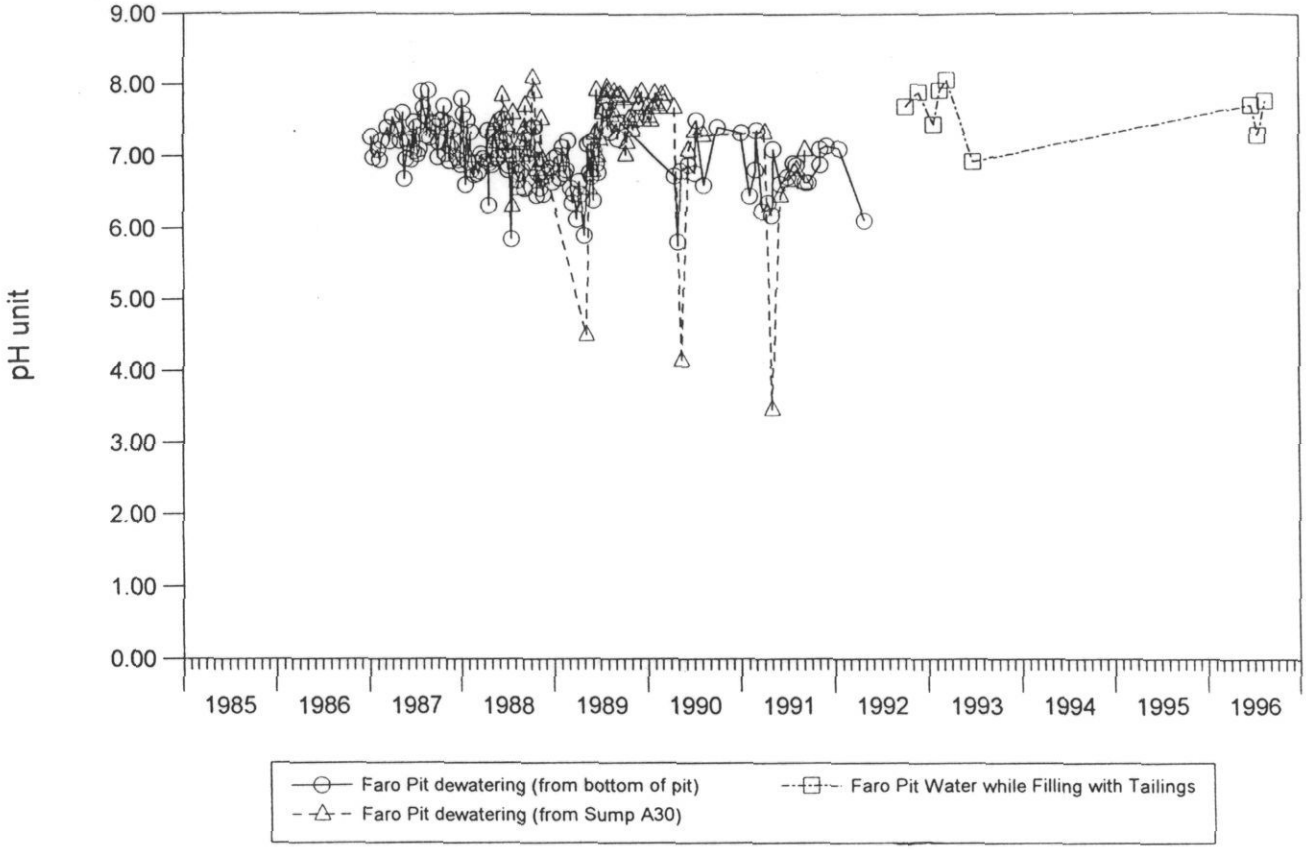


Figure 4-15
Alkalinity time trend, Faro Pit dewatering and Faro Pit water while filling with tailings

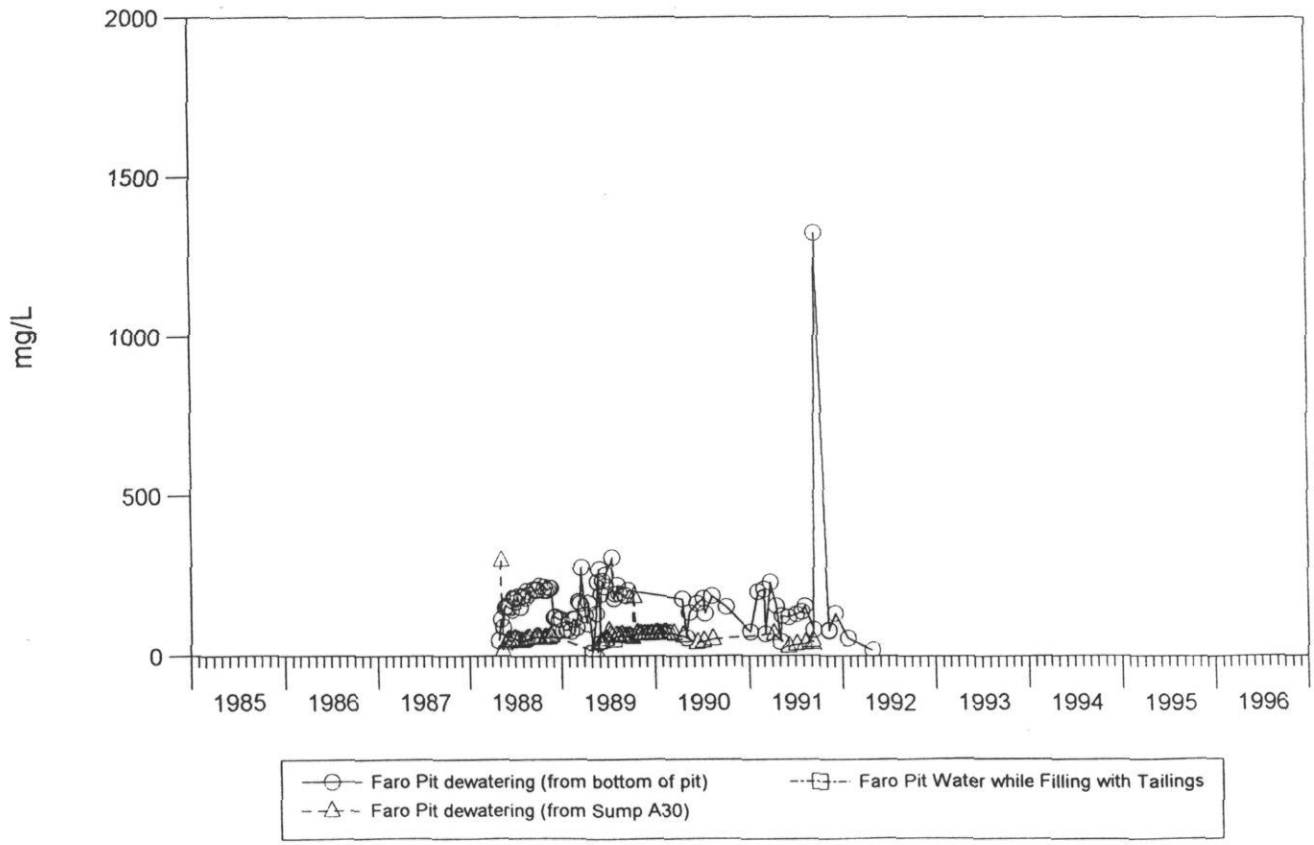


Figure 4-16
Total sulphate time trend, Faro Pit dewatering and Faro Pit water while filling with tailings

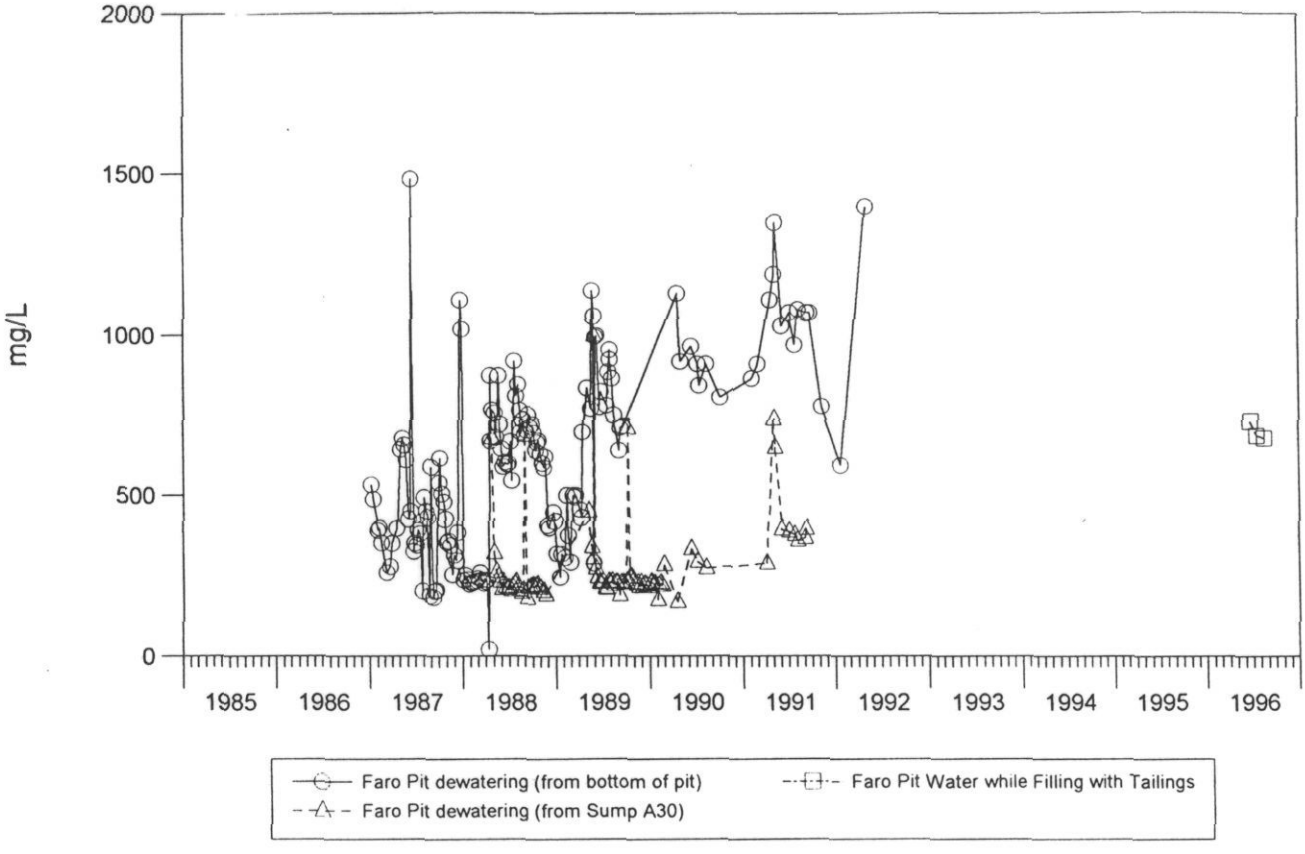


Figure 4-17
Total zinc time trend, Faro Pit dewatering and Faro Pit water while filling with tailings

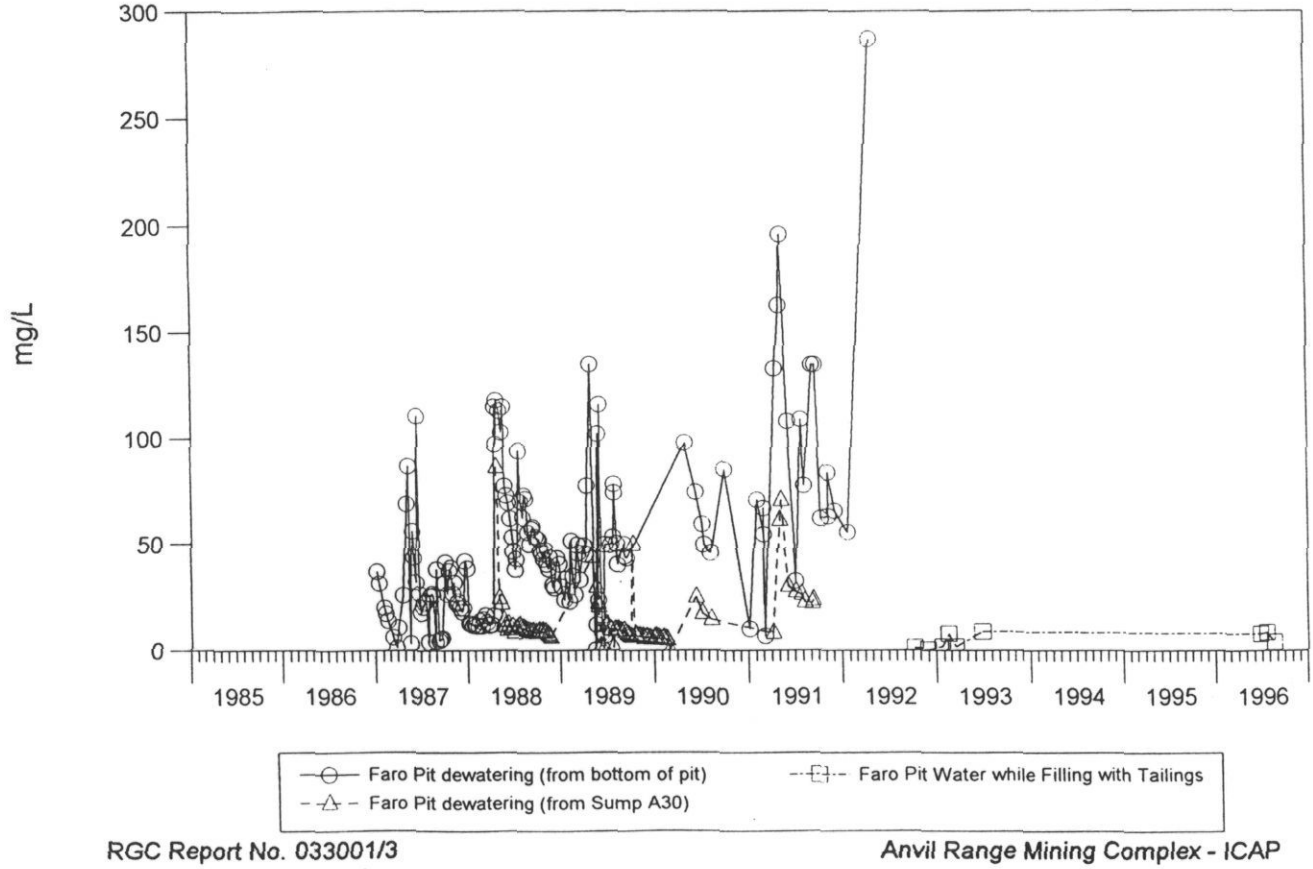


Figure 4-18
History of Dump Construction

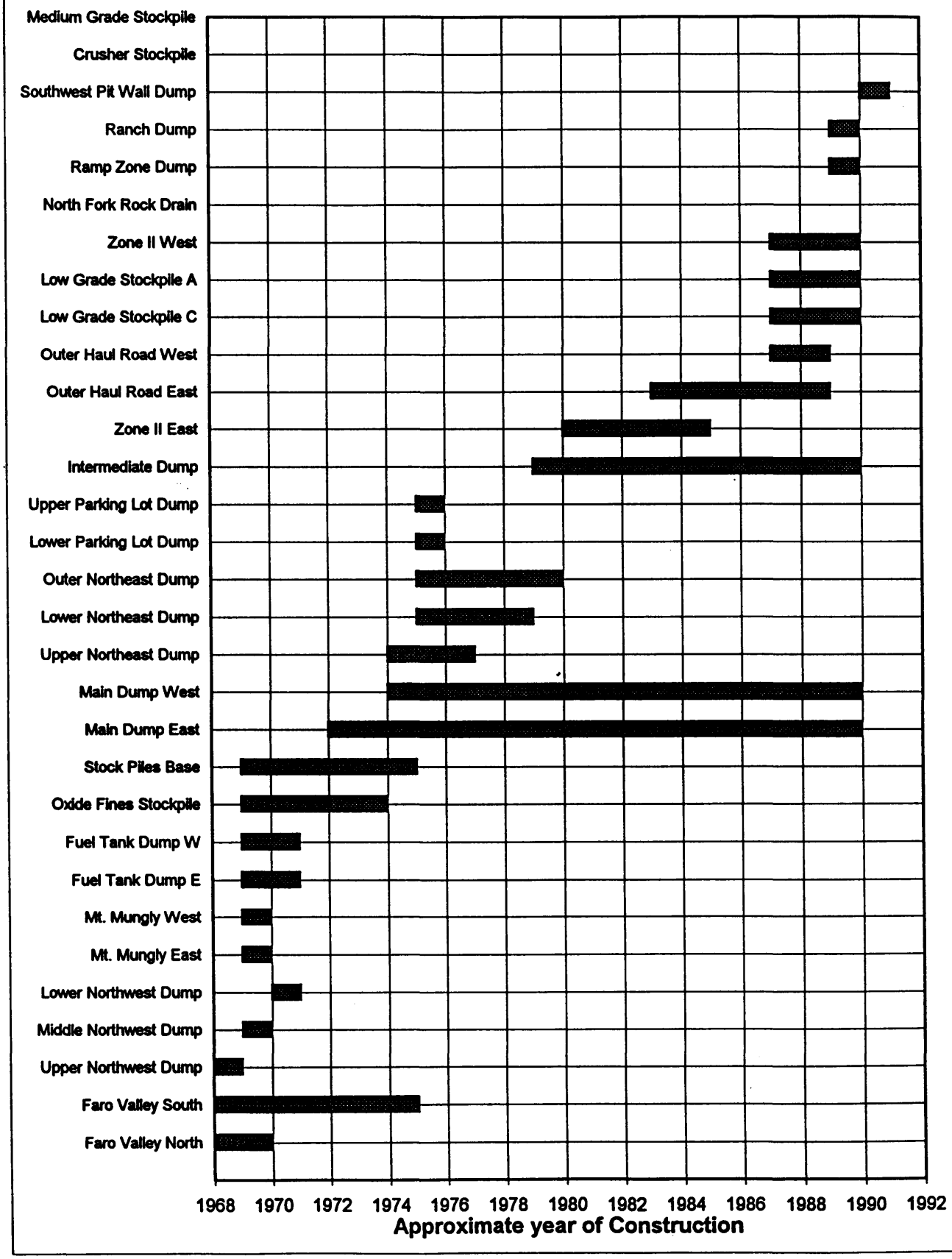


Figure 4-19
Dump Tonnage

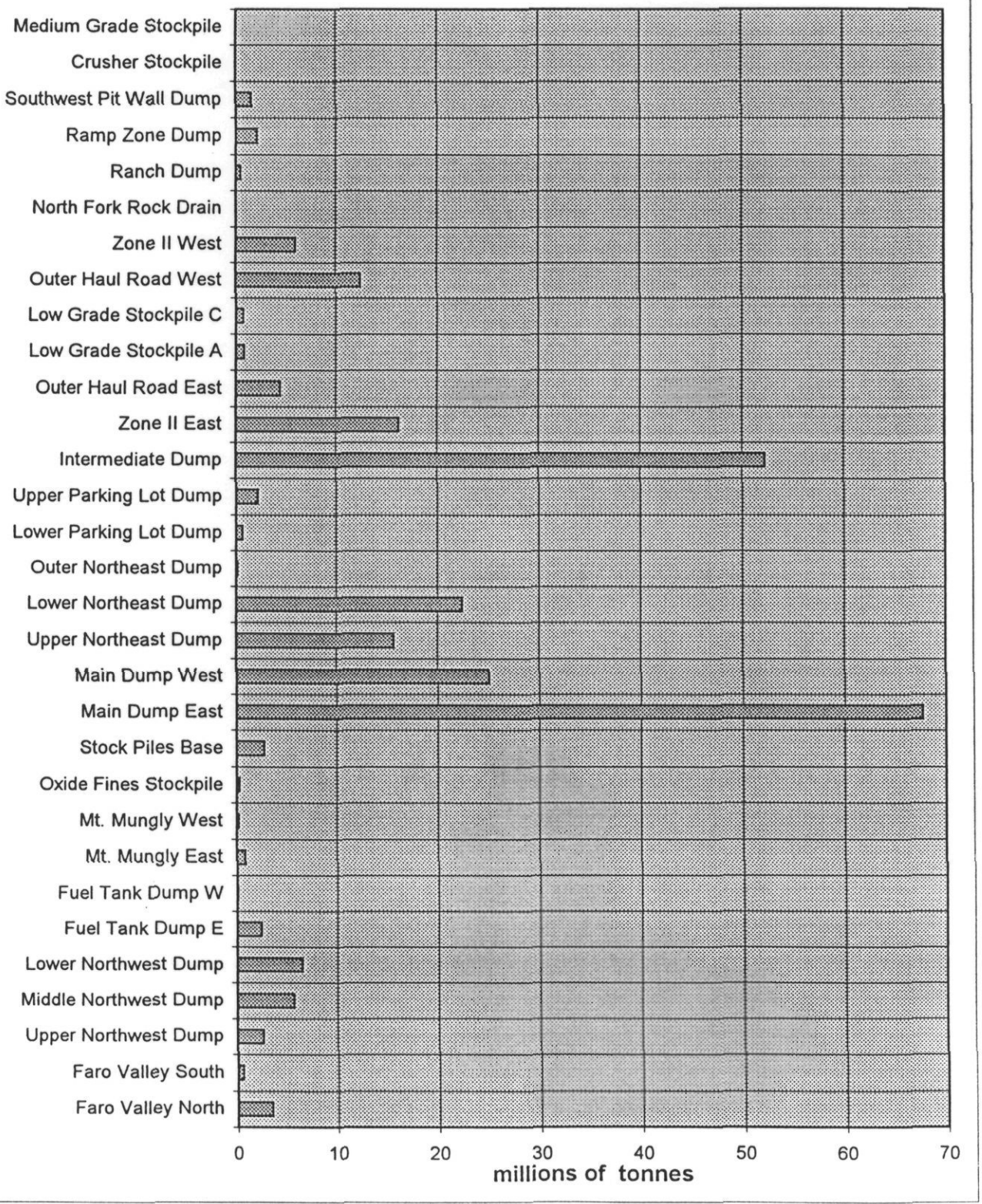


Figure 4-20
Stored Sulphate Load

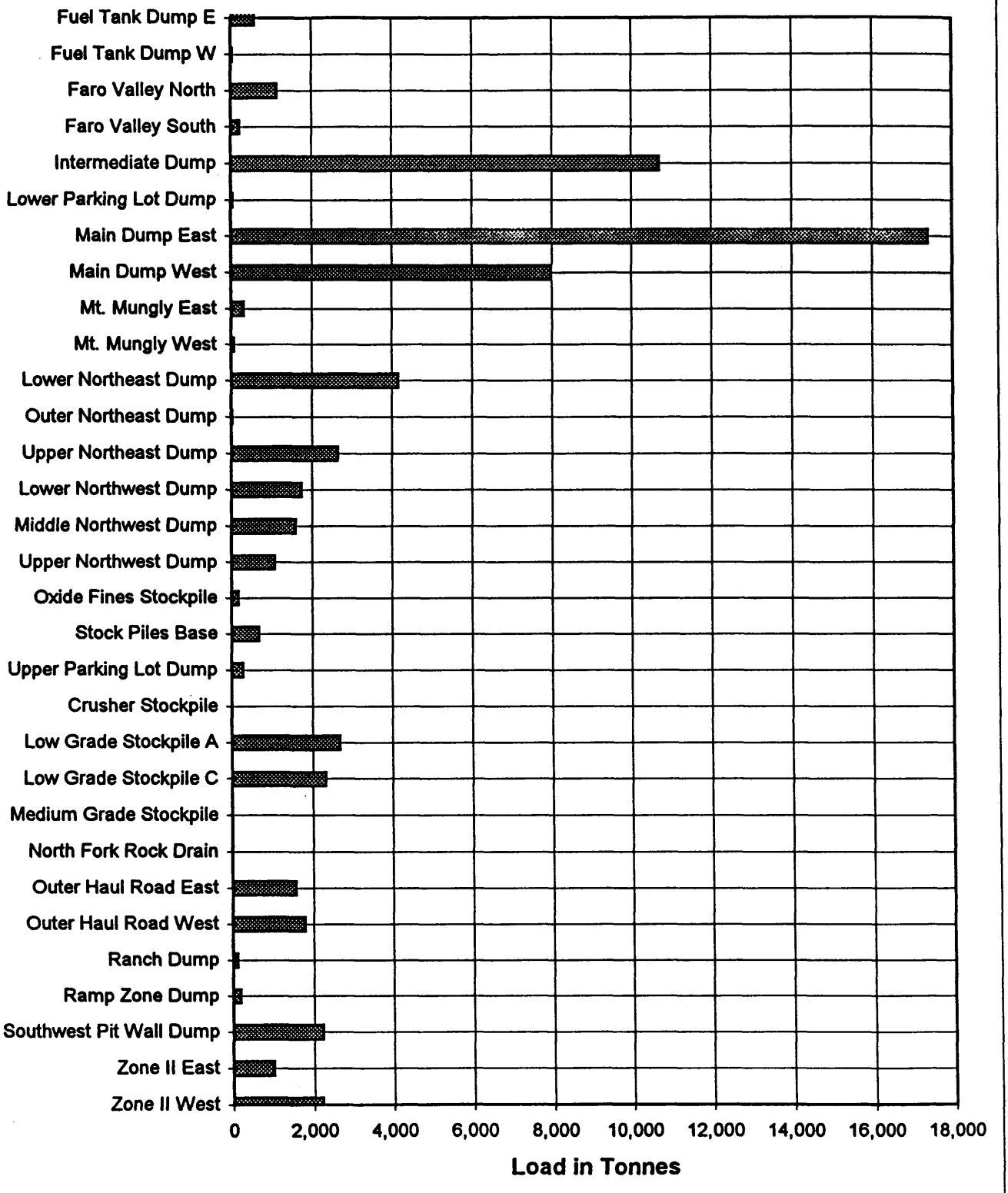


Figure 4-21
Stored Zinc Load

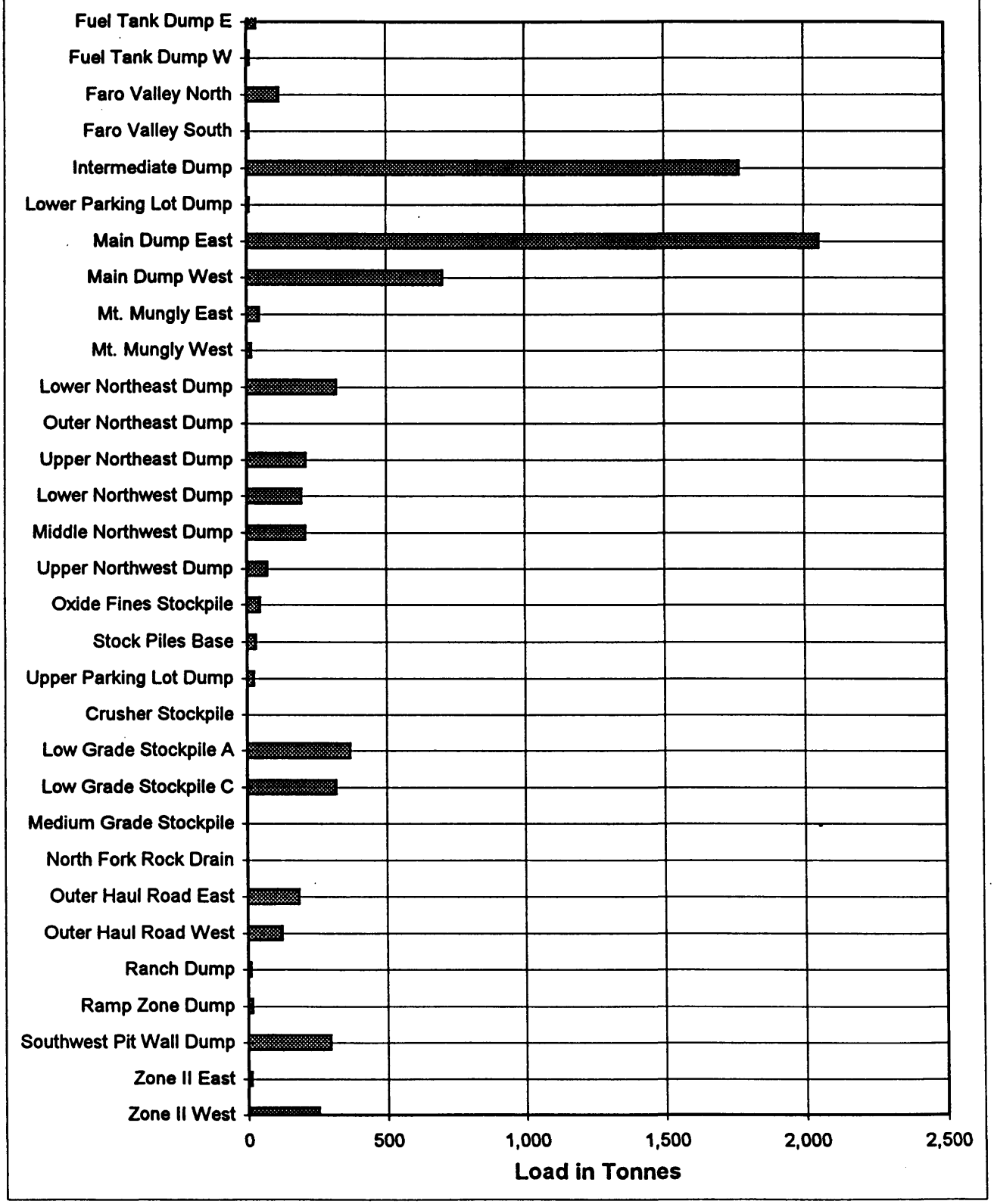
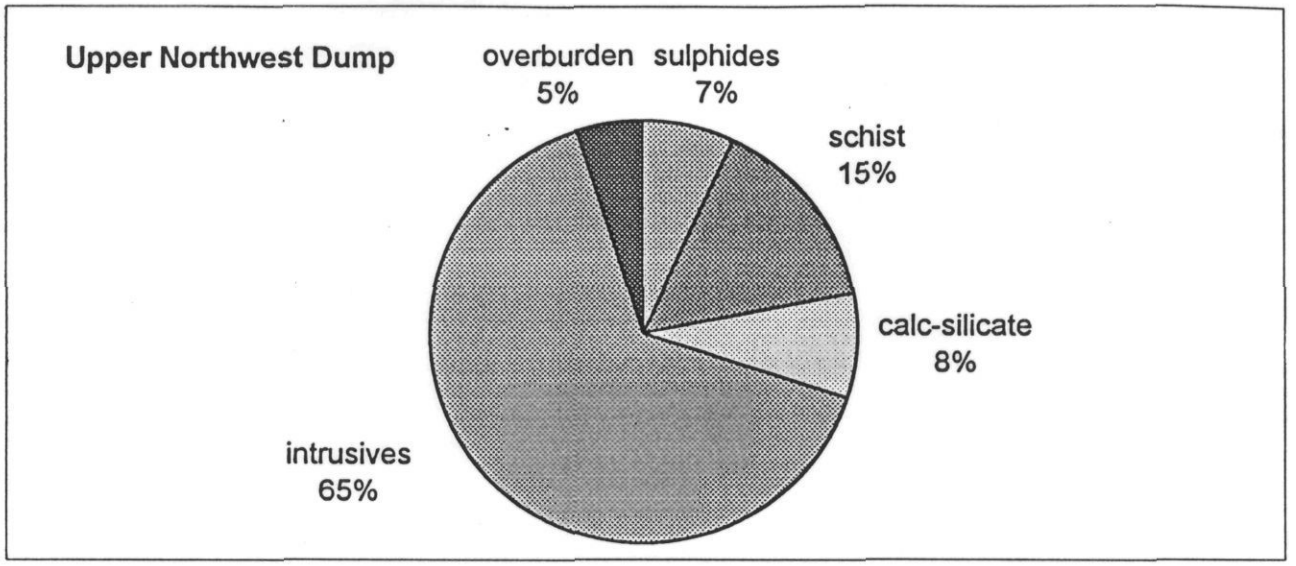
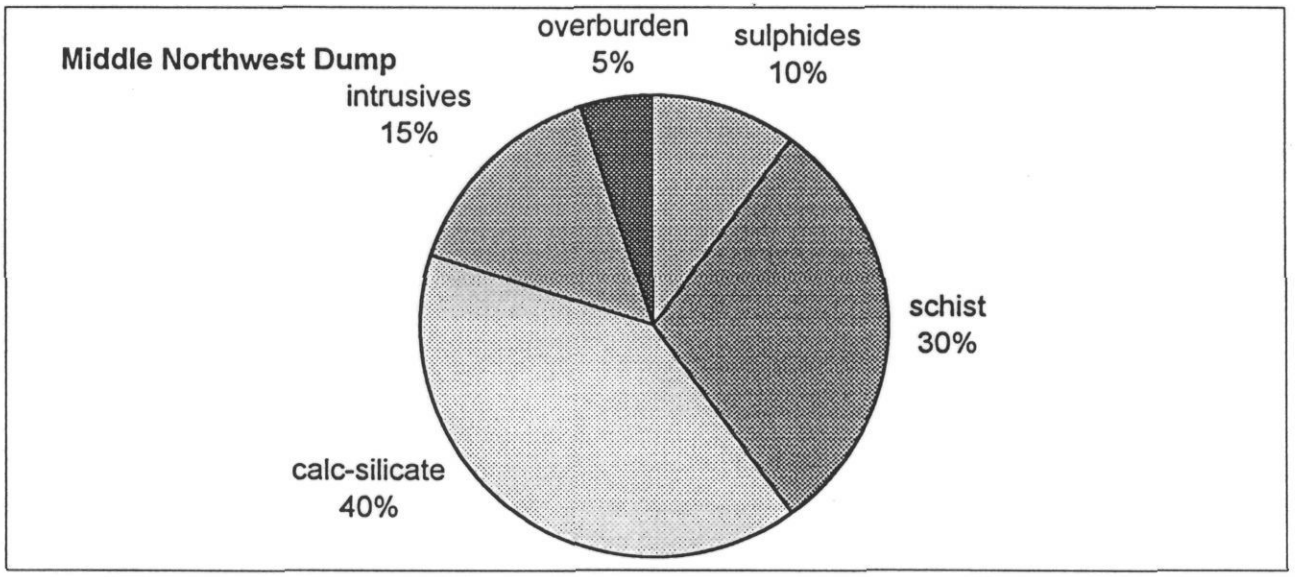


Figure 4-22 Northwest Dumps





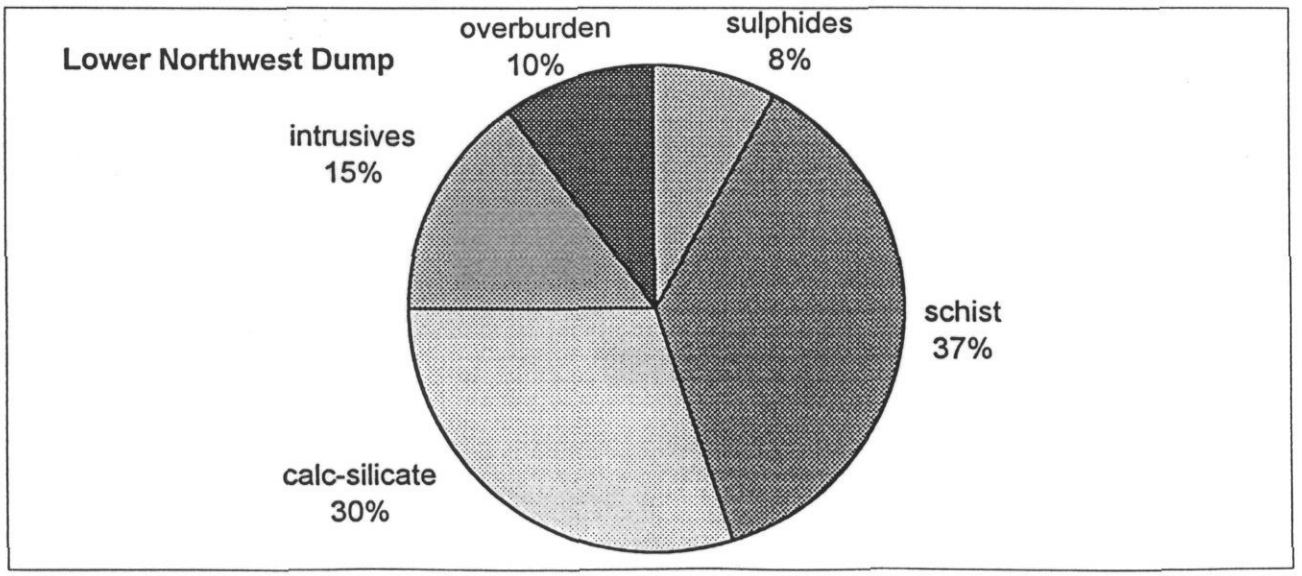
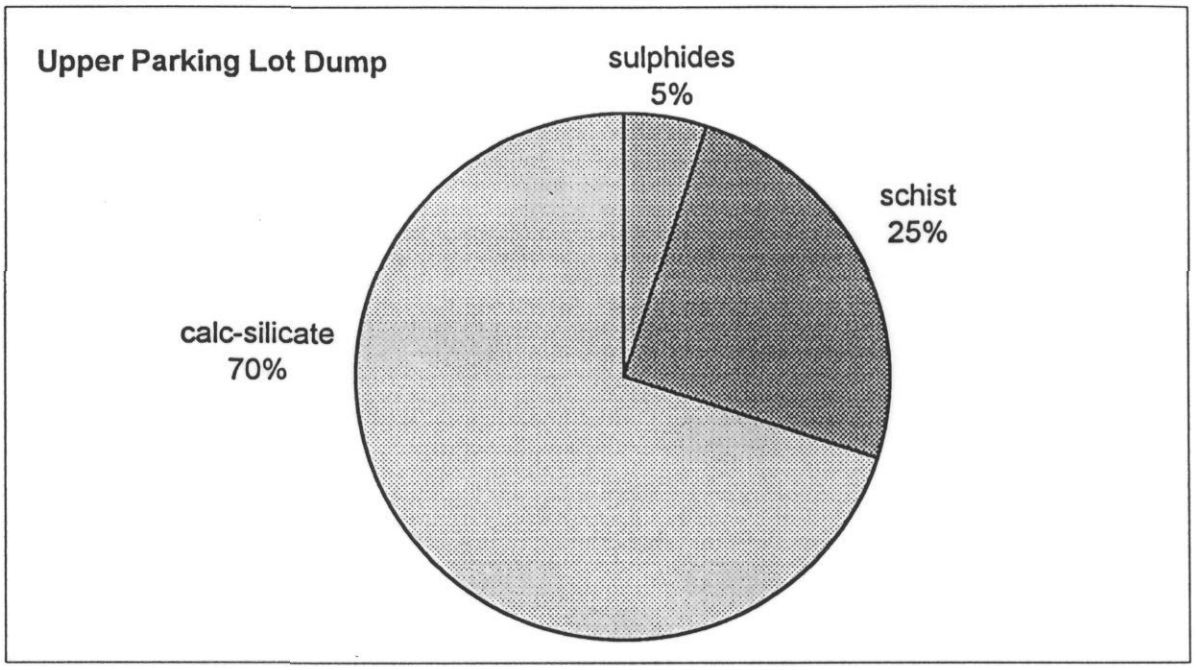


Figure 4-23
Parking Lot Dumps



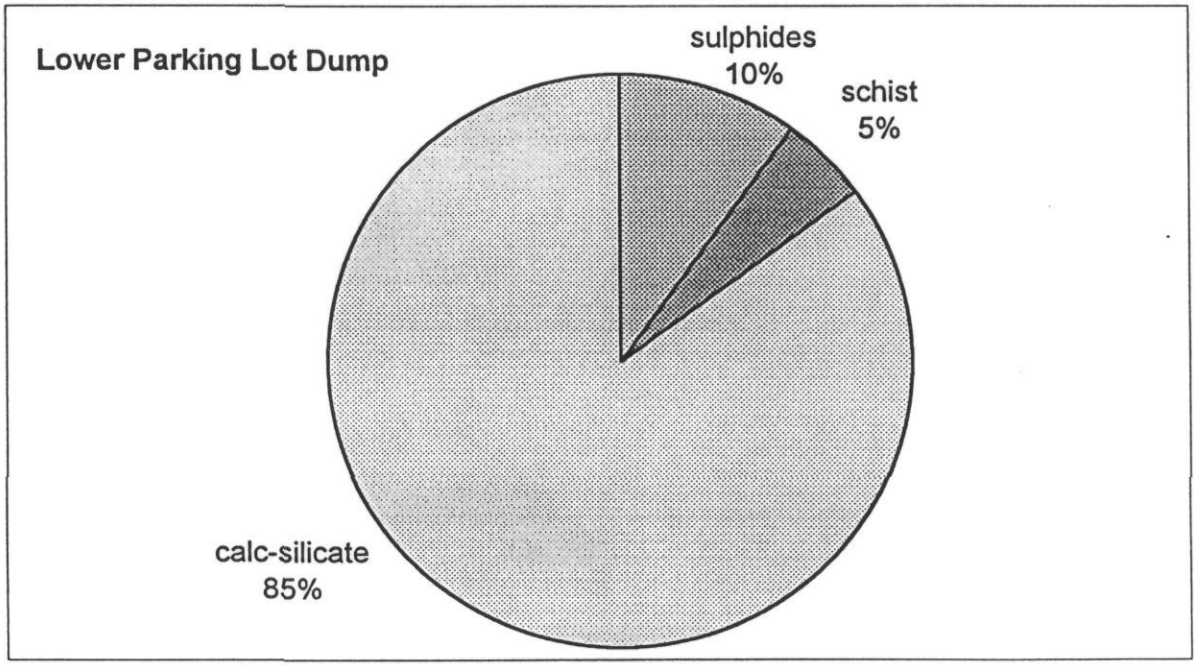
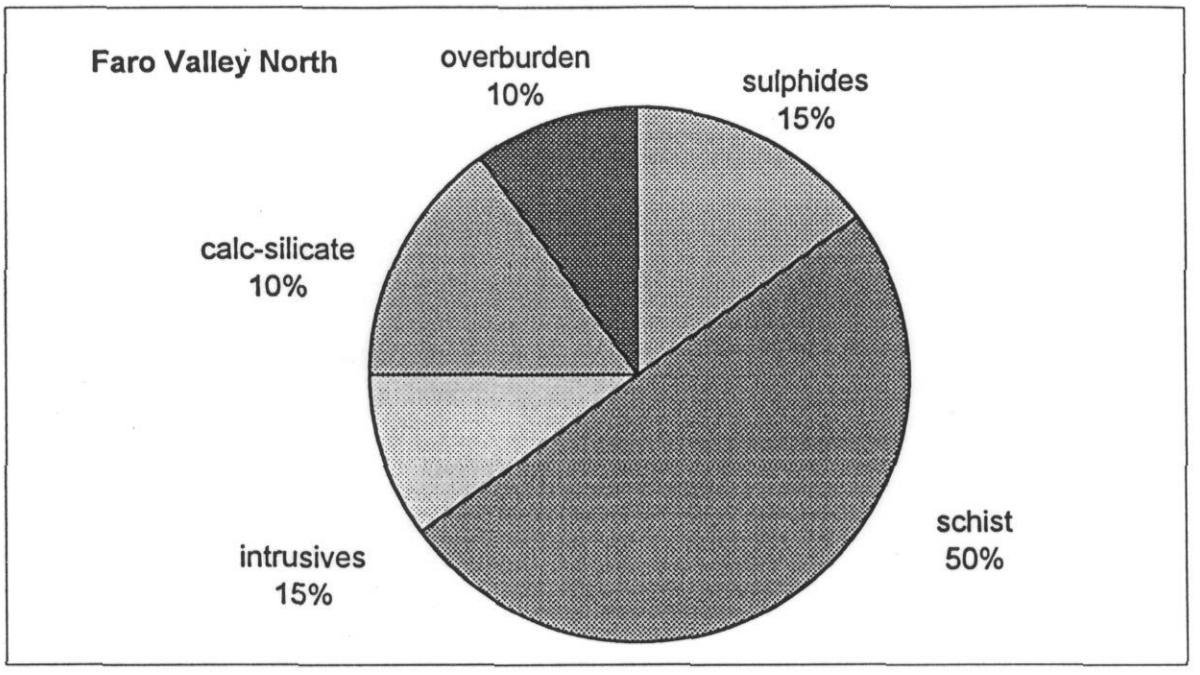


Figure 4-24
Faro Valley Dumps



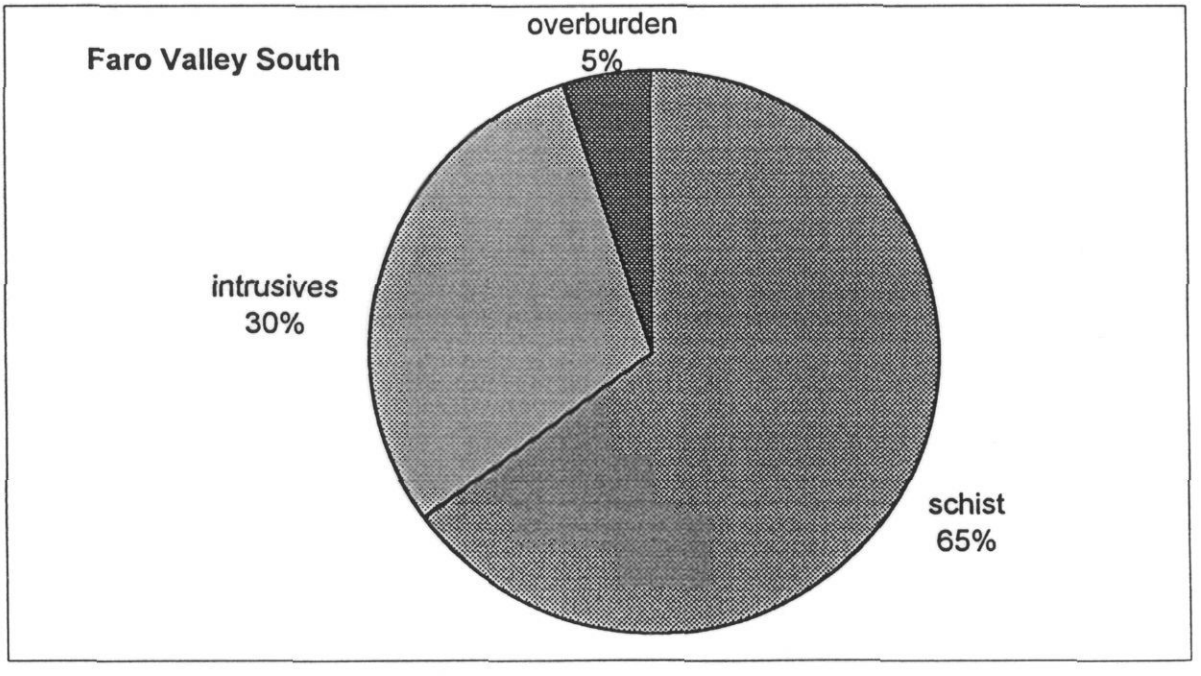
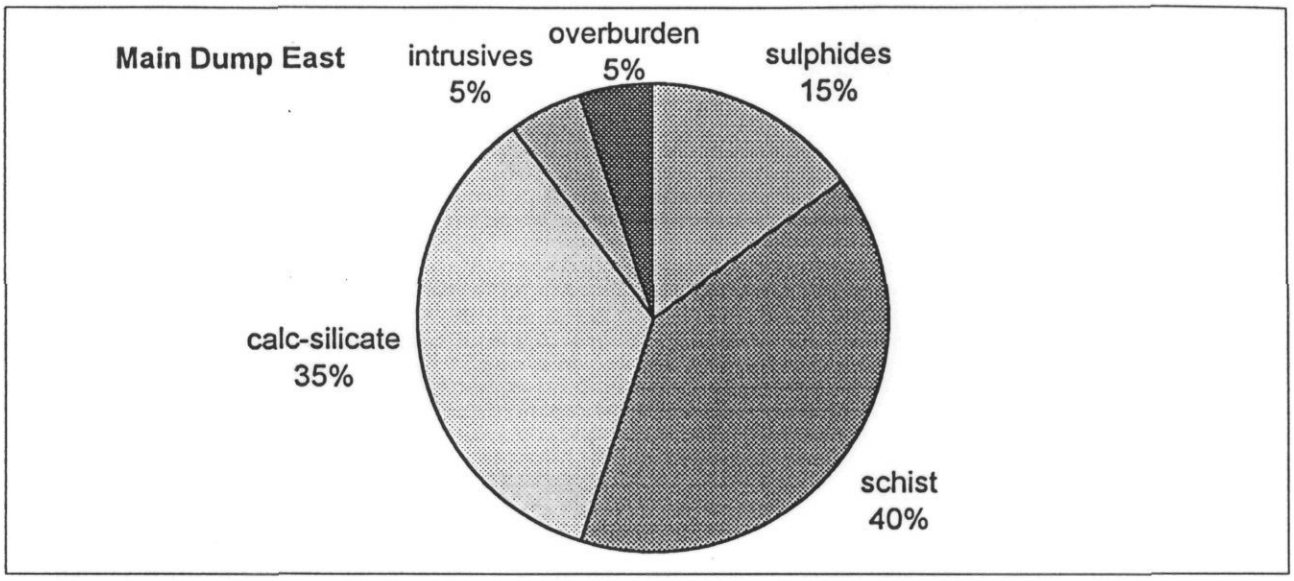
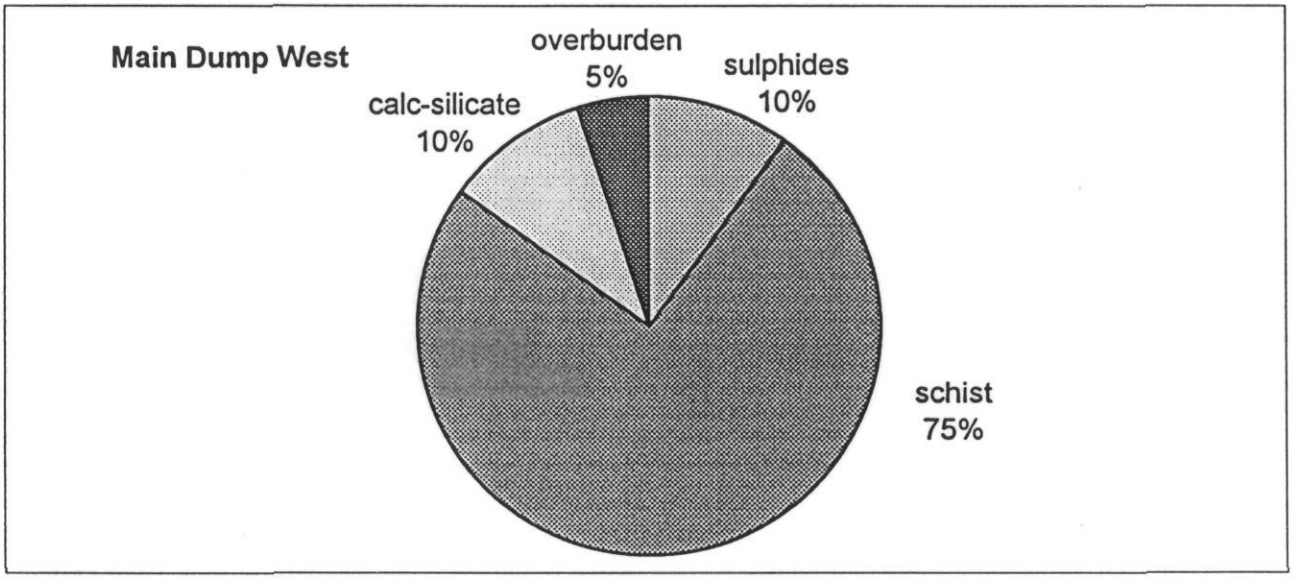


Figure 4-25
Main and Intermediate Dumps





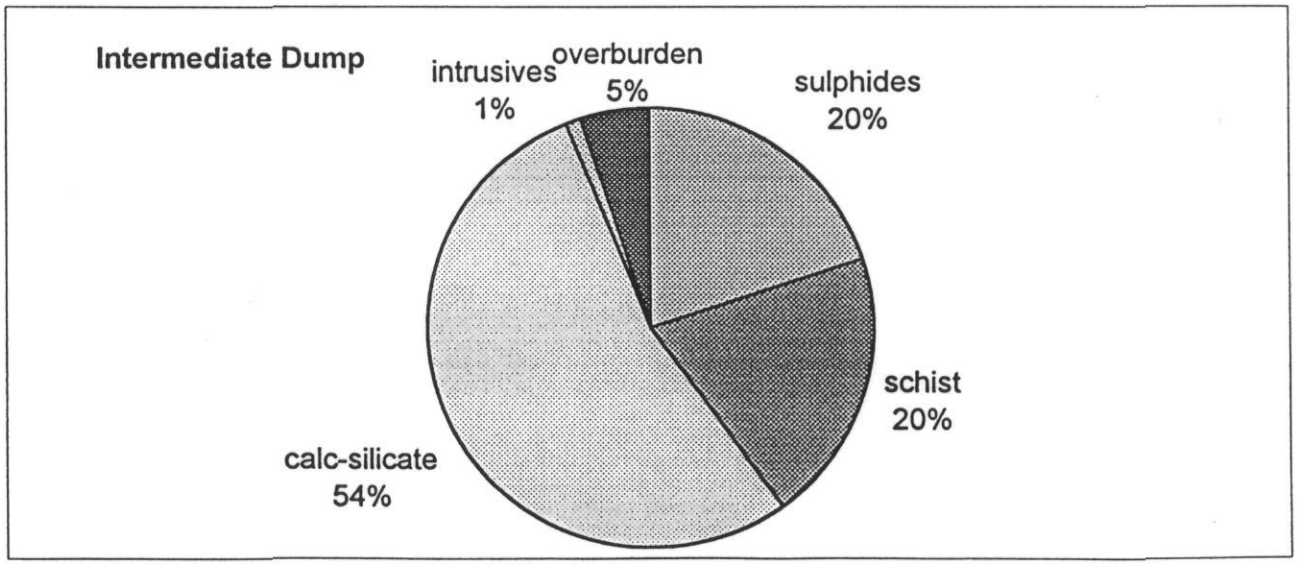
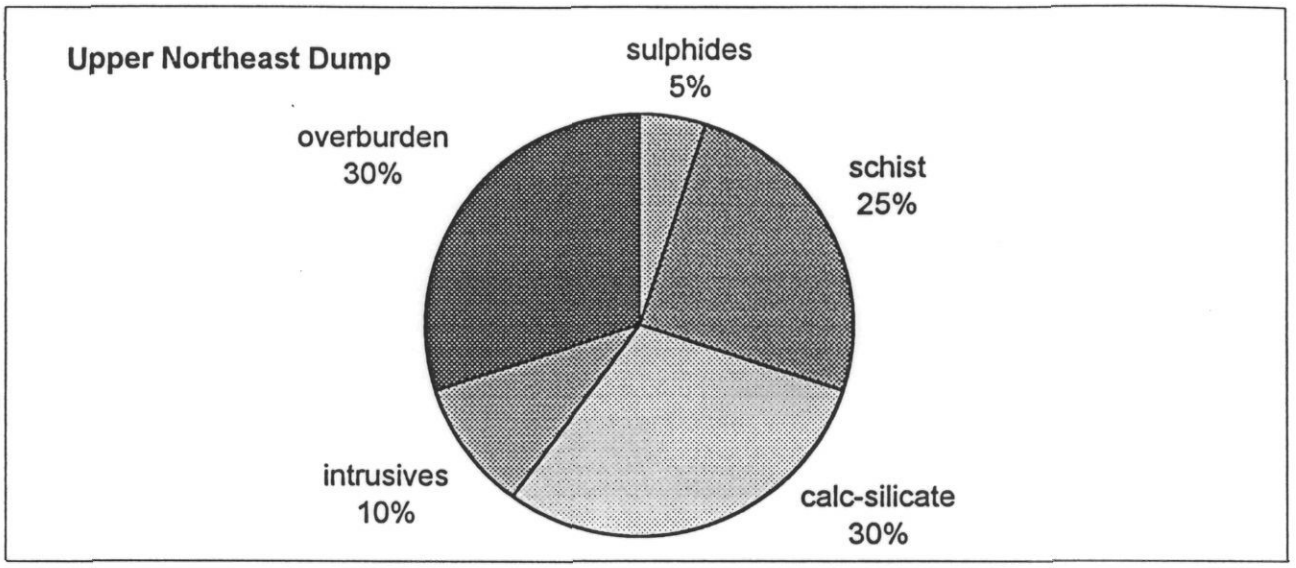
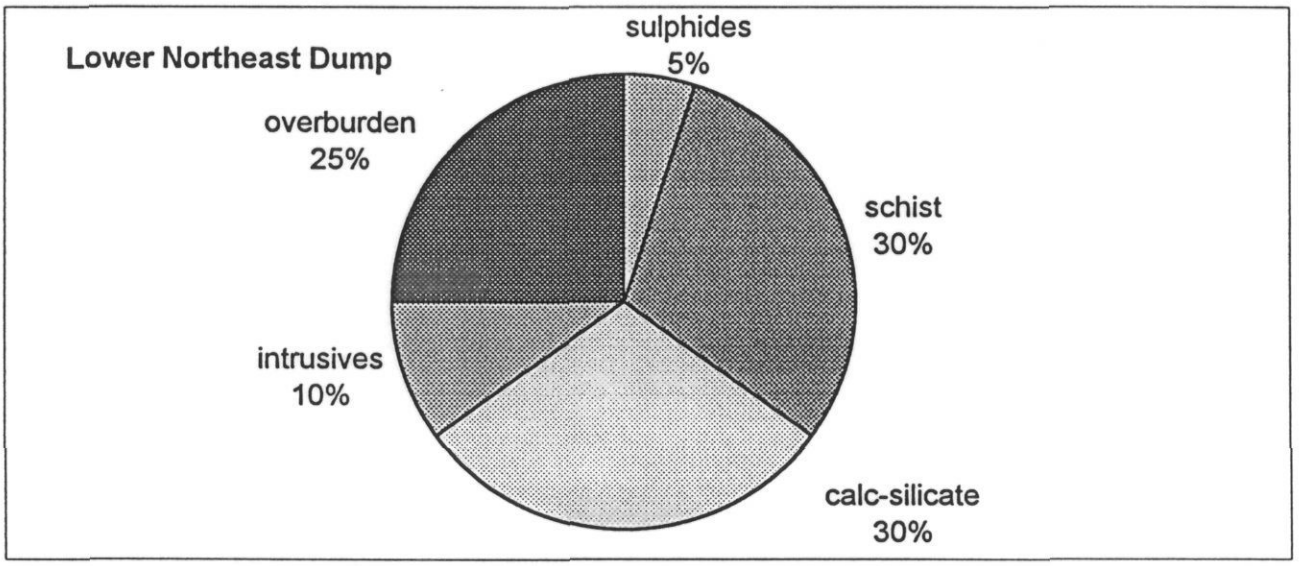


Figure 4-26
Northeast Dumps





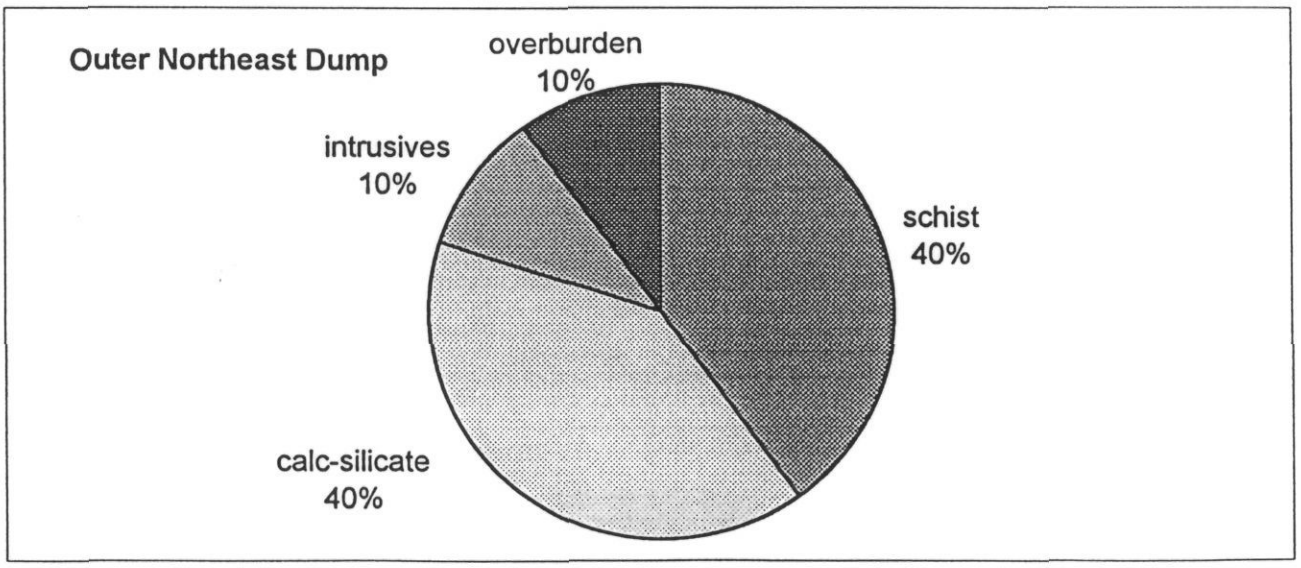
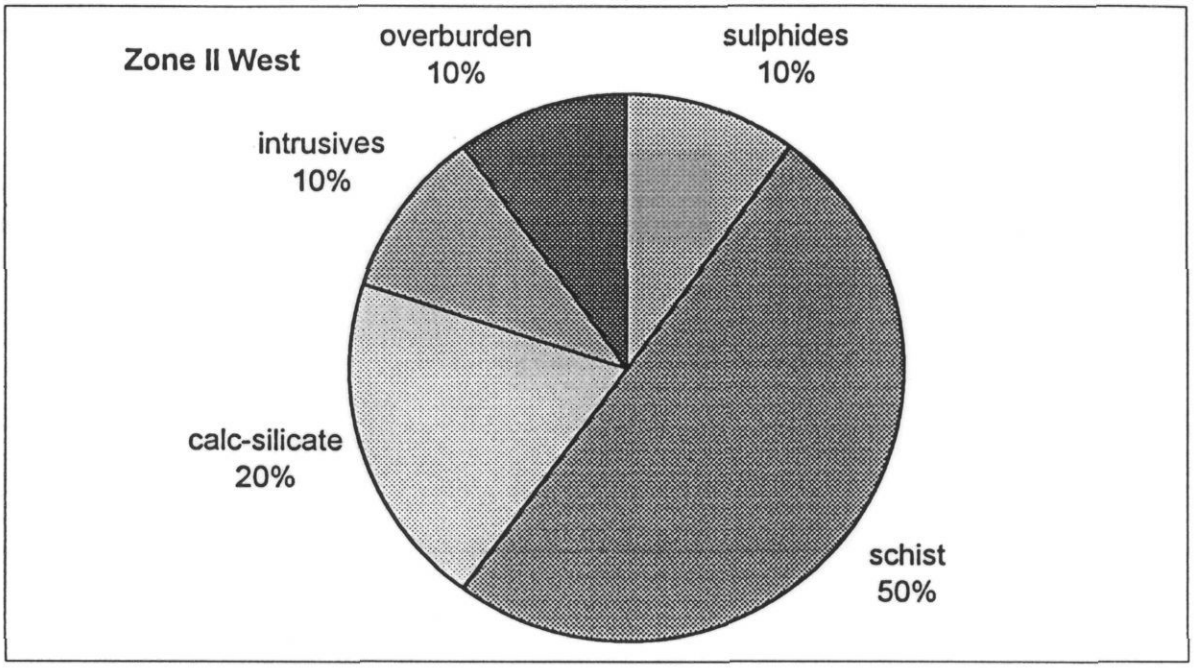


Figure 4-27
Zone II Dumps



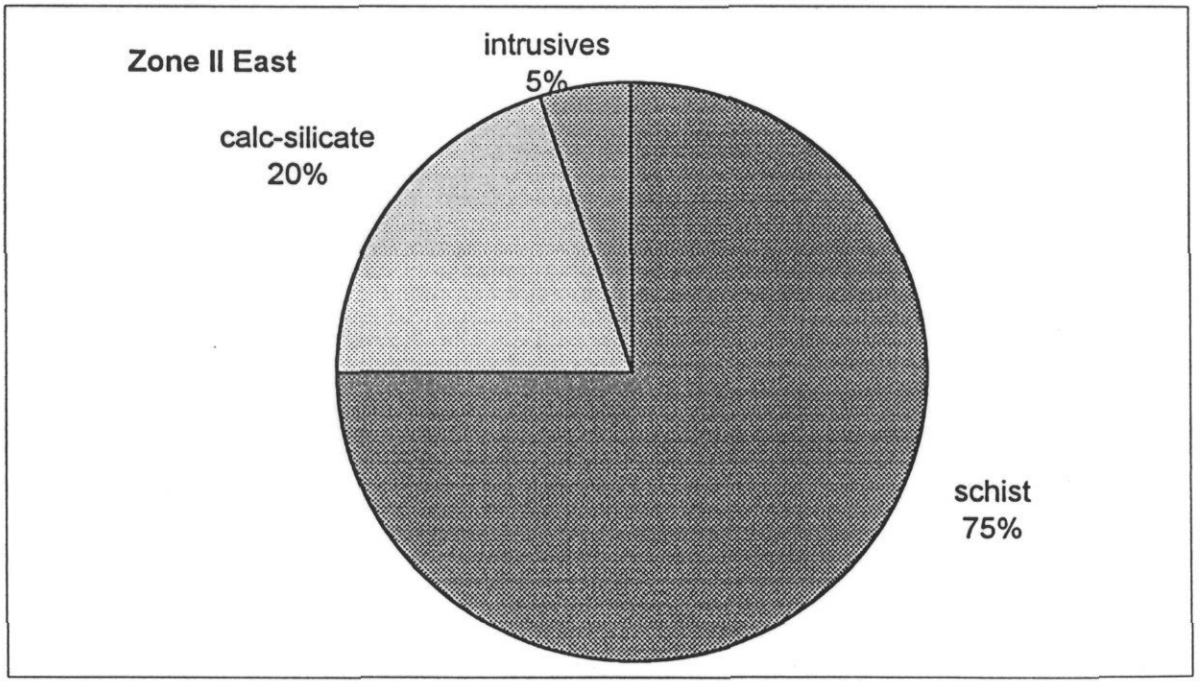


Figure 4-28
Near Pit Dumps

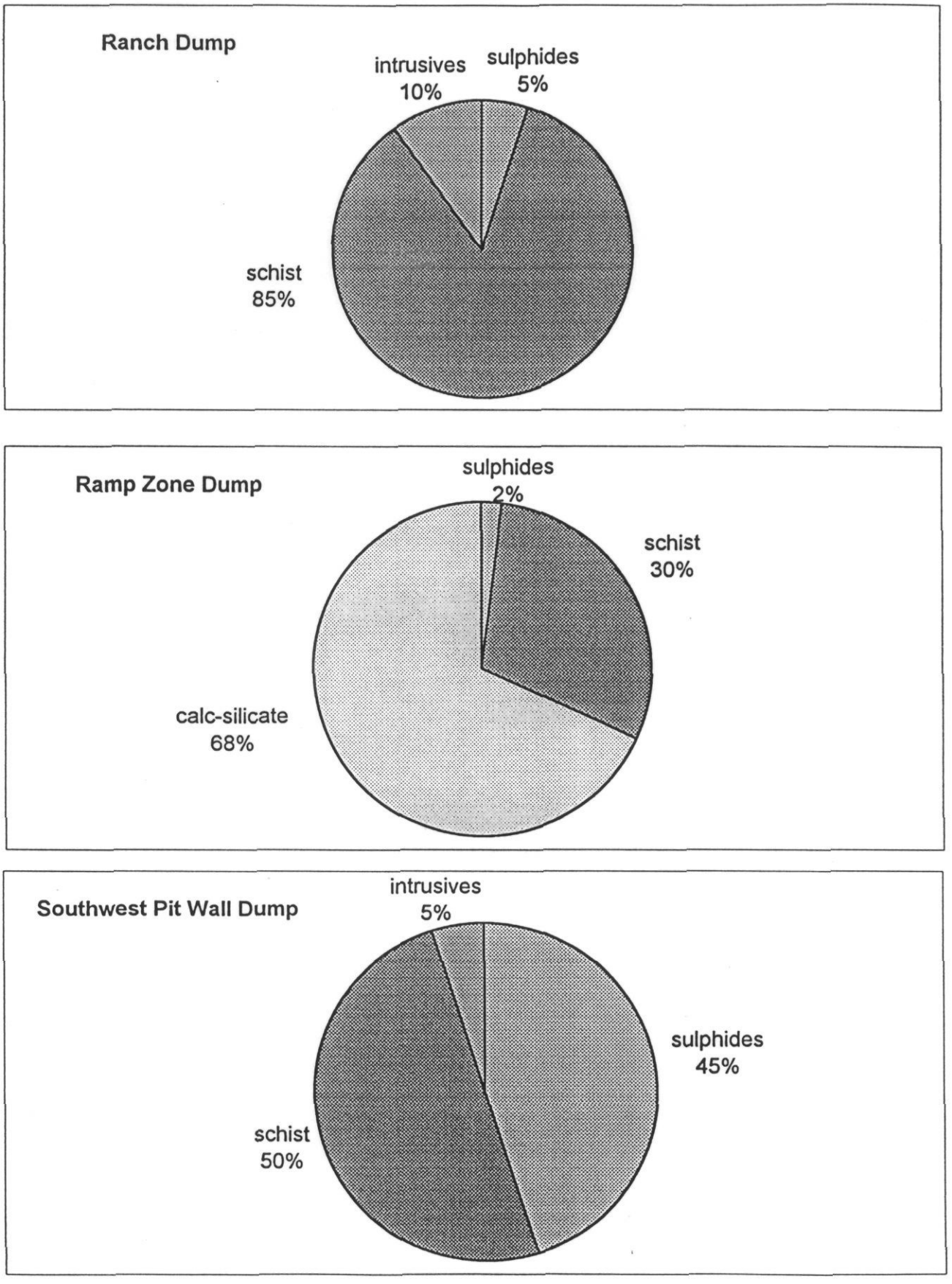
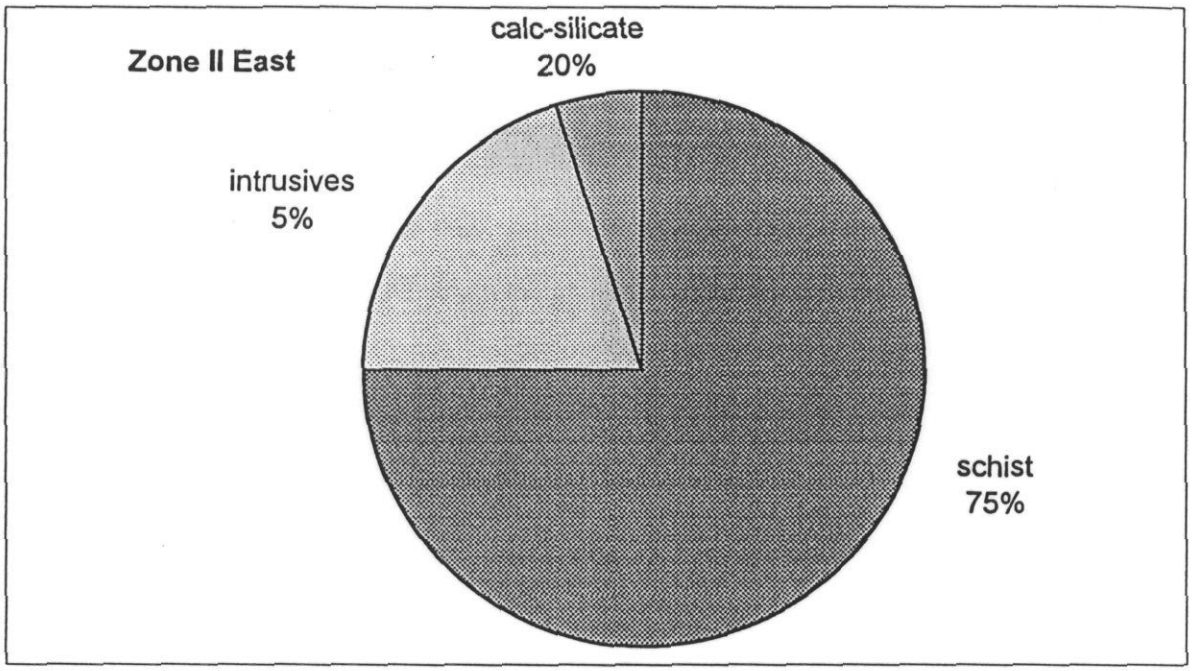


Figure 4-29
Outer Haul Road



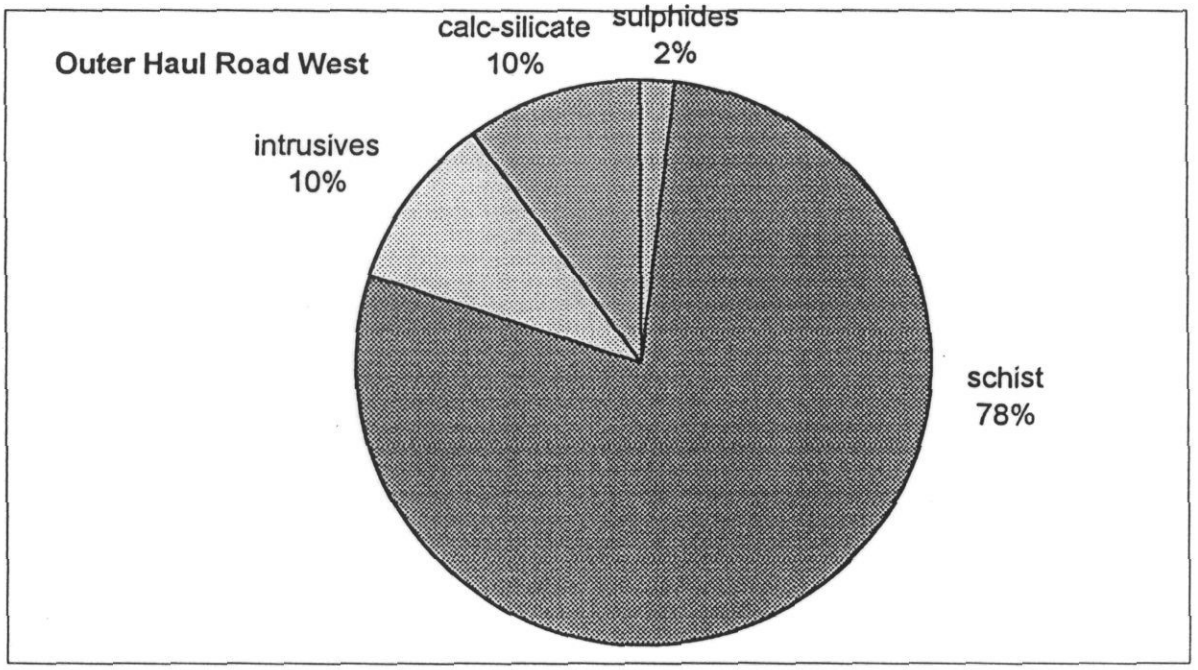


Figure 4-30 pH time trend, Faro Creek above diversion channel, diversion upstream, and diversion adjacent to NE waste dumps

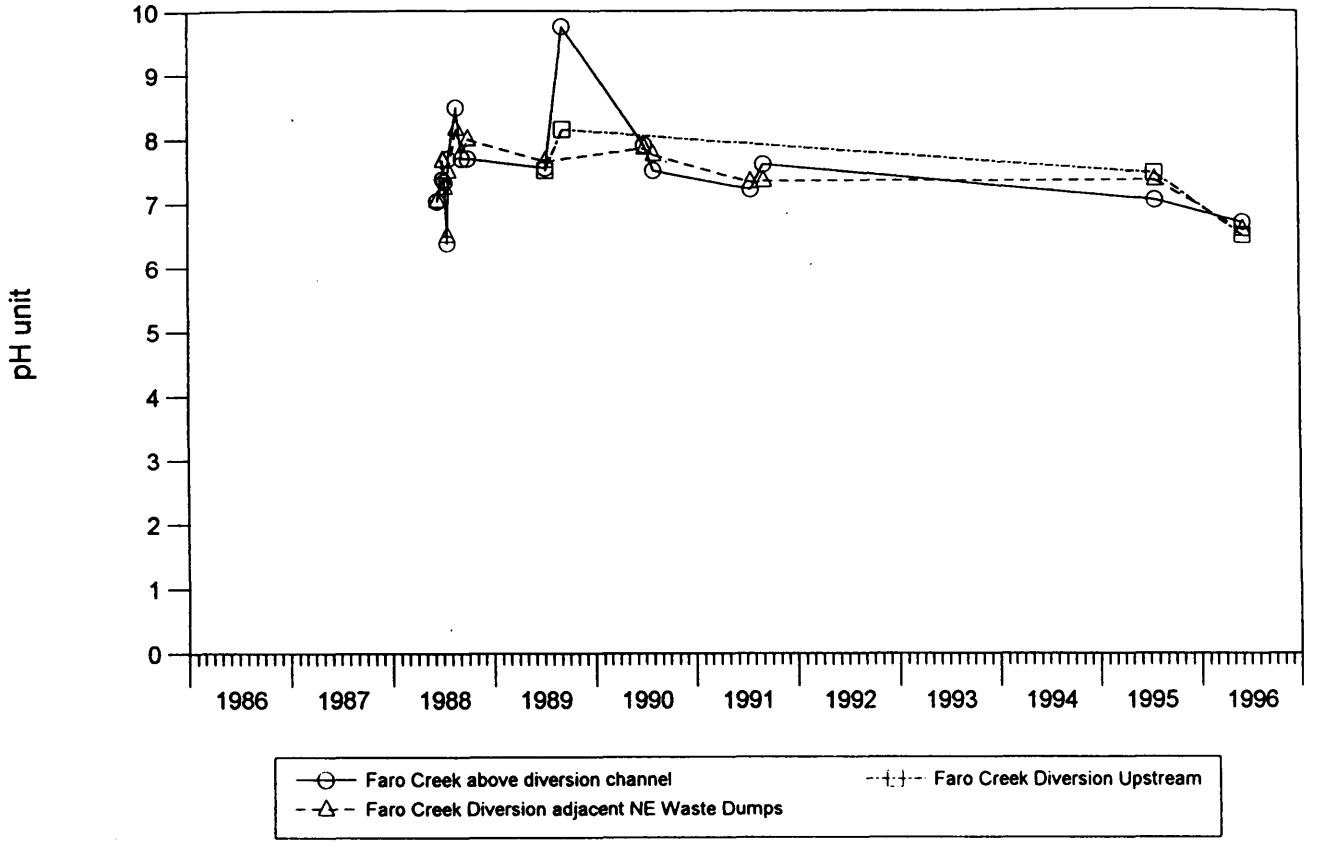


Figure 4-31 Total sulphate time trend, Faro Creek above diversion channel, diversion upstream, and diversion adjacent to NE waste dumps

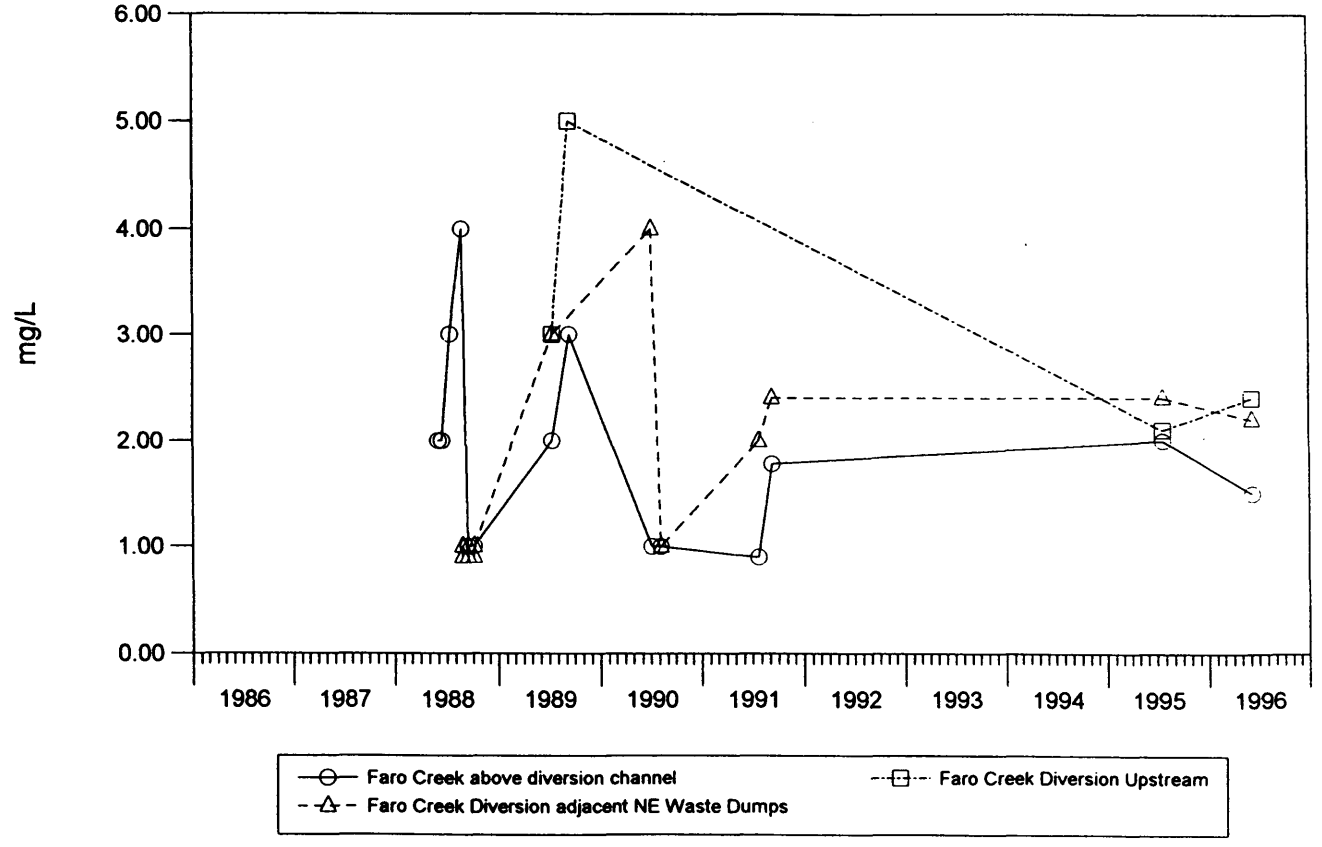


Figure 4-32 Total zinc time trend, Faro Creek above diversion channel, diversion upstream, and diversion adjacent to NE waste dumps

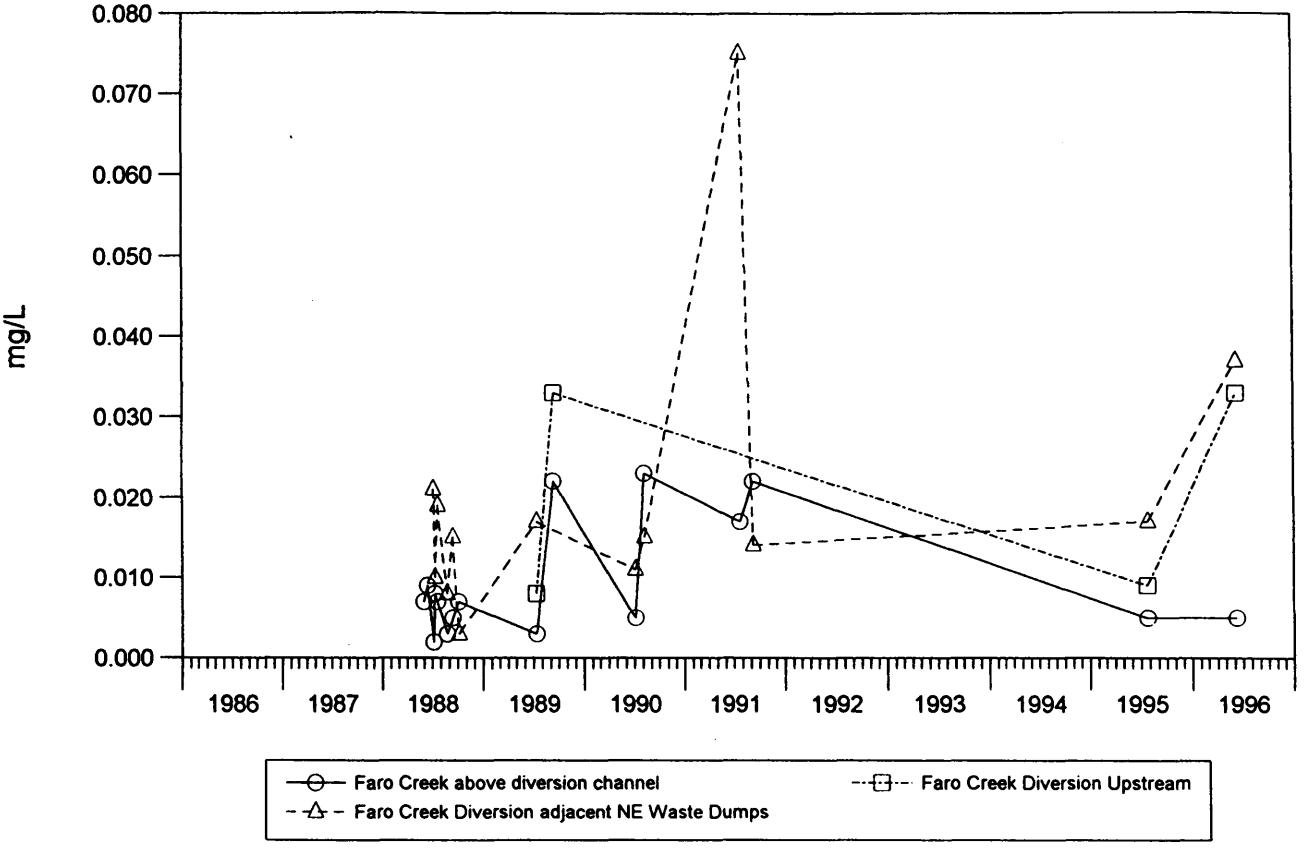


Figure 4-33 Dissolved zinc time trend, Faro Creek above diversion channel, diversion upstream, and diversion adjacent to NE waste dumps

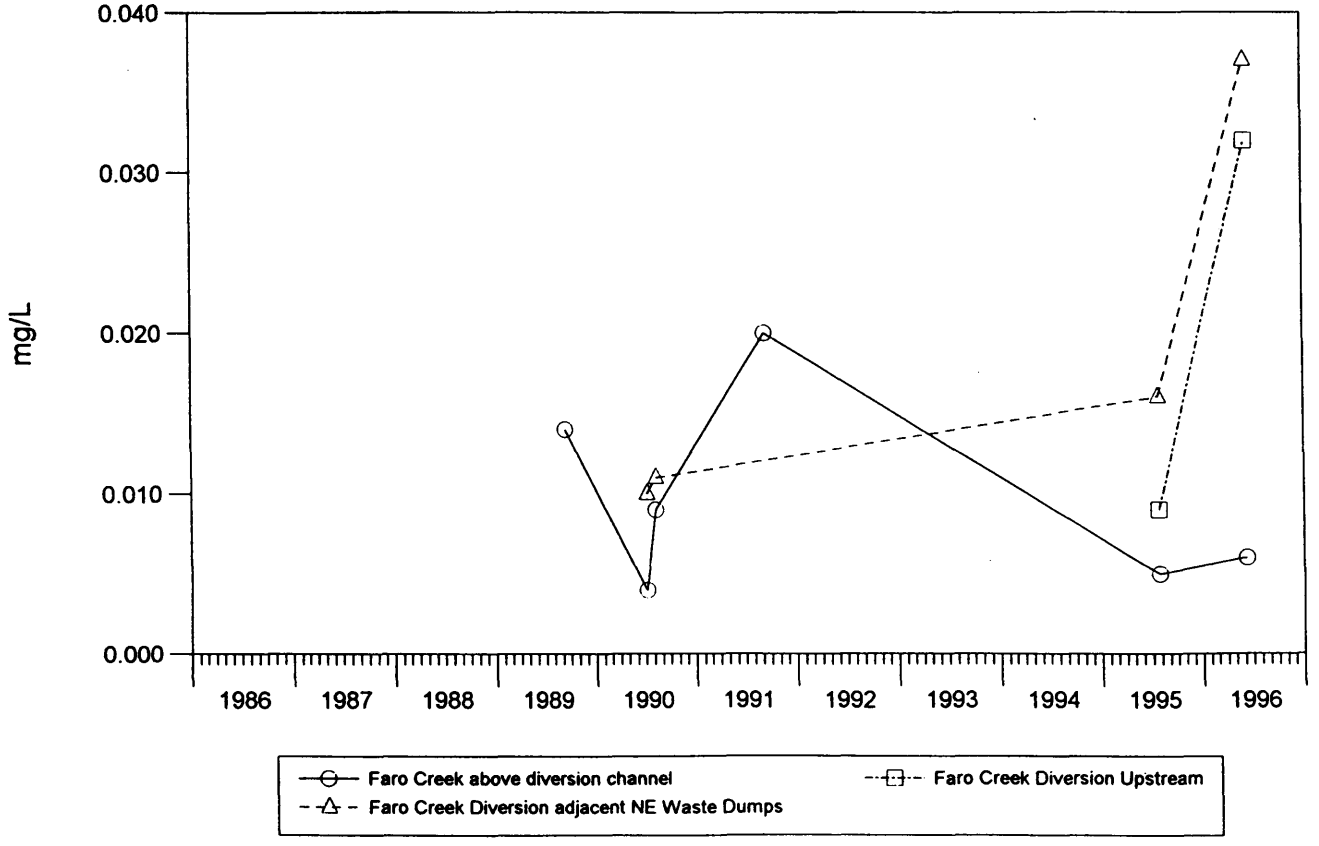


Figure 4-34
pH time trend, Upper pit Wall Zone MPA6 Sump and Old Faro Creek Site 29/30

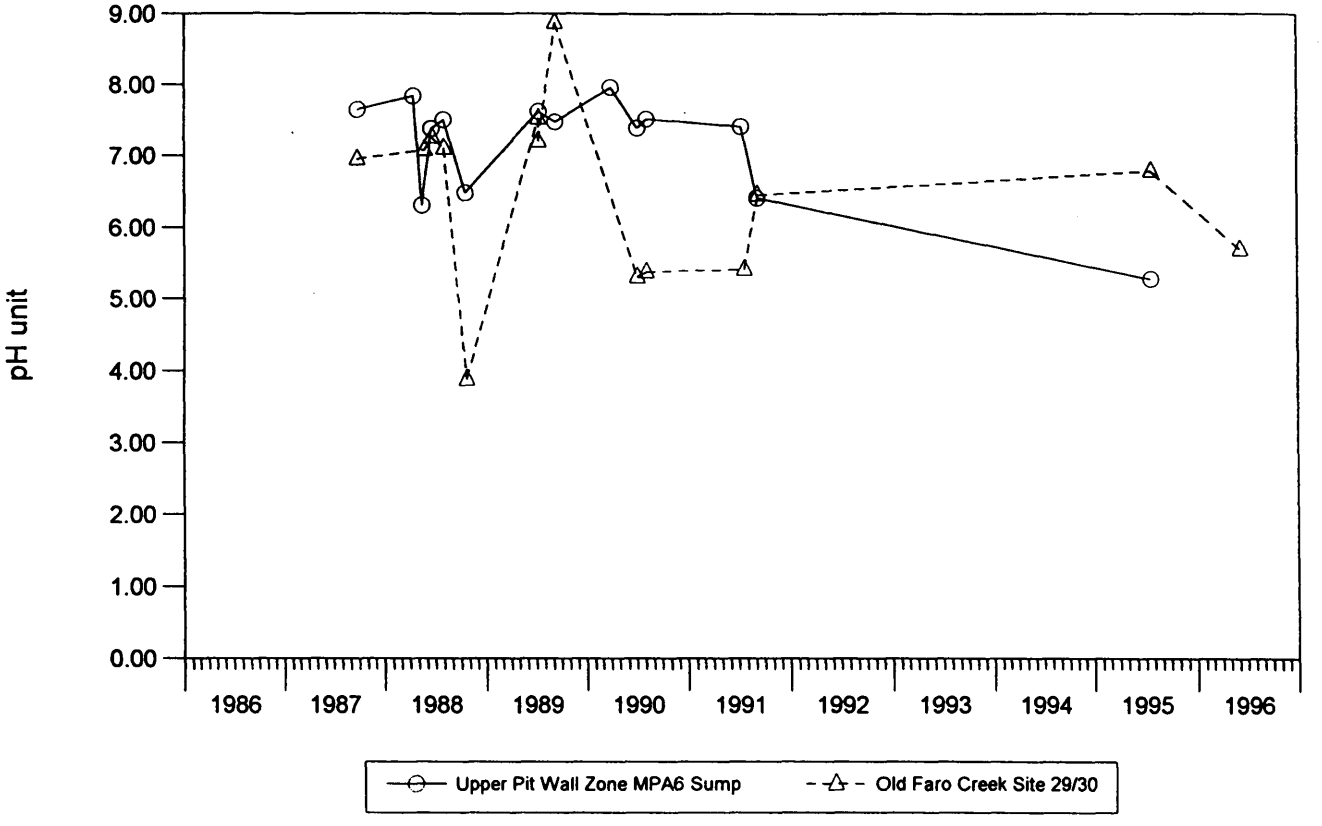


Figure 4-35
Alkalinity time trend, Upper pit Wall Zone MPA6 Sump and Old Faro Creek Site 29/30

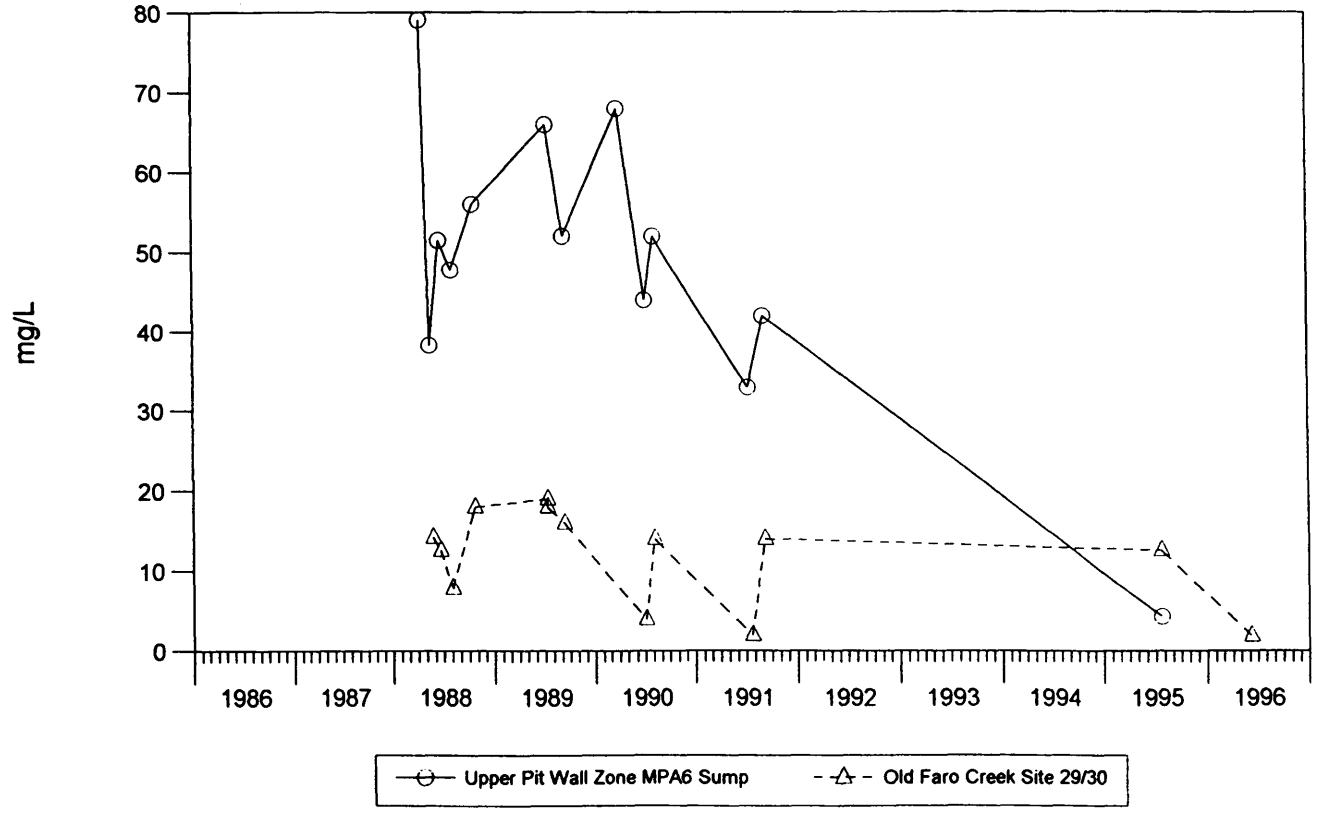


Figure 4-36
Total sulphate time trend, Upper pit Wall Zone MPA6 Sump and Old Faro Creek Site 29/30

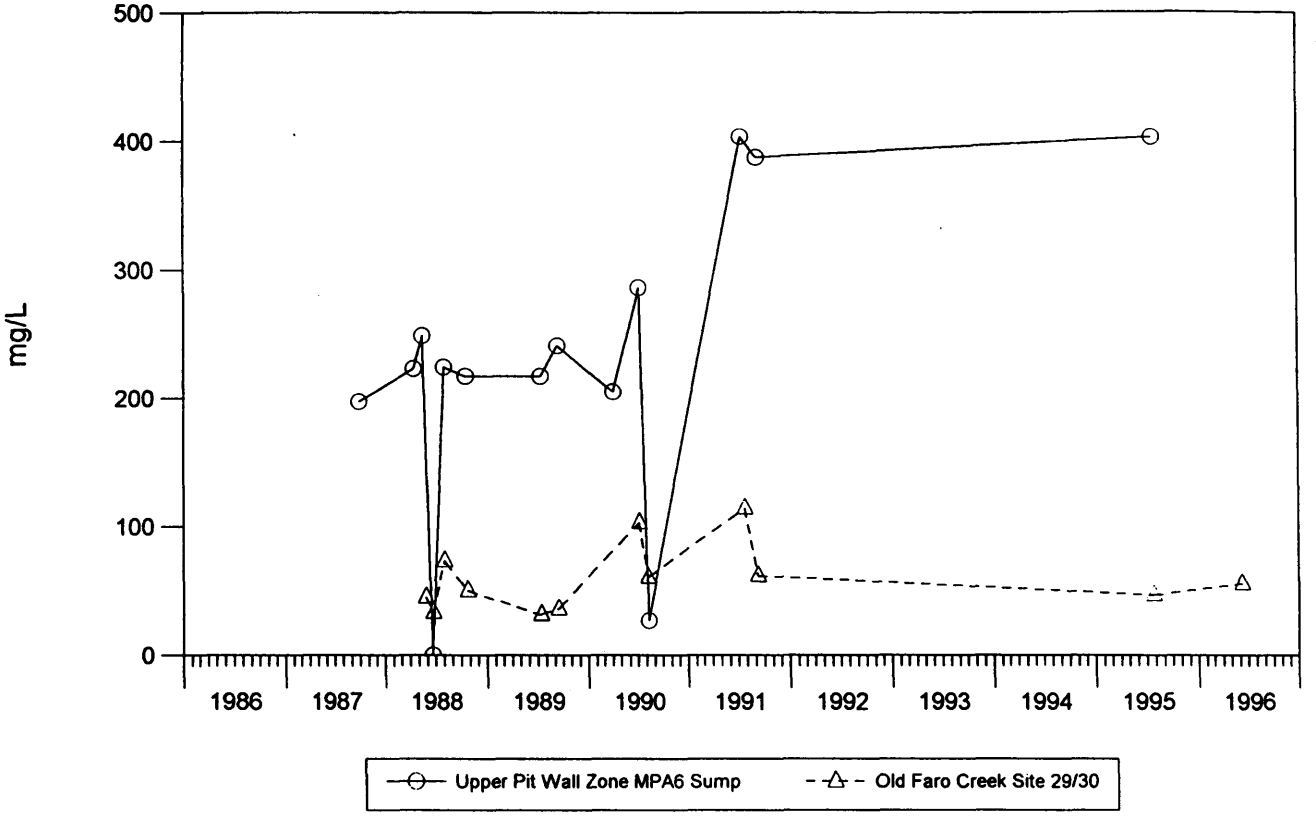


Figure 4-37
Total zinc time trend, Upper pit Wall Zone MPA6 Sump and Old Faro Creek Site 29/30

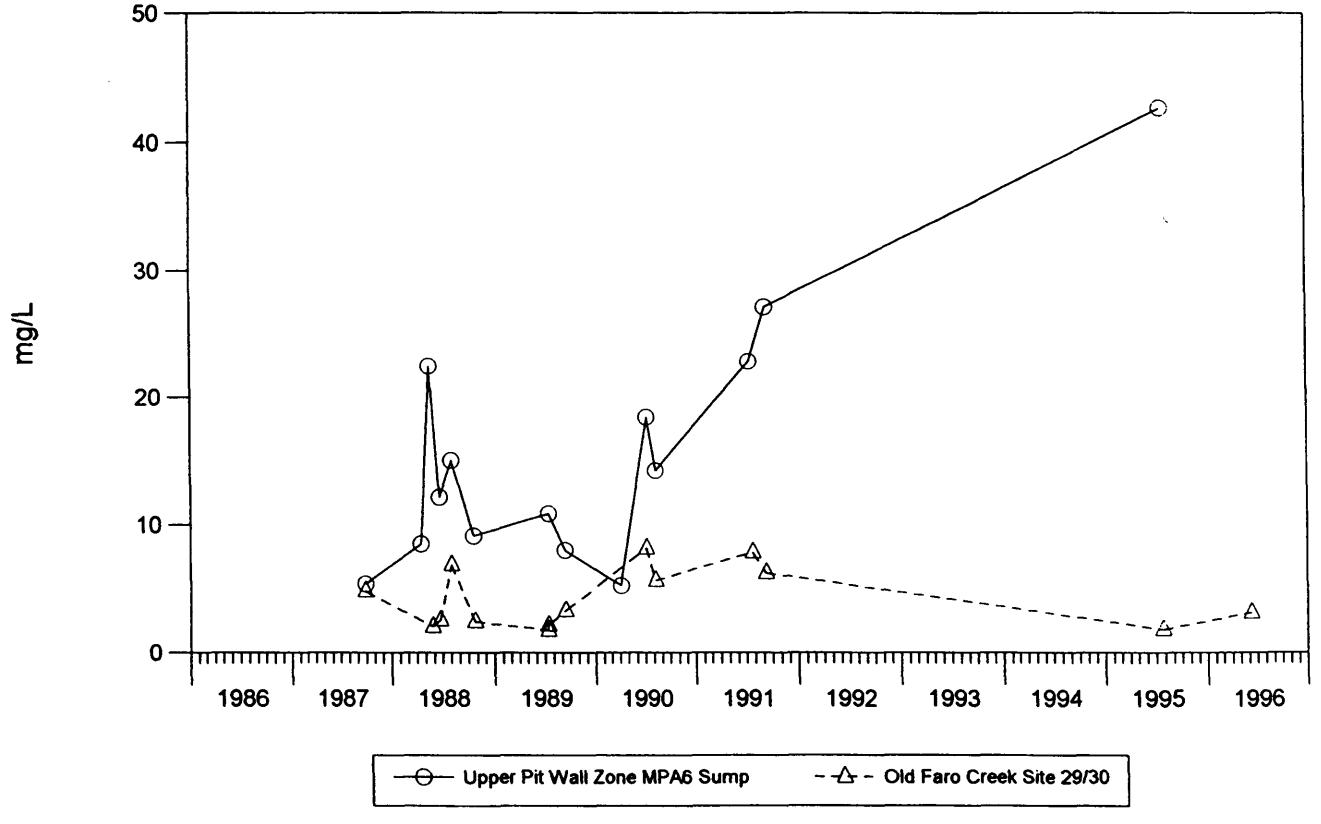


Figure 4-38
Dissolved zinc time trend, Upper pit Wall Zone MPA6 Sump and Old Faro Creek Site 29/30

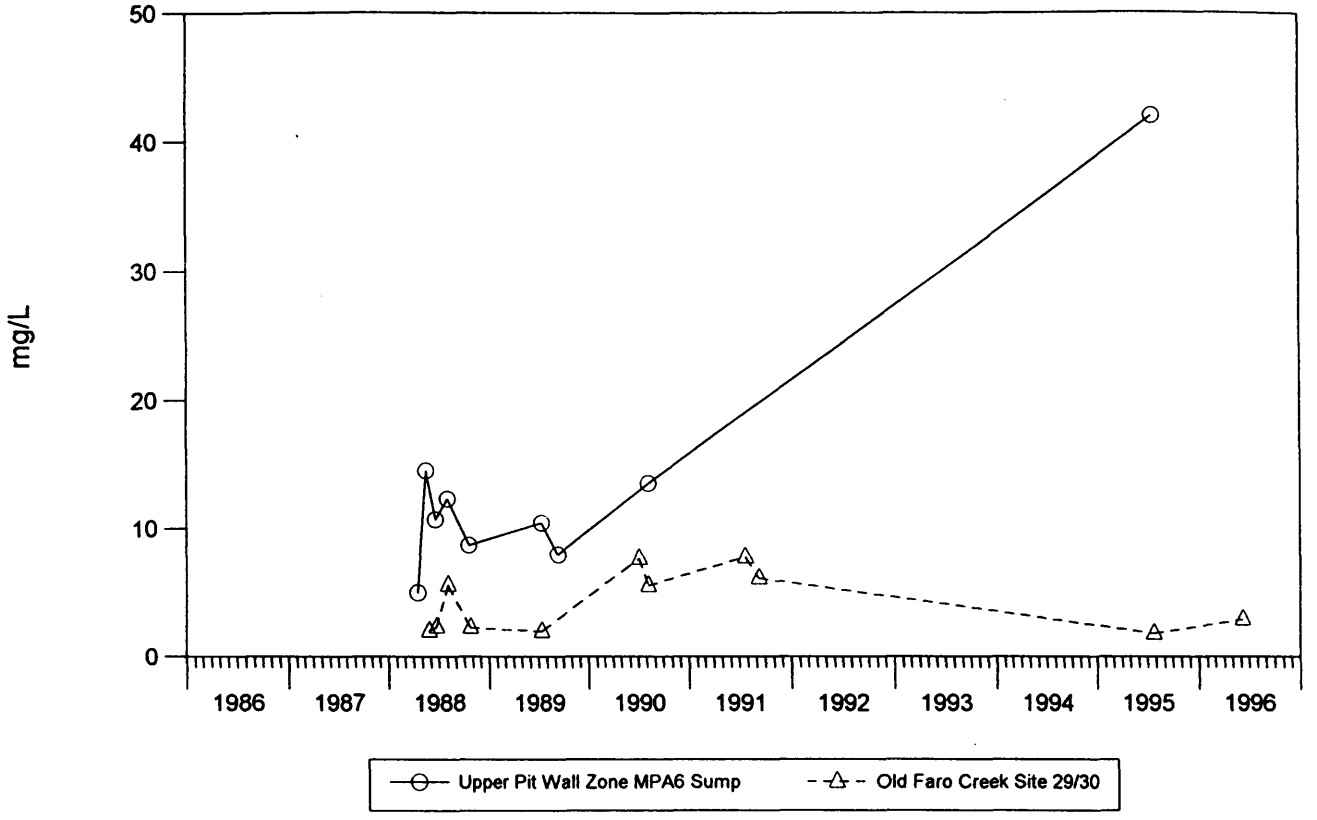


Figure 4-39
pH time trend, Upper Guardhouse Creek downstream of NW Dump

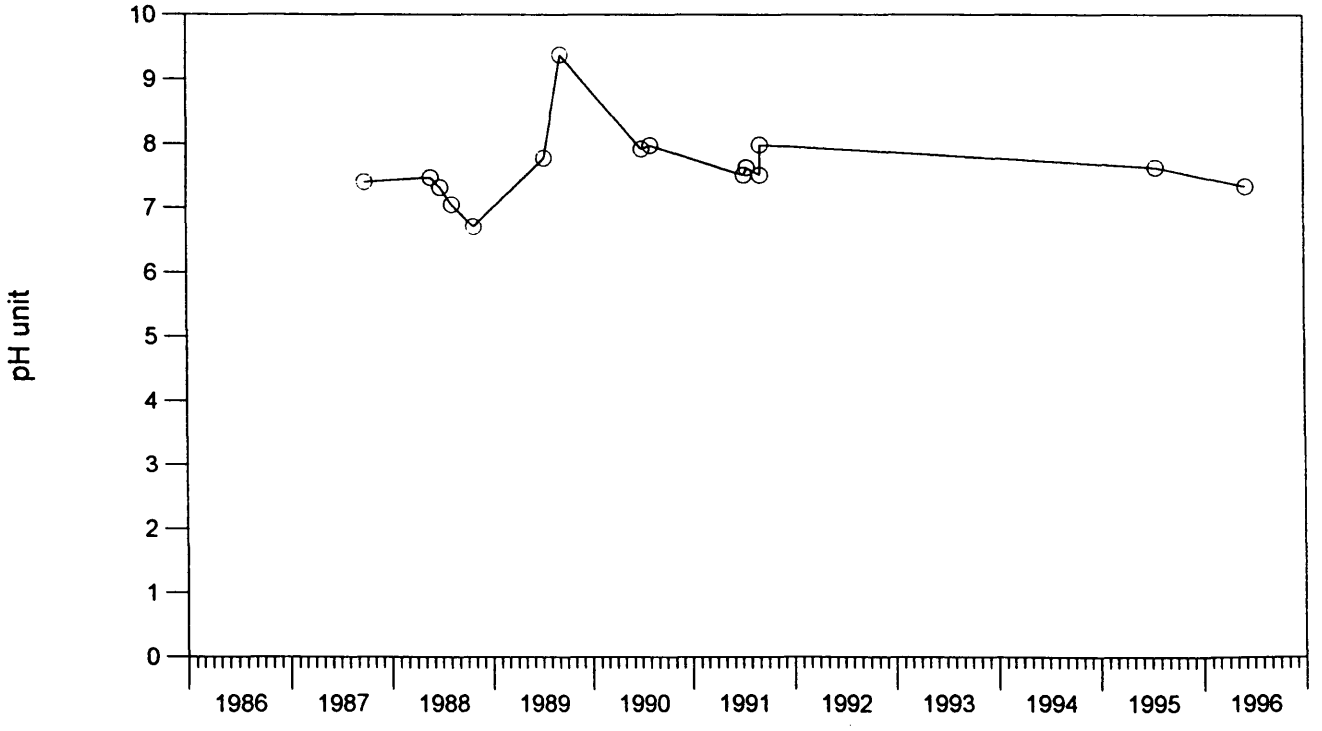


Figure 4-40
Alkalinity time trend, Upper Guardhouse Creek downstream of NW Dump

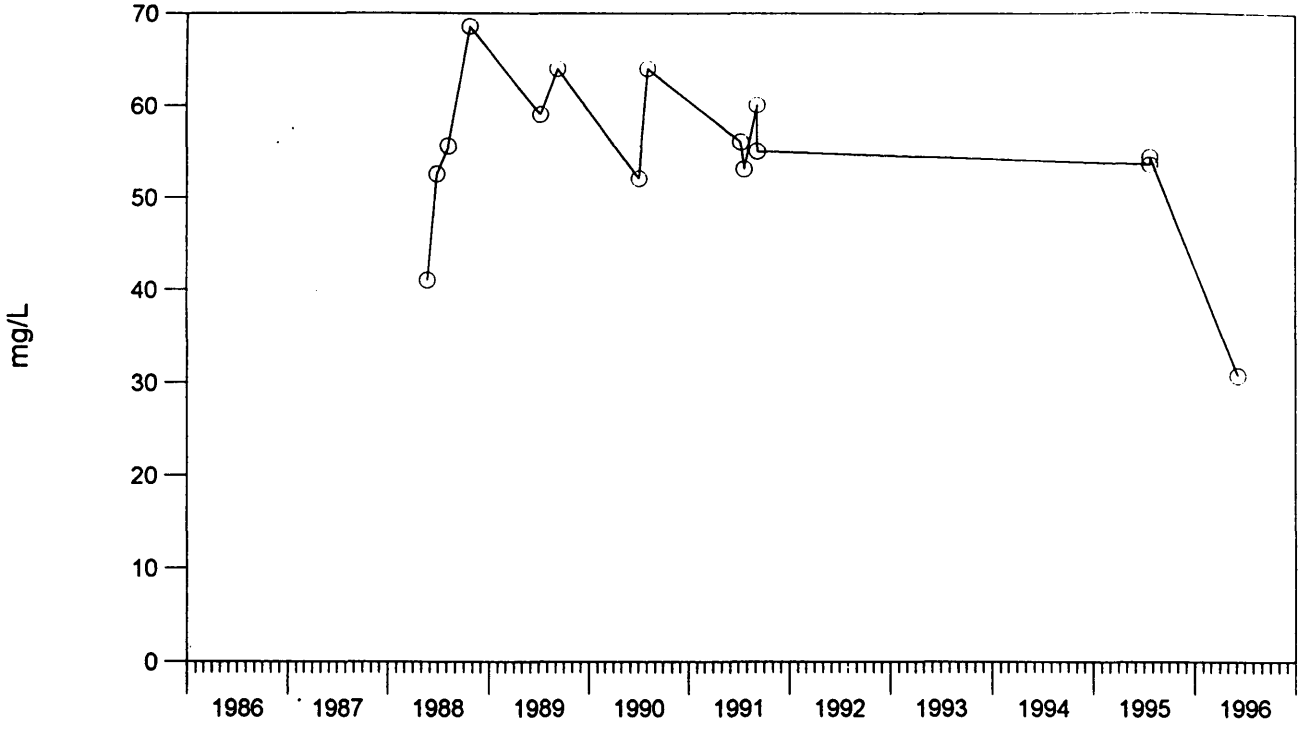


Figure 4-41
Total sulphate time trend, Upper Guardhouse Creek downstream of NW Dump

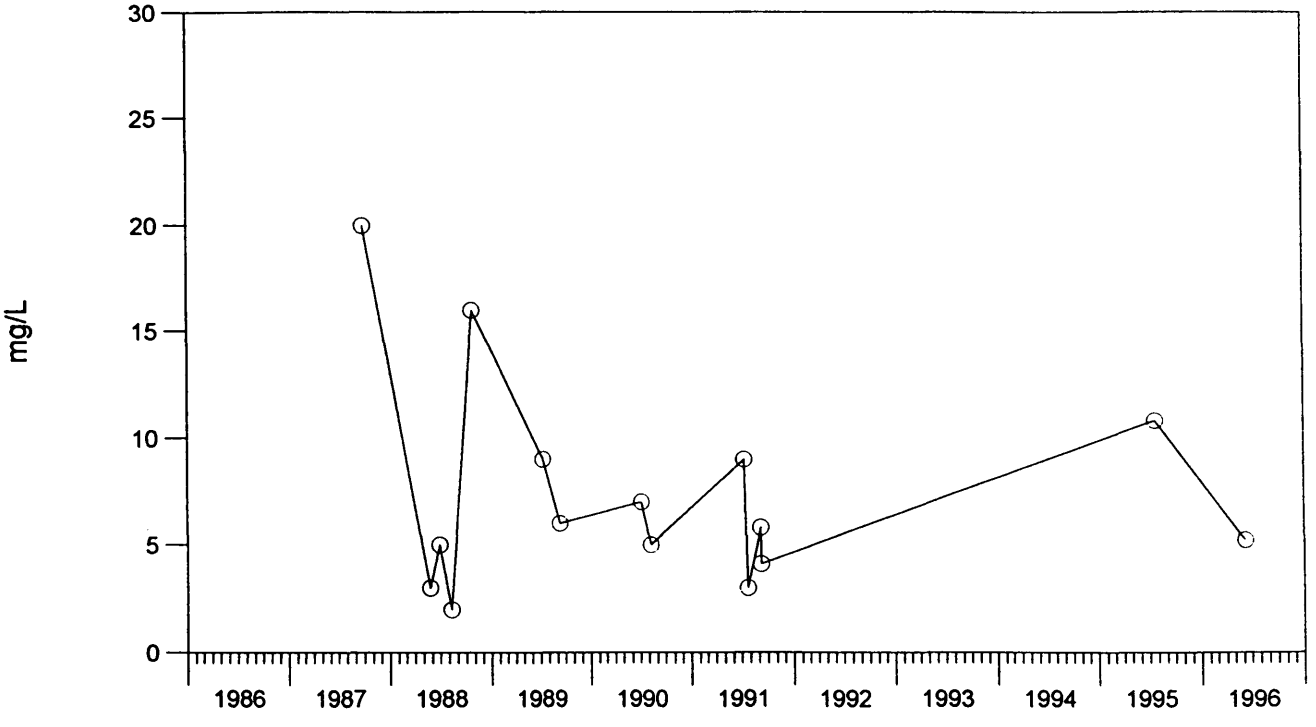


Figure 4-42
Total zinc time trend, Upper Guardhouse Creek downstream of NW Dump

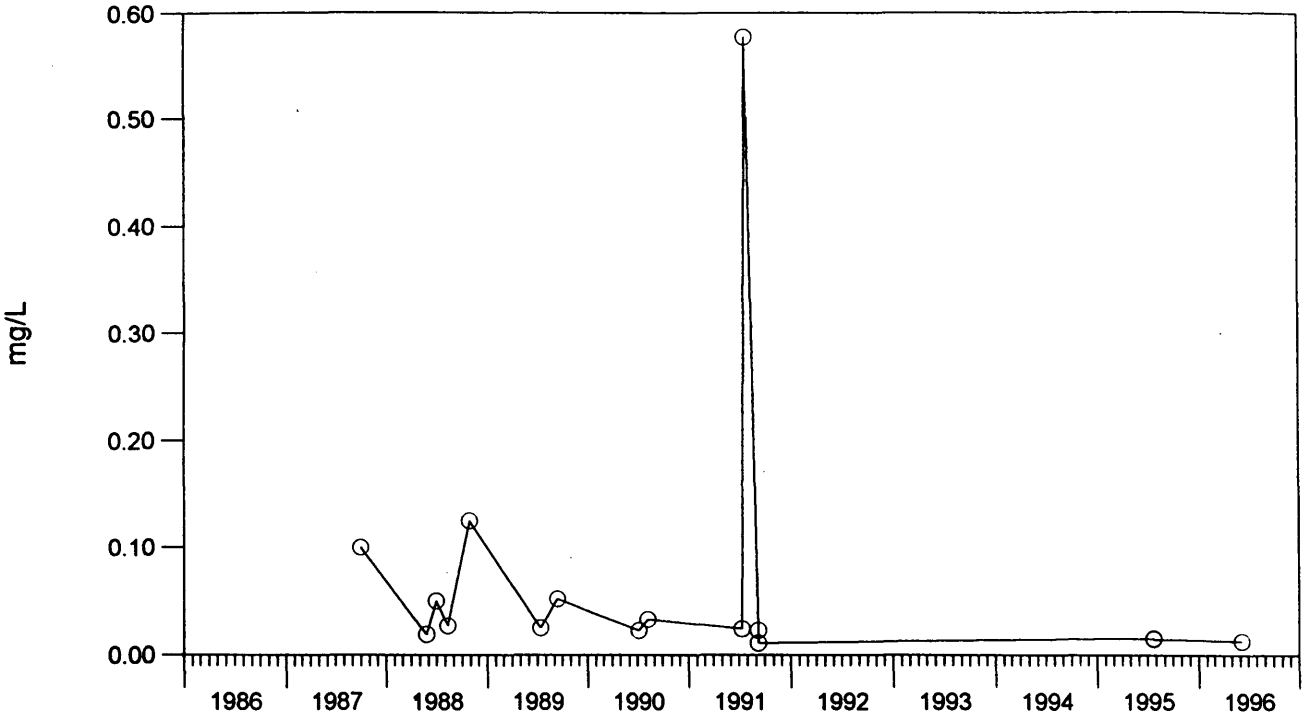


Figure 4-43 pH time trend, North Fork Rose Creek site 1 upstream of Haul Road and site 2 downstream of Haul Road

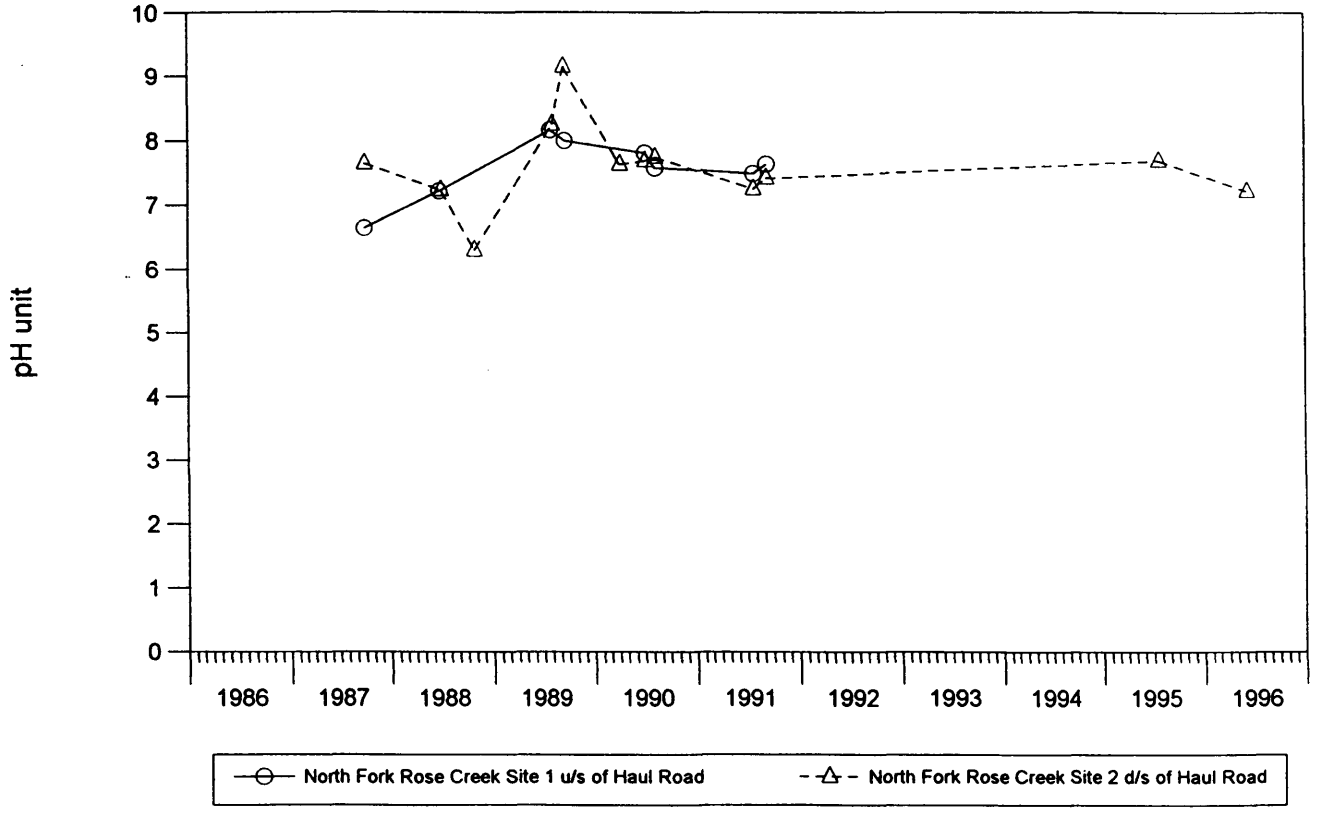


Figure 4-44 Alkalinity time trend, North Fork Rose Creek site 1 upstream of Haul Road and site 2 downstream of Haul Road

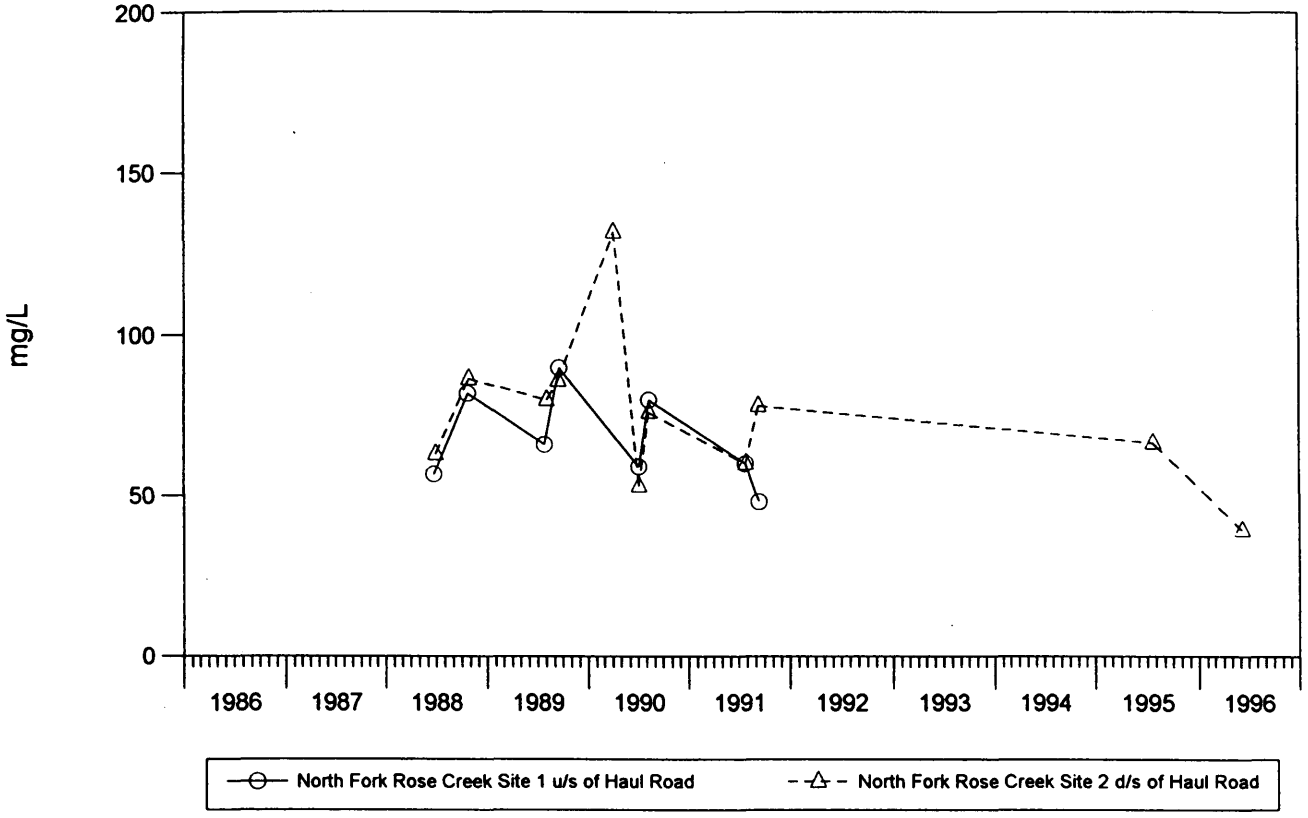


Figure 4-45 Total sulphate time trend, North Fork Rose Creek site 1 upstream of Haul Road and site 2 downstream of Haul Road

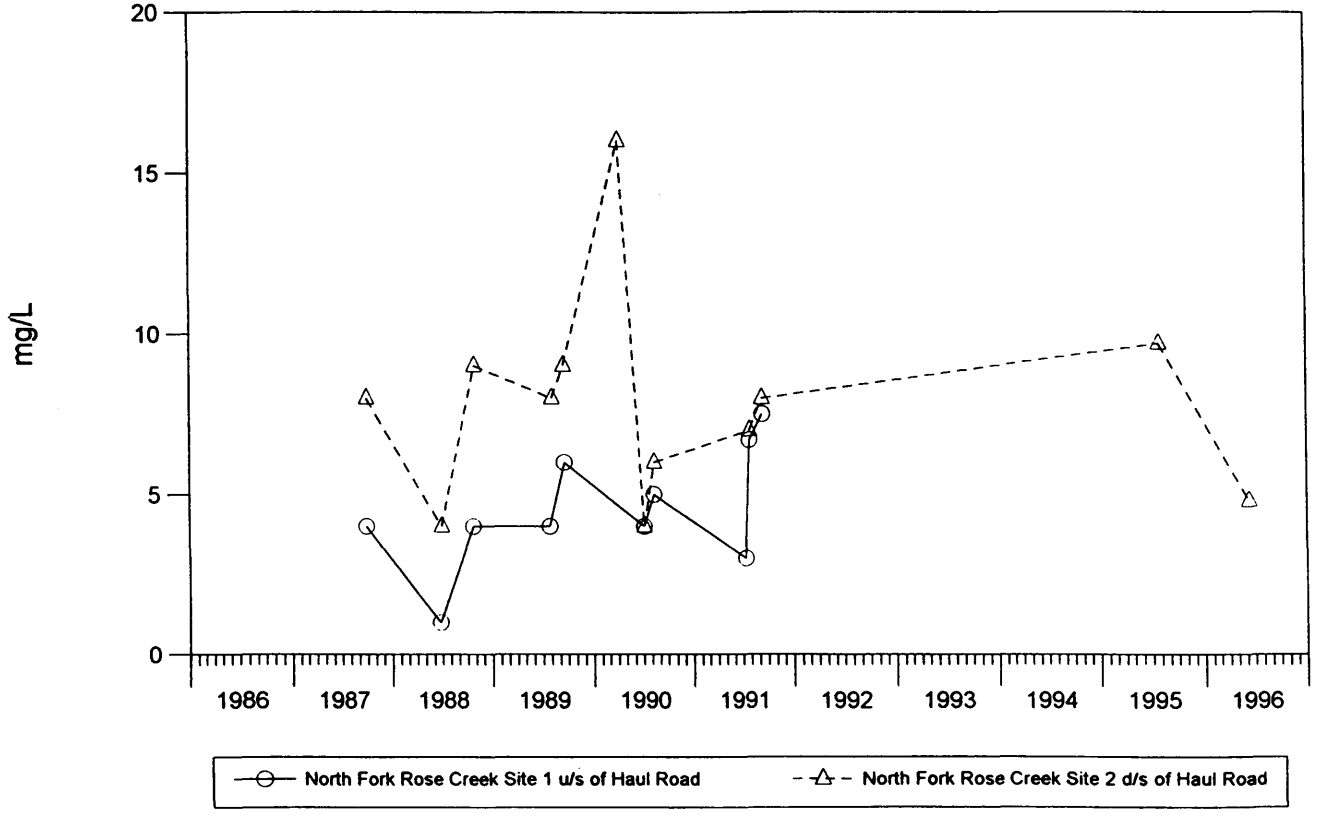


Figure 4-46 Total arsenic time trend, North Fork Rose Creek site 1 upstream of Haul Road and site 2 downstream of Haul Road

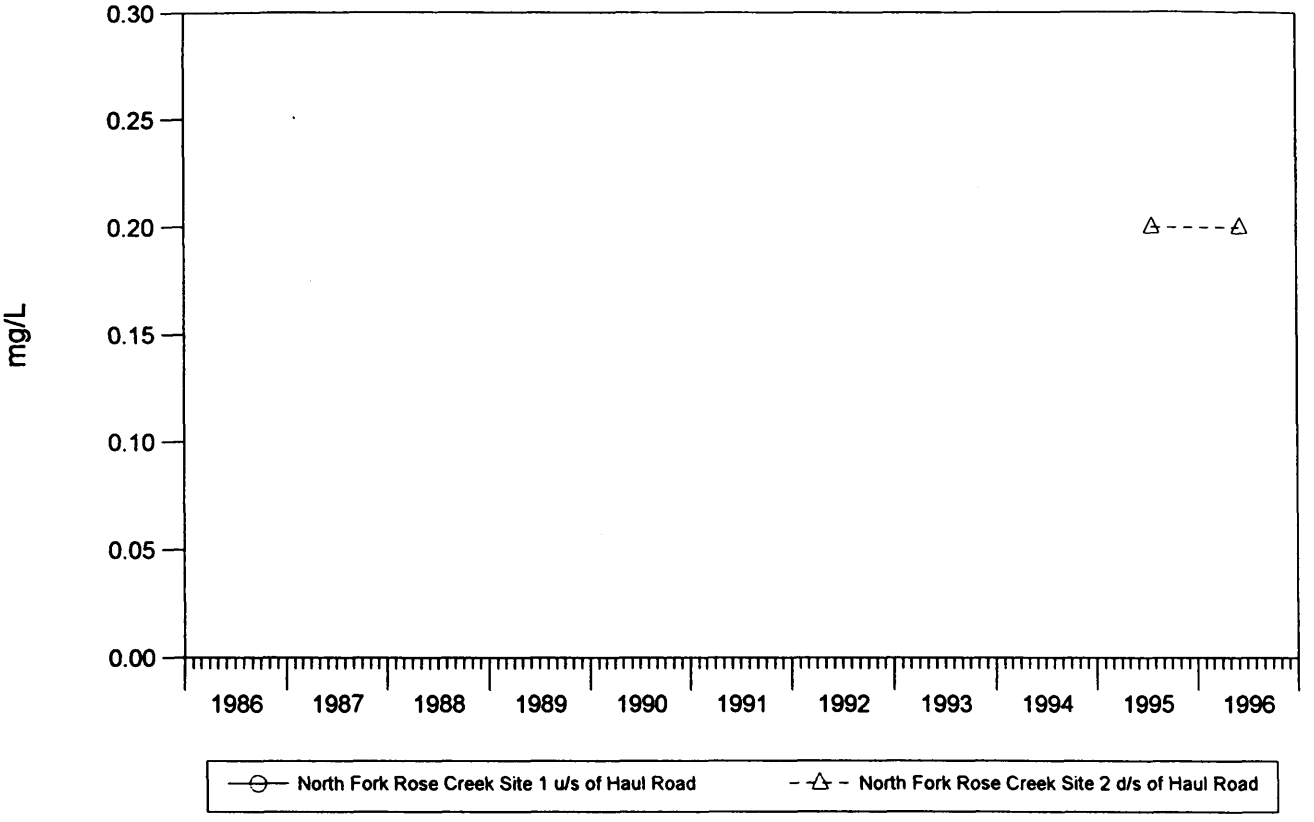


Figure 4-47 Total iron time trend, North Fork Rose Creek site 1 upstream of Haul Road and site 2 downstream of Haul Road

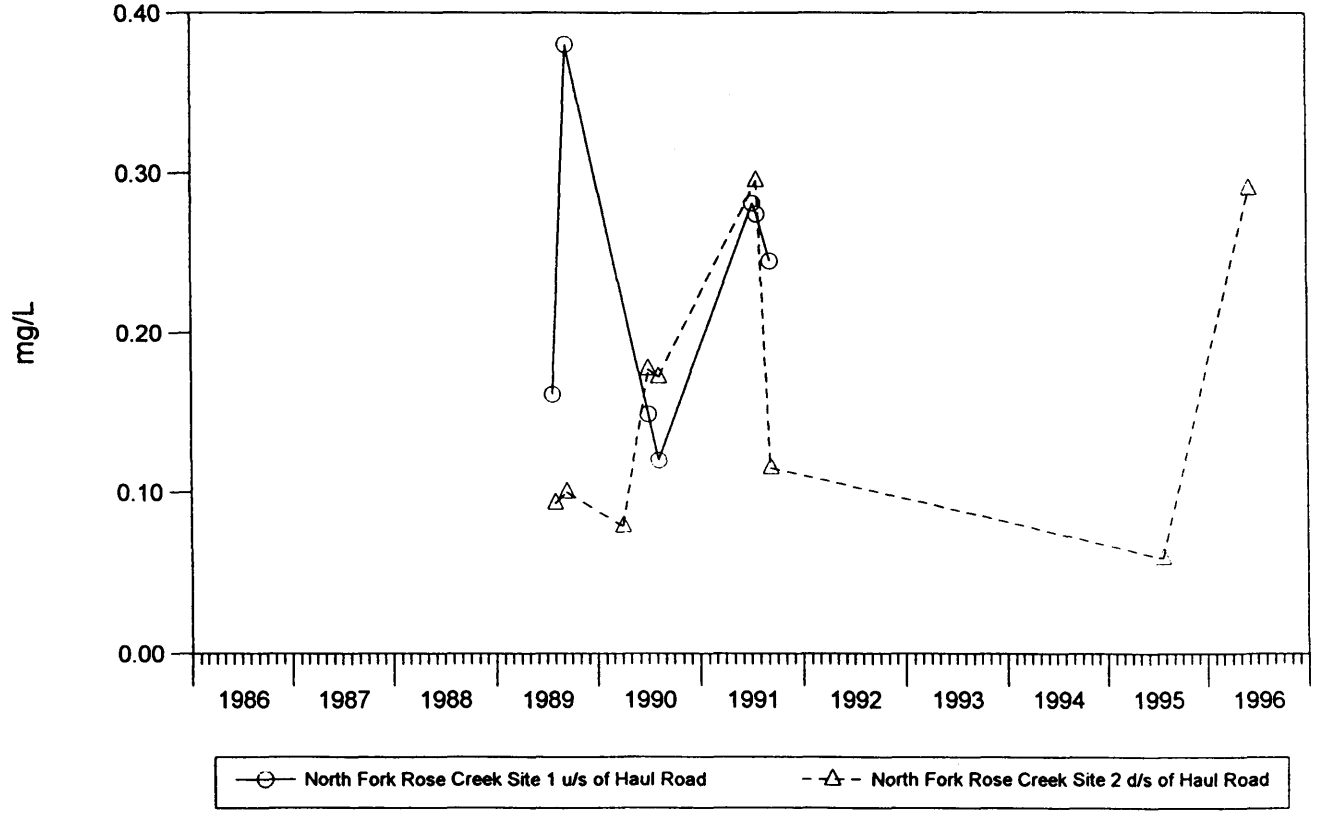


Figure 4-48 Total manganese time trend, North Fork Rose Creek site 1 upstream of Haul Road and site 2 downstream of Haul Road

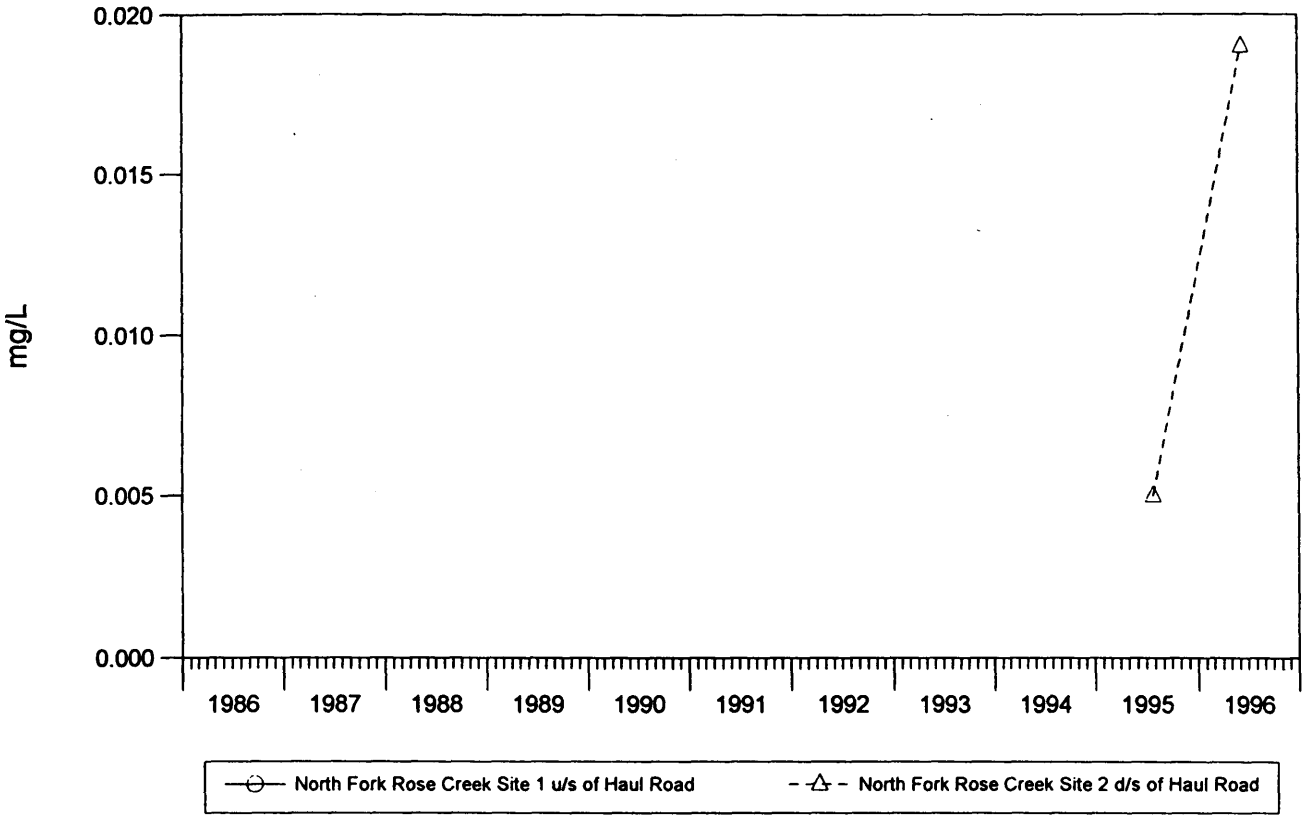


Figure 4-49 Total zinc time trend, North Fork Rose Creek site 1 upstream of Haul Road and site 2 downstream of Haul Road

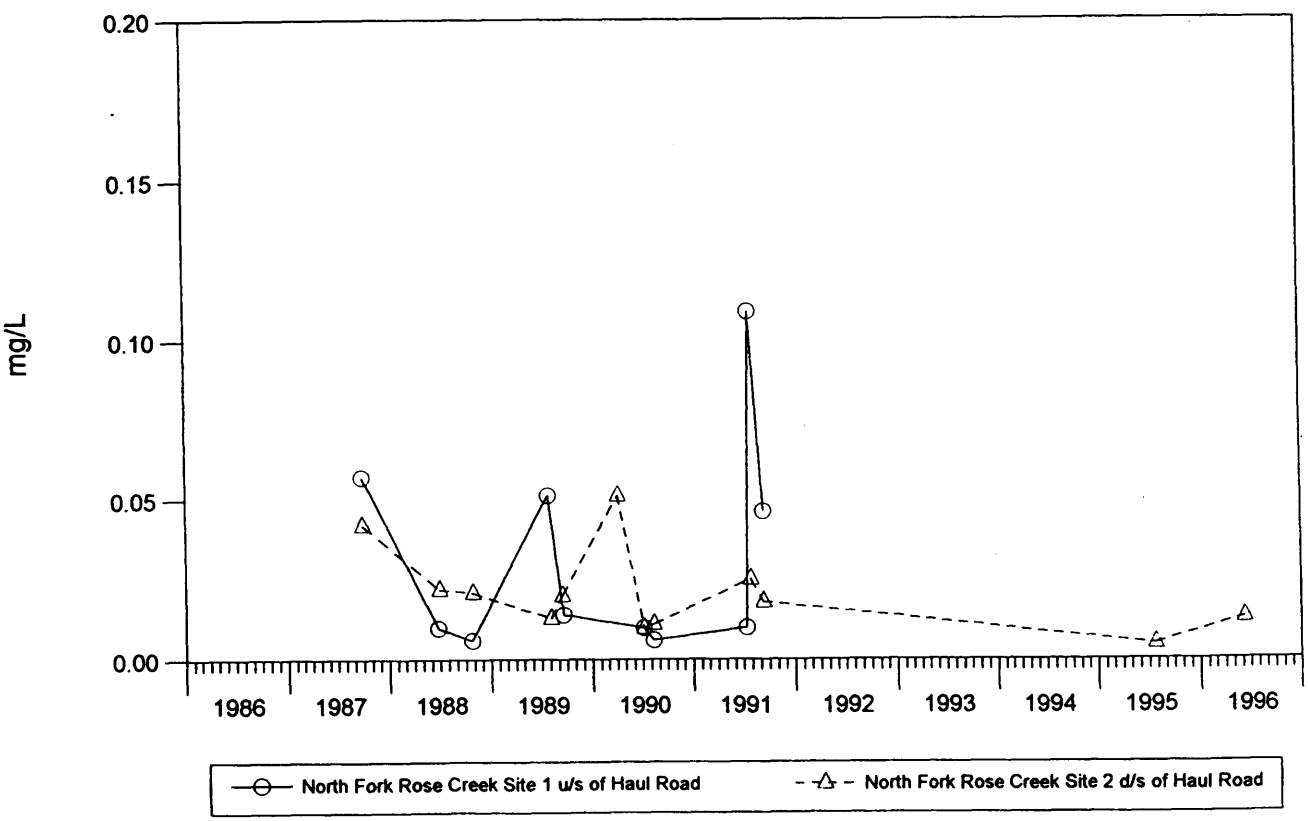


Figure 4-50 Total zinc loading time trend, North Fork Rose Creek site 1 upstream of Haul Road and site 2 downstream of Haul Road

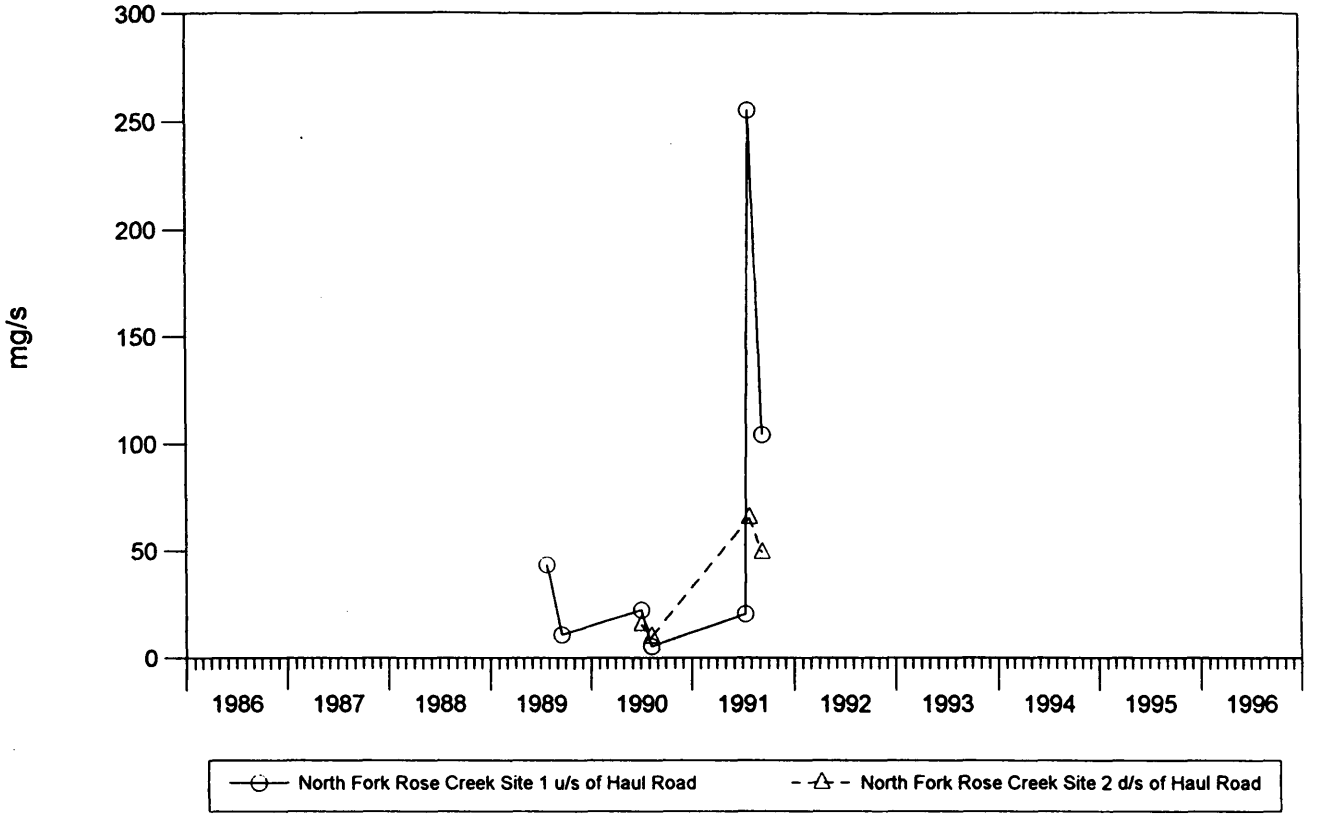


Figure 4-51 pH time trend, North Fork of Rose Creek upstream of mine access road

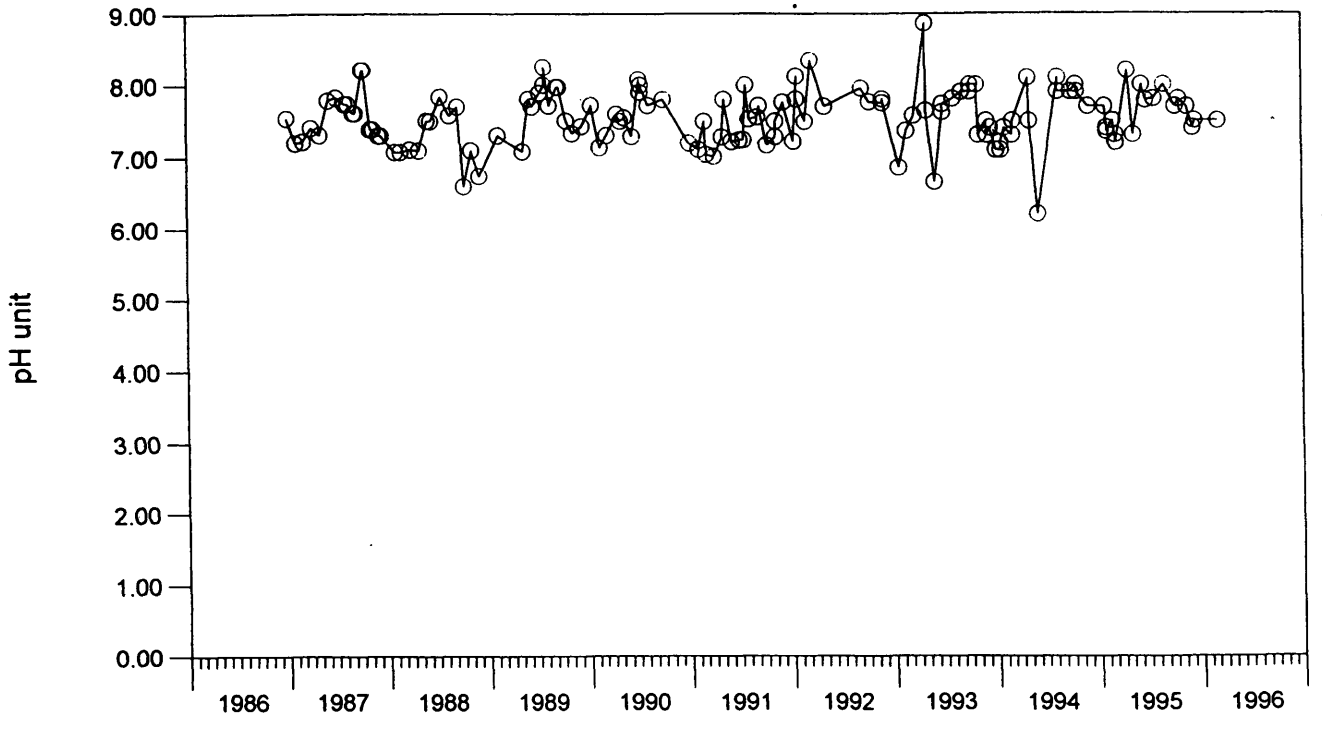


Figure 4-52
Total alkalinity time trend, North Fork of Rose Creek upstream of mine access road

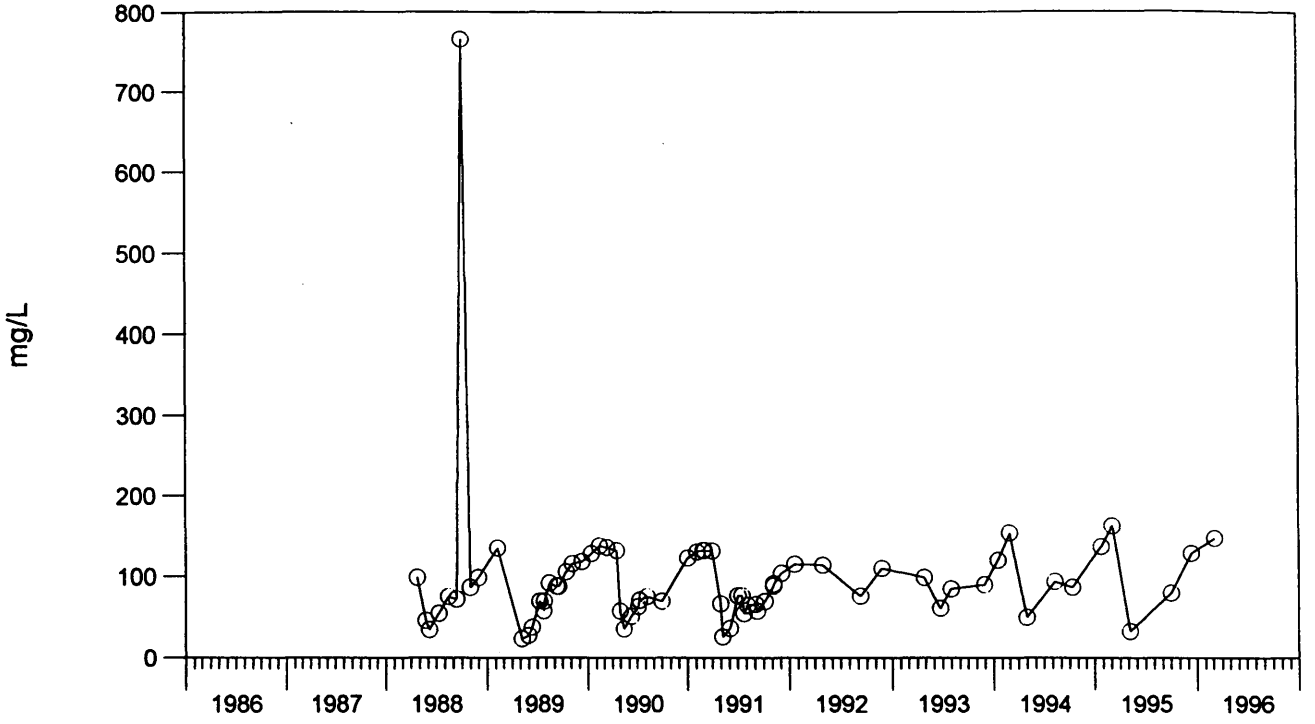


Figure 4-53
Total sulphate time trend, North Fork of Rose Creek upstream of mine access road

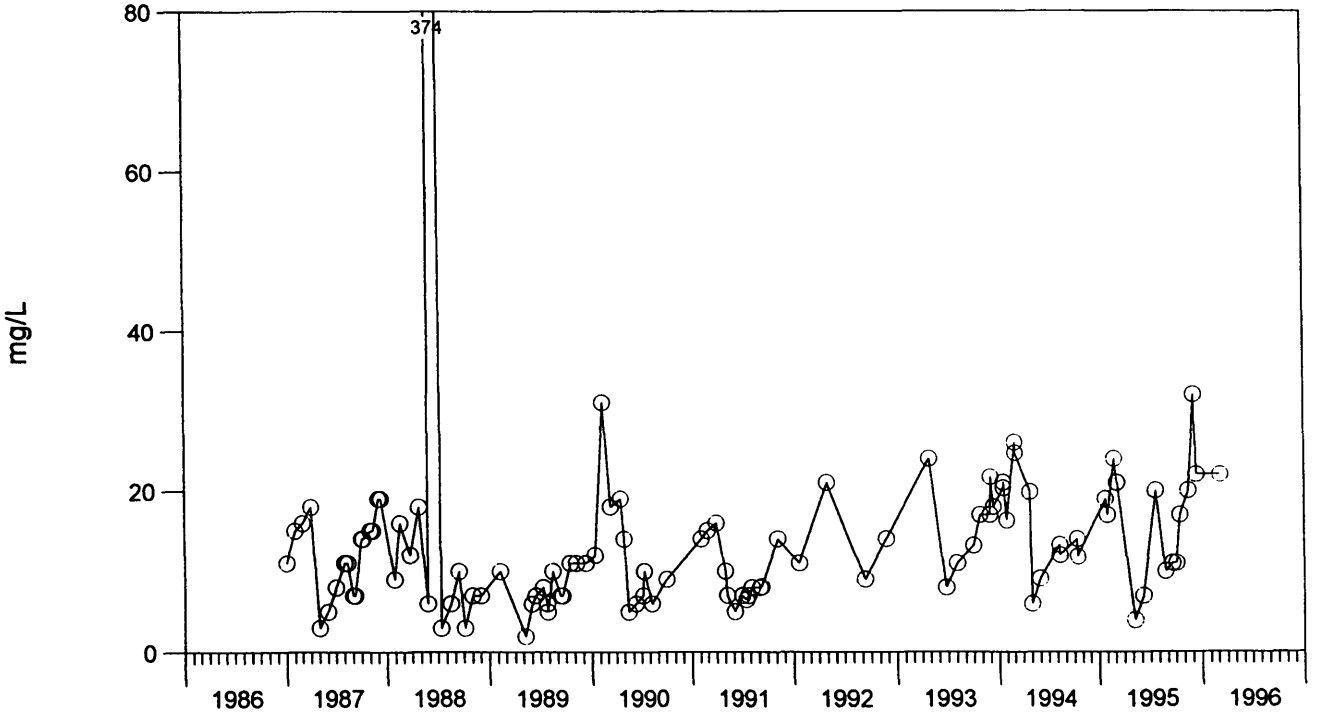


Figure 4-54
Total arsenic time trend, North Fork of Rose Creek upstream of mine access road

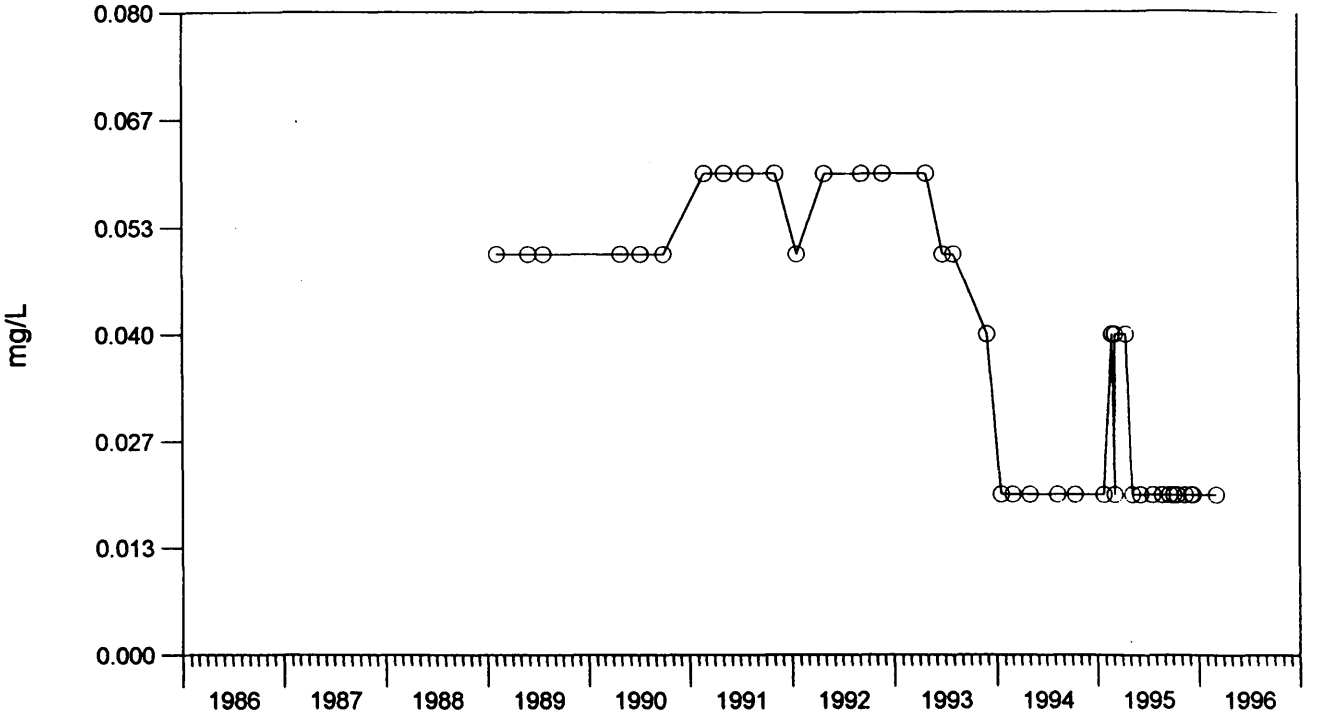


Figure 4-55
Total iron time trend, North Fork of Rose Creek upstream of mine access road

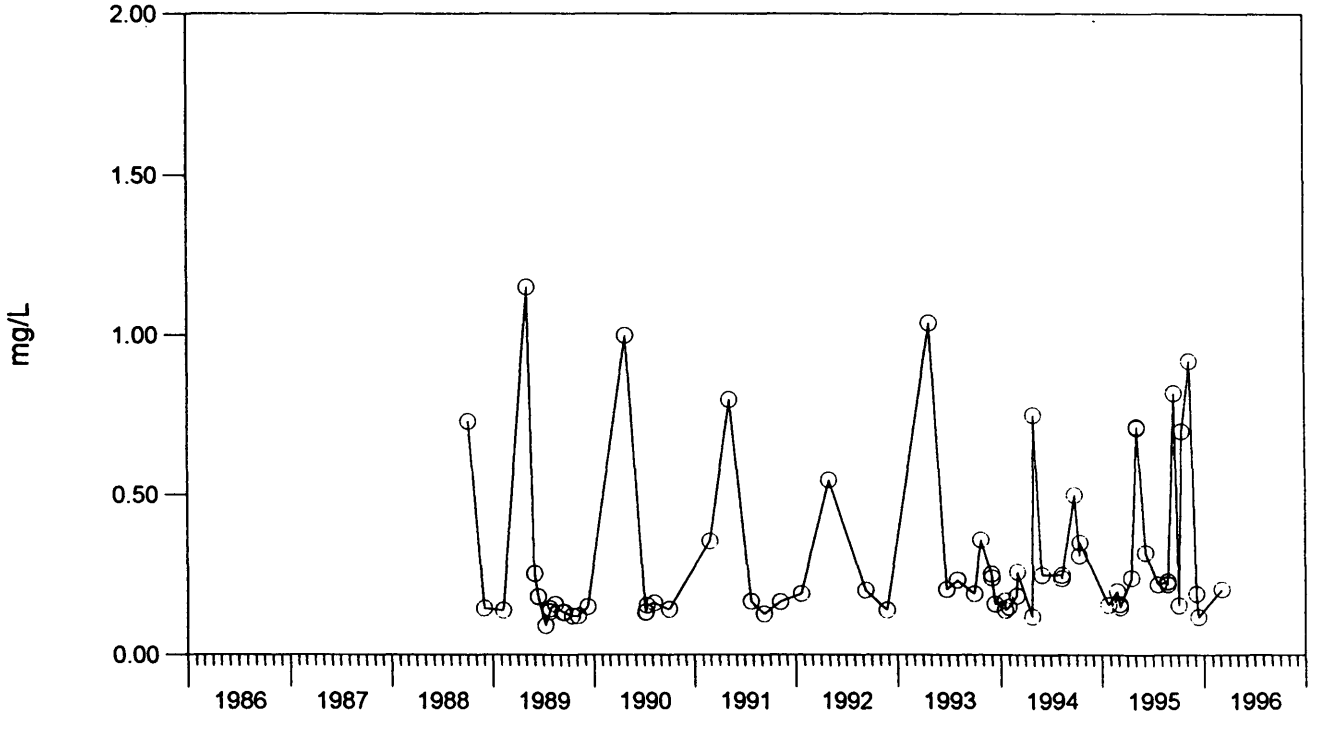


Figure 4-56
Total manganese time trend, North Fork of Rose Creek upstream of mine access road

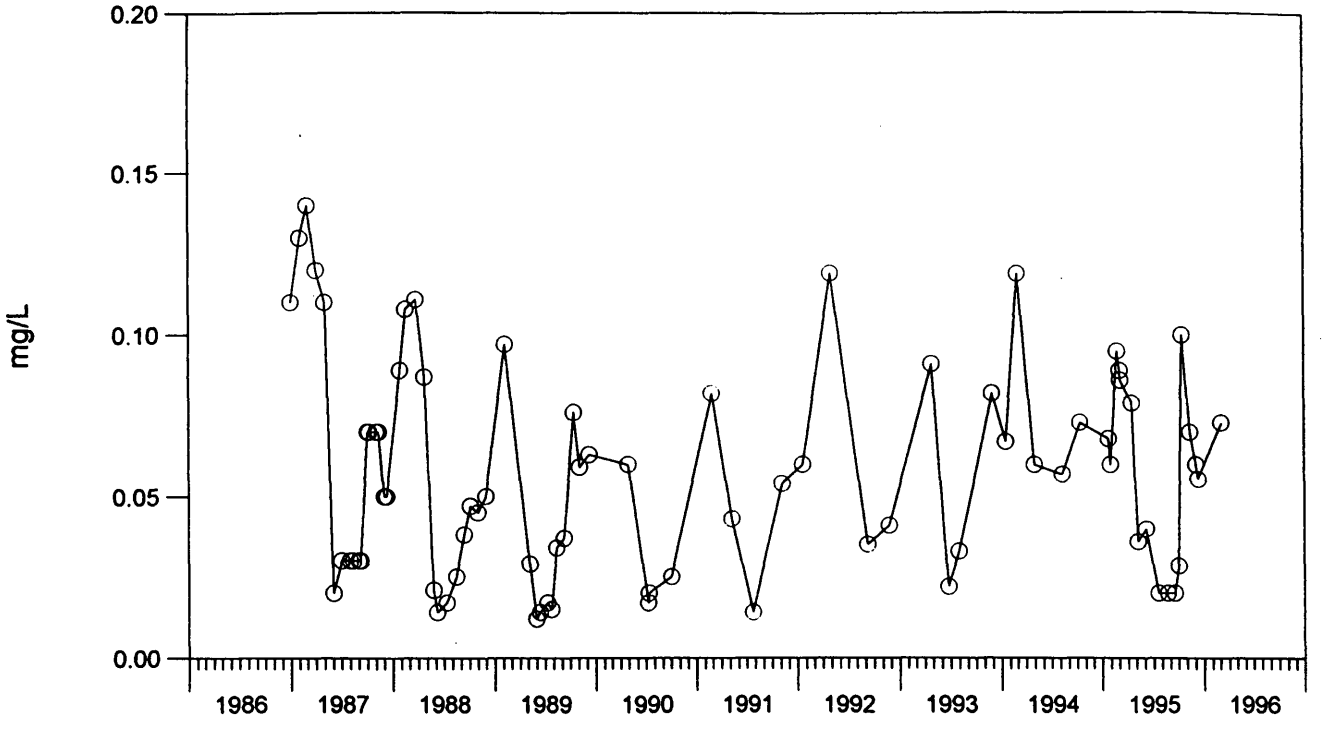


Figure 4-57
Total zinc time trend, North Fork of Rose Creek upstream of mine access road

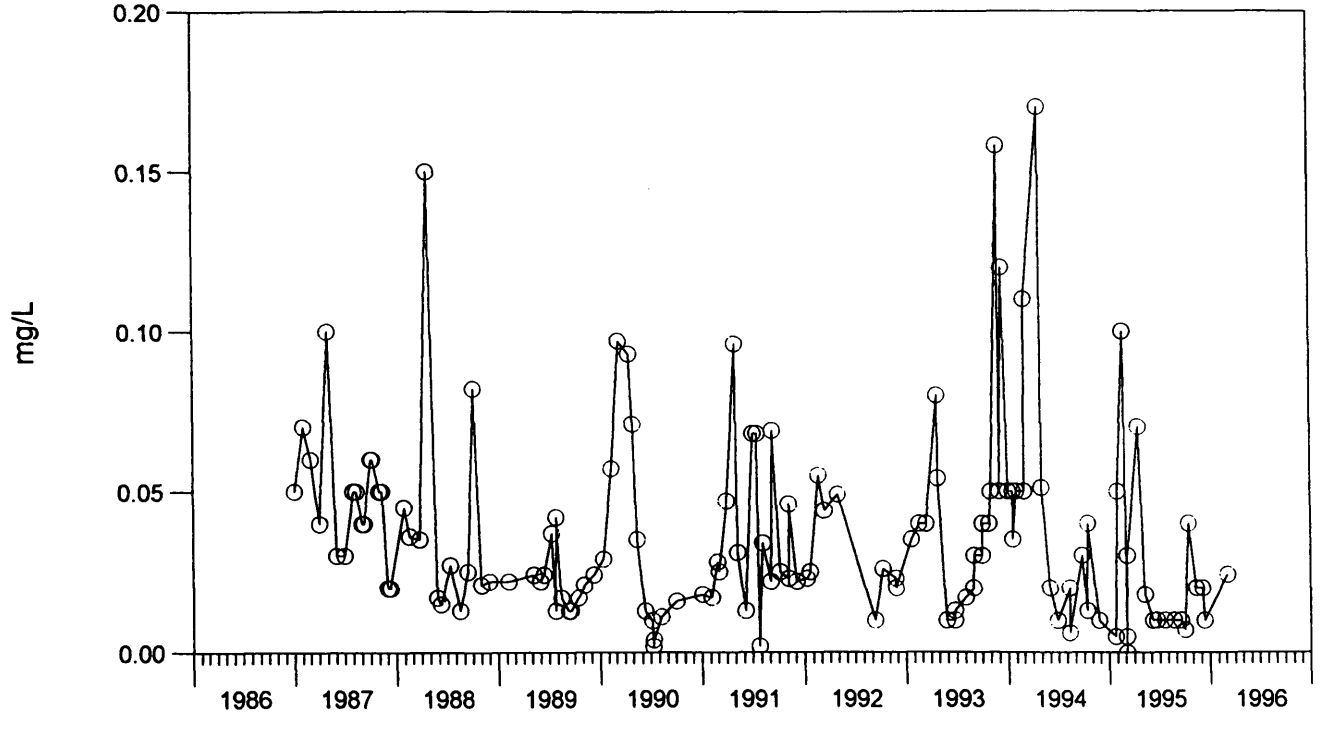


Figure 4-58
pH time trend, Old Faro Creek channel at toe of waste dumps

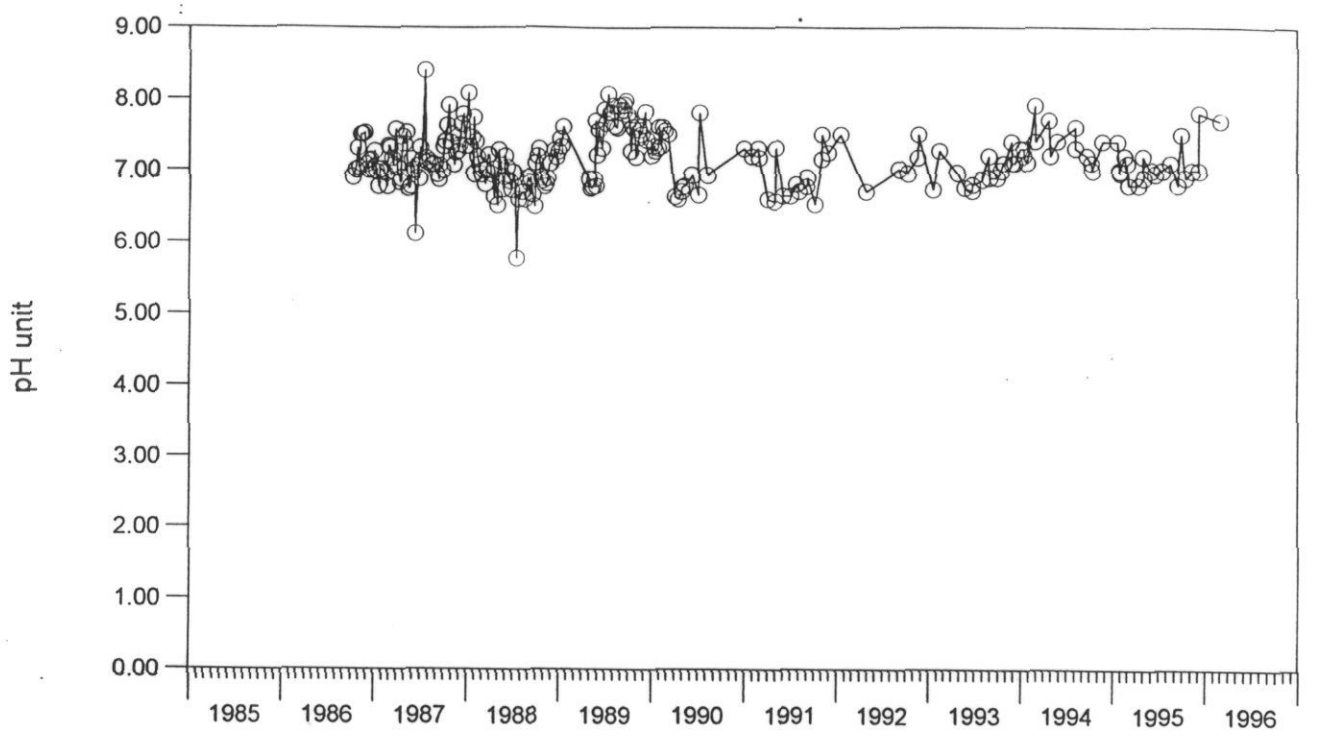


Figure 4-59
pH seasonal trend, Old Faro Creek channel at toe of waste dumps

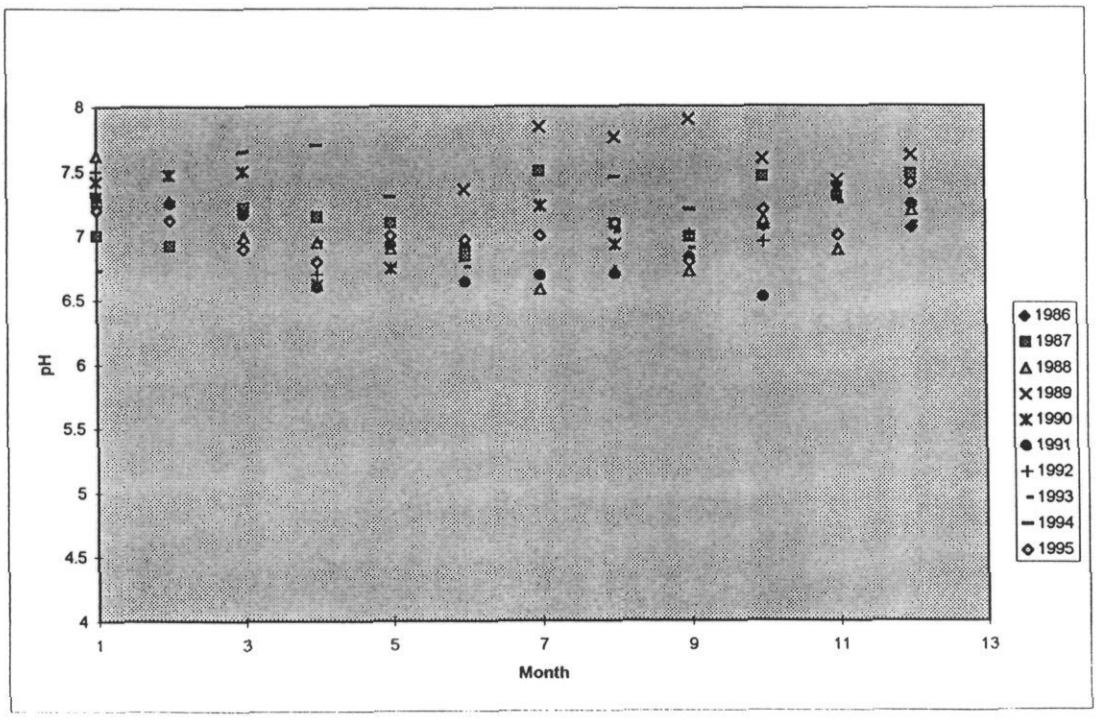


Figure 4-60
Total sulphate time trend, Old Faro Creek channel at toe of waste dumps

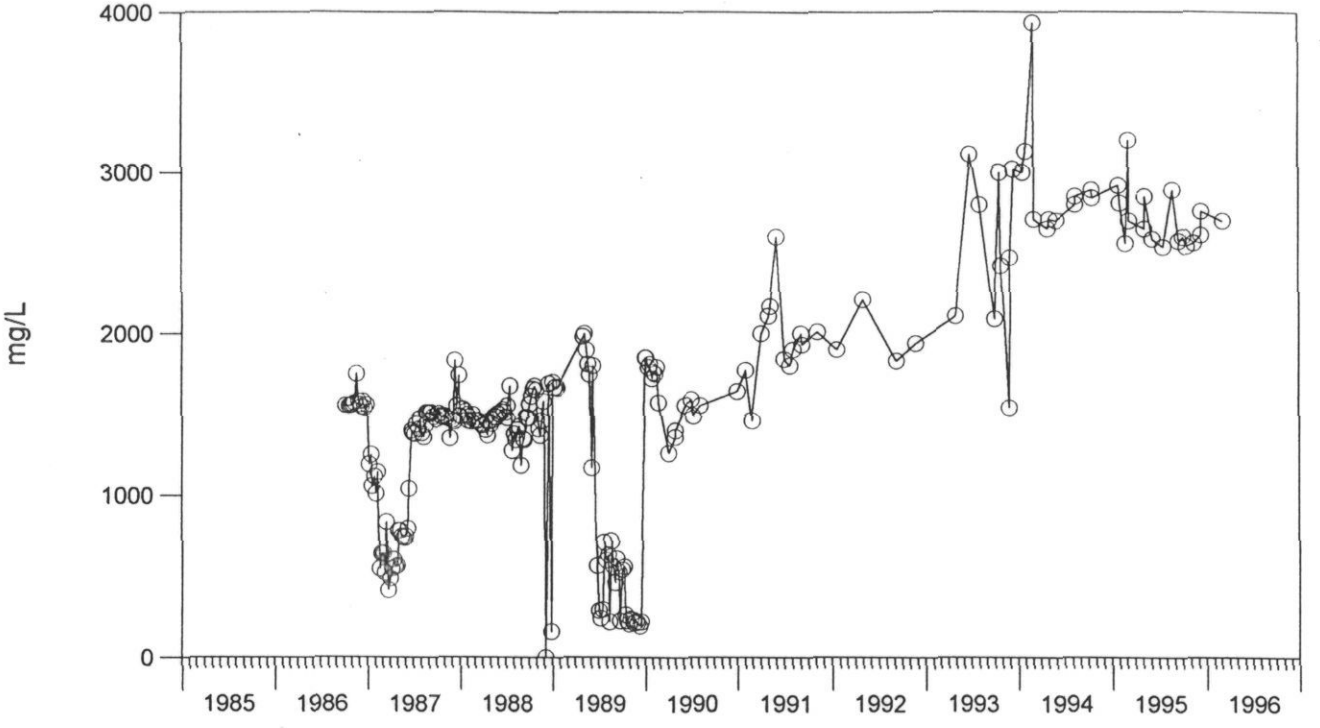


Figure 4-61
Sulphate seasonal trend, Old Faro Creek channel at toe of waste dumps

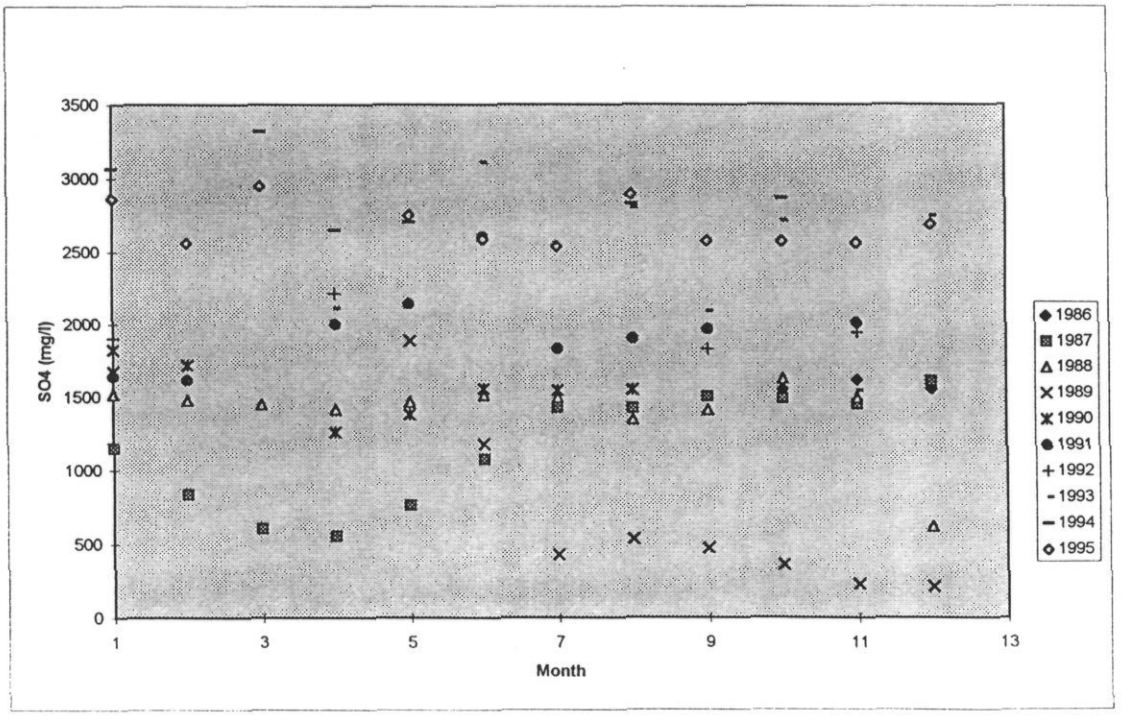


Figure 4-62
Total arsenic, Old Faro Creek channel at toe of waste dumps

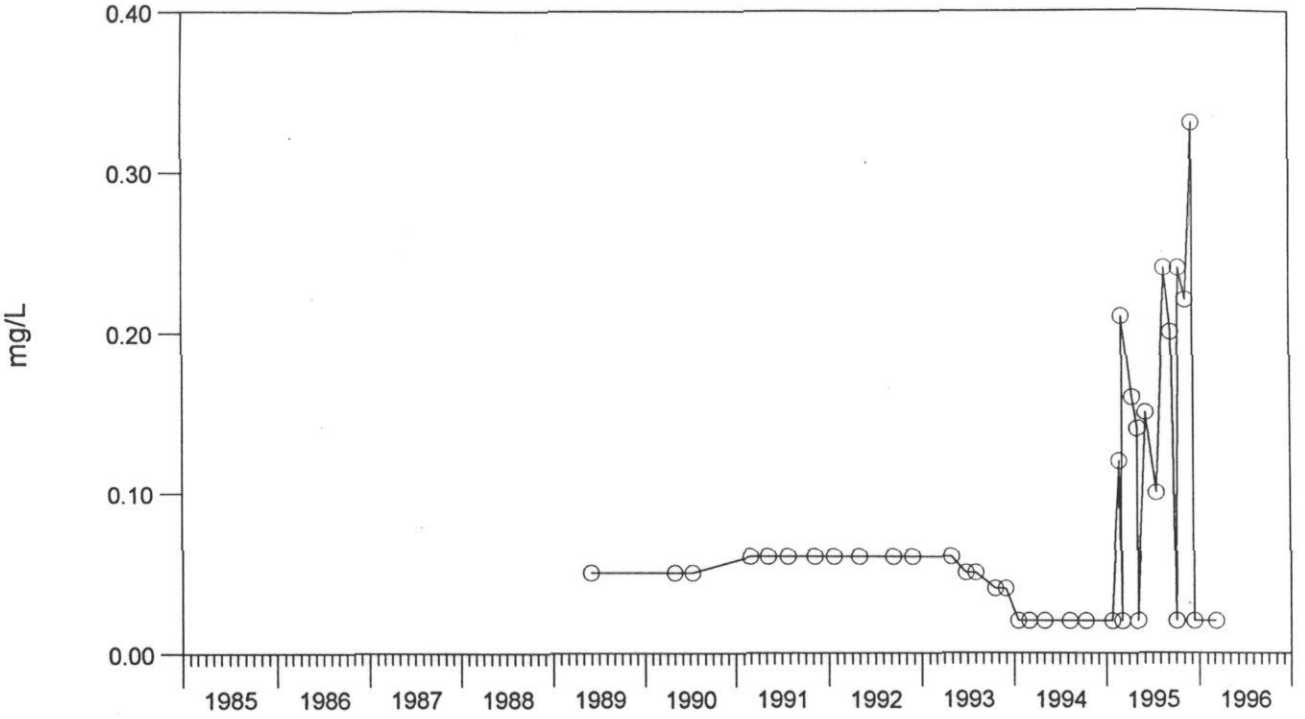


Figure 4-63
Total lead time trend, Old Faro Creek channel at toe of waste dumps

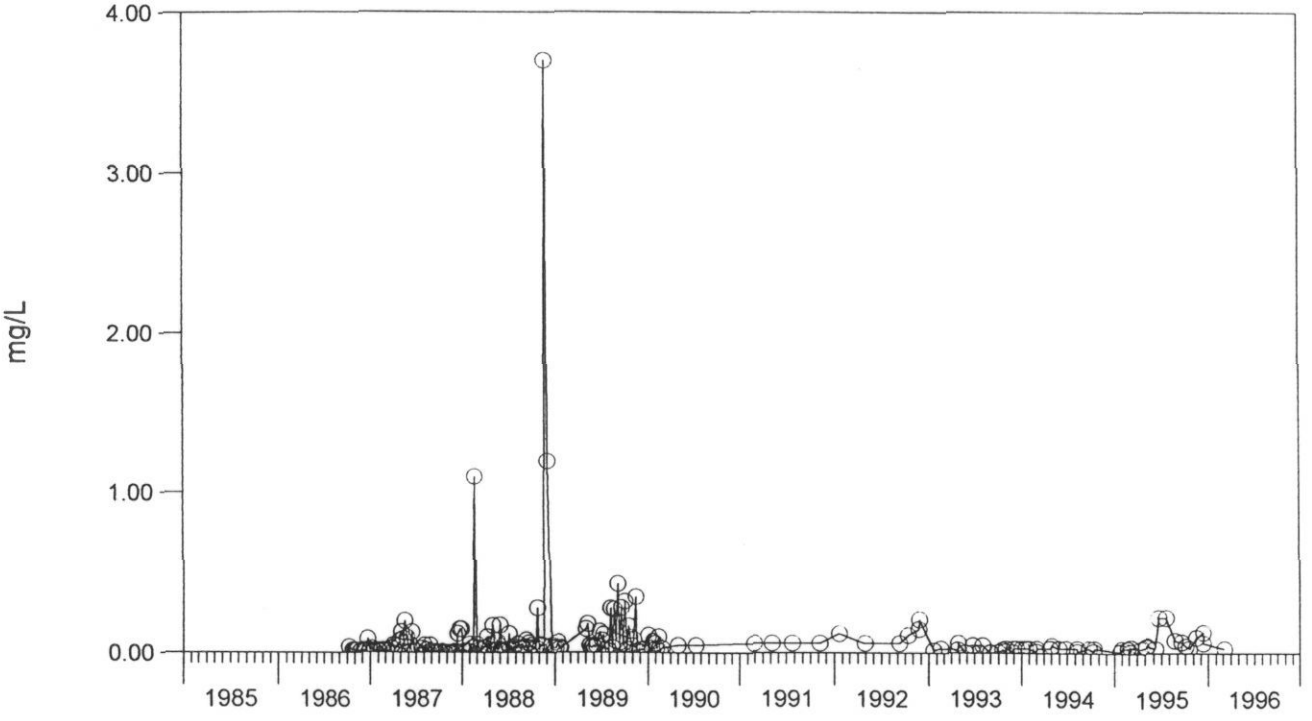


Figure 4-64
Total zinc time trend, Old Faro Creek channel at toe of waste dumps

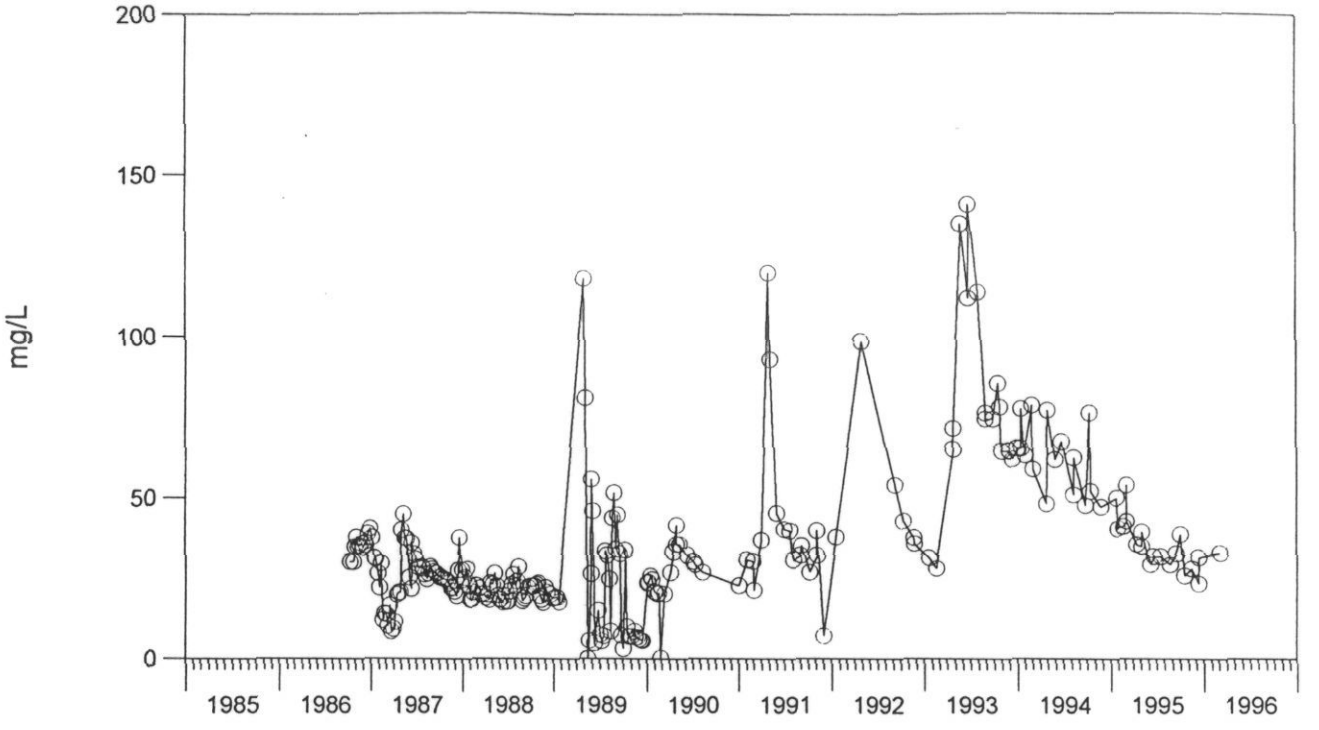


Figure 4-65
Zinc seasonal trend, Old Faro Creek channel at toe of waste dumps

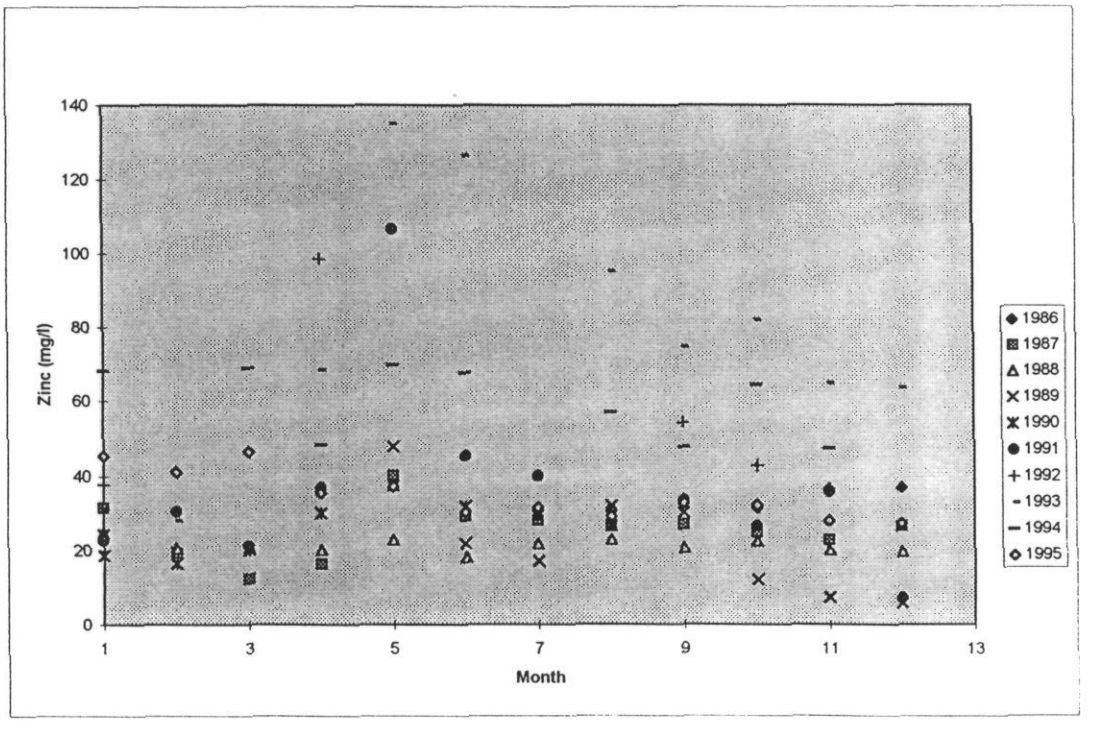


Figure 4-66 Total zinc time trend, Old Faro Creek Channel at mine access road and at the toe of the waste dumps

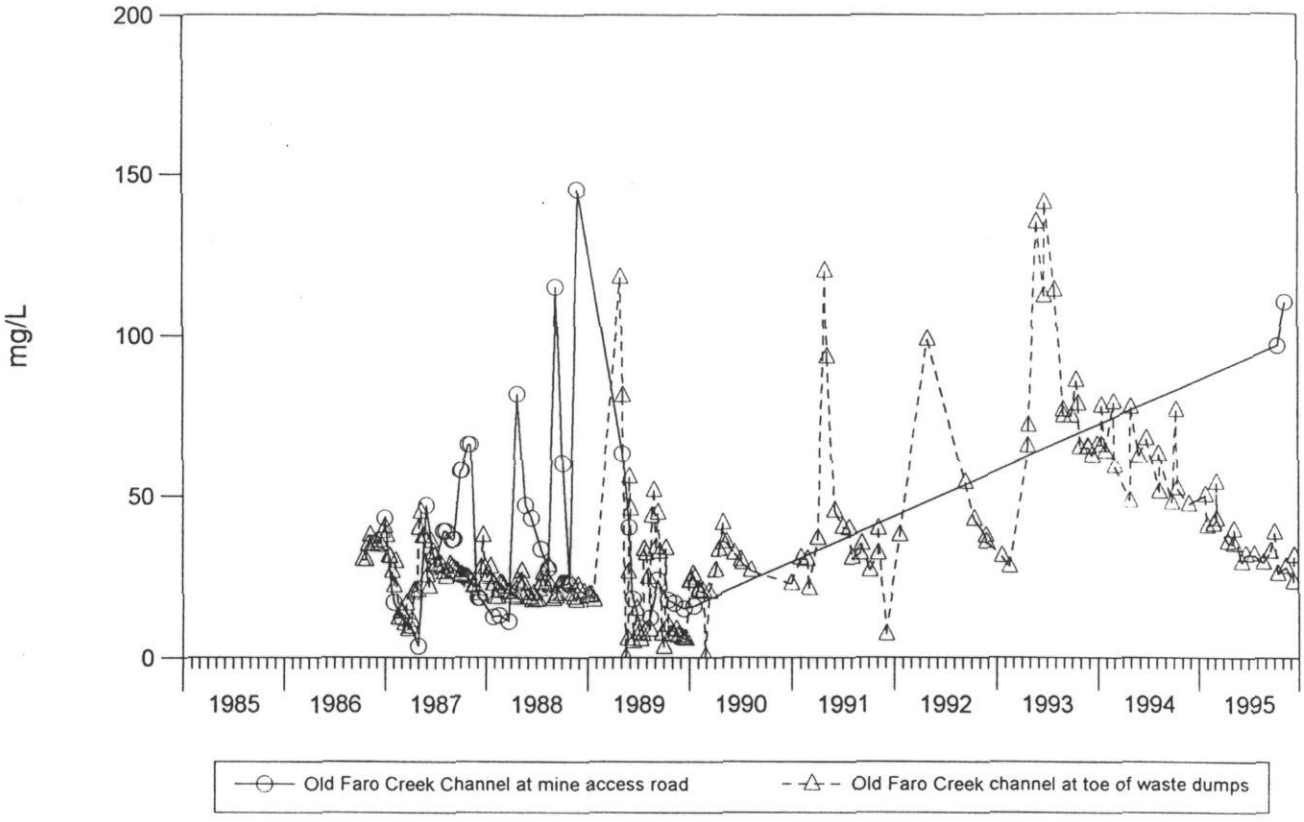
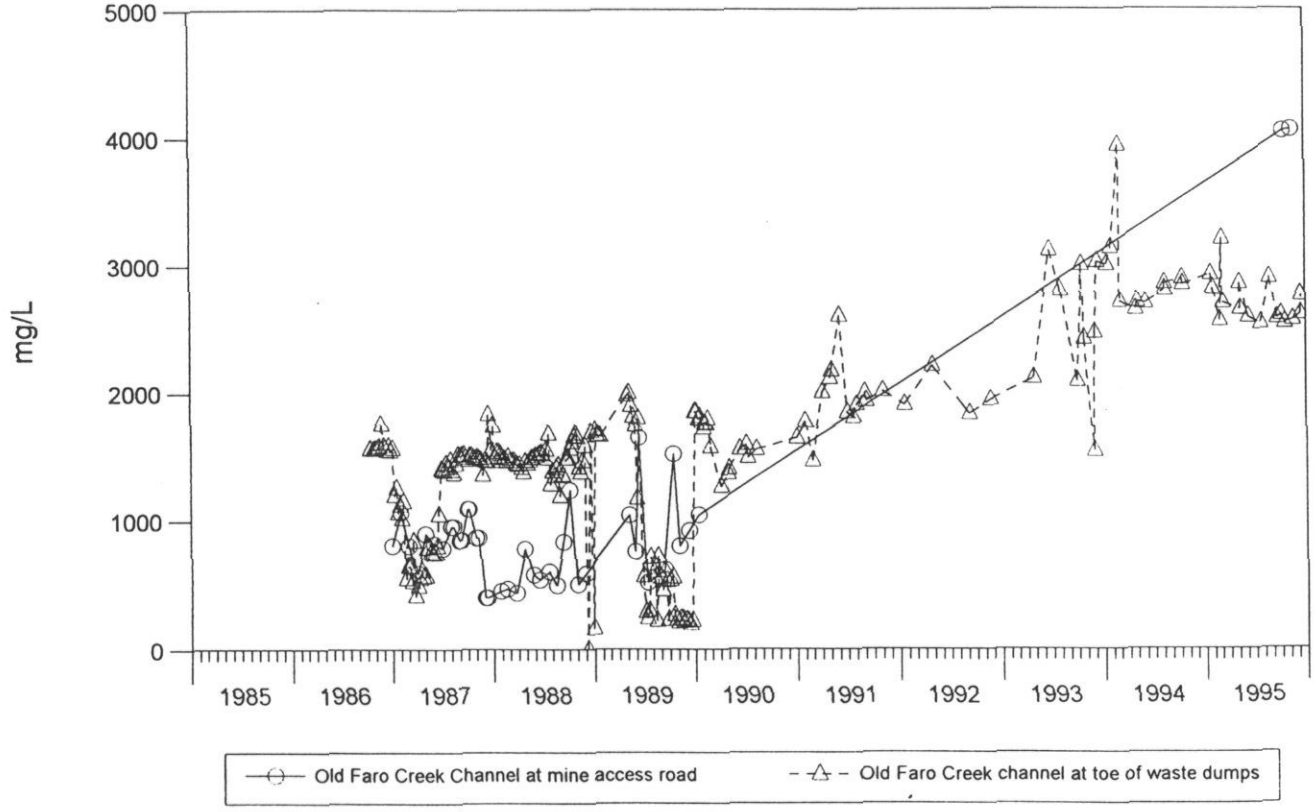
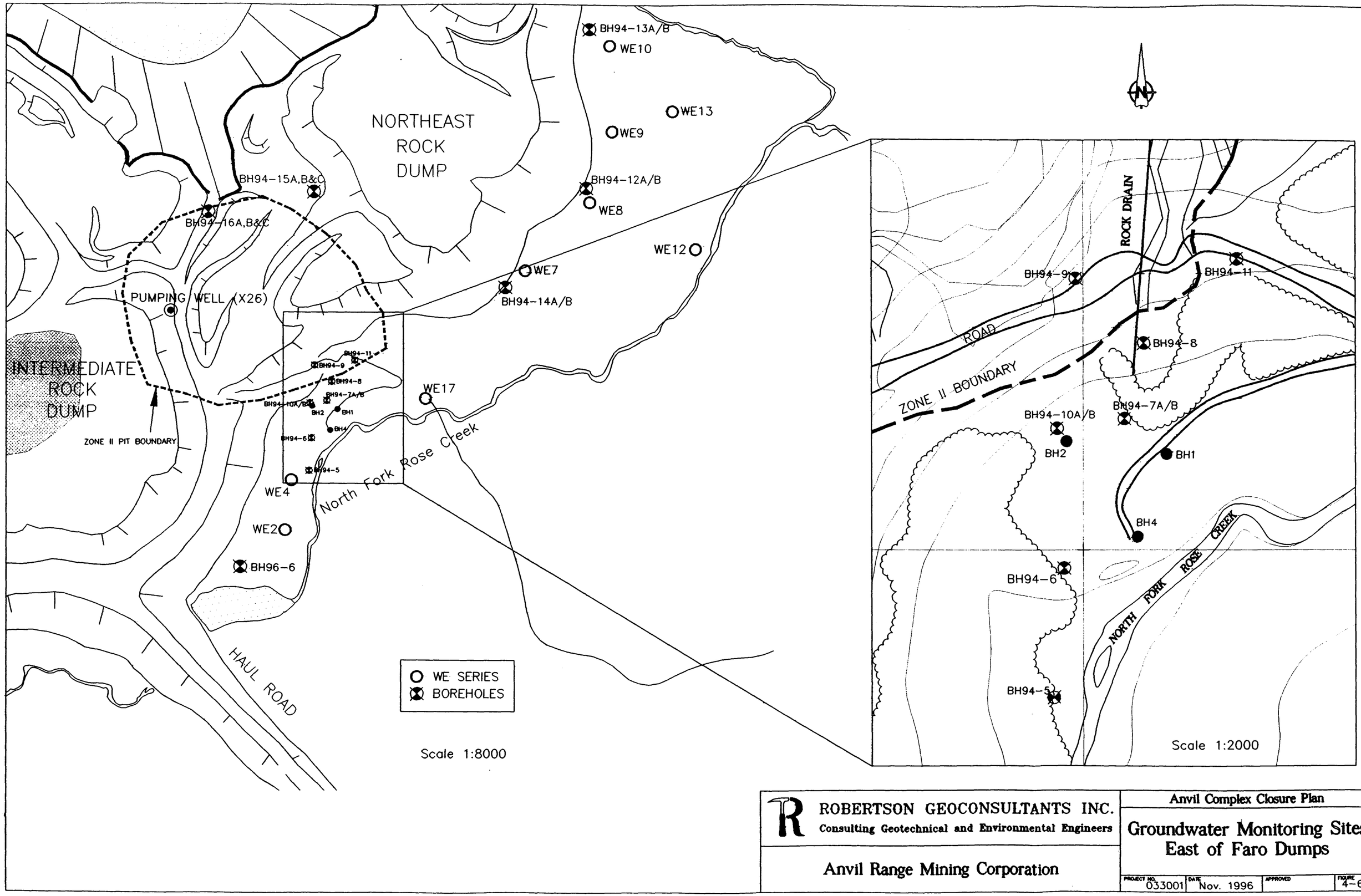


Figure 4-67 Total sulphate time trend, Old Faro Creek Channel at mine access road and at the toe of the waste dumps





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Anvil Range Mining Corporation

Anvil Complex Closure Plan

Groundwater Monitoring Sites East of Faro Dumps

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-68

Figure 4-69 Groundwater Quality Trends in Boreholes BH1 and BH2.

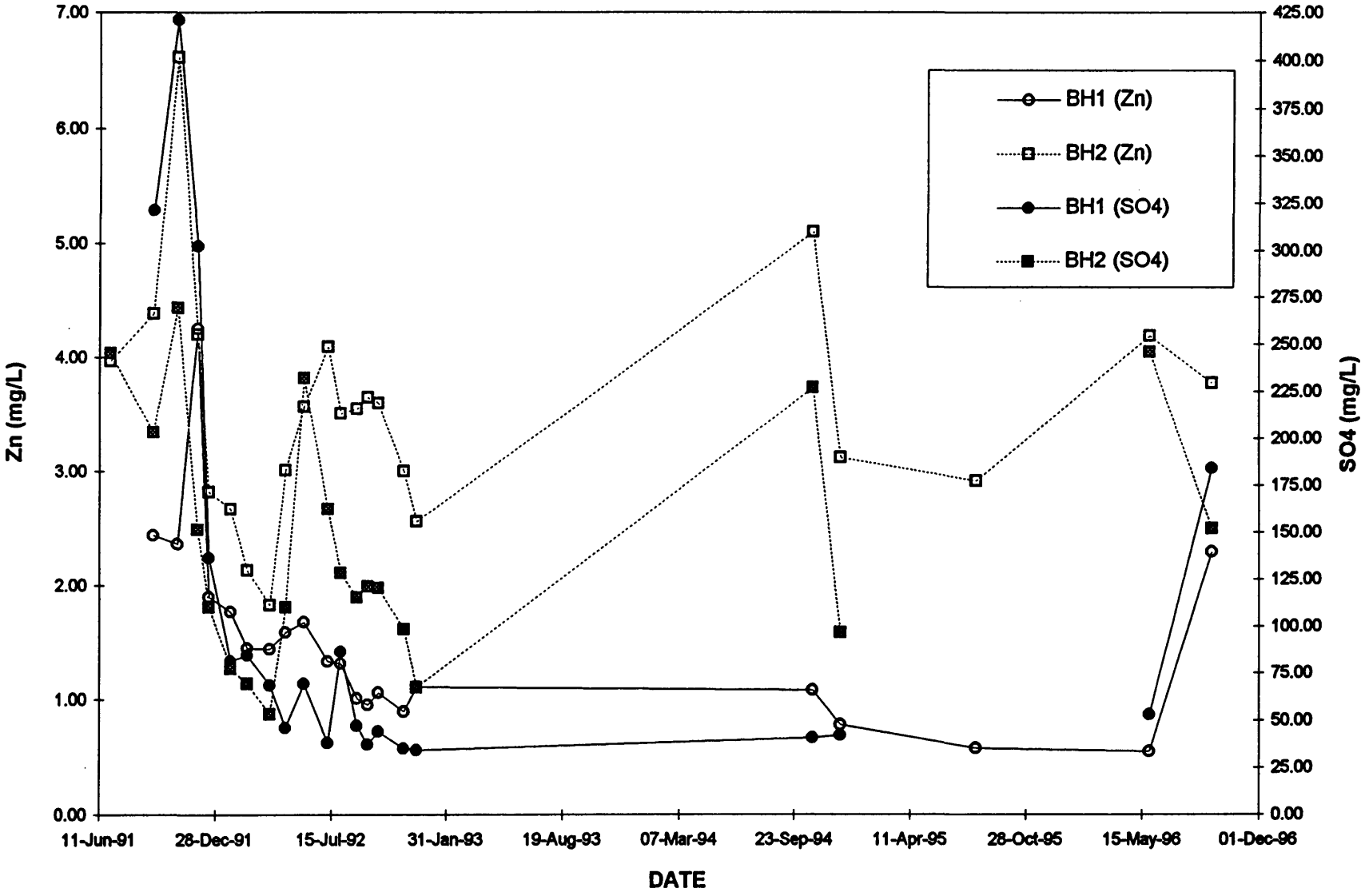
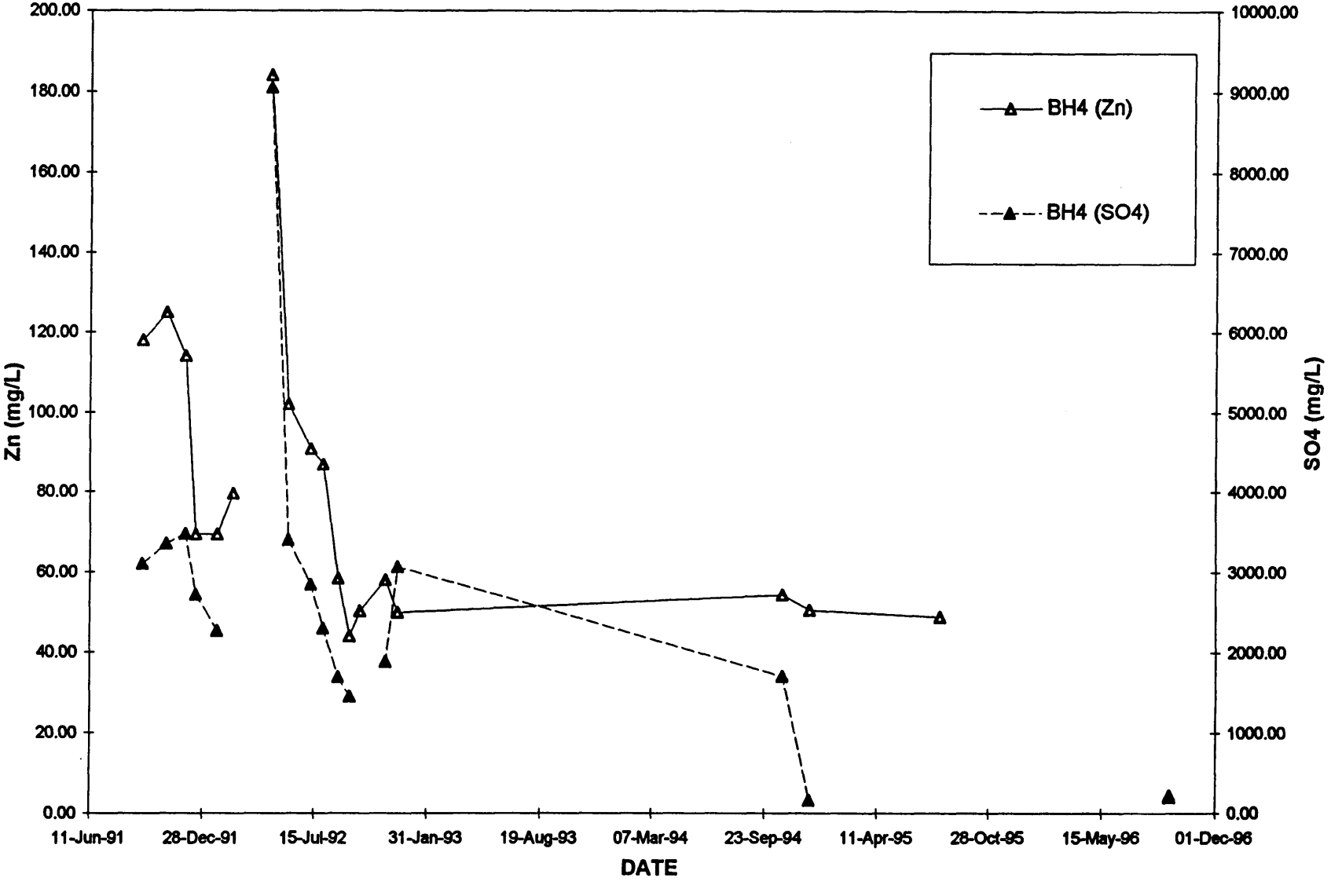
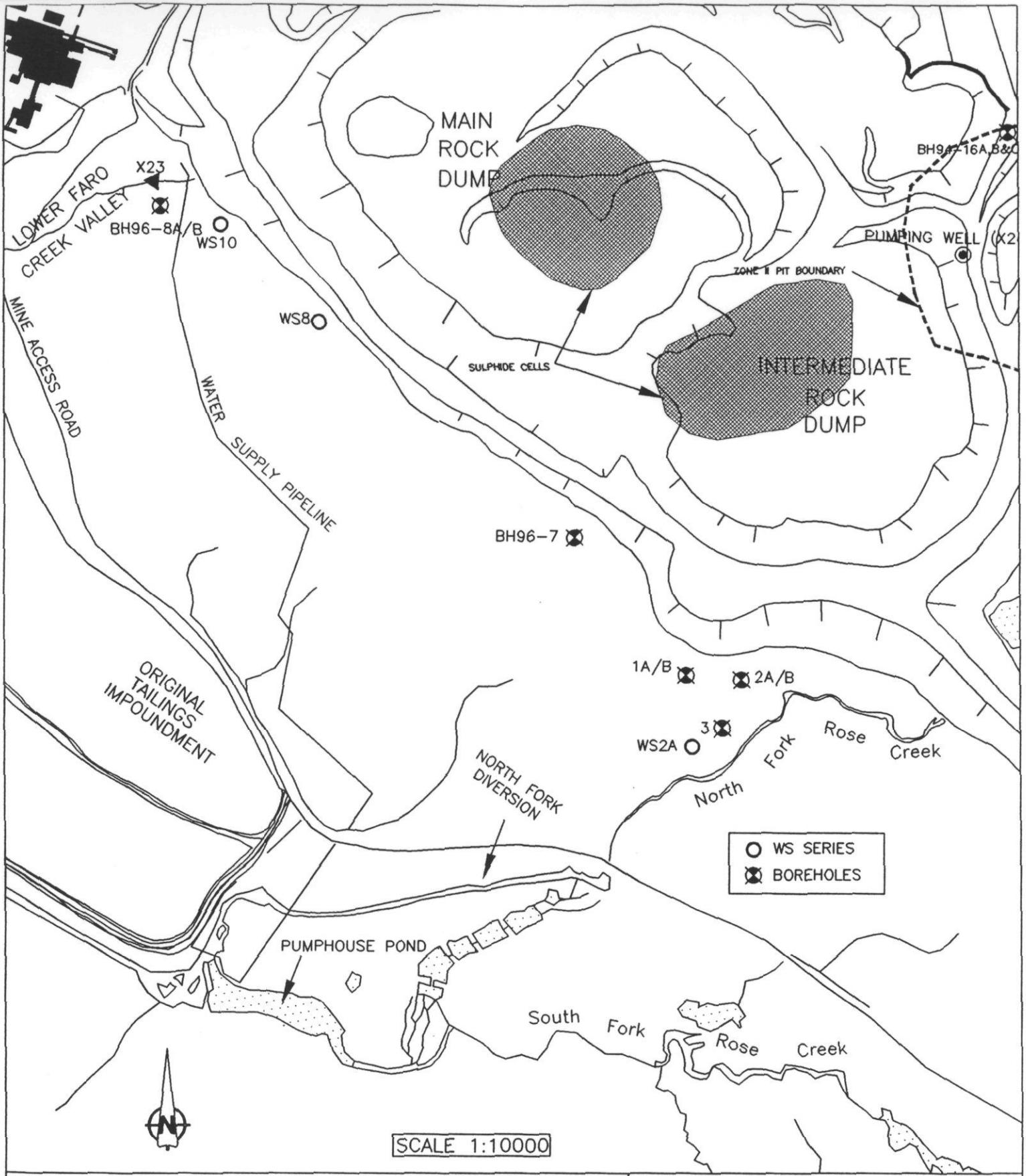


Figure 4-70 Groundwater Quality Trends in Borehole BH4.





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Groundwater Monitoring Sites South of Faro Dumps

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-71

iroLane\4-71FaroWaterPiezometers Wed Nov 27 08:52:57 1996

Figure 4-72a Groundwater Quality Trends in Boreholes 1A and 1B

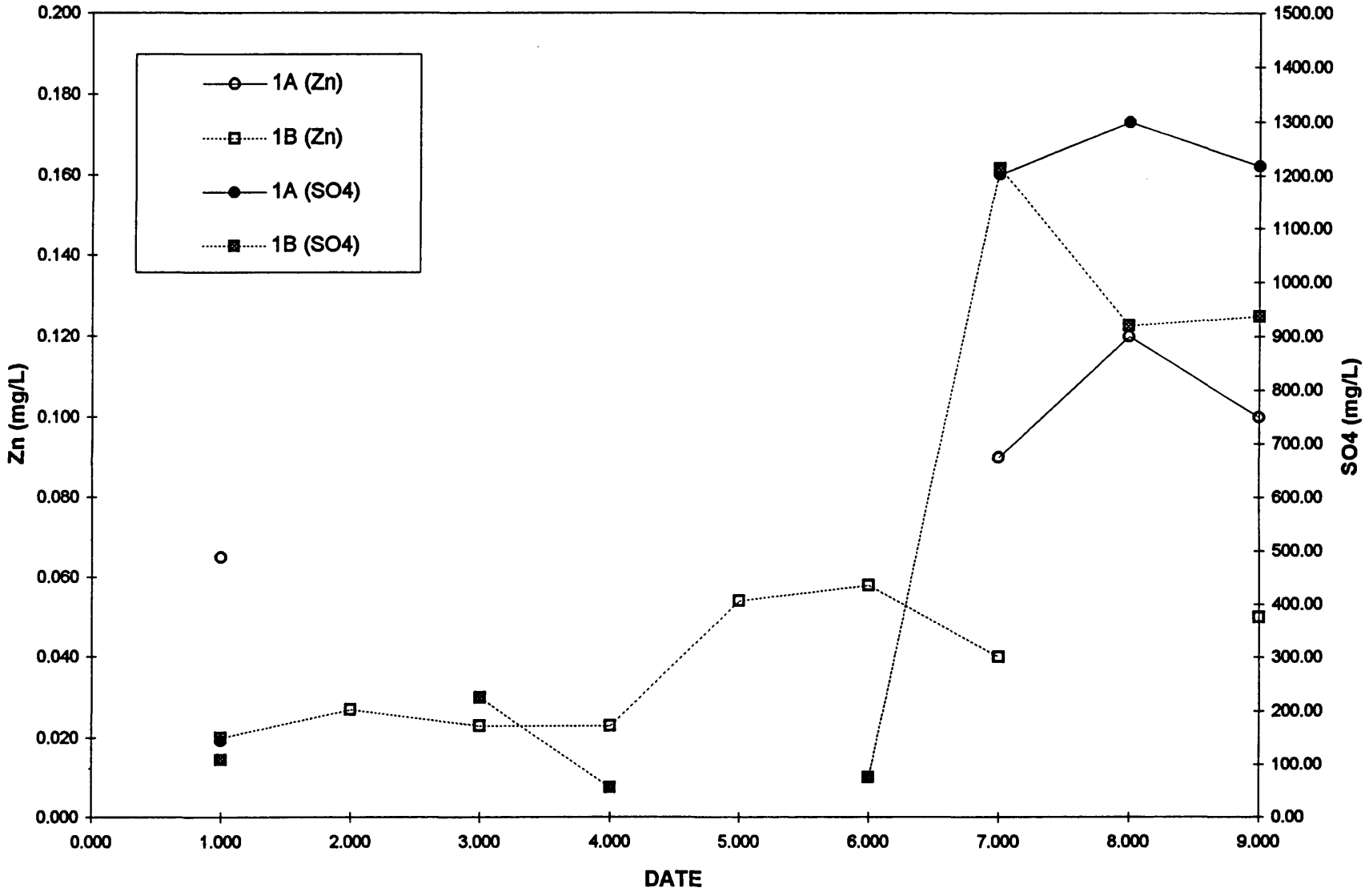
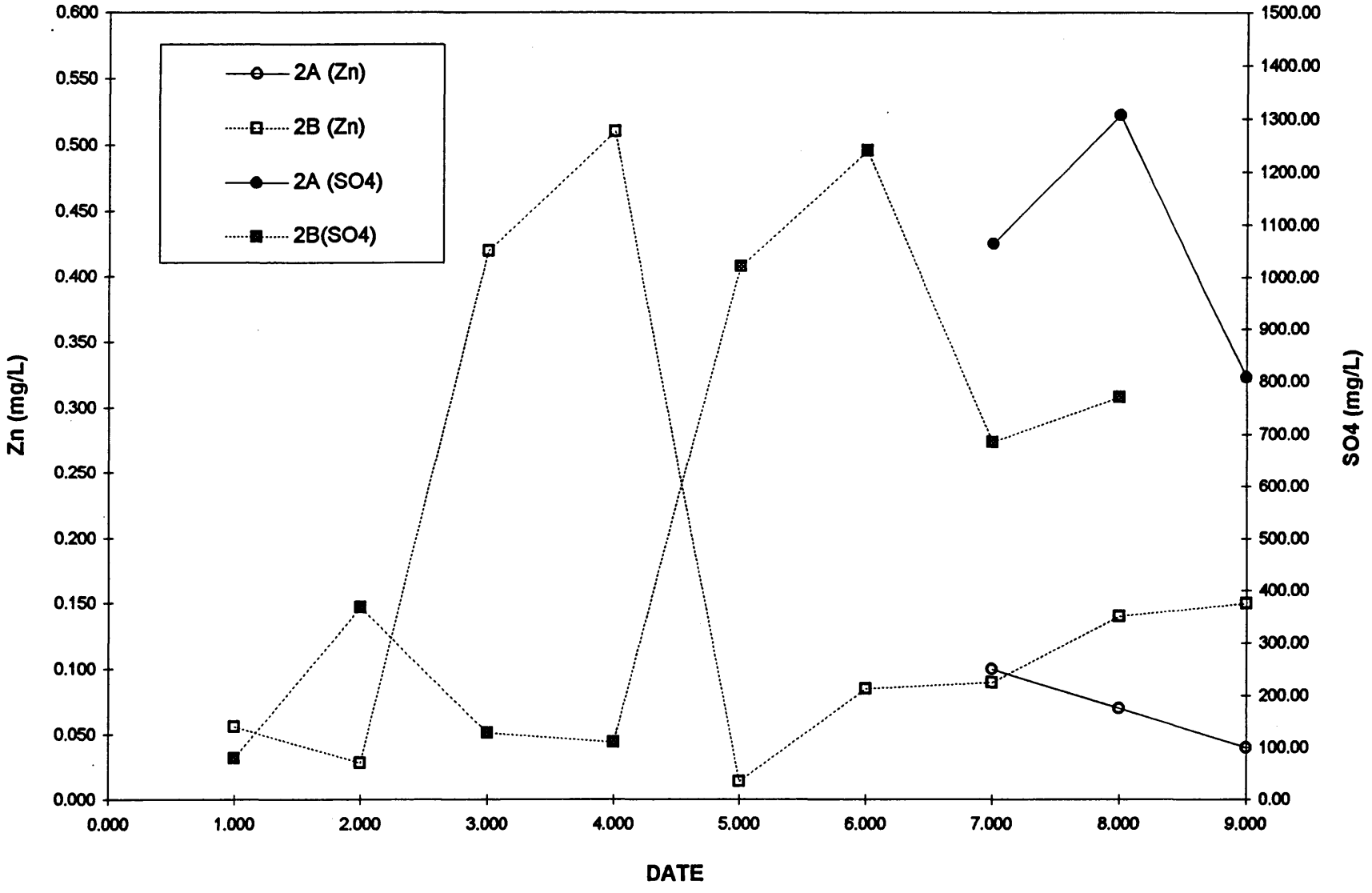
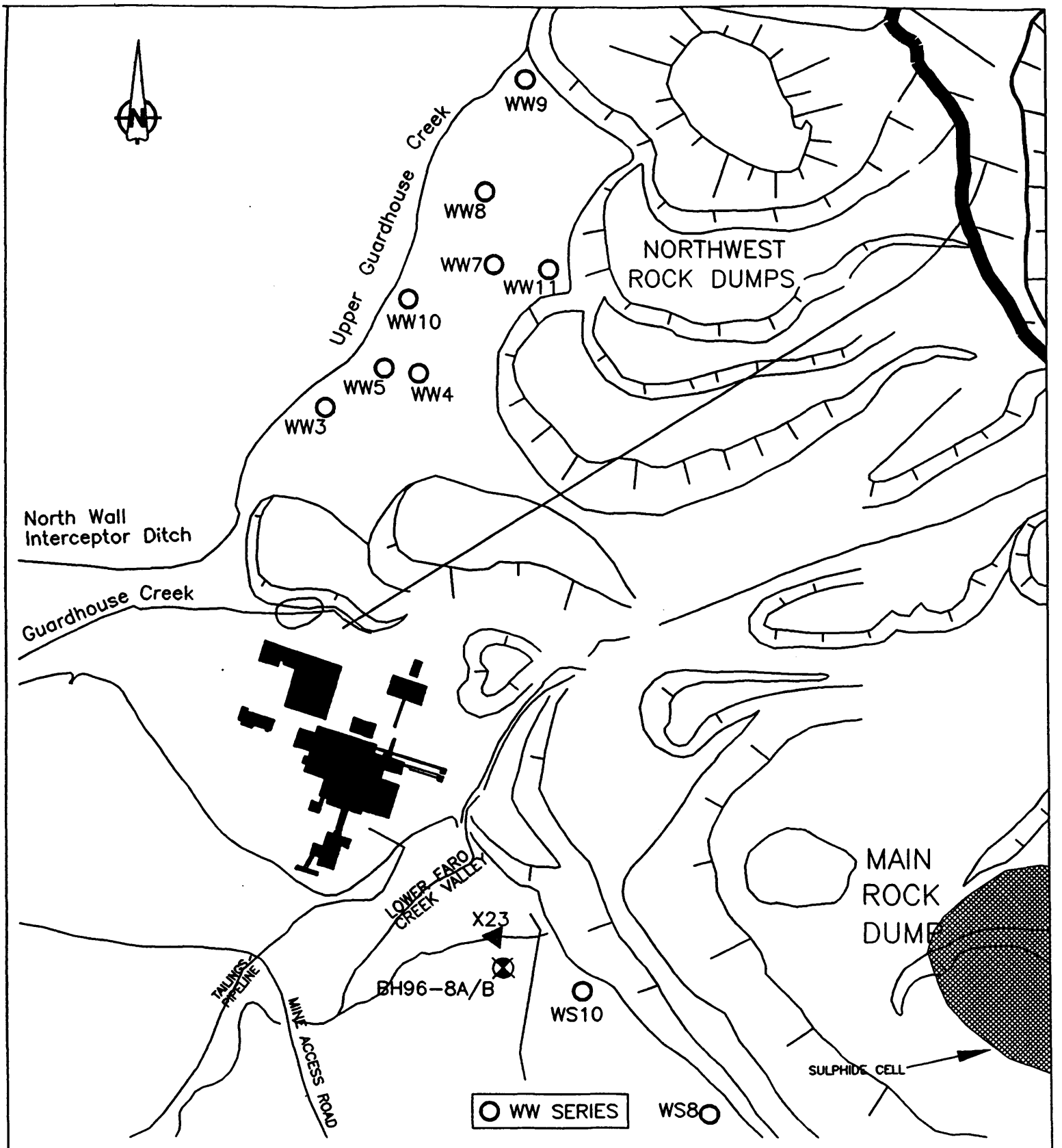


Figure 4-72b Groundwater Quality Trends in Boreholes 2A and 2B





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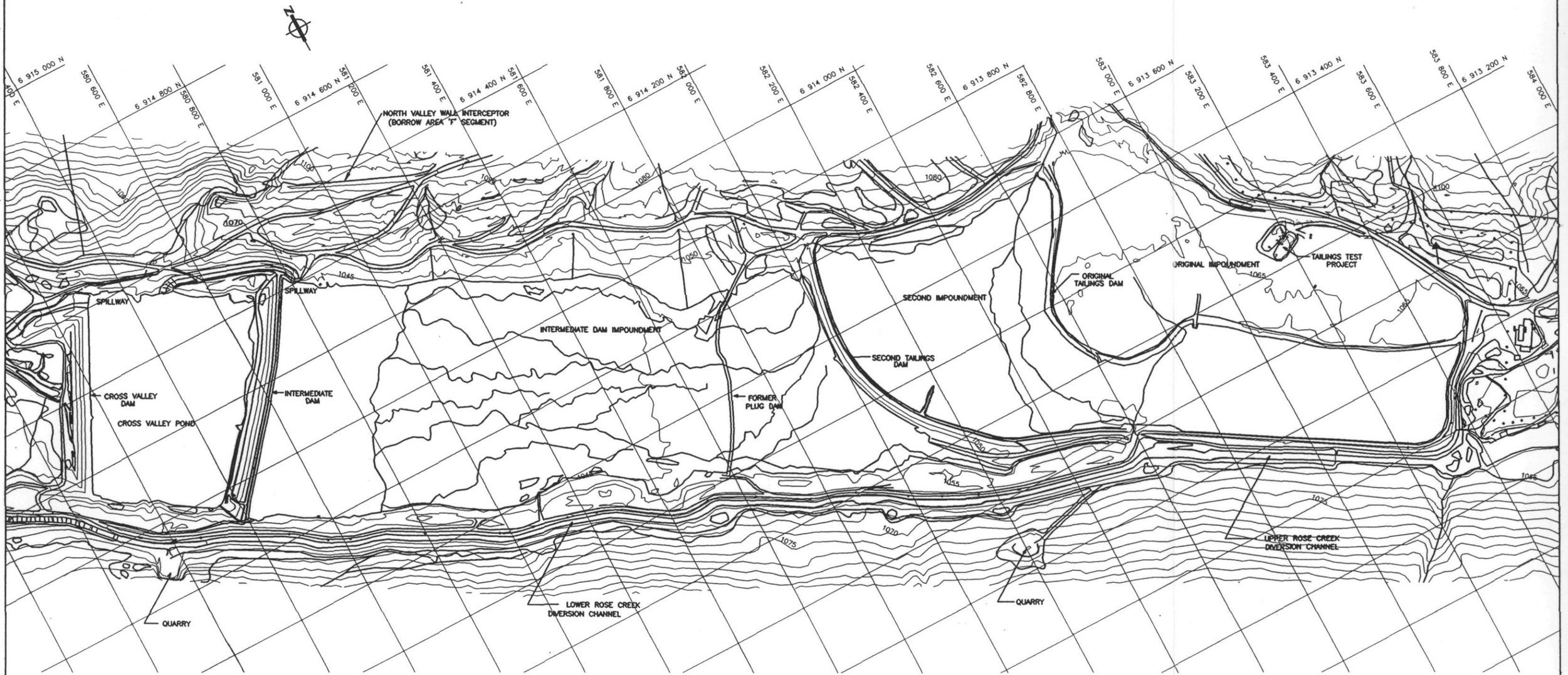
Anvil Range Mining Corporation

Anvil Complex Closure Plan

Groundwater Monitoring Sites West of Faro Dumps

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-73

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MAPPING COMPILED BY THE ORTHOSHOP, CALGARY, ALBERTA, JANUARY, 1991. BASED ON AERIAL PHOTOGRAPHY, SEPTEMBER 17, 1990, AT A SCALE OF 1 : 10 000. SURVEY CONTROL WAS SUPPLIED BY CURRAGH RESOURCES INC.

CONTOUR INTERVAL = 5 METRES.
COORDINATES ARE UTM IN METRES.
ELEVATIONS ARE RELATIVE TO MEAN SEA LEVEL (NAD27)

0 100 200 300 400 Metres

SCALE 1 : 10 000

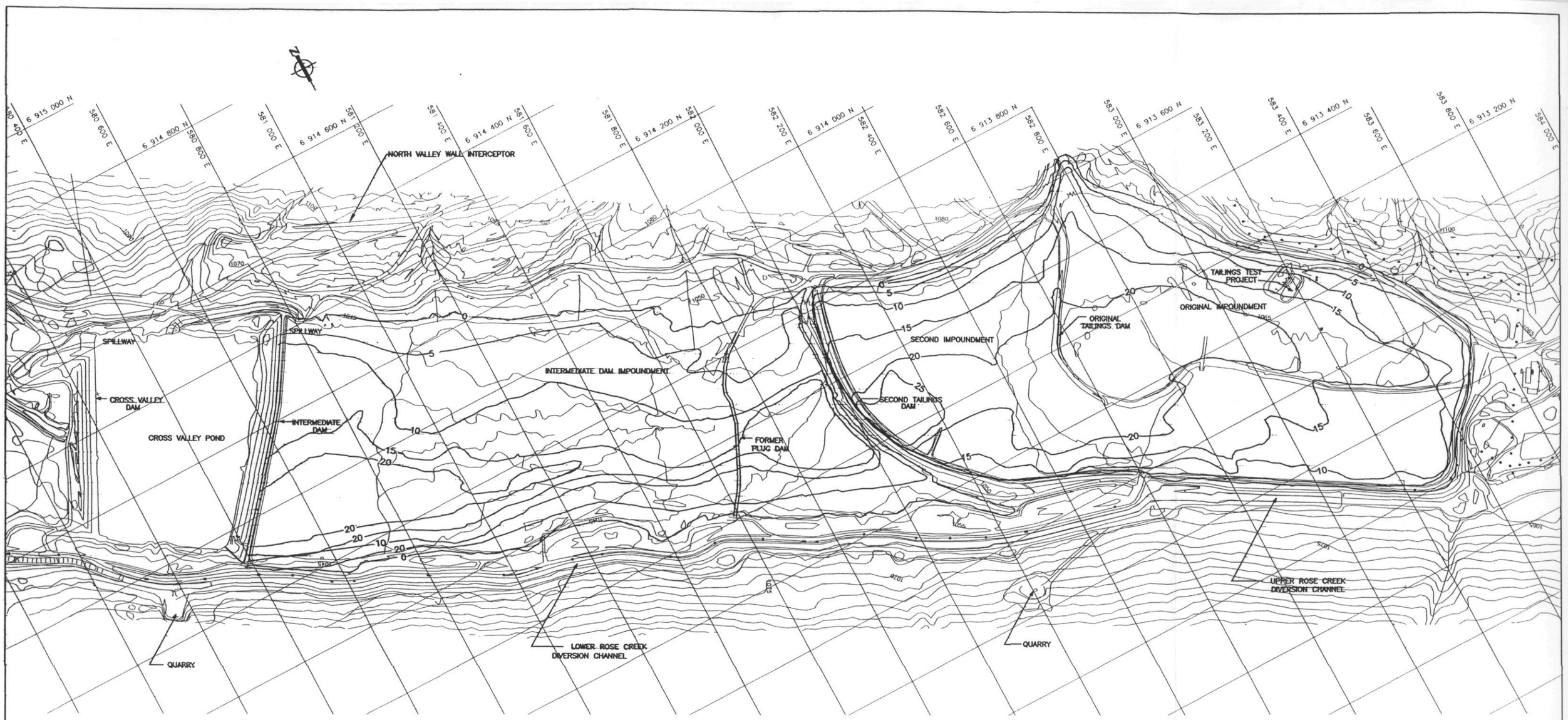
ROBERTSON GEOCONSULTANTS INC.
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Anvil Range Mining Corporation

Anvil Complex Closure Plan

ROSE CREEK TAILINGS FACILITY - OVERVIEW

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-74



MAPPING COMPILED BY THE ORTHOSHOP, CALGARY, ALBERTA, JANUARY, 1991. BASED ON AERIAL PHOTOGRAPHY, SEPTEMBER 17, 1990, AT A SCALE OF 1 : 10 000. SURVEY CONTROL WAS SUPPLIED BY CURRAGH RESOURCES INC.

CONTOUR INTERVAL = 5 METRES.
COORDINATES ARE UTM IN METRES.
ELEVATIONS ARE RELATIVE TO MEAN SEA LEVEL (NAD27)
TAILINGS THICKNESSES SHOWN ARE AS OF SEPT. 17, 1990.

0 100 200 300 400 Metres

SCALE 1 : 10 000

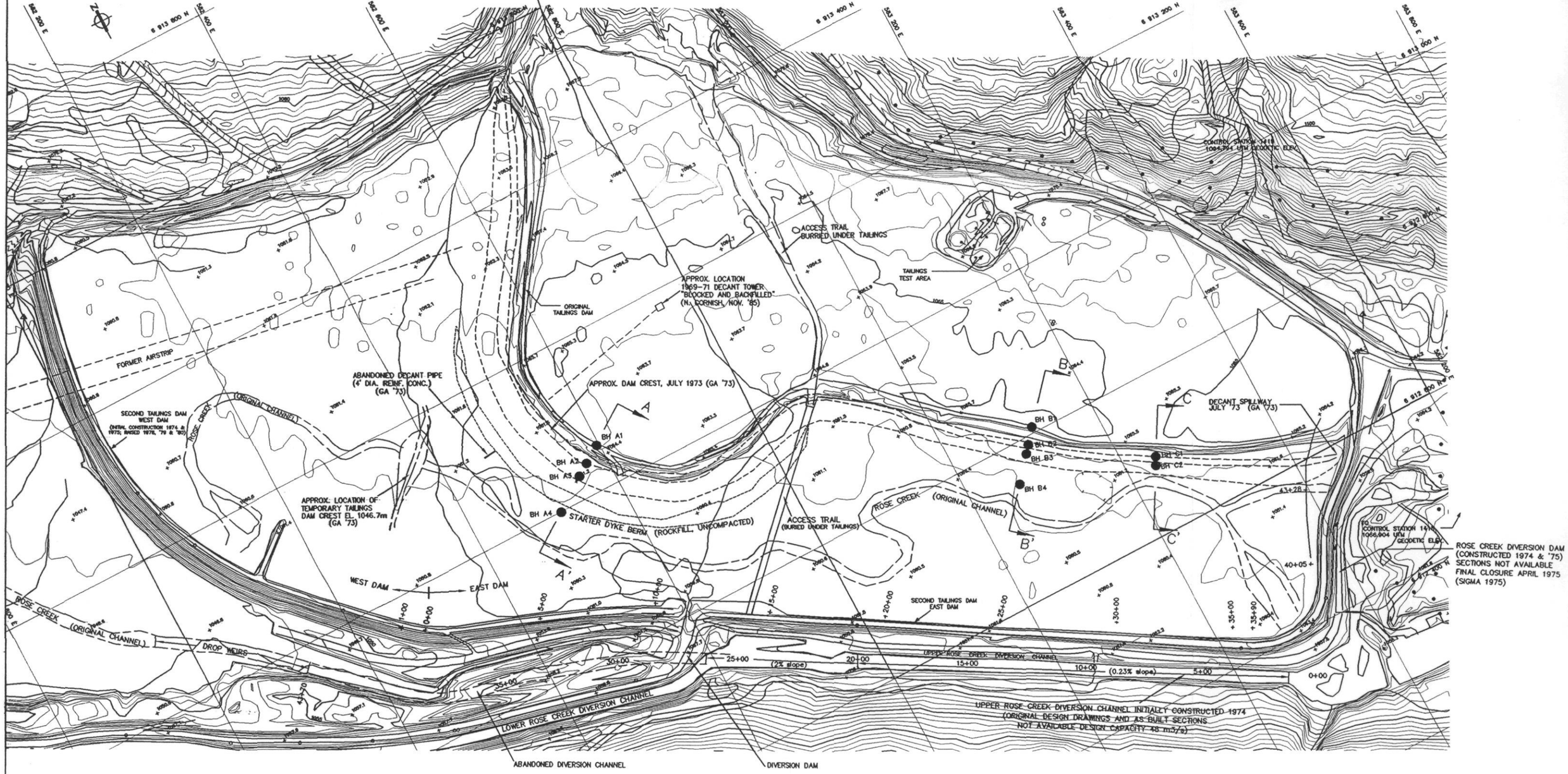
ROBERTSON GEOCONSULTANTS INC.
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Anvil Complex Closure Plan

ROSE CREEK TAILINGS FACILITY TAILINGS THICKNESS ISOPACHS

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-75



PLAN

SCALE 1 : 5 000

MAPPING COMPILED BY THE ORTHOSHOP, CALGARY, ALBERTA, JANUARY, 1991. BASED ON AERIAL PHOTOGRAPHY, SEPTEMBER 17, 1990, AT A SCALE OF 1 : 10 000. SURVEY CONTROL WAS SUPPLIED BY CURRAGH RESOURCES INC.

NOTES

1. STATIONING IS IN FEET AS INDICATED ON GOLDER ASSOCIATES DRAWINGS IN THE REFERENCE LIST.
2. ELEVATIONS ON THIS DRAWING ARE IN METRES AND ARE RELATED TO UTM DATUM. ADD APPROXIMATELY 32.3 METRES TO THE ELEVATIONS TO OBTAIN THE EQUIVALENT ELEVATIONS BASED ON DOWN VALLEY DATUM.
3. FOR CONVENIENCE MINE DATUM ELEVATIONS SHOWN IN BRACKETS (1078) ARE METRIC ELEVATIONS AS USED IN INTERMEDIATE DAM AND CROSS VALLEY DAM 1988 AND 1989 CONSTRUCTION CONTRACT DRAWINGS.
4. CONTOUR INTERVAL = 1m.

REFERENCES

1. GOLDER ASSOCIATES: Rpt. V72337, "REPORT TO ANVIL MINING CORPORATION LIMITED ON THE TAILINGS CONTAINMENT FACILITY, ANVIL MINE, FIGURE 6, "SITE PLAN SHOWING LOCATIONS OF SECTIONS AND BOREHOLES", SEPT. 1973.
2. GOLDER ASSOCIATES: DWG. V74011, "CYPRUS ANVIL MINING CORPORATION, TAILINGS CONTAINMENT CONSTRUCTION, CONSTRUCTION CARRIED OUT TO END OF 1975", SEPT. 1975.
3. GOLDER ASSOCIATES: Rpt. 782-2736, "1978 DYKE RAISING FOR THE TAILINGS RETENTION FACILITY, FARO, YUKON TERRITORY", FIGURE 2, "TAILINGS DAM - 1978 CONSTRUCTION".
4. NEWT CORNISH: "HISTORY OF ANVIL TAILINGS DEPOSITION", NOV. 5, 1985.

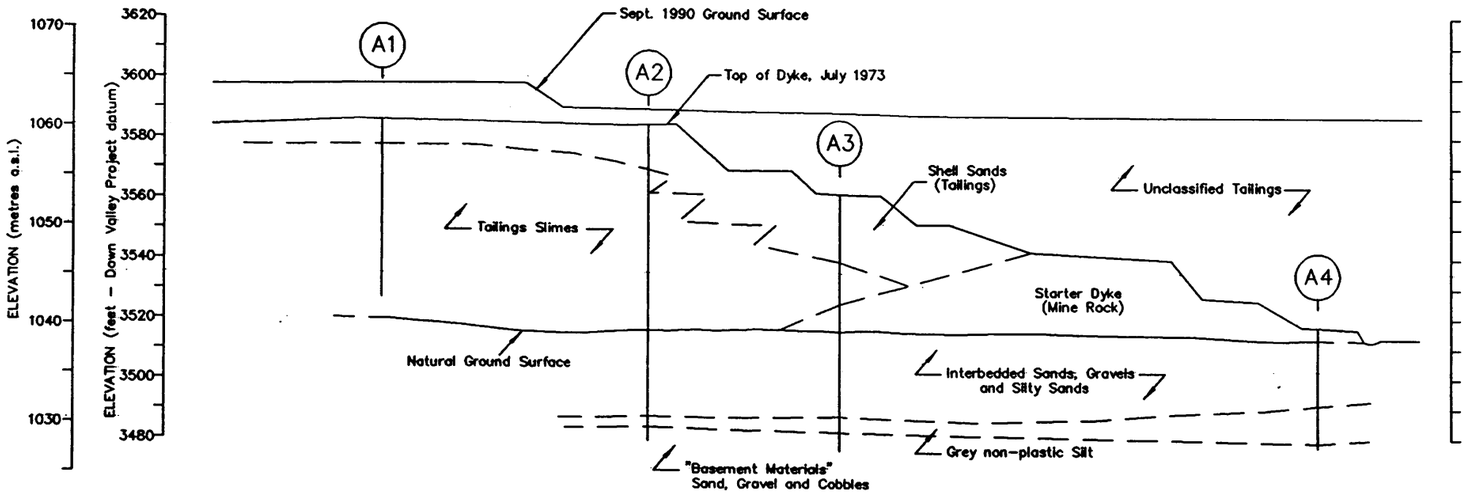
ROBERTSON GEOCONSULTANTS INC.
Consulting Geotechnical and Environmental Engineers

Anvil Range Mining Corporation

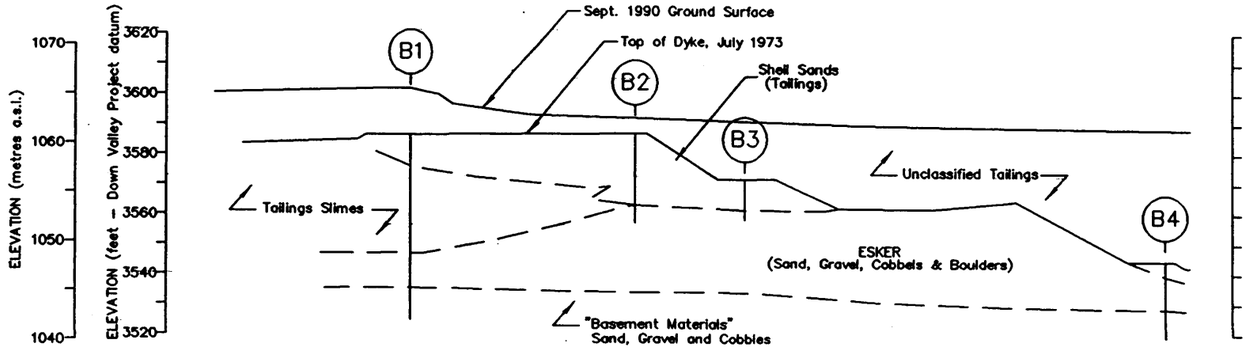
Anvil Complex Closure Plan

Original and Second Tailings Impoundments - Plan

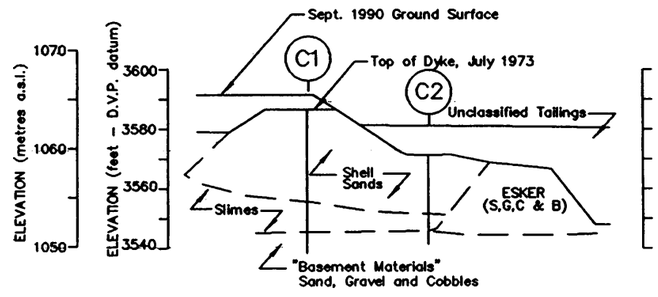
PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-76



SECTION A–A



SECTION B–B



SECTION C–C

LEGEND

(A4) Golder Brawner Associates borehole location, July 1973. (ref. Golder Brawner Associates report V72337, "Report to Anvil Mining Corporation Limited on the Tailings Containment Facility, Anvil Mine, Faro, Yukon", Oct. 1973)

NOTES

1. Stratigraphy indicated has been interpolated from available borehole data. Statigraphy between boreholes may differ from that shown. See Golder Brawner Associates report V72337 for detailed borehole logs.
2. See Figure 4–76 for section locations.

0 20 40 60 80 feet
Scale 1"= 50'

Reference: Golder Associates (1973). Report to Cyprus Anvil Mining Corporation Limited on the Tailings Containment Facility, Oct., Figures 7,8 and 9

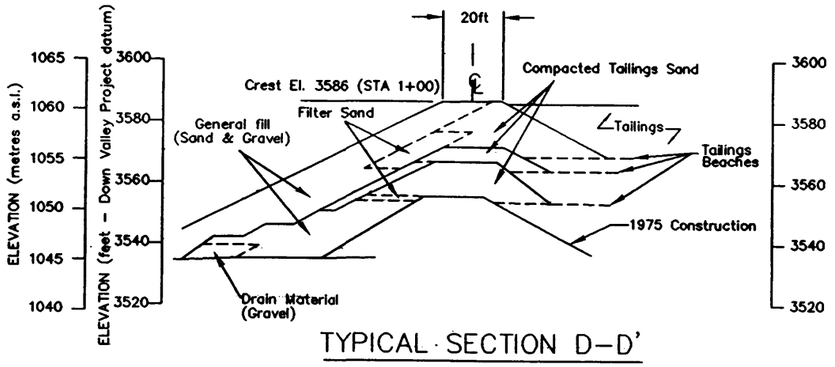
ROBERTSON GEOCONSULTANTS INC.
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Anvil Range Mining Corporation

Anvil Complex Closure Plan

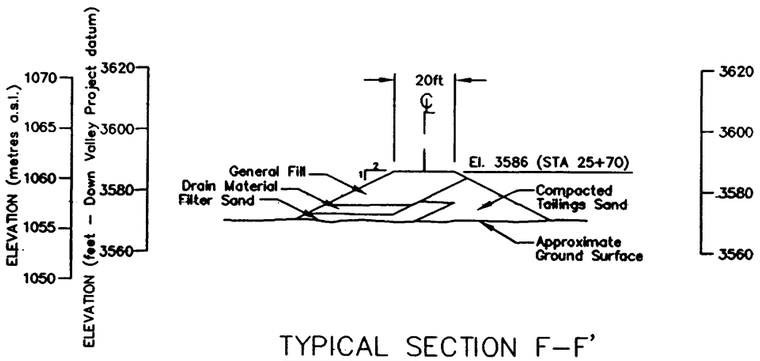
ORIGINAL FARO TAILINGS EMBANKMENT SECTIONS

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4–77



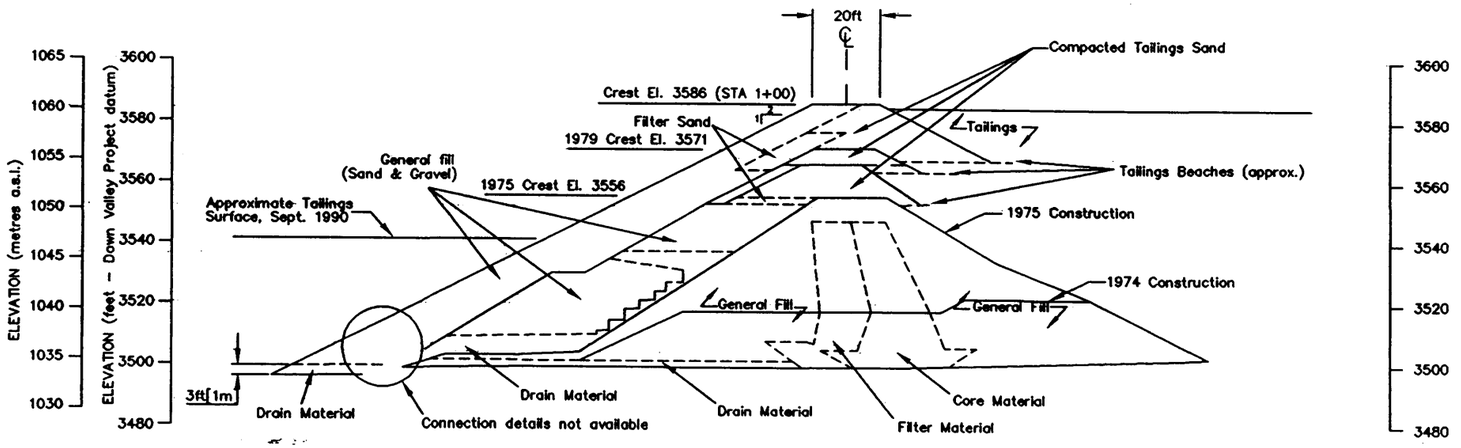
TYPICAL SECTION D–D'

(REPRESENTS STATIONS 0+00 TO 10+00, AND 19+00 TO 26+00)



TYPICAL SECTION F–F'

(REPRESENTS STATIONS 26+00 TO 27+86)



TYPICAL SECTION E–E'

(REPRESENTS STATIONS 10+00 TO 19+00)

REFERENCES

1. Golder, Brawner & Associates Dwg. V74011, "Construction Carried Out to End of 1975"' Sept. 1975.
2. Golder Associates, Report No. 782–2736, "1978 Dyke Raising for the Tailings Retention Facility, Faro, Yukon Territory", April 1979, Figure 2 "Tailings Dam, 1978 Construction".
3. Golder Associates, Report no. 802–2024, "Construction Report, Cyprus Anvil Mining Corporation, 1980 Raising of the Tailings Dam, Faro, Yukon Territory", Appendix II, (Tender Drawings).

NOTES

1. All elevations are in feet relative to the Down Valley Project datum. To convert to sea level datum (NAD 27), subtract 106ft (32.3m). To convert from feet to metres, multiply by 0.3048m/ft.

0 20 40 60 80 feet

Scale 1"= 50'

ROBERTSON GEOCONSULTANTS INC.

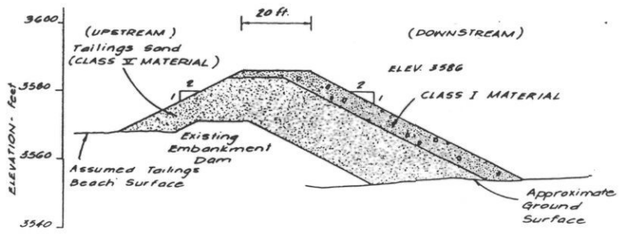
Consulting Geotechnical and Environmental Engineers

Anvil Range Mining Corporation

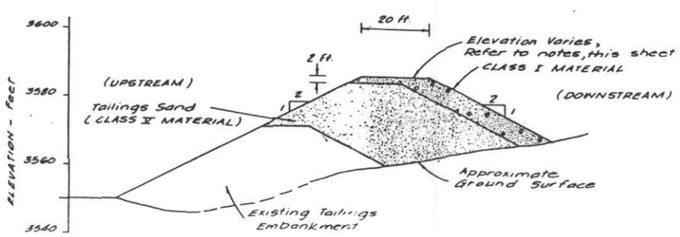
Anvil Complex Closure Plan

ROSE CREEK TAILINGS FACILITY
SECOND IMPOUNDMENT - WEST DAM
TYPICAL SECTIONS

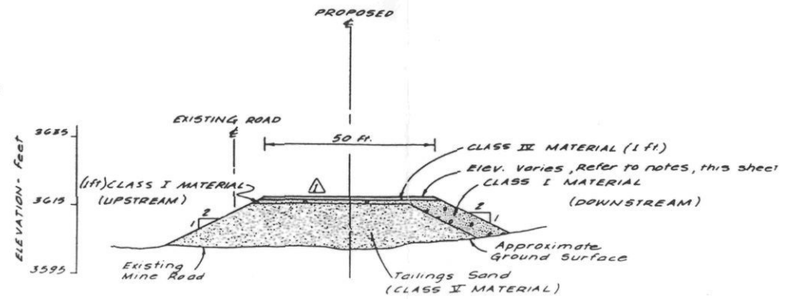
PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov 1996		4–78



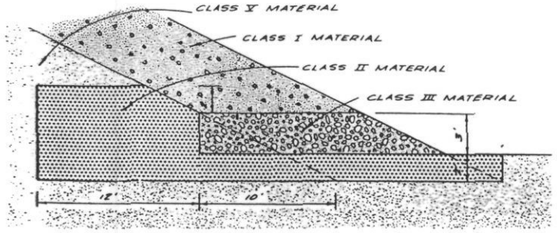
TRANSITION ZONE E-E'
TYPICAL SECTION
(REPRESENTS STATIONS 0+00 to 6+30)



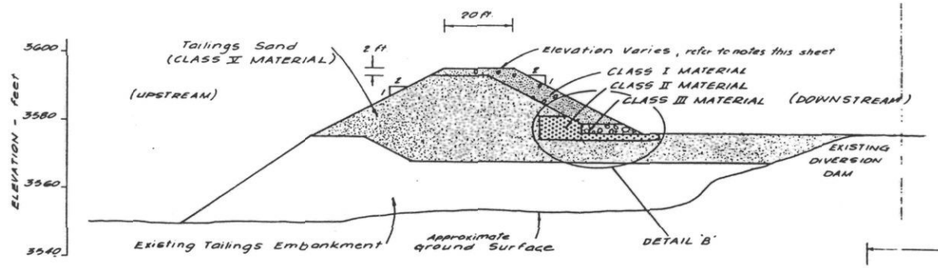
TYPICAL SECTION F-F'
(REPRESENTS STATIONS 6+30 to 28+00
AND 46+00 to 51+00)



TYPICAL SECTION H-H'
(REPRESENTS STATIONS 51+00 to 61+30)



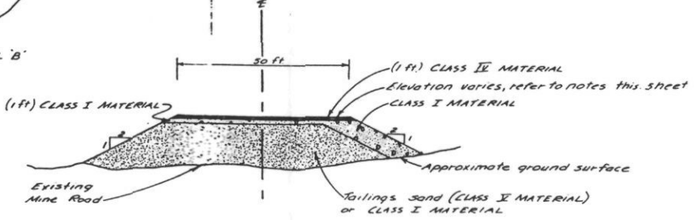
DETAIL 'B'



TYPICAL SECTION G-G'
(REPRESENTS STATIONS 28+00 to 46+00)

△2 Mine road elevations
316, 4100 S at El. 3574.45
to 314. 1700 S (Station E) at El. 3606.65 > at 3.4 % slope

11. Construct new mine road 316 4100 S to 59+65 E so that surface, crown and superelevation conform with the curvature and sloping of the existing mine road profile.



△2 TYPICAL SECTION I-I'
(Represents stations 1+37.5 (49+60 E) to 41+00 S on existing mine road toward Faro.)

NOTES

1. East Dam Station 0+00 to 6+30 is a transition zone which evenly shifts the centerline construction of the West Dam to downstream construction on the East Dam.
2. East Dam crest elevations:

   Station 0+00 at El. 3586 feet
   to Station 20+00 at El. 3586 feet > at 0.40% slope
   to Station 30+00 at El. 3590 feet > at 0.67 % slope △2
   to Station 45+00 at El. 3600 feet
   to End of Construction at El. 3620 feet > at 1.33% slope

3. Area beneath proposed new construction Foundation to be cleared of all deleterious materials and proof rolled with a heavy vibrating roller to the satisfaction of the engineer. Areas found to be unsuitable are to be subexcavated and replaced with Class I material compacted to 100% of the Standard Proctor Maximum dry density at optimum moisture content.
4. Materials used in 1980 construction to be compacted in 12" loose lift thickness to 100% of the standard Proctor maximum dry density at the optimum moisture content.
5. Class III material not to be placed in direct contact with Class X material without a minimum 1 foot thick layer of Class II filter material in between.
6. All sections looking in direction of increasing Station numbers.
7. All distances and elevations are subject to final field survey.
8. Top surface of existing embankment dams to be scarified for a depth of 0.5 feet prior to placement of new fill.

△2 9. Construct new dam top road widths as follows so as to conform with existing mine road width at 60+00 E

Station	Width (ft)	Station	Width (ft)
50+00 E	50	54+00 E	44
51+00 E	48	57+00 E	43
52+00 E	48	58+00 E	42
53+00 E	47	59+00 E	41
54+00 E	46	59+65 E	40.35
55+00 E	45		

DRAWING TAKEN FROM:

Scale: 1 inch to 20 feet	CYPRUS ANVIL Faro Yukon Territory Cyprus Anvil Mining Corporation	TYPICAL EAST DAM SECTIONS	Golder Associates Golder Geotechnical Consultants Ltd.
Job No. 792-2041			
Drawing No. 5			
Drawn by R.W. / Reviewed by			
Date April 1980 / Date April 1980		1980 EMBANKMENT DAM RAISING	Sheet 5 of 5

NOTES

1. ALL ELEVATIONS ARE IN FEET RELATIVE TO DOWN VALLEY PROJECT DATUM. SUBTRACT 106 FT, AND MULTIPLY BY 0.348 TO CONVERT TO METRES ABOVE MEAN SEA LEVEL.

DISCLAIMER

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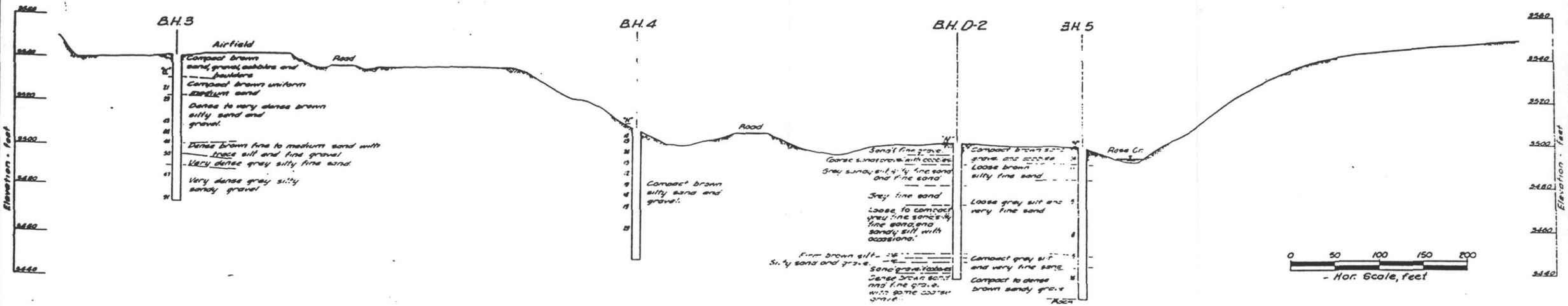
ROBERTSON GEOCONSULTANTS INC.
Consulting Geotechnical and Environmental Engineers

Anvil Range Mining Corporation

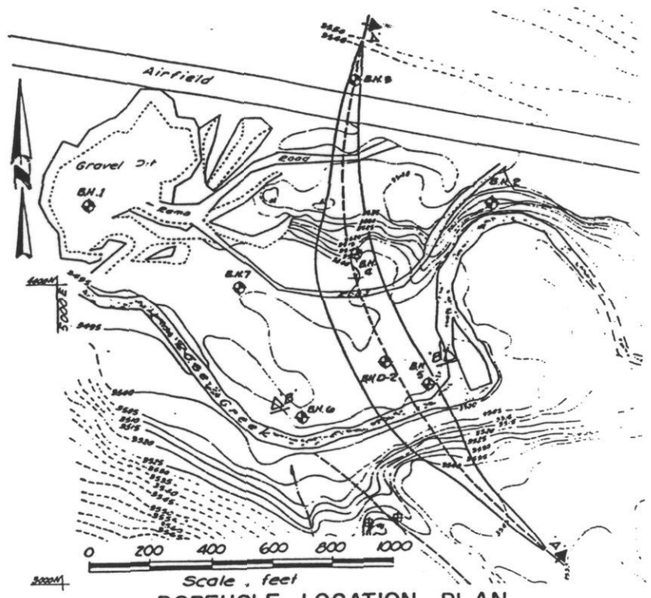
Anvil Complex Closure Plan

SECOND TAILINGS IMPOUNDMENT
EAST DAM SECTIONS
(after Golder Associates)

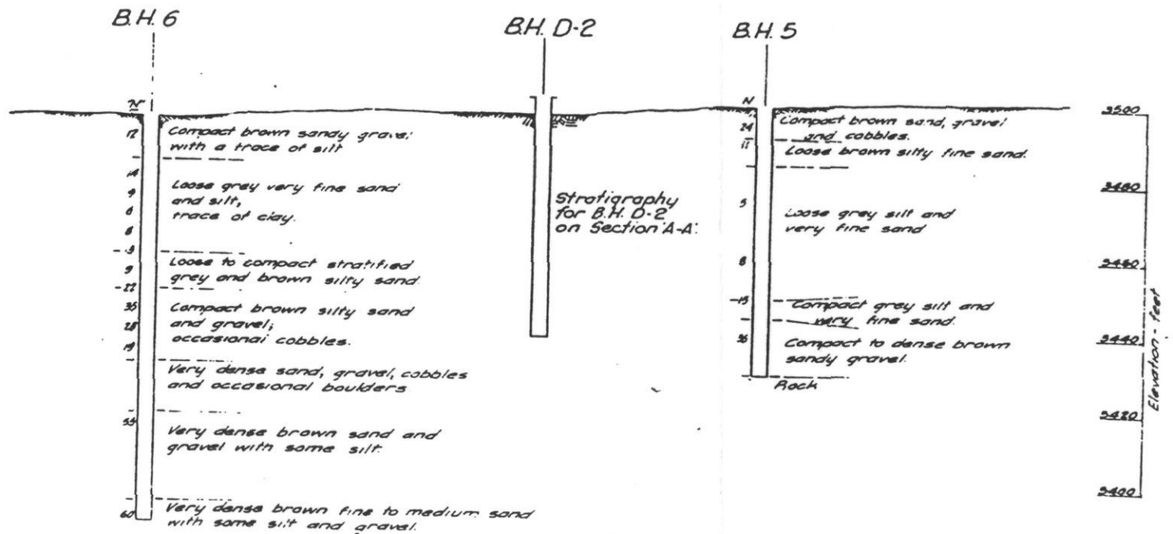
PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-79



SECTION A-A



BOREHOLE LOCATION PLAN



SECTION B-B

DISCLAIMER

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NOTE: REDUCED FROM ORIGINAL GOLDER ASSOCIATES DRAWING, ENTITLED "FIGURE 5, DOWNSTREAM DAMSITE, STRATIGRAPHIC AND MATERIALS DETAILS". REPORT REFERENCE UNKNOWN.

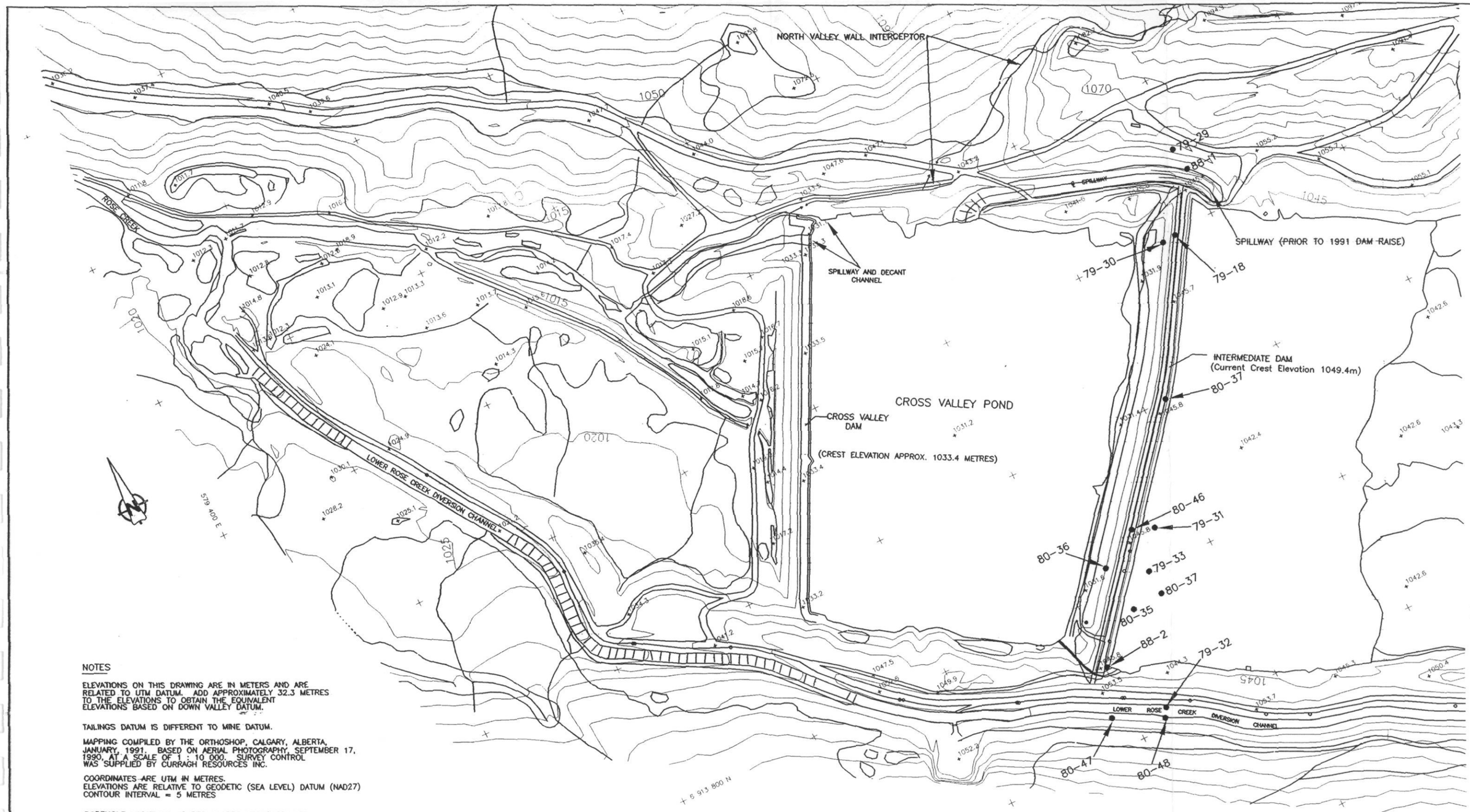
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Anvil Range Mining Corporation

Anvil Complex Closure Plan

SECOND TAILINGS IMPOUNDMENT WEST DAM FOUNDATION CONDITIONS

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-80



NOTES

ELEVATIONS ON THIS DRAWING ARE IN METERS AND ARE RELATED TO UTM DATUM. ADD APPROXIMATELY 32.3 METRES TO THE ELEVATIONS TO OBTAIN THE EQUIVALENT ELEVATIONS BASED ON DOWN VALLEY DATUM.

TAILINGS DATUM IS DIFFERENT TO MINE DATUM.

MAPPING COMPILED BY THE ORTHOSHOP, CALGARY, ALBERTA, JANUARY, 1991. BASED ON AERIAL PHOTOGRAPHY, SEPTEMBER 17, 1990, AT A SCALE OF 1 : 10 000. SURVEY CONTROL WAS SUPPLIED BY CURRAGH RESOURCES INC.

COORDINATES ARE UTM IN METRES.
ELEVATIONS ARE RELATIVE TO GEODETIC (SEA LEVEL) DATUM (NAD27)
CONTOUR INTERVAL = 5 METRES

BOREHOLE LOCATIONS AS PER GOLDER ASSOCIATES LTD. DRAWING NUMBER 912-2402-2, JULY 1991.

0 80 160 240 320 400 METRES

SCALE 1:5000

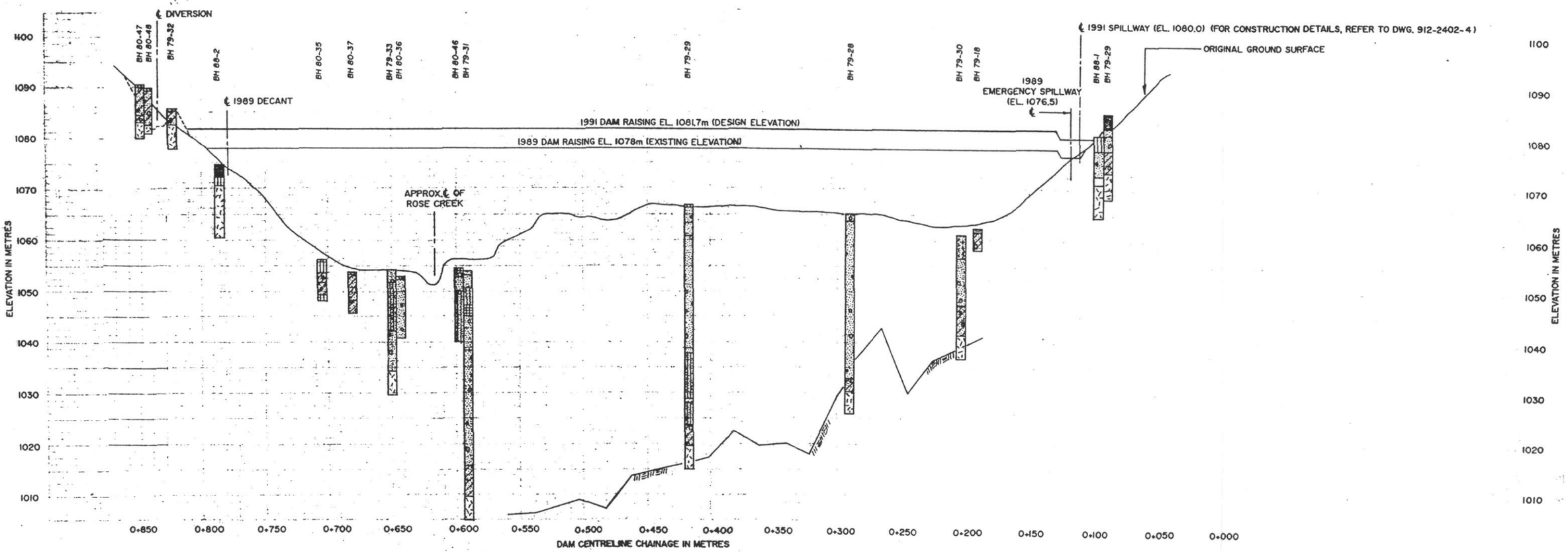
ROBERTSON GEOCONSULTANTS INC.
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Anvil Range Mining Corporation

Anvil Complex Closure Plan

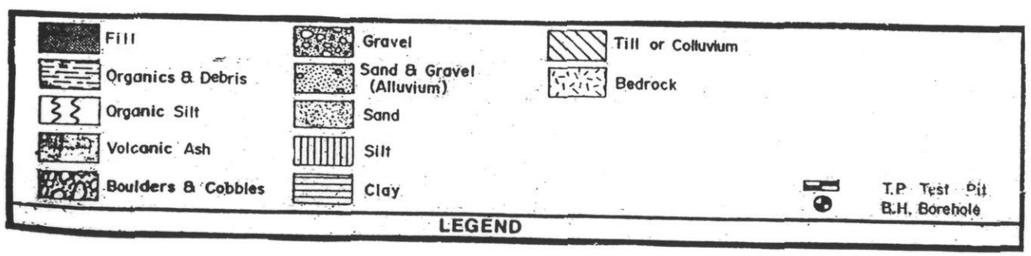
INTERMEDIATE & CROSS VALLEY DAMS - OVERVIEW -

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-81



SECTION A – A'

SCALE: 1 : 3000 HORIZONTAL
1 : 750 VERTICAL



DRAWING TAKEN FROM:

NOTES

1. ALL ELEVATION ARE REFERENCED TO THE DOWN VALLEY PROJECT DATUM. SUBTRACT 32.3 m FROM ELEVATIONS SHOWN TO CORRECT TO SEA LEVEL DATUM.

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PROFESSIONAL ENGINEER H.G. GILCHRIST YUKON TERRITORY

Scale: as shown	
Job No. 912-2402	
Drawing No. 912-2402-2	
Drawn by RK	Reviewed by PW/HGG
Date JUL 91	Date JUL 91

CURRAGH RESOURCES INC.
Faro
Yukon Territory

INTERMEDIATE DAM
1991 RAISING
PLAN AND PROFILE

INT. DAM RAISING & CV DAM TOE DRAIN...

Golder Associates Ltd.

Sheet 2 of 8

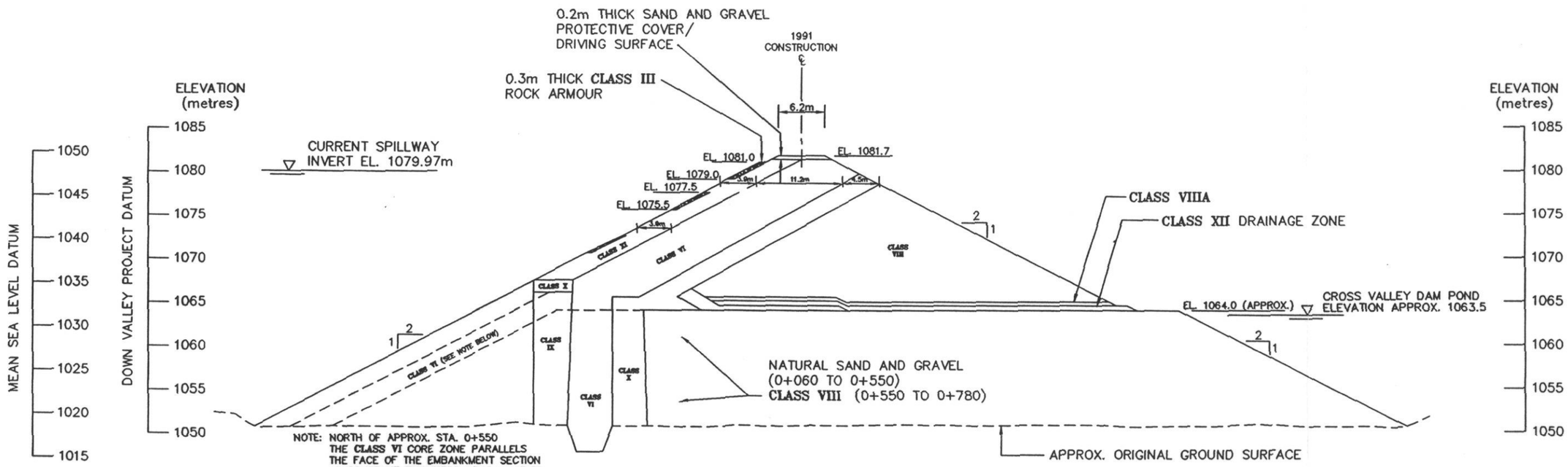
ROBERTSON GEOCONSULTANTS INC.
Consulting Geotechnical and Environmental Engineers

Anvil Range Mining Corporation

Anvil Complex Closure Plan

INTERMEDIATE DAM LONGITUDINAL PROFILE

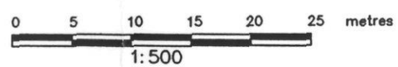
PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-82



NOTES :

1. Embankment geometry and internal zoning as shown in Golder Associates Drawing 912–2402–3, Int. Dam Raising & C.V. Dam Toe Drain, Cross Section and Detailed Plan, Rev. 1, Aug. 8, 1991.
2. All elevations are referenced to Down Valley Project Datum. Subtract 32.3m from elevations shown to convert to mean sea level (NAD27) datum.
3. Refer to Golder Associates as built reports for detailed descriptions of material classes. General descriptions as follows :

CLASS VI	Dam Core (glacial till)
CLASS VII	Upstream Shell (silty sand and gravel)
CLASS VIII	Downstream Shell (sand and gravel)
CLASS VIIIA	Drainage Filter (sand and gravel)
CLASS IX	Upstream Filter (silty sand)
CLASS X	Downstream Filter (sand and gravel)
CLASS XI	Tailings Sand (fine to medium sand)
CLASS XII	Drainage Zone (gravel)



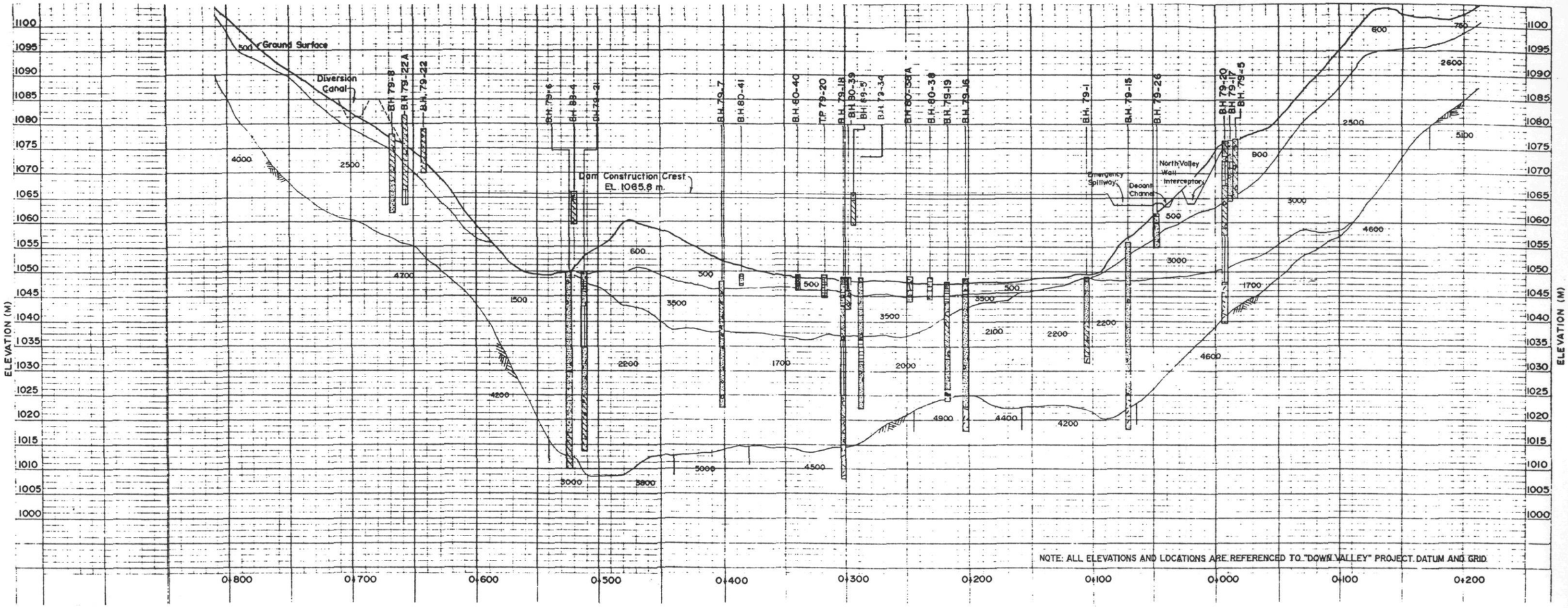
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Anvil Range Mining Corporation

Anvil Complex Closure Plan

INTERMEDIATE DAM CURRENT CONFIGURATION

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4–83



SECTION A – A'

SCALE: 1 : 3000 HORIZONTAL
1 : 750 VERTICAL

Fill
Organics & Debris
Organic Silt
Volcanic Ash
Boulders & Cobbles
Gravel
Sand & Gravel (Alluvium)
Sand
Silt
Clay
Till or Colluvium
Bedrock
Seismic Boundary
T.P. Test Pit
B.H. Borehole
Seismically inferred bedrock surface
2000 Denotes seismic velocity m/sec.

LEGEND

DRAWING TAKEN FROM:

NOTES

1. PLAN AND PROFILE "BASE PLANS" HAVE BEEN TAKEN FROM 1980 DOWN VALLEY TAILINGS PROJECT CONSTRUCTION.

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PROFESSIONAL ENGINEER H.G. GILCHRIST YUKON TERRITORIES

Scale: as shown	
Job No. 912-2402	
Drawing No. 912-2402-6	
Drawn by RK/CG	Reviewed by LFG/HGG
Date JUL 91	Date JUL 91

CURRAGH RESOURCES INC.
Faro
Yukon Territory

CROSS VALLEY DAM
PLAN AND PROFILE
INT. DAM RAISING & CV DAM TOE DRAIN

Golder Associates Ltd.

Sheet 6 of 8

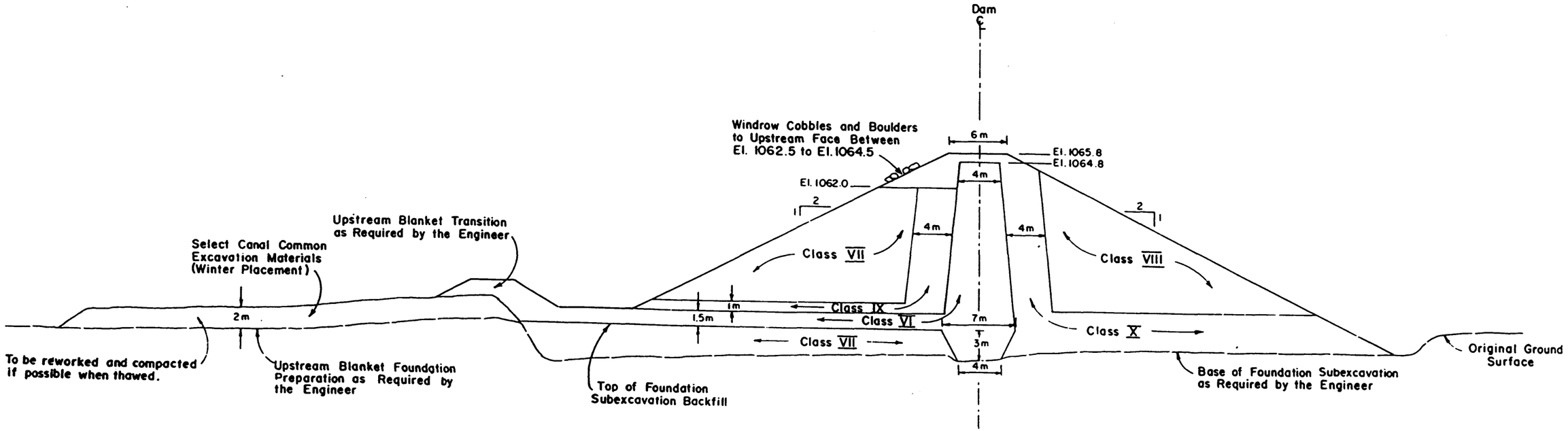
ROBERTSON GEOCONSULTANTS INC.
Consulting Geotechnical and Environmental Engineers

Anvil Range Mining Corporation

Anvil Complex Closure Plan

CROSS VALLEY DAM LONGITUDINAL PROFILE

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-84a



CROSS VALLEY DAM TYPICAL CROSS - SECTION

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DRAWING TAKEN FROM:

	Scale 1:300 Job No 792-2025 Drawing No 7922025-9 Drawn by W.T. / Reviewed by D.G. Date August 1980 / Date August 1980	CYPRUS ANVIL Faro Yukon Territory Cyprus Anvil Mining Corporation	CROSS VALLEY DAM AND SPILLWAYS CROSS - SECTIONS	Golder Associates Golder Geotechnical Consultants Ltd. Sheet 9 of 12

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Consulting Geotechnical and Environmental Engineers

Anvil Range Mining Corporation

Anvil Complex Closure Plan

CROSS VALLEY DAM TYPICAL SECTION

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-84b

Figure 4-85
Total zinc time trend, Cross Valley Dam decant

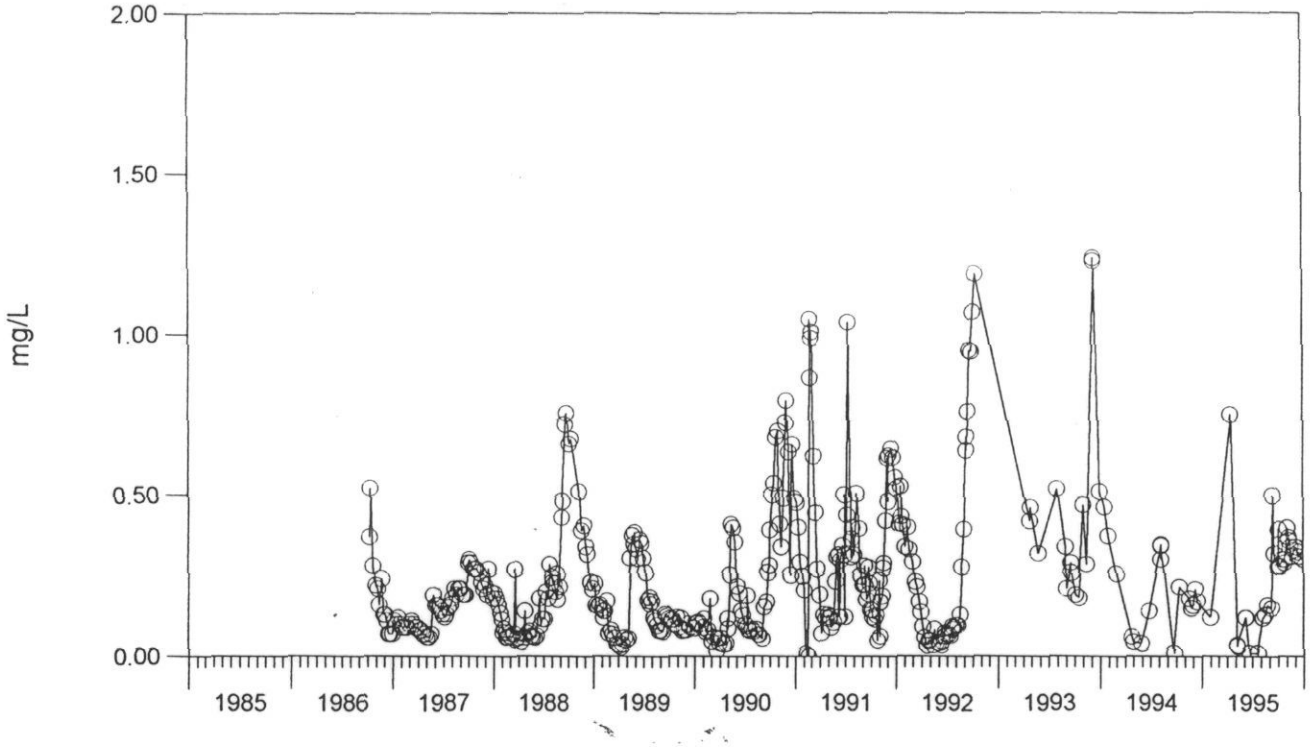


Figure 4-86 pH time trend, from north and south toes of the Cross Valley Dam and combined seepage from the Cross Valley Dam

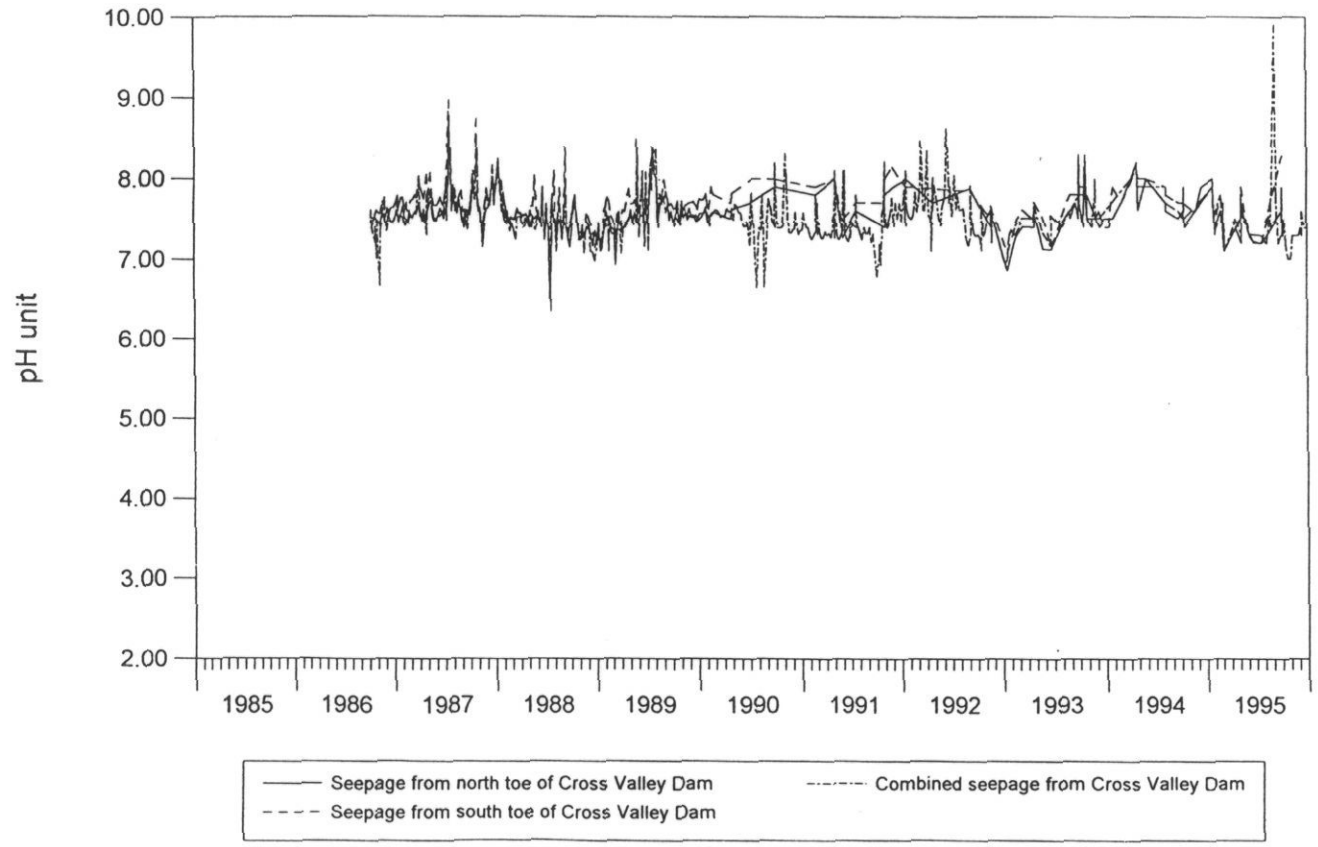


Figure 4-87 Alkalinity time trend, from north and south toes of the Cross Valley Dam and *combined seepage from the Cross Valley Dam*

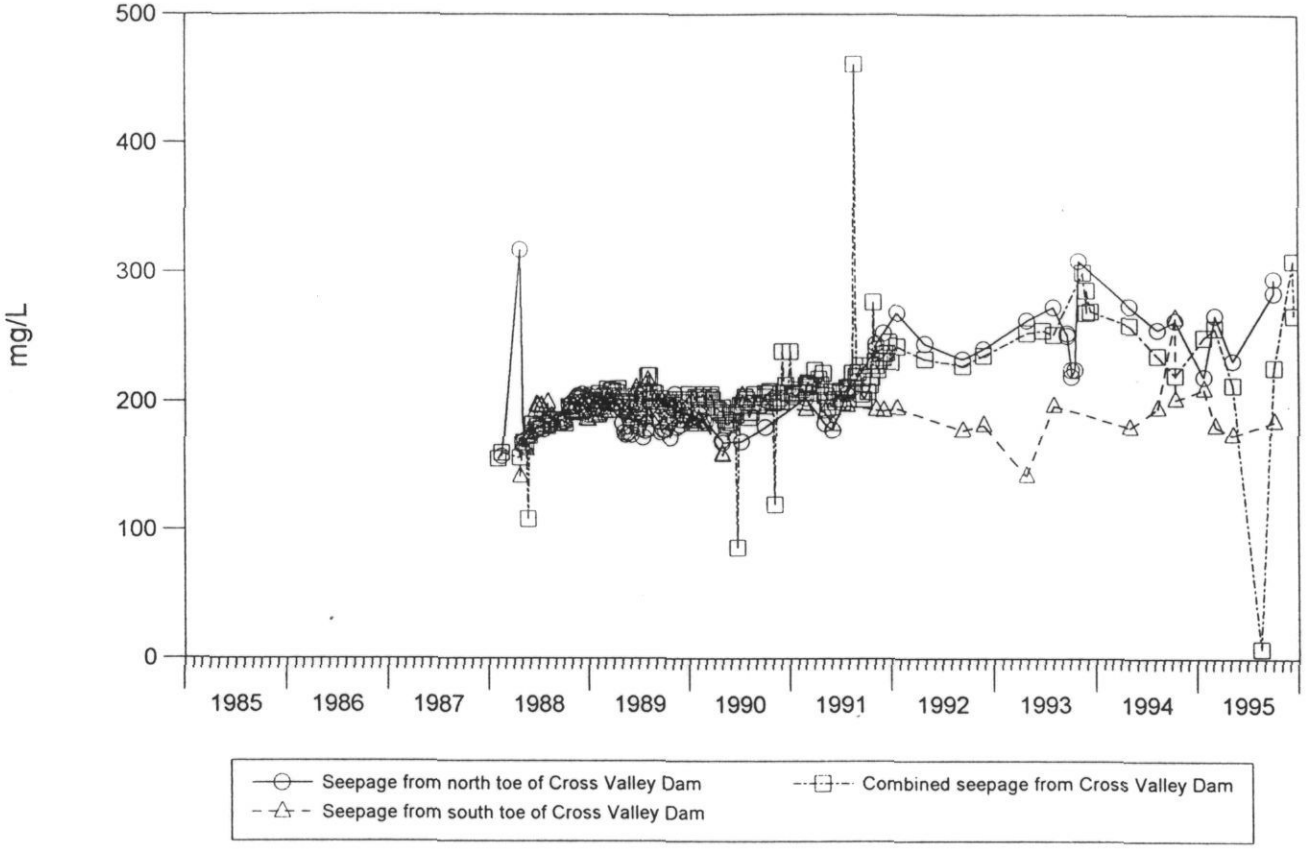


Figure 4-88
(SO4/(Ca+Mg)) time trend, from north and south toes of the Cross Valley Dam

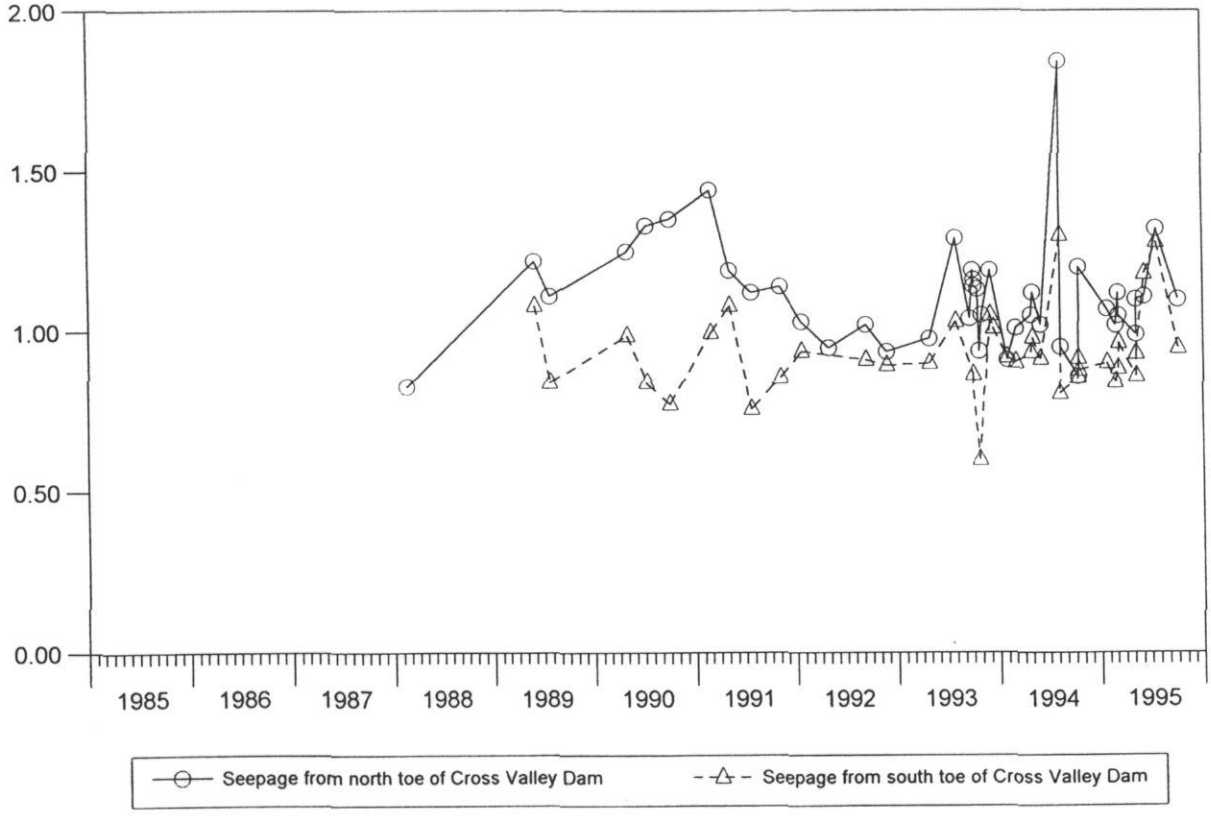


Figure 4-89 Total iron time trend, from north and south toes of the Cross Valley Dam and combined seepage from the Cross Valley Dam

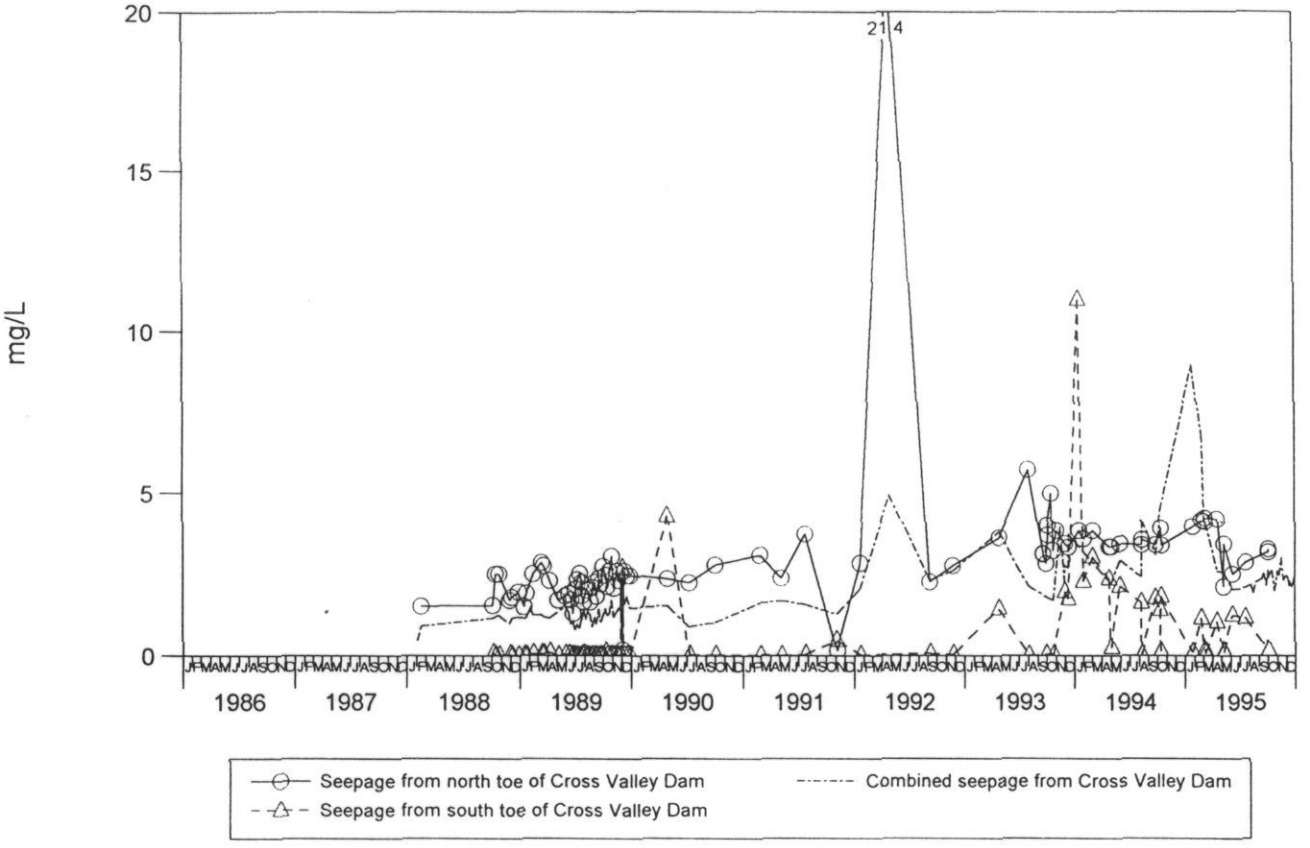


Figure 4-90 Total zinc time trend, from north and south toes of the Cross Valley Dam and combined seepage from the Cross Valley Dam

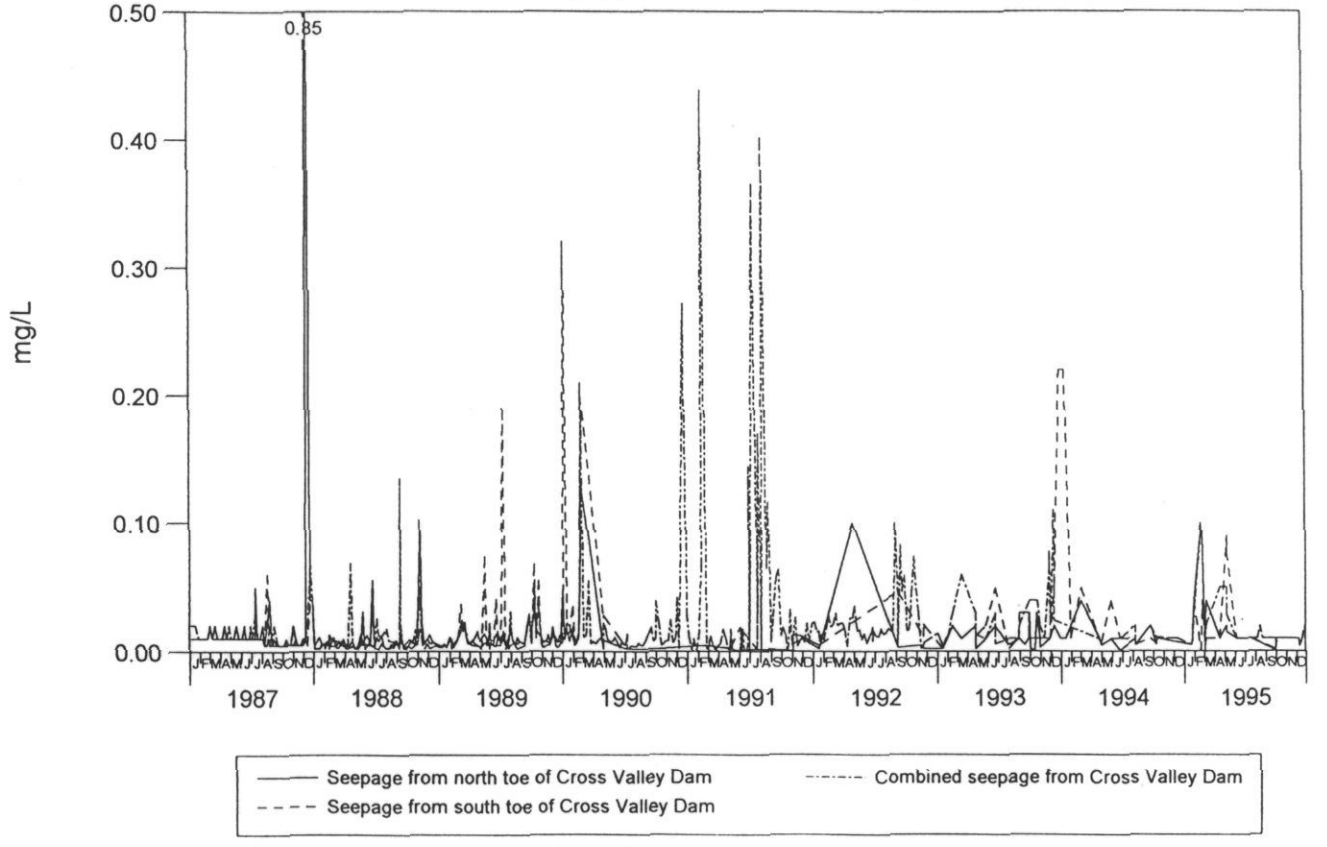


Figure 4-91
Sulphate seasonal trend, Rose Creek downstream of the diversion channel

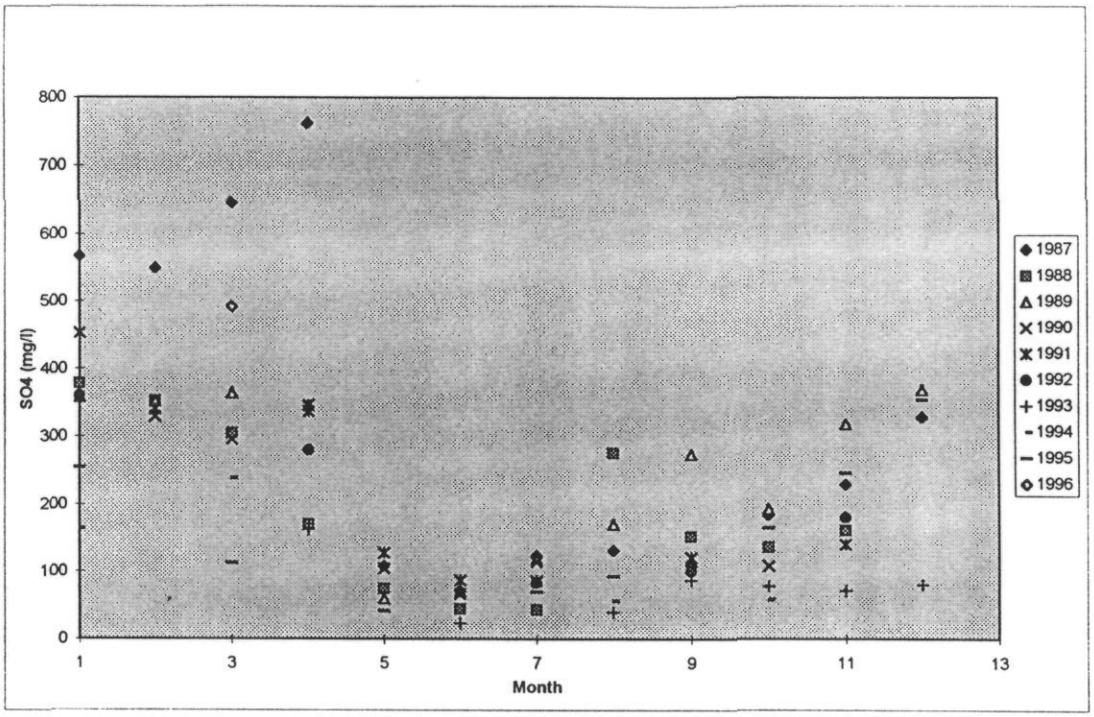


Figure 4-92
pH time trend, Rose Creek downstream of the diversion channel

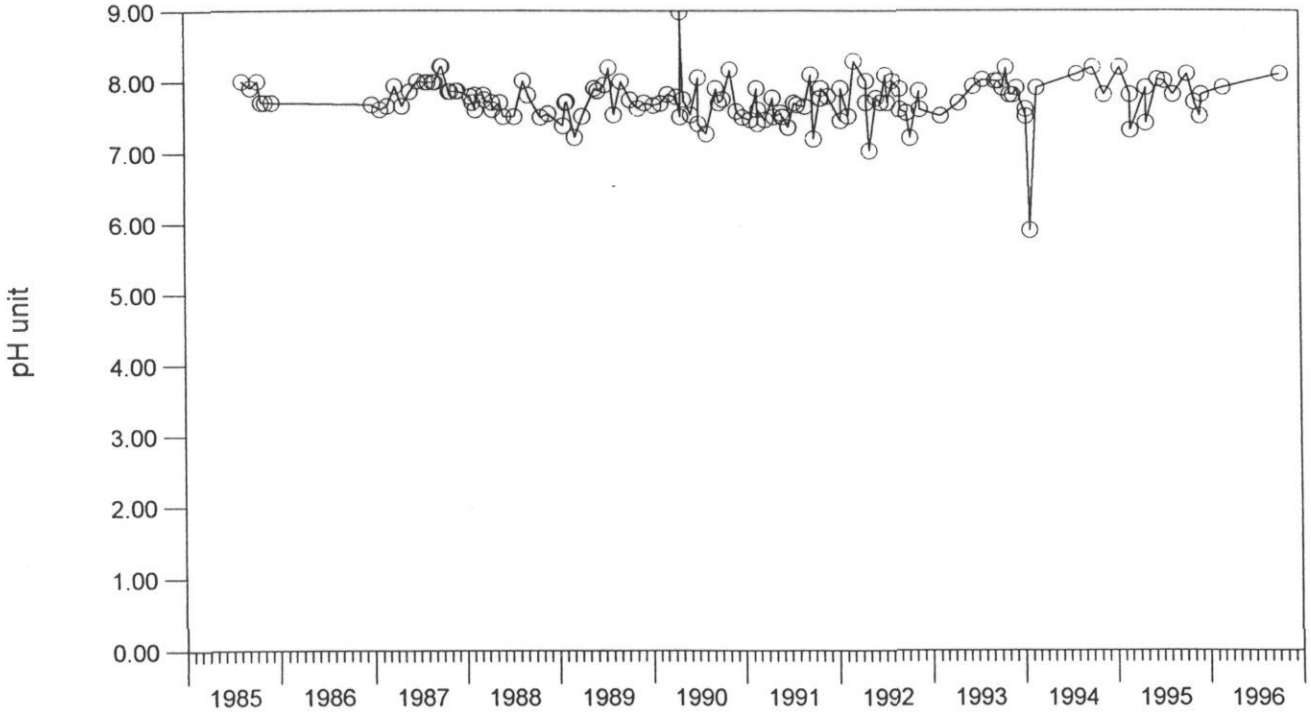


Figure 4-93
Total iron time trend, Rose Creek downstream of the diversion channel

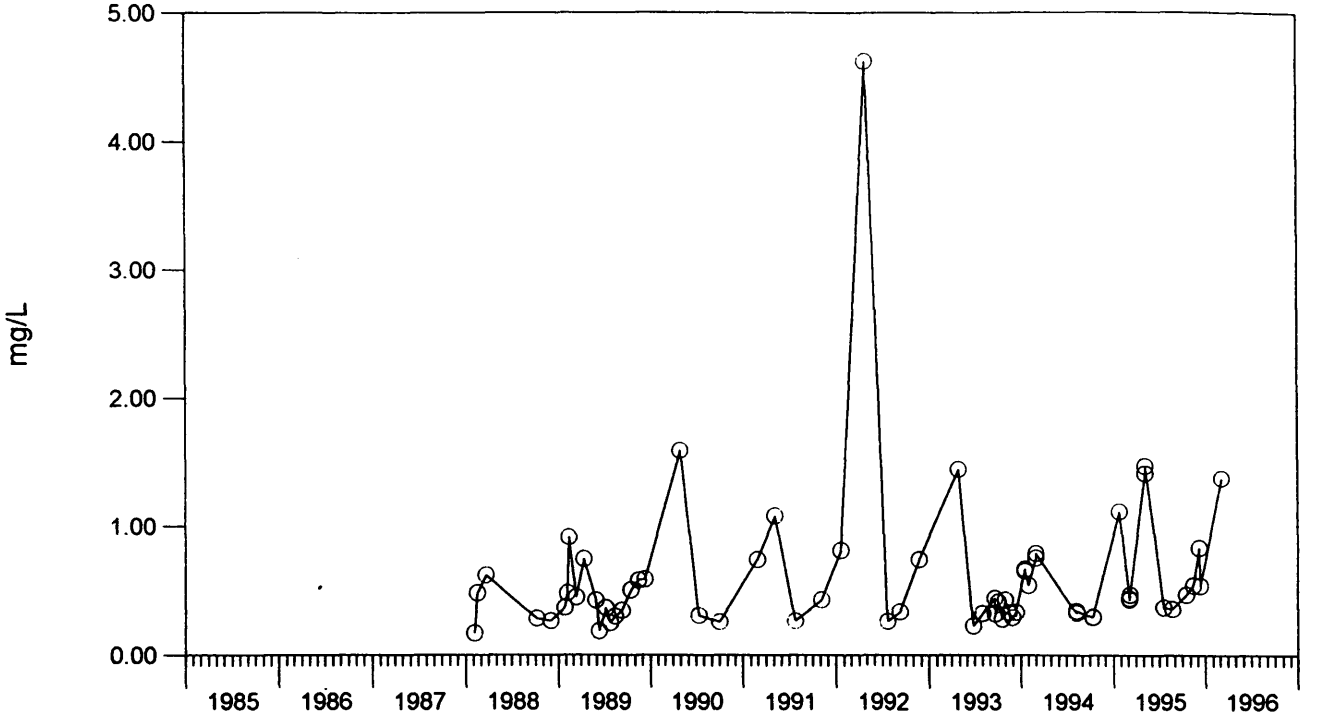
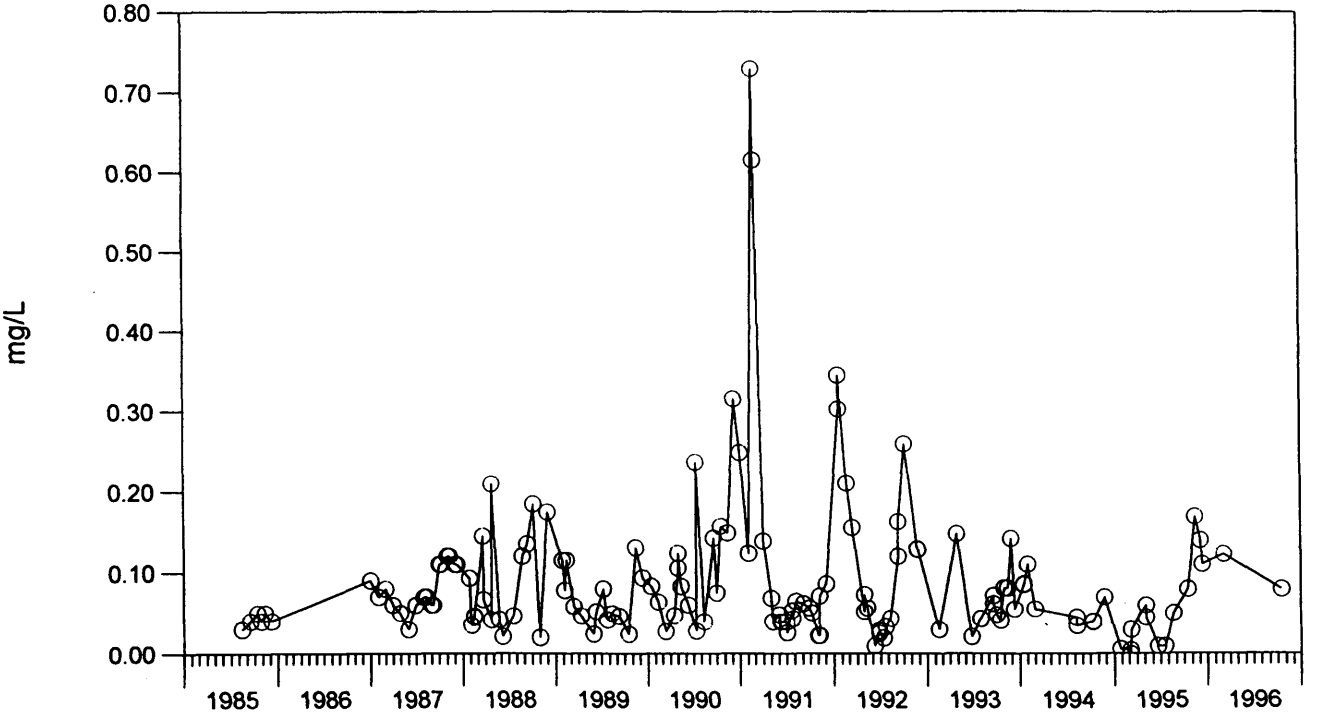
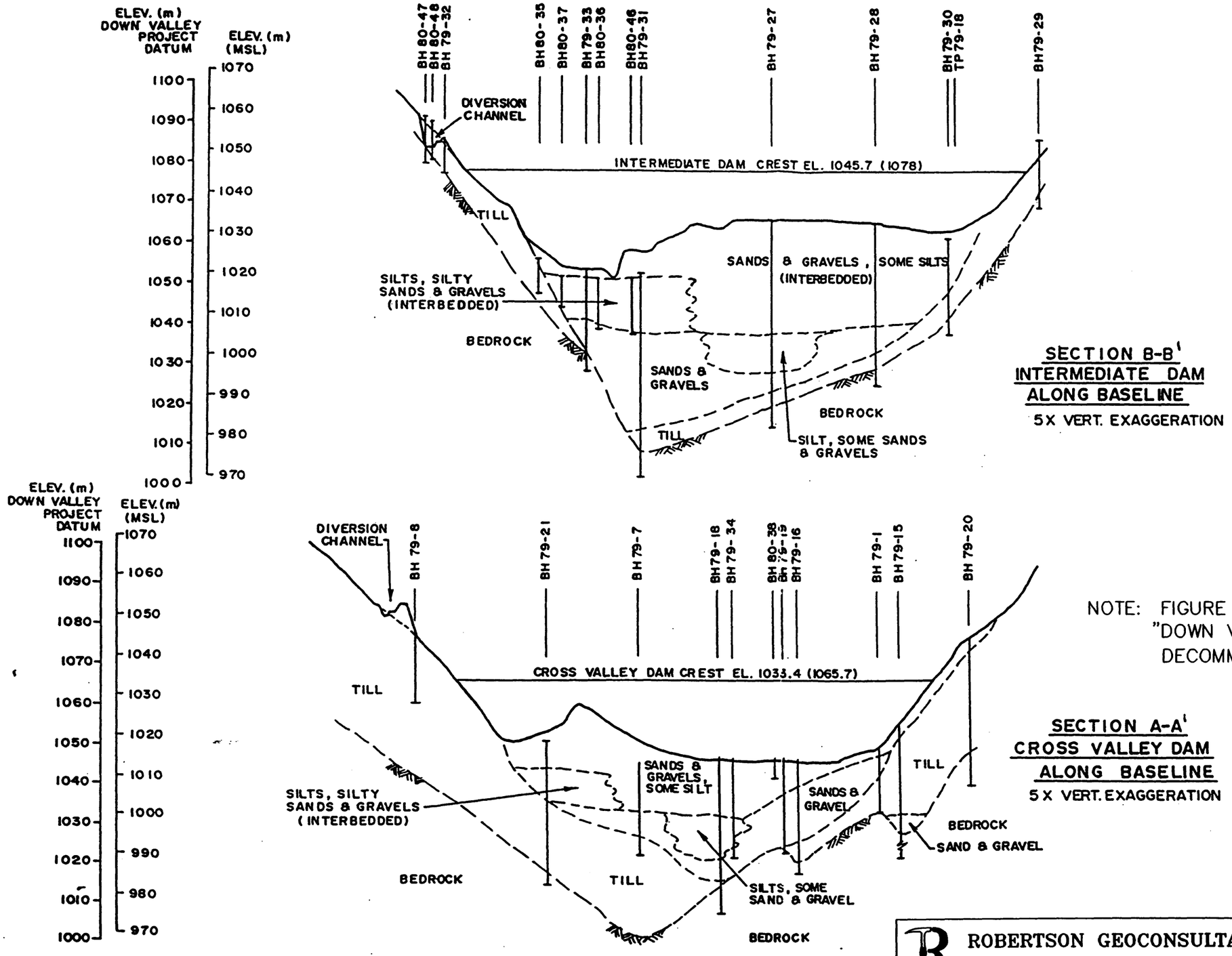


Figure 4-94
Total zinc time trend, Rose Creek downstream of the diversion channel





SECTION B-B' INTERMEDIATE DAM ALONG BASELINE

5X VERT. EXAGGERATION

SECTION A-A' CROSS VALLEY DAM ALONG BASELINE

5 X VERT. EXAGGERATION

NOTE: FIGURE TAKEN FROM SRK REPORT 60635 "DOWN VALLEY TAILINGS IMPOUNDMENT DECOMMISSIONING PLAN", APRIL 1991.

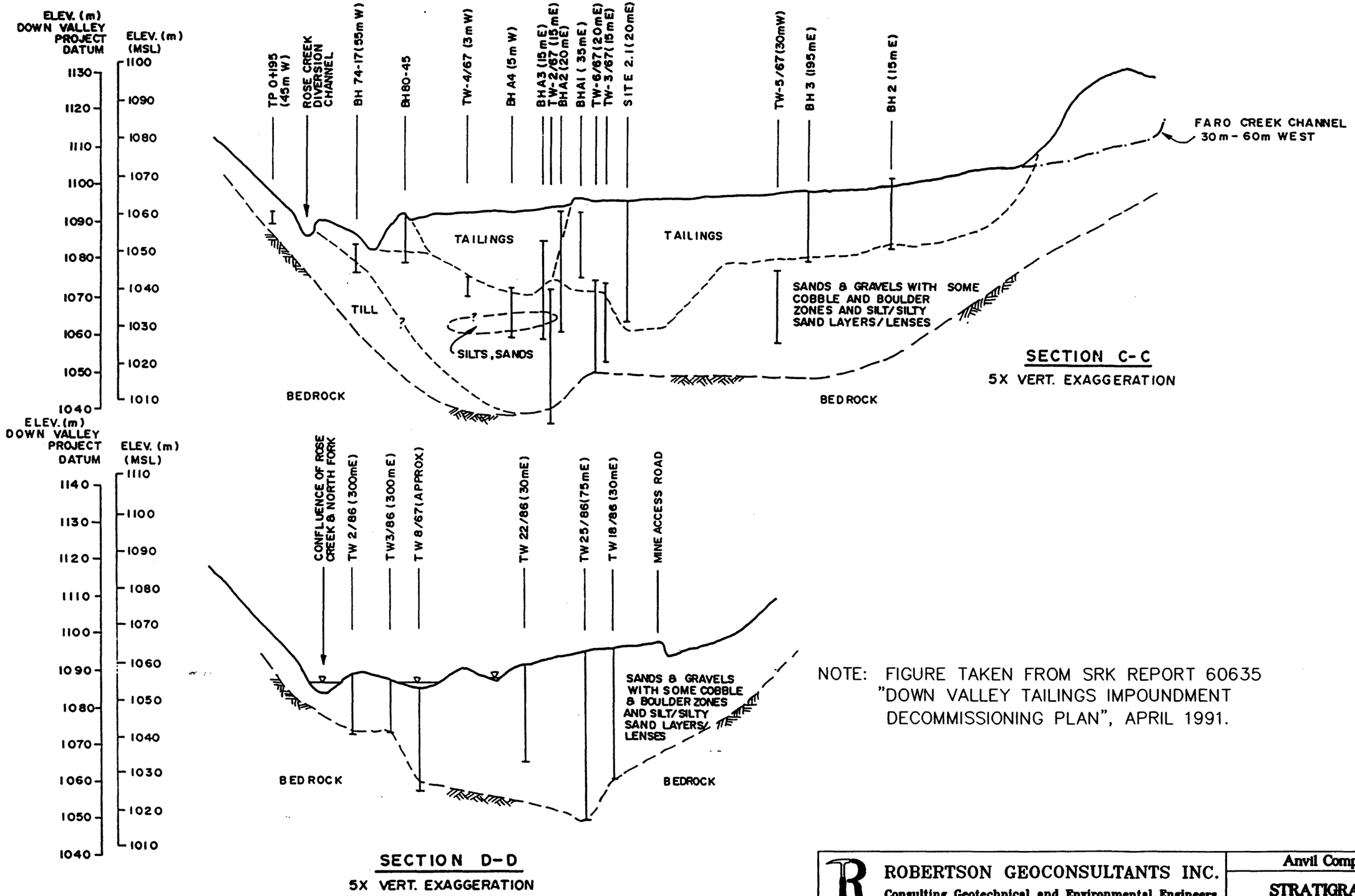
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Consulting Geotechnical and Environmental Engineers

Anvil Range Mining Corporation

Anvil Complex Closure Plan

STRATIGRAPHIC SECTIONS THROUGH IMPOUNDMENT AREA SECTIONS A - A AND B - B

PROJECT NO. 033001 | DATE Nov. 1996 | APPROVED | 14-95



NOTE: FIGURE TAKEN FROM SRK REPORT 60635 "DOWN VALLEY TAILINGS IMPOUNDMENT DECOMMISSIONING PLAN", APRIL 1991.

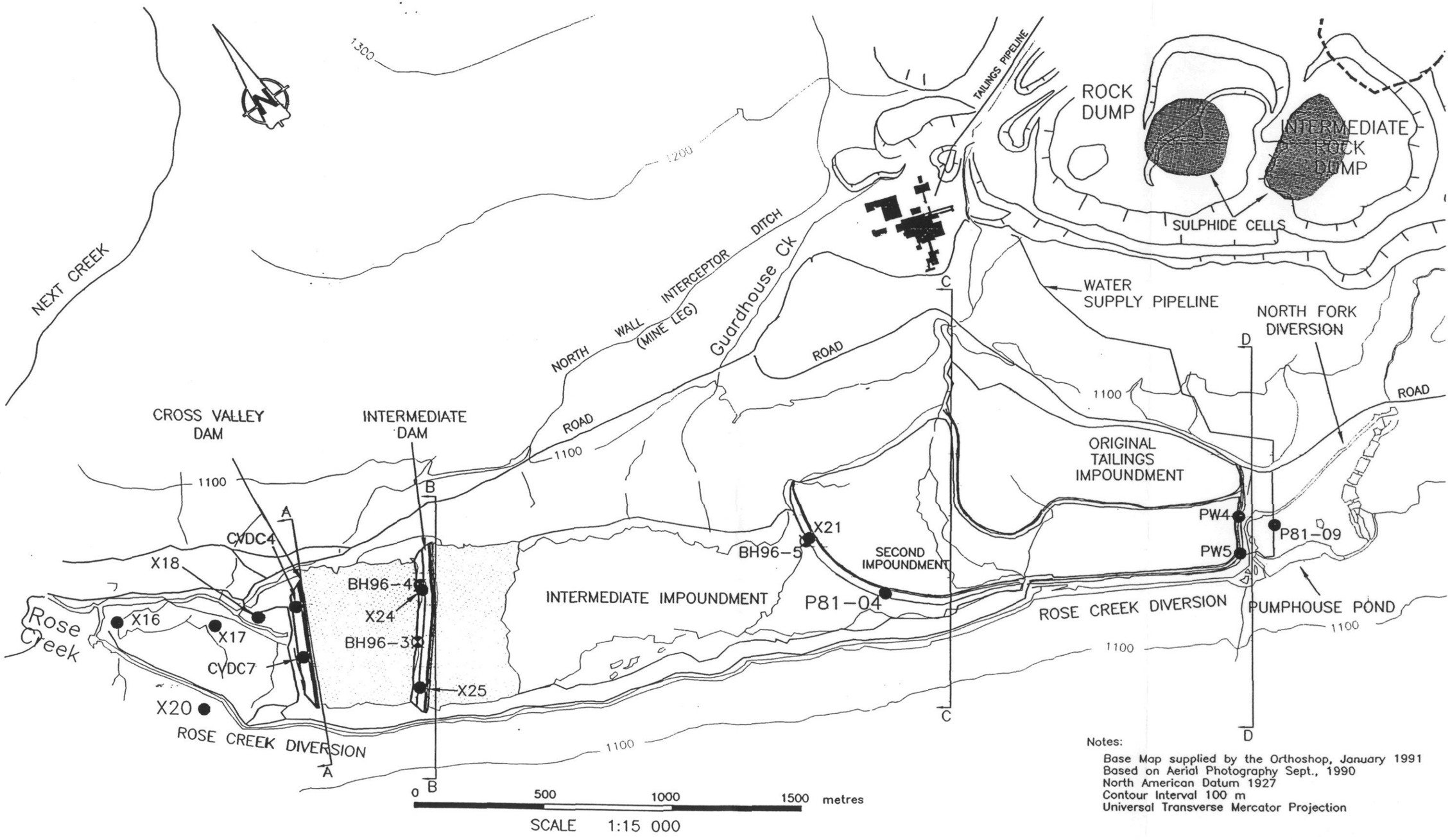
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STRATIGRAPHIC SECTIONS THROUGH IMPOUNDMENT AREA SECTIONS C - C AND D - D

PROJECT NO. 033001 | DATE Nov. 1996 | APPROVED | FIGURE 4-96



Notes:

Base Map supplied by the Orthoshop, January 1991
Based on Aerial Photography Sept., 1990
North American Datum 1927
Contour Interval 100 m
Universal Transverse Mercator Projection

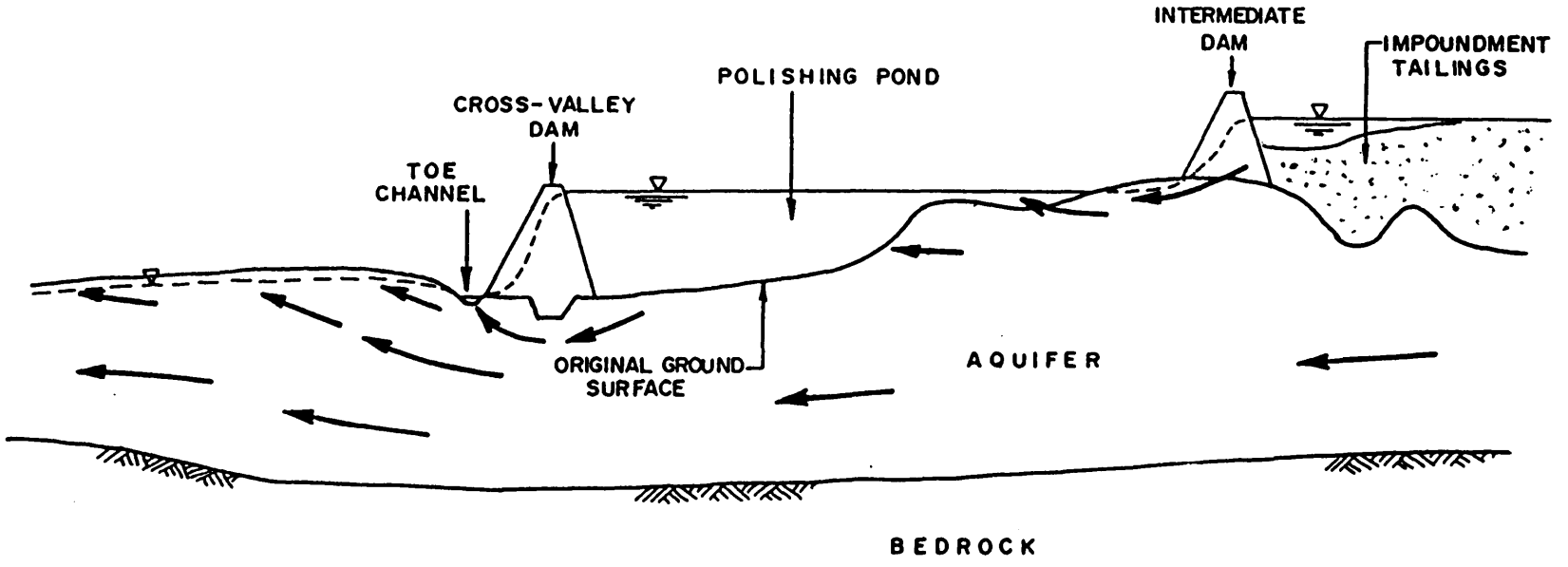
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Anvil Complex Closure Plan

Groundwater Monitoring Sites Rose Creek Valley

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-97



VERT. SCALE = 1:1250
HORIZ. SCALE = 1:5000

NOTE: FIGURE TAKEN FROM SRK REPORT 60635 "DOWN VALLEY TAILINGS IMPOUNDMENT DECOMMISSIONING PLAN", APRIL 1991.

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Anvil Complex Closure Plan

SCHEMATIC OF PRESENT FLOW REGIME BELOW POLISHING POND

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-98

\\westdale\FLOW-REGIME1 Thu Nov 21 17:45:07 1996 L. du Toit

Figure 4-99 X21 Time Trends

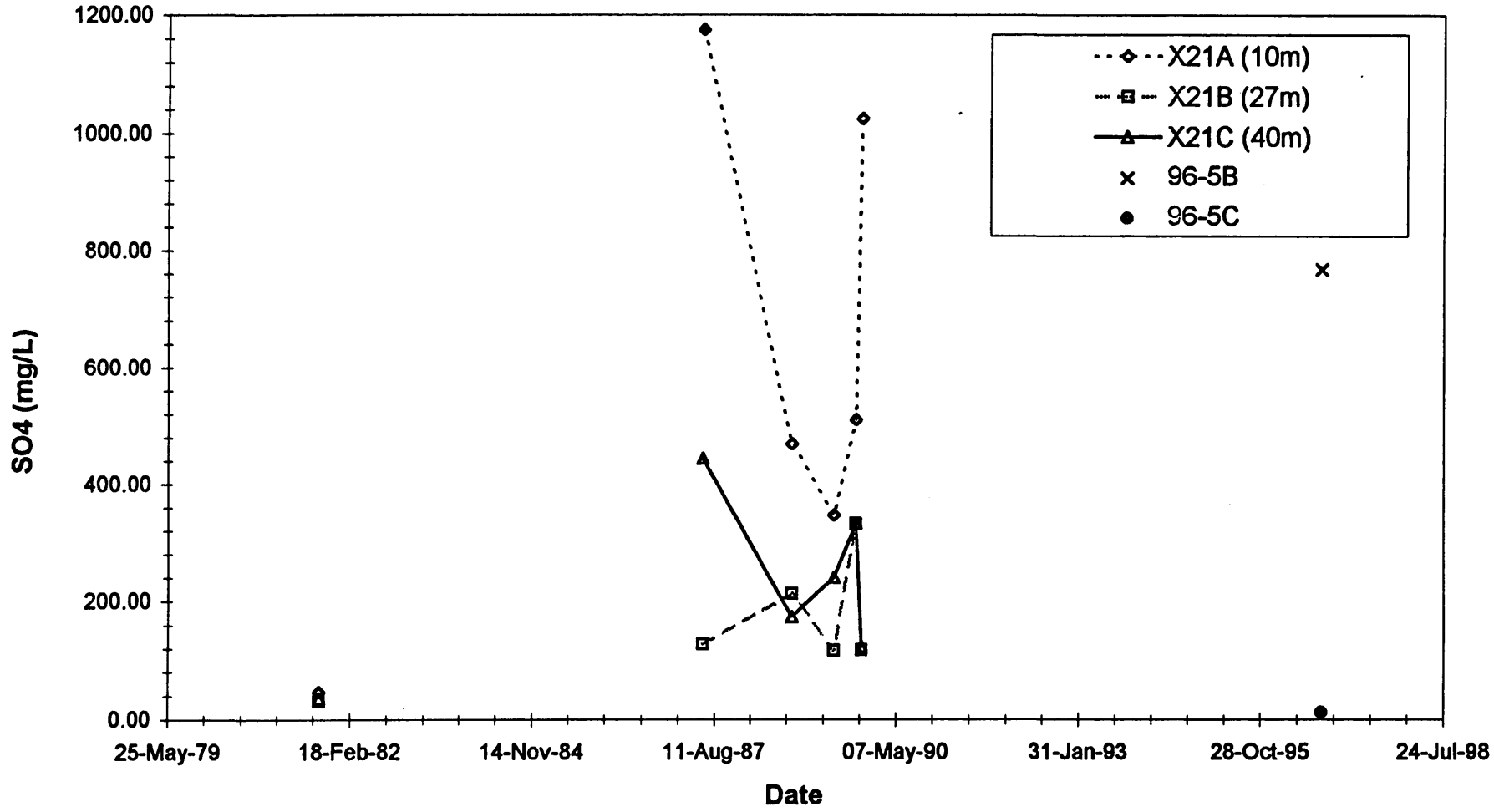


Figure 4-100 X24 Time Trends

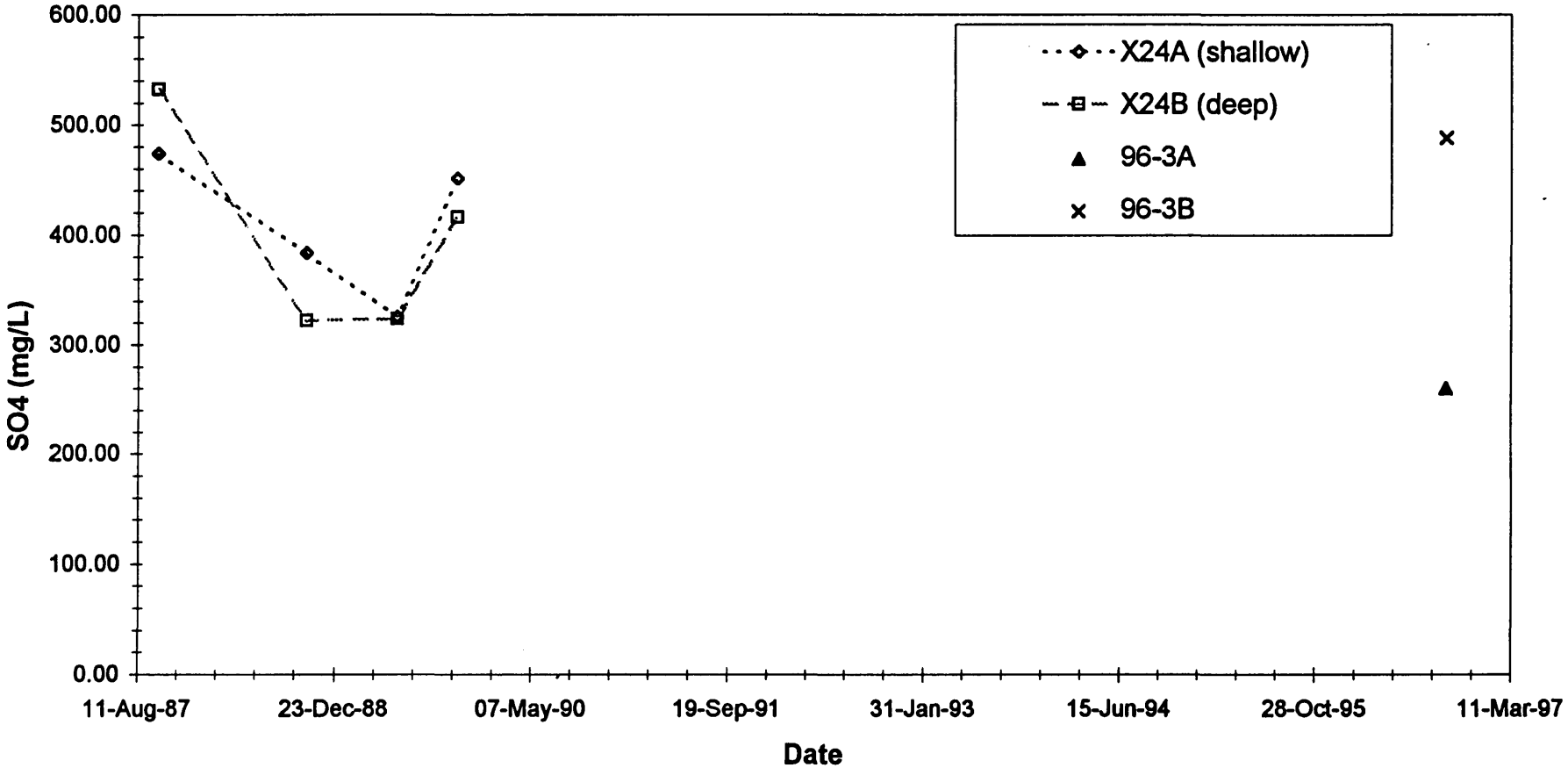


Figure 4-101 X25 Time Trends

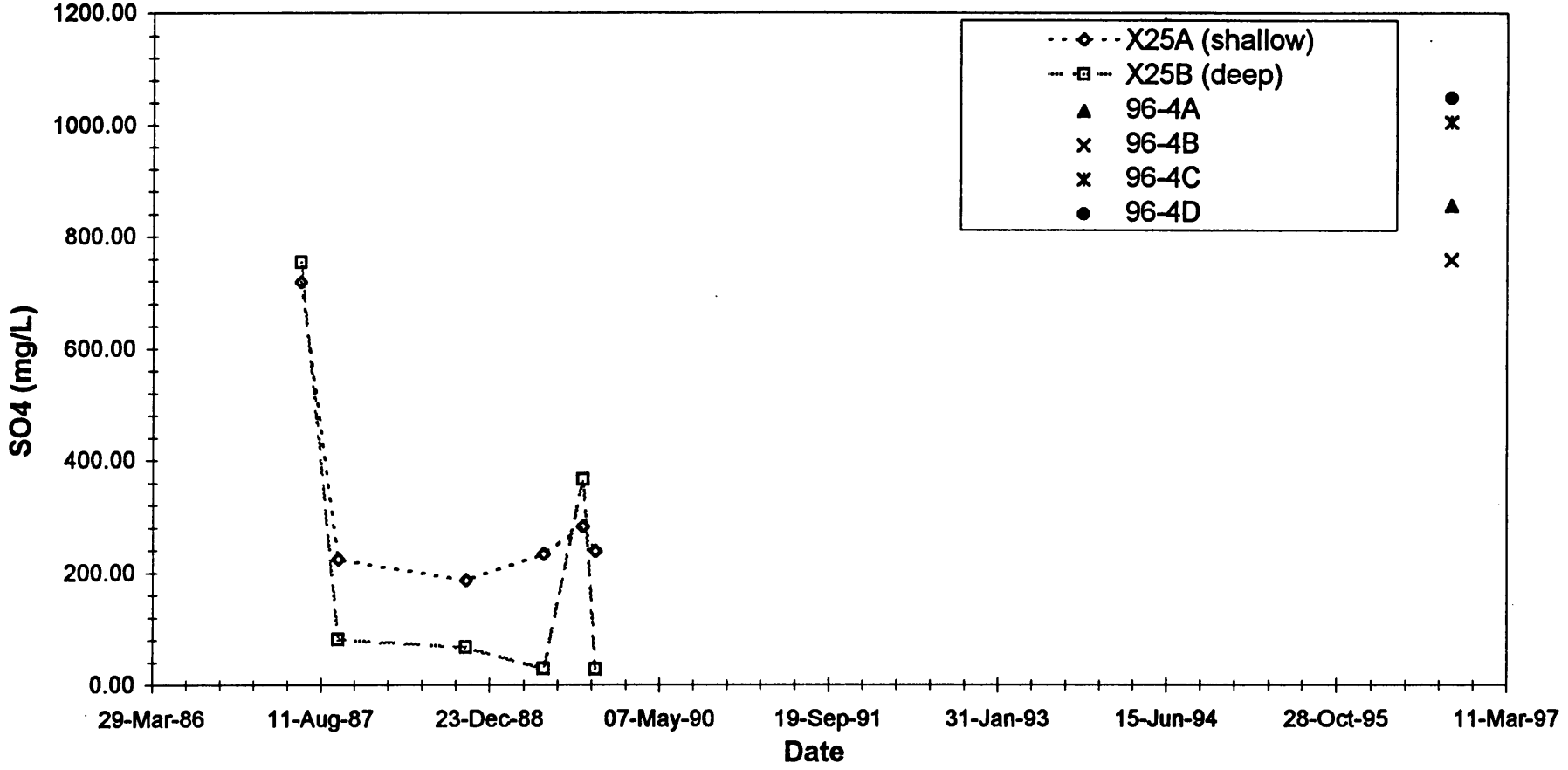


Figure 4-102 X18 Time Trends

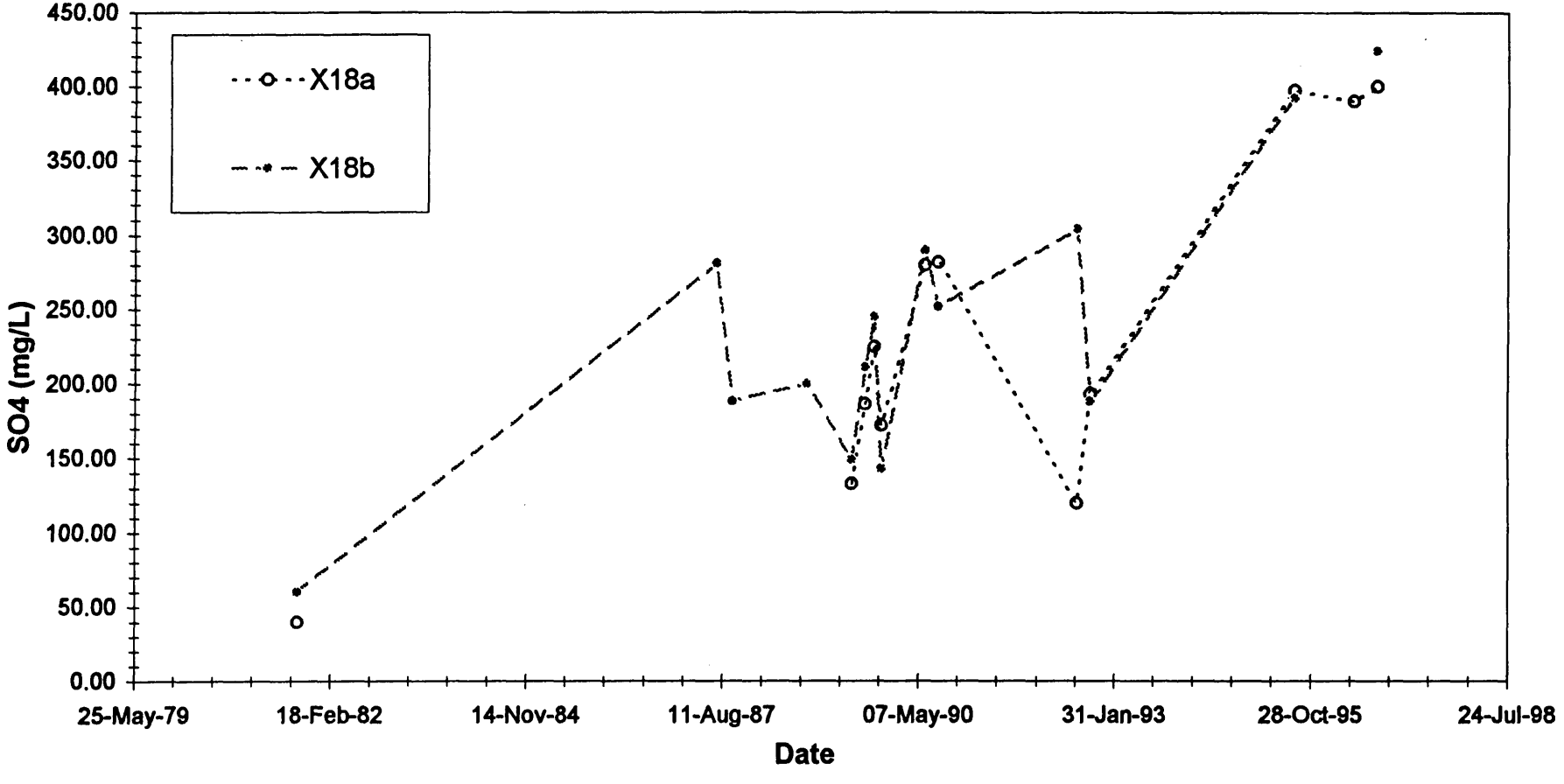


Figure 4-103 X17 Time Trends

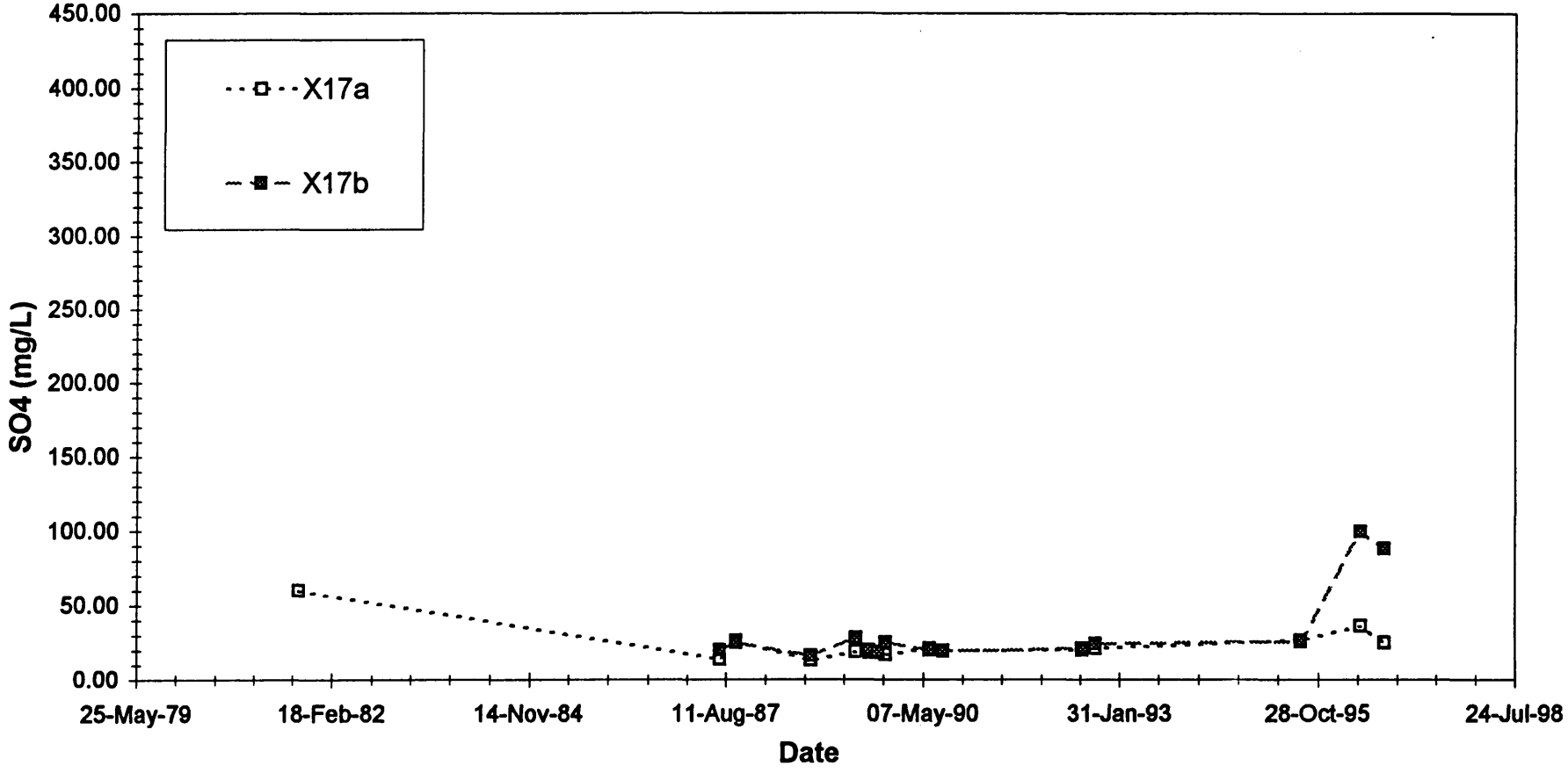


Figure 4-104 X16 Time Trends

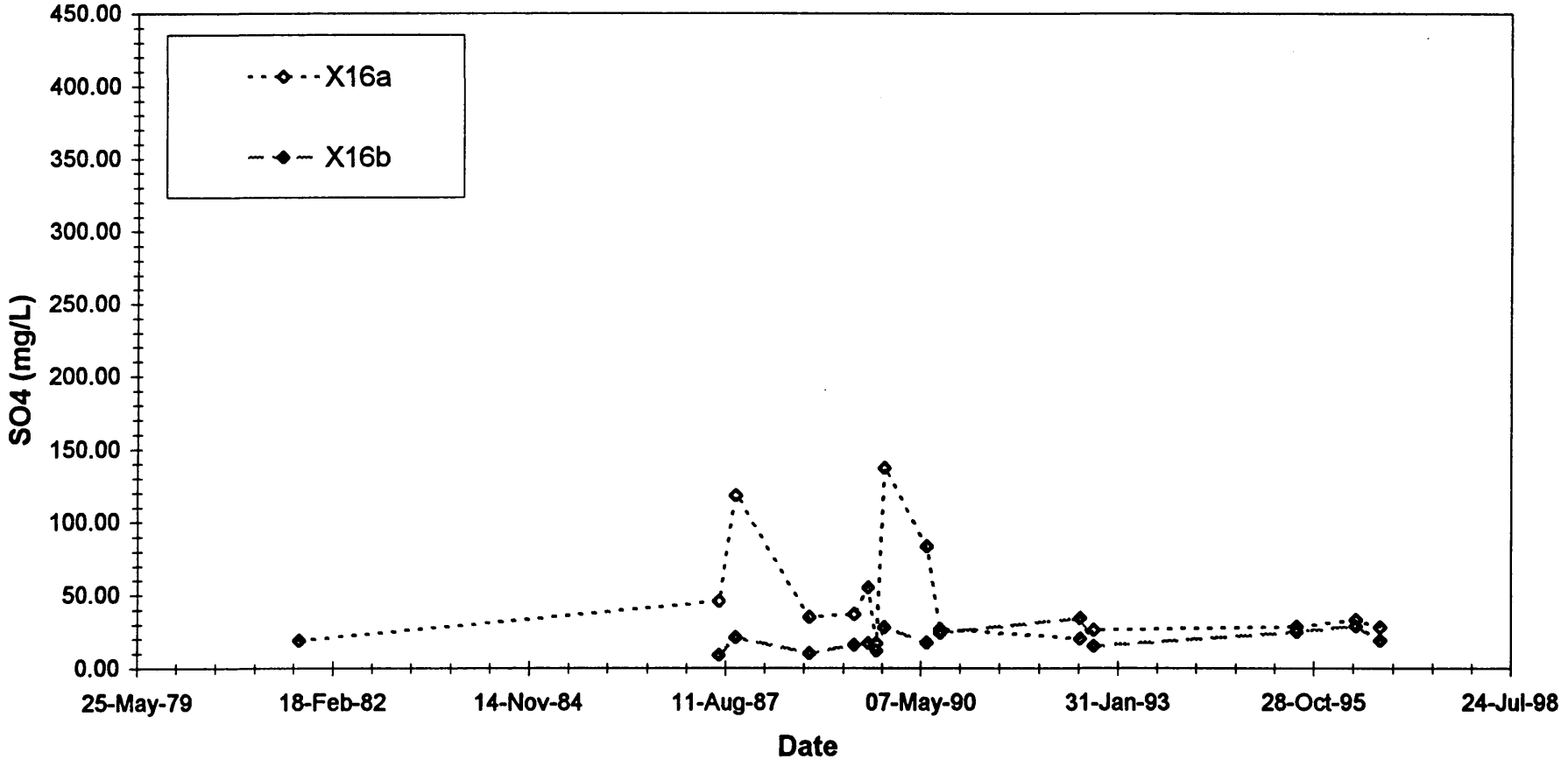
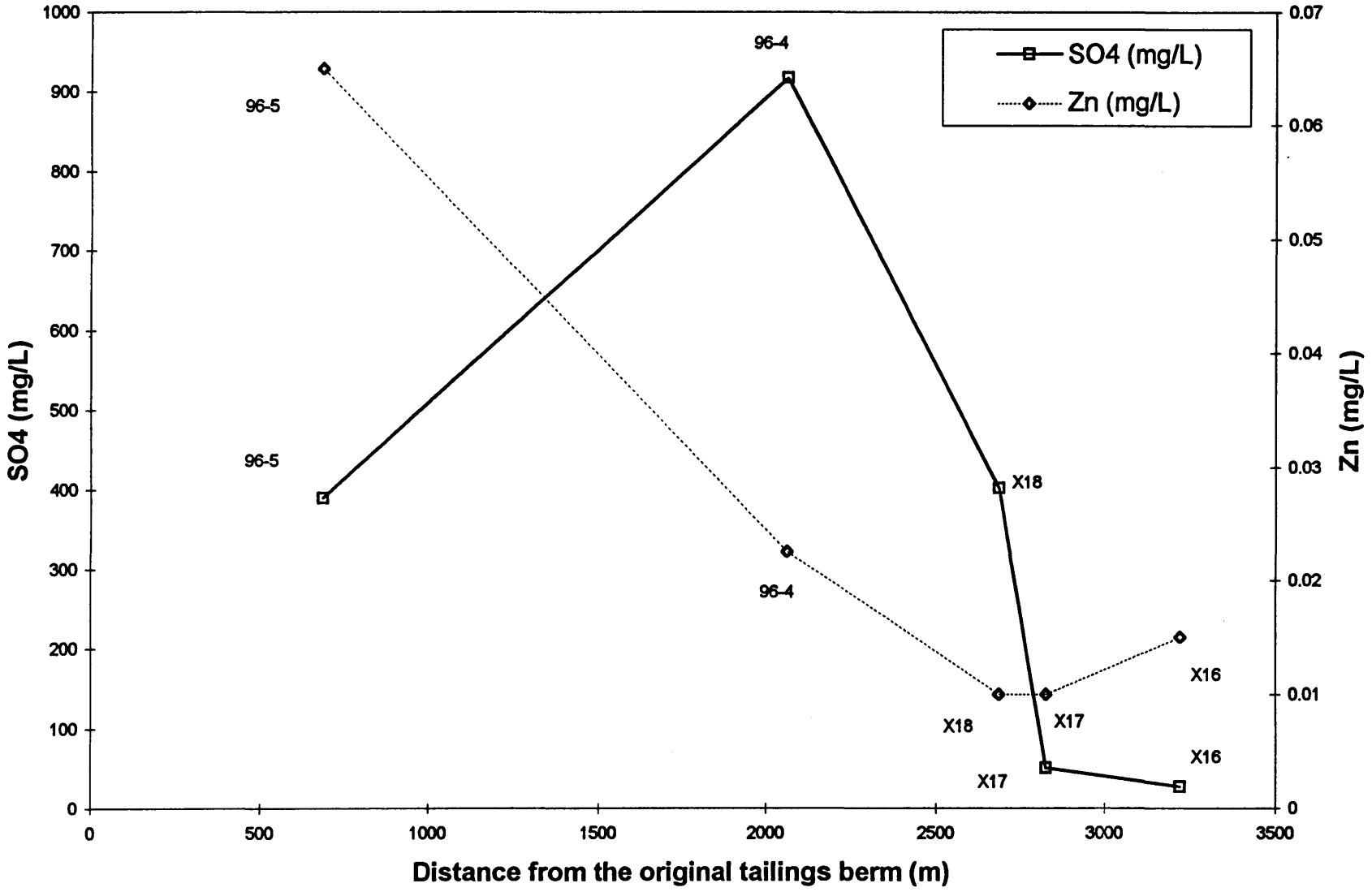
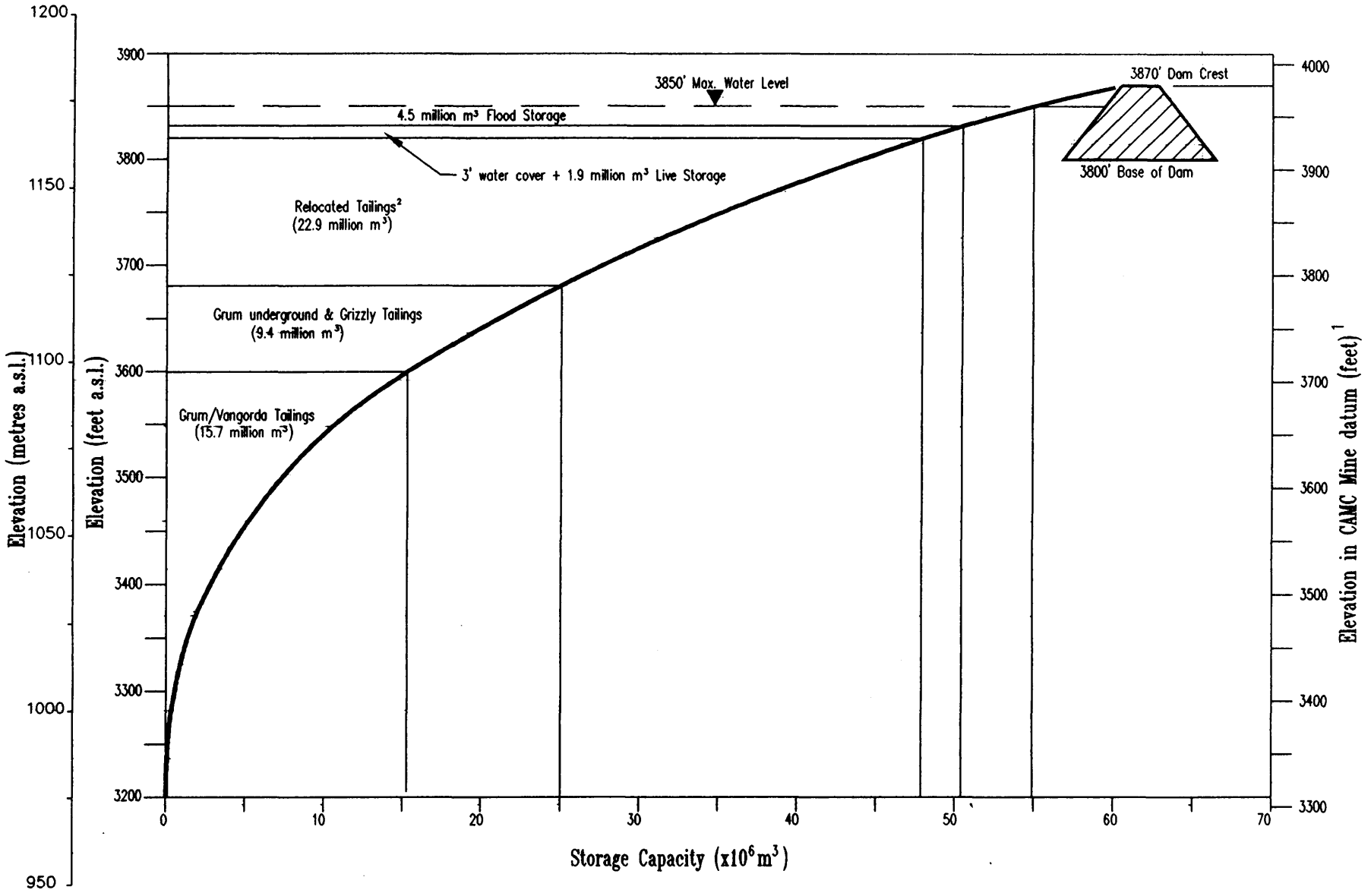


Figure 4-105 Zn and Sulphate vs Distance from Source





Note:
1. CAMC Mine Datum = UTM Datum (a.s.l.) + 109.57'
2. Tailings above elevation 1042m in Intermediate & Original/Second Impoundment

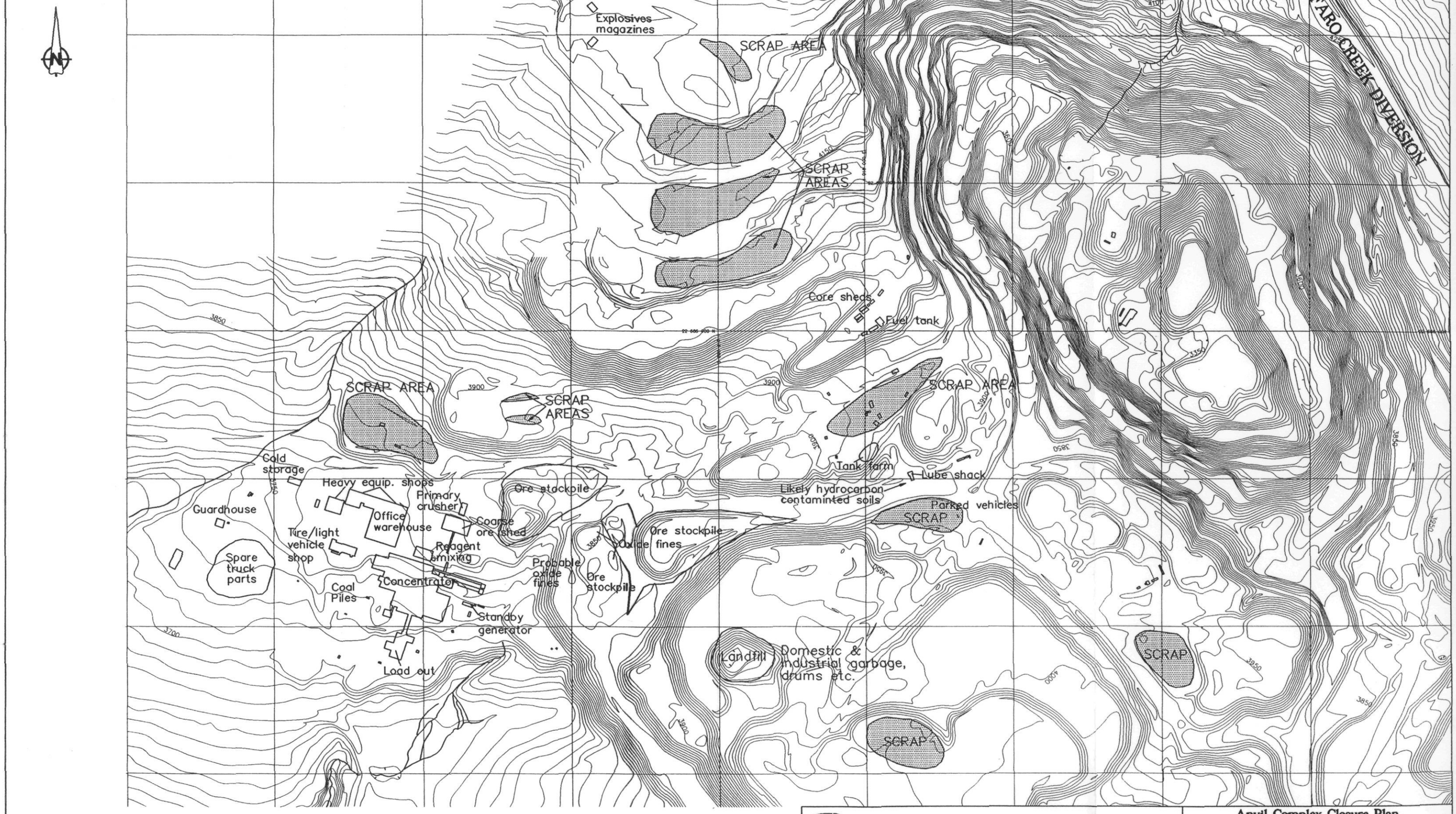
ROBERTSON GEOCONSULTANTS INC.
Consulting Geotechnical and Environmental Engineers

Anvil Range Mining Corporation

Anvil Complex Closure Plan

Height Capacity Curve
Faro Main Pit

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-106



Scale 1 inch = 600 feet

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Consulting Geotechnical and Environmental Engineers

Anvil Range Mining Corporation

Anvil Complex Closure Plan

Faro Mill Site General Plan Arrangement

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		4-107

5. Vangorda Plateau - Site Characterization

The Vangorda Plateau mine complex has been develop comprises the consists of the Vangorda Pit and Dumps and the Grum pit and dumps. Figure 5-1 gives an overview of this minesite and shows the extent of pit excavation and construction of the rock dumps in early 1996. A brief description of the history of development, geology and hydrogeology of the Vangorda Plateau area is given below. The remainder of this chapter describes the various components of the Vangorda Plateau minesite.

5.1 History of Development

The Vangorda deposit was discovered in 1953 and drilled several times between then and the late 1980's when it was finally developed for production. At the same time Vangorda was discovered two small occurrences, Champ and Firth, were found. Twenty years later the Grum deposit was found between these two minor occurrences. During the years 1975 to 1977 extensive work programs were carried out at Grum to define the ore deposit. This work included an underground exploration program. The deposit was accessed by a ramp from a portal elevation of about 1265 m. and twin declines followed the ore zone for 700 m., extensive definition drilling as done from these declines. There has been no underground exploration at Vangorda.

The Vangorda Plateau site was first developed in 1988 when efforts to de-water tills of the Vangorda and Grum sites began. Several drainage ditches were dug at Vangorda and Doal Lake (actually a shallow pond) was drained. Stripping at the Grum site began first with the wet soils from the vicinity of Doal Lake being placed in the wet dump, immediately southwest of the pit area.

Mining on the site got off to an erratic start due to production difficulties with other mining sites in the area. Mining began at Vangorda in 1990 following issuance of a water license. Between 1990 and 1993 Curragh Inc. mined 5.7 million tonnes or ore from Vangorda and several times that amount of waste. The Vangorda pit was redesigned shortly after mining began resulting in a larger pit, this combined with a lessening in the amount of till being excavated due to iron contamination of the basal till, led to a redesign of the waste dumps, in particular, to the covers proposed for the dumps.

Stripping was carried out intermittently at Grum during the same interval resulting in the excavation of approximately 22 million tonnes of glacial overburden and rock. Only minor ore (52,000 tonnes) was mined from the Grum pit by Curragh.

The site was under temporary closure from 1993 to late 1994. During this time D.I.A.N.D. undertook construction of the Vangorda collector ditch and re-sloped a portion of the rock dump. A 2 m. thick till cover was placed on half of the area re-sloped. Four groundwater wells were placed around the rock dump during the shutdown.

Anvil Range took over the site in November 1994 and resumed pre-production stripping at Grum. Anvil completed the Overburden Dump and concentrated its dumping on the Main and Southwest Dumps. Anvil re configured the Grum Dumps in response to higher than anticipated amounts of

sulphide bearing material. Anvil continues to mine at Vangorda when production allows but approximately 400,000 tonnes of ore remains there in late 1996.

During the 1993 to 1994 shutdown, water accumulated in the Vangorda Pit. This has given an indication of water quality that can be expected in pit water and also has allowed demonstration that the Vangorda Water Treatment plant is capable of operating at capacity for extended periods.

5.2 Geology of the Vangorda Plateau

5.2.1 Vangorda Deposit Geology

The general nature of the ore deposits of the Anvil District is described in Appendix G. The Vangorda deposit is a small deposit for this area and has a number of characteristics that make it unusual. These include:

- shallow depth and greater weathering
- abundance of barren foot wall sulphides
- degree of development of strongly altered phyllites

The Vangorda deposit was relatively close to the surface and was more affected by weathering than the other ore deposits of the district. The top 6 to 10 m. of the deposit was moderately oxidized and contained cyanide soluble copper which interfered with the selective flotation of lead and zinc in the mill. This material was placed in the Vangorda dump on the sulphide cell side of the dump. Later the material was screened and the coarse fraction sent to the mill (since it was less oxidized). The fine fraction of this oxidized ore remains in the dump and is referred to as "oxide fines" or "baritic fines".

The thickest part of the ore body occurred below a ridge of highly compacted till east of Vangorda creek. This material was used to construct the berms around the periphery of the Vangorda dump. The basal 1 to 5 m. of the till is commonly stained and cemented by iron oxides thought to be derived from oxidation of the ore or sulphides associated with graphitic rocks or both. This material was rejected for use in construction of the till berms as it was thought likely to be charged with soluble metals.

The proximity of the deposit to the surface may have other implications for the geochemical behaviour of the Vangorda sulphides. Marcasite associated with pyrrhotite is described in polished section of many ore types at Vangorda. It is suspected that the presence of marcasite is due to surficial weathering. The apparent rapid oxidation of Vangorda sulphides and formation of extensive white powder on rock surfaces during periods of dry weather is possibly due to weathering of marcasite.

The Vangorda deposit consists of one major sulphide horizon, the "main horizon", about 50 to 120 m beneath the basal carbonaceous member of the Vangorda formation [unit 5A], as shown on Figure 5-2. The host rocks for the deposit are dominantly non-calcareous phyllites [unit 3G], probably part of the Mt. Mye formation. However, formational assignments near this deposit are ambiguous, largely due to the strong wall rock alteration developed around the deposit. Most phyllites, especially

in the main horizon foot-wall, are bleached, locally silicified and/or chloritic and sulphide bearing [unit 4L].

A number of thin horizons occur above the main horizon, see Figure 5-2. These horizons were too thin or too low grade to be mineable, with the exception of the south-east end of the deposit where they were shallow and the stripping ratio was low.

The deposit consists of the same sulphide rock types as the other Anvil District deposits as outlined in Appendix G. Two rock types are particularly prominent. The foot-wall of the main horizon of the deposit is an assemblage of sulphide-rich quartzite [unit 4C], quartz bearing semi-massive sulphides [unit 4EC] and massive pyrite [unit 4E0] which will be referred to as the "foot-wall pyritic quartzite". The rocks of the foot-wall pyritic quartzite grade downwards from hard pyrite - quartz rocks over an interval of 50 to 100 m into flaggy, moderately hard, siliceous phyllite and, ultimately, into soft non-siliceous altered phyllite. Parallel to this downward decrease in silica in the foot-wall pyritic quartzite is a downward decrease in the abundance of sulphides from massive barren pyrite (with over 80 % pyrite) at the top through quartz-rich semi-massive sulphide (with approximately 50 % pyrite) to weakly pyritic altered phyllite (with a few % pyrite) at the base. Most of the sulphides in the quartzite are pyrite, however pyrrhotite is generally present and locally abundant or dominant in this foot-wall zone. Magnetite is unusually well developed in the foot-wall pyritic quartzite. The foot-wall pyritic quartzite contains only minor lead and zinc but has similar copper (approx. 0.2%) and gold (approximately 0.75 g/t) grades as high grade Pb-Zn ore. Most of this unit is sulphide waste on the basis of lead-zinc content and mining of this material is minimal except as required to gain access to deeper high grade baritic sulphides of the main horizon. Since the copper and gold grade of this material suggests possible milling at some point this material has been segregated on the north side of the Vangorda waste dump.

The main horizon massive sulphides that overlie the pyritic quartzite are commonly baritic and rich in lead and zinc. The contact between the high grade ore and underlying barren foot-wall is commonly sharp and visually obvious. Of the mineralization that exceeded 5% Pb+Zn at Vangorda, 90% was barite bearing massive sulphides [unit 4EG].

Of the other sulphide rock types, only 4A is of any importance. As is usual for these deposits, it tends to be low grade and peripheral to the deposit. Much of the 4A is actually part of the uppermost horizon associated with the carbonaceous phyllite at the base of the Vangorda formation.

The Vangorda deposit occurs in the hinge of a large second phase fold, as shown on Figure 5-2. However, there is considerable uncertainty in the details of fold morphology. The deposit is elongated in the north-west to south-east direction, parallel to the second phase fold axis, and has been traced over an area of 1300 m by 200 m.

The north-west half of the deposit plunges about 10 degrees towards the north-west but the south-east half is sub-horizontal, as shown on Figure 5-3. An axial planar foliation (S2) dips shallowly toward the south-west but is locally quite variable. This foliation is the dominant plane of failure for the host rocks of the deposit and is a principal factor in the slope stability of pit walls.

The deposit is truncated by a steep normal fault at its north-west end. Steep normal or transcurrent faults offset the ore near cross sections 10E and 12E, as shown on Figure 5-4. Both these faults are late stage post-folding and post-metamorphism structures.

A plan of the geology of the open pit in 1992 is shown on Figure 5-4. The final open pit will differ little from this map since most of the mining since the map was made has concentrated in the deep part of the pit below 1100 m. elevation. Due to the overall south-west dip of the ore body the north east pit walls have well developed exposures of the foot wall pyritic quartzite and are strongly acid generating. The opposite walls consist of various altered phyllites and thin sulphide bands, these walls also must be considered acid generating but not as strongly acid generating as the rocks in the north east wall. In the deeper part of the pit the sulphides do not extend all the way up the pit walls thus there is potential to flood the sulphides. A major fault is found at the north west end of the Vangorda pit. This fault truncated the ore body and juxtaposed black graphitic phyllite (unit 5A) of the basal member of the Vangorda formation against the ore body. These graphitic rocks are well developed in the pit wall and Vangorda Creek diversion canal rock cuts. The graphitic rocks contain minor pyrite and seepages from them are iron rich.

All phyllites mined from the Vangorda pit are altered and many contain at least minor pyrite or pyrrhotite with relatively little calcite. For this reason all phyllites from the Vangorda were considered acid generating and placed in the south-west portion of the bermed Vangorda rock dump. As in all of the Anvil Range, the dominant meta-sedimentary phyllites contain layers of mafic meta-igneous intrusives. At Vangorda the meta-intrusives are altered like all other phyllites and may thus have more leachable Ni and Co, elements normally enriched in mafic rocks.

5.2.2 Grum Deposit Geology

The Grum deposit with mineable reserves of approximately 25 million tonnes was much larger than Vangorda and has a number of unusual characteristics that make it stand out among the deposits. Particularly outstanding are:

- the high proportion of disseminated sulphide ore types compared to massive ores
- the generally weak alteration overprint
- the complex, large scale, fold structure

The sub crop of the ore deposit is covered by up to 100 m of till and better sorted glacio-fluvial silts, sands and gravels. The material fills a buried channel trending north-south through the pit area, as shown on Figure 5-5. There have been no noteable weathering features encountered at Grum.

The Grum deposit consists of three to five highly contorted layers of massive and disseminated sulphide mineralization within a 150 m thick section of barren phyllite. The most important mineralised horizon occurs just beneath the basal carbonaceous member of the Vangorda formation [unit 5A]. There are thin low-grade horizons within the Vangorda formation and more important horizons in the upper part of the Mt. Mye formation.

A feature unique to Grum among the district deposits is the relative abundance of quartzose ore types, particularly carbonaceous pyritic quartzites [unit 4A] which comprise about 50% of the

reserves above 4% Pb+Zn. Other ore types are similar to the remainder of the deposits as described in Appendix G. Thick sections of barren pyritic quartzite and barren massive pyrite such as found at Vangorda and Faro are not conspicuous at Grum. Altered phyllites are also not prominent at Grum and where present they contain relatively little sulphide. The sulphide waste at Grum thus tends to consist of low grade ore types mixed with phyllites rather that the phyllites themselves.

At Grum, the Vangorda formation consists of soft, highly fissile, calcareous phyllites [unit 5B0]. Mafic meta-igneous rocks in the Grum area are volumetrically minor and tend to be highly foliated chlorite phyllite (unit 5D0) rather than blocky, massive greenstones (unit 5C0) that typify the Vangorda formation elsewhere. Carbonated versions of these chloritic phyllites (unit 5D4*) are widespread near the ore and commonly these pale coloured muscovite - ankerite rich rocks contain bright green minerals similar in appearance to fuchite. The basal carbonaceous member of the formation [unit 5A0] thickens across the deposit from about 10 m in the north-east to as much as 80 or 100 m south west of the deposit. The sulphide horizons appear to be associated with the north-east pinch-out of this unit. Immediately above the main ore horizon, the carbonaceous rocks are soft, highly sheared and gouged. Elsewhere they are moderately hard, highly fractured, black siliceous phyllites. The Mt. Mye formation also consists of soft phyllites [unit 3G0] which are distinguished from those of Vangorda formation by being non-calcareous and less distinctly banded. Phyllites at Grum are not as strongly altered as at Vangorda and many are strongly calcareous, consequently considerable effort is being made to separate the phyllites from acid generating rocks in the dumps to concentrate the acid generating material in a place where drainage control can be provided with greatest effectiveness.

There are no significant post-metamorphic dykes at Grum. The Anvil Batholith crops out 1.5 km north-east of the deposit but is separated from it by major faults. The Batholith is unrelated to the deposit and does not appear to have significantly affected it.

The ore layers at Grum are contorted into a complex, shallowly north-west plunging, polyphase fold structure. The prominent S-shaped folds as shown on Figures 5-4 are second phase structures. They are superimposed on a larger Z-shaped first phase fold. The dominant plane of fissility or cleavage in the phyllites at Grum is axial planar to the second phase folds and dips shallowly (10 degrees to 30 degrees) generally to the south-west. This fissility is a major factor in assessing slope stability for the pit. The overall deposit elongation parallels the axial direction of the second phase folds (315 degree trend, 11 degree plunge).

There are several important faults at Grum. The largest displacements occur on moderately dipping (35 degree to 45 degree) structures that truncate the deposit at both its north-west and south-east ends but will not crop out in the pit. The rocks of these faults vary from intact fault rock with characteristics identical to the enclosing phyllites to approximately 3 to 10 m. thick intervals of gouge and surrounding fractured rock. A steeply north-west dipping fault set trending about 060 degrees, passes between sections 68W and 72W and down drops the deposit about 60 m to the north-west. A myriad of smaller faults were mapped underground by Kerr Addison, and by later operators in the pit, trending on the average 080 degrees and dipping steeply. Joints mapped underground and on surface tend to strike 060 degrees and dip sub-vertically.

A plan of the open pit geology of the ultimate Grum pit is shown on Figure 5-5. The geometry of the deposit, covering glacial till and the pit walls is such that the Grum pit final walls will contain no appreciable sulphides above the level of eventual flooding.

5.3 Hydrogeology

The mine development at the Vangorda/Grum mine site has influenced groundwater flows in the immediate vicinity of the two open pit mines:

- Vangorda Open Pit and Dumps (in vicinity of Vangorda Creek);
- Grum Open Pit (in vicinity of former "Doal Lake" and Grum Creek)

The hydrogeology in these two areas is briefly described below.

5.3.1 Vangorda

The Vangorda Pit is located on the Vangorda Plateau on a hillside gently sloping to the southwest (Figure 5-1). Prior to pit development, the surface drained towards Vangorda Creek which flowed in a westerly direction through the northern pit area, and to a small tributary creek which was flowing along the west side of the pit.

The overburden soils consist of compacted silty and sandy glacial till which has a relatively low permeability and forms an upper confining layer. The competent impervious phyllite bedrock forms a lower confining layer. Interbedded between these layers is a water-bearing zone of sands and gravel over weathered bedrock. Piezometer readings indicated that the general direction of groundwater flow in the water-bearing layer was from north-east to southwest (Steffen Robertson & Kirsten, 1989).

Little is known about the valley sediments in Vangorda Creek. The sediments of Vangorda Creek valley are in the order of 20m deep, over a nominal valley width of 30m to 40m. It is likely that the valley sediments contain more permeable material than the surrounding glacial till. The northwestern section of Vangorda Pit intersected the Vangorda Creek Valley and Vangorda Creek was diverted along the the North wall of the pit. A sump was in use during active mining to collect groundwater inflows from the valley sediments.

Groundwater conditions in the bedrock at Vangorda pit were discussed in Piteau and Associates (1990). Piezometric levels measured in piezometers installed in bedrock prior to pit development indicated much higher piezometric heads along the east wall than in the central portion of the pit, indicating that groundwater flow from the phyllites behind the east wall into the pit area is impeded, possibly by the anisotropic nature of the phyllites. The hydraulic gradients within the central portion of the pit were very low, indicating that the hydraulic conductivity of the rock mass in the ore zone was greater than that of the surrounding rock mass. Limited falling head test data that is available for some of the piezometers installed in the Vangorda Pit area in 1987, indicate that the hydraulic conductivity of the rock mass is in the order of 10^{-5} cm/sec (Piteau and Associates, 1990). However, several fault zones have been mapped in the Vangorda Pit area,

typically striking north-northwest (Figure 5-4). Permeability along fracture zones associated with major structures could be much greater, but these permeable zones would likely be less than a few tens of metres thick (Piteau and Associates, 1990).

The development of the Vangorda Pit resulted in partial dewatering ("depressurization") of the surrounding overburden soils and bedrock. A review of water level and pumping records in Vangorda Pit suggest that total groundwater inflows to Vangorda Pit averaged about 350 dam^3/yr during the wet period of 1992 -1993 and about 220 dam^3/yr over the drier period from 1994-1995. It is unknown how much of this inflow is coming from shallow subsurface flow and how much from deep groundwater flow entering the pit via bedrock. The significant yearly variations in inflows to the pit suggest that shallow subsurface flows, possibly originating from the Vangorda Creek sediments andor from the Vangorda Creek Diversion (ditch leakage), is a major component. It is estimated that current inflows to Vangorda Pit from deep groundwater along structural features in bedrock are less than 100 dam^3/yr.

The Vangorda Rock Dumps are located southwest of the Vangorda pit on a hillside draining in part to Vangorda Creek and in part to Dixon Creek. Results of drilling at the toes of the dumps indicate that the overburden soils in this area typically consist of very compacted glacial till. The till is silty in most places and generally has a low permeability. Groundwater levels are typically well below natural ground surface at a depth of 8-14 meters below ground. Artesian conditions were observed in one well located downslope towards Vangorda Creek suggesting a confined layer of more permeable till material at depth (SRK-Robinson, 1994).

5.3.2 Grum

The Grum pit is located on the Vangorda Plateau to the northwest of Vangorda Pit (Figure 5-1). The area in immediate vicinity of the Grum Pit represents a small depression in the hillside and a small lake (Doal Lake) was located here prior to excavation. The Grum Dumps to the south are located on much steeper terrain which is sloping south-southeast towards Vangorda Creek. A small tributary, Grum Creek, is draining much of this area and is flowing into Vangorda Creek.

Surficial deposits in the area of the Grum Pit are very deep ranging from more than 100 metres depth in the east corner of the pit to less than 40 metres in the northeast and southeast of the pit area. The deep surficial deposits pinch out towards the western section of the pit area. The majority of surficial sediments which overlie bedrock in the Grum Pit area are low permeability till with very low rates of groundwater flow. However, a confined layer of saturated sands and gravel varying in thickness from 20-30 meters was encountered at depth. This aquifer appears to be confined to a prominent linear depression in the bedrock surface ("thalweg") which runs in a southerly direction across the southeastern part of the pit area (Piteau Associates, 1991). Recent drilling suggests that a second "basal" aquifer is present at the sediment-bedrock interface (Piteau Associates, 1995). This basal aquifer is hydraulically connected to the upper aquifer in some areas and appears to extend throughout the thalweg and possibly the surrounding areas. This system of two aquifers represents the major water-bearing hydrogeological unit in the Grum Pit area.

Piezometric data for the various piezometers and pumping wells indicate that groundwater flows in a southerly direction, with groundwater recharge occurring to the north and northeast of the proposed pit, and groundwater discharge occurring in the gentle valley to the south of the developed pit (Piteau and Associates, 1995). Artesian heads measured in the aquifer under the area of former Doal Lake indicate that groundwater flow to the discharge area is restricted, possibly due to low permeability zones within the aquifer. Several dewatering wells were drilled into this thalweg aqifer and upper bedrock just off the southeast wall of Grum pit. When the wells were completed, all were flowing between 1 and 100 liters per minute. This aquifer has been depressurized substantially due to pumping and these wells are not artesian any longer.

Pump tests have been performed in these sands and gravels to characterize the aquifer and to design a dewatering system for pit excavation (Hydrogeological Consultants Ltd., 1989). The data suggested that this deep aquifer is of fairly limited extent and recharge is limited as well (Piteau and Associates, 1991). However, it was noted that there is a large quantity of water stored in the aquifer and overlying sediments which would require significant pumping to depressurize and dewater the aquifer.

Several dewatering wells have been installed and are being pumped since 1990 to control groundwater seepage into the Grum Pit (Piteau Associates, 1995). In the period from January 1990 to December 1995, an estimated 1.5 million m^3 were pumped from dewatering wells in the area of Grum pit. It is estimated that approximately 60% of this pumped water (or 900,000 m^3) was derived from storage in the aquifer whereas the remaining 40% (or 600,000 m^3) came from recharge.

It is unclear whether this thalweg aquifer extends further south beyond the perimeter of the Grum Pit. Significant discharge of groundwater was observed south of the Grum Pit area prior to dewatering the thalweg aquifer (Piteau, 1991) suggesting that the aquifer may pinch out in this area. It is not clear whether the section of the aquifer daykighting in the Grum Pit is hydraulically connected to this groundwater discharge area.

Most of the Grum rock dumps (with the exception of the till dump) were placed in areas underlain by only a thin colluvium over impermeable bedrock. It has to be assumed that the only significant flows of groundwater south of the Grum Pit wil occur in the valley depressions of Grum Creek and other small tributaries where overburden is relatively thick and permeable stream sediments may be present near surface or confined at depth.

5.4 Groundwater Chemistry

The groundwater chemistry data for the Grum/Vangorda mine site are very limited. There are no licence specified sampling sites at this mine site. However, Anvil Range Mining Corporation is currently monitoring groundwater quality at several monitoring wells at the Vangorda Rock Dump and one nested piezometer downstream of the Grum Dump (see Figure 5-6 for locations of these groundwater monitoring sites). Groundwater from several pumping wells upstream of the Grum Pit is also being sampled for water quality. However, sampling takes place in a ditch into which the pumped groundwater is discharged. It can not be excluded that surface water is also being

collected in this ditch; hence these data may not be representative of the actual groundwater quality and are not reviewed here.

All monitoring wells at the Vangorda Plateau mine site were installed in the last three years (1994-1996). Hence available groundwater quality data are limited to a short time period and do not allow the evaluation of time trends such as at the Faro mine site. The groundwater quality data for the Vangorda Plateau site are briefly summarized below.

A total of 14 boreholes were drilled in 1994 at the toe of the Vangorda Dump as part of the rehabilitation work undertaken at the Vangorda Dump during the period of receivership (SRK-Robinson, 1994).

Five monitoring wells (GW-94 series) were completed in the overburden soil (silty till) on the downstream side of the seepage collection ditch (see Figure 5-6). These monitoring wells are screened in glacial till material at a depth ranging from 11.5 m to 16 m below ground surface. The water quality data for the GW-94 series are summarized in Table K13 (see Appendix K for summary tables of groundwater quality). The pH of the groundwater downstream of the Vangorda Dump has a pH of ~8.0 and high alkalinities (~450 mg/L). Presently, there are no signs of significant acid rock drainage in these wells. In GW-94-01 and GW-94-04 sulphate concentrations are very low (~20-90 mg/L) and are probably close to background. The other three boreholes show slightly elevated sulphate concentrations (typically 100-200 mg/L). The high alkalinity of the groundwater in the till buffers any possible input of ARD from the dumps and as a result dissolved Zinc concentrations are near the detection limit (<0.03 mg/L). Iron concentrations are more variable ranging from <0.03 mg/L to 0.9 mg/L.

Nine boreholes were drilled near the toe of Vangorda Dump through the rock dump material a few meters into natural ground (till) (Figure 5-6). Water quality data collected from these shallow piezometers is very similar to that collected from the deeper monitoring wells. Again, the groundwater is alkaline and well-buffered with pH values near 8.0 and alkalinities around 400 mg/L (Table K14). Dissolved Zinc is also near the detection limit (<0.02 mg/L).

The groundwater data collected below the Vangorda Dump suggest that any ARD which may have occurred thus far is well buffered in the rock dump itself and/or in the shallow overburden till. Future monitoring of the exisitng piezometers should continue in order to evaluate the potential for future ARD at this site.

One borehole (BH 96-9) was drilled in a small valley next to a tributary to Grum Creek which drains the central portion of the Grum Dump (Figure 5-6). The overburden soils are relatively thick (~17 m) in this valley (RGC, 1996). Two permeable, water-bearing units were encountered at this site. The upper layer consists predominantly of coarse gravel and boulders and the lower layer of sand and some silt above bedrock. The lower layer is confined below by phyllite bedrock and above by a low-permeability silt layer (RGC, 1996).

Monitoring piezometers were installed in each of the two water-bearing units and first sampled in September 1996. The water quality data from these two units are shown in Table K14. The groundwater quality in the upper, unconfined layer shows no signs of ARD with pH at 8.0 and an alkalinity of 178 mg/L. The concentrations of dissolved metals are also very low (Iron ~0.23 mg/L

and Zn below the detection limit). As expected sulphate concentrations are also low (54 mg/L) suggesting that there has been little or no release of contaminated seepage from the Grum Dump to date. The groundwater quality in the lower confined gravel layer is similar with pH at 8.2 and an alkalinity of 87 mg/L. Sulphate concentrations are somewhat higher (156 mg/L) but dissolved Iron and Zinc are both very low (Fe = 0.19 mg/L and Zn < 0.01 mg/L). The piezometers 96-9 is located in a valley immediately downstream of the sulphide cells in the Grum Main Dump. The permeable gravel layers in this area are more likely to transmit groundwater downstream than the side valleys where only a thin veneer of low permeability till is present. Hence the groundwater sampled in these piezometers is msot likely to be impacted by ARD in the future. Continued monitoring of groundwater quality in these boreholes is recommended to detect any potential ARD well before it reached the downstream receiving water.

5.5 Vangorda Open Pit

5.5.1 General Description

The Vangorda Pit is 1,150 m in length, is 200 to 350 metres wide and up to 100 metres in depth. The longitudinal axis of the pit is approximately north-west / south-east with the deepest portion to the north-west end of the pit. The south-east half of the pit is a narrow slot about 200 m. wide and only 50 m. deep. Access to the pit was by a ramp with the entrance at the south-east end of the pit. and down the slot to the deeper north-west pit where the thickest ore was mined. Vangorda Creek originally passed directly over the thickest part of the ore body and had to be diverted to mine the pit. A site plan is presented on Figure 5-1.

The geology of the pit is shown on Figure 5-4 and has been discussed in section 5.2.1. The north-east pit wall is mostly in massive and disseminated sulphide bearing rock. Due to the plunge of the deposit and offset by faults the north-west part of the deposit is deeper, consequently the pit is considerably broader there and remnant sulphide exposures on the pit walls are correspondingly deeper. The upper north-east walls are weakly altered phyllites of the Mt. Mye formation that pose no significant acid generation concern. The south-west pit walls are mostly phyllites; in the slot area there are minor sulphide lenses interlayered but in the deeper part of the pit there are no known sulphides in the phyllites. The north-west end of the pit is in black, carbonaceous phyllite with moderate to weak acid generation potential but low base metal sulphide contents. The weathering of these rocks appears to be the source on iron in the ferricrete at the base of the till in the area but seepage from them was not high in Zn or other metals.

Two small in pit rock dumps were placed by Anvil on either side of the haul road near the pit entrance, the size of these dumps is not well known but there are probably not more than a few 10's of thousands of tonnes in each. The rocks are likely to be mixtures of sulphides and phyllites in the proportion 50:50.

The Vangorda pit walls have experienced local instability. In particular, the north-west end of the pit in the carbonaceous phyllites, near several faults, where wall failures forced the diversion canal to be moved early in it's life. This immediate area appears to have stabilised however ongoing movement

is occurring on the south walls of the pit. The north-east walls, particularly the north-east wall on the slot do not appear to be unstable.

The height capacity data for the Vangorda Pit at ultimate extent are shown in Table 5.1 which were obtained from pit bench plans (Anvil Range Mining Corporation, pers. comm.).

Table 5.1.

Height-Capacity Data for Vangorda Pit - Ultimate Pit Design

Vangorda Pit (ultimate extent)			
Bench (m AMSL.)		Volume	Cum. Vol.
toe	crest	m^3	m^3
1023	1032	7,390	7,390
1032	1038	10,140	17,530
1038	1044	35,760	53,290
1044	1050	58,040	111,330
1050	1056	83,230	194,560
1056	1062	127,660	322,220
1062	1068	183,280	505,500
1068	1074	224,640	730,140
1074	1080	262,280	992,420
1080	1086	321,540	1,313,960
1086	1092	384,730	1,698,690
1092	1098	444,310	2,143,000
1098	1104	495,850	2,638,850
1104	1110	555,230	3,194,080
1110	1116	631,520	3,825,600
1116	1122	731,980	4,557,580
1122	1128	828,040	5,385,620
1128	1134	941,730	6,327,350
1134	1140	1,019,230	7,346,580

5.5.2 Water Balance

The Vangorda Pit has a total catchment area of some 22 km^2. The Vangorda Creek Diversion Channel and the two interceptor ditches together control virtually all of this area. Only 0.8 km^2 of the total catchment area lies below the diversion and interceptor ditches.

Inflows to the pit originate from four sources, namely:

- direct precipitation on the incremental catchment below the ditches (0.8 km^2);
- ditch leakage from the Vangorda Creek Diversion Channel and the two interceptor ditches;
- shallow groundwater flows through the alluvium in the Vangorda Creek valley; and,
- some deep groundwater flowing into the pit along fault zone predominantly from the southeast pit wall.

The only significant losses during operations are evaporation and pit dewatering.

The temporary shutdown of the mine from April 1993 to November 1994 afforded a good opportunity to accurately estimate the net inflow rate to the pit (i.e., the total of all four inflow components minus evaporation). Near the end of the shutdown period, the water level in the pit was surveyed. With this single survey value, the open pit effectively acted as a giant "bucket and stopwatch" measurement. Based on the geometry of the pit at the time of the shutdown, the surveyed water level was converted into an equivalent volume of 820,000 m^3 of water. Given that this volume of water accumulated over two spring freshets, the average net inflow rate to the pit was assessed at about 12 L/s.

Net precipitation on the pit catchment (i.e., precipitation minus evapotranspiration) was estimated to contribute a long-term average flow of about 5 L/s to the pit inflows. Accordingly, the remaining 7 L/s of the total inflow rate may be attributed to ditch leakage and groundwater sources. It is interesting to note that this level of combined ditch leakage and groundwater flow is only about 15% of the level observed for the Main Pit at the Faro site.

5.5.3 Geochemistry

The Vangorda pit water is sampled at Station V22. Vangorda pit water is pumped to Little Creek Dam for storage prior to pumping to the treatment plant (Figure 3-20).

Water chemistry data for this station are summarized in Table 5.2, and shown graphically in Figures 5-7 through 5-10. More detailed data summaries are provided in Appendix E on Vangorda Plateau Water Chemistry Data.

The water chemistry in the Vangorda pit is influenced by:

- groundwater seepage through pit walls which contain pyrite and leachable metals;

- precipitation and runoff from pit walls; and
- drainage from stockpiles of low grade ore and, more recently waste dumps, located around the perimeter of the pit.

The geology of the Vangorda pit is discussed in section 5.2.1, and is important to the understanding of the pit water chemistry, particularly in the longer term. The geology of the Vangorda pit results in a much higher potential for long term leaching from oxidized wall rock, and for oxidation and acid generation from much of the pit walls, than for Grum, as discussed in section 5.2.1. Approximately 30% of the pit wall is comprised of sulphides and 70% is phyllites with about 10% of the total being carbonaceous phyllites.

During the shutdown in 1993 and 1994, water was not pumped from the Vangorda pit. Dewatering began in 1995 to allow further mining in the pit. This dewatering will affect the data for the chemistry of the pit water when compared to the period of no pumping; less dilution is available within the pit so seepage from pit walls, dumps or stockpiles would more markedly affect the measured pit water chemistry. A sudden change in measured concentrations within the pit in 1995 therefore does not necessarily mean that there is a sudden change in the rate or extent of oxidation and leaching processes.

The data in Table 5.2 show that the pit water remains near neutral, with consistent pH values averaging 7.6 over the period from 1991 through 1995. Alkalinity values however appeared to be decreasing during the shutdown period but remain above 100 mg/L, as shown in Figure 5-7. Sulphate concentrations also increase during the shutdown period to an annual average of 422 mg/L in 1994, and increase more rapidly in 1995 (as shown in Figure 5-8). Sulphate concentrations are still below 1992 levels however. The peaks in sulphate and metal loading measured during the early years of mining probably reflect both the release of soluble contaminants from the oxidized upper portion of the pit (as discussed in section 5.2.1) and total suspended solids in the pit water. This oxidized material has been processed in the mill, or removed to the stockpiles by the pit or on the Vangorda dumps although some oxide material is exposed in the upper pit walls at the south-east end of the pit.

The concentrations of metals within the Vangorda pit water are generally higher than seen at Grum, and a wider suite of metals occur above detection limits. In addition to lead and zinc, water in the Vangorda pit contains other metals of concern including copper, nickel and cobalt. Both nickel and cobalt concentrations increase during 1993/94, as shown in Figure 5-9, indicating an accumulation of these metals in the pit water. Copper values decreased in 1995 from the 1993/94 peak. Zinc concentrations are the most elevated of metals in the pit, reaching peak concentrations of 20 mg/L in 1994 and 1995. Average annual concentrations are summarized in Table 5.2, and presented in more detail in Appendix E. Dissolved metal concentrations are not routinely measured in pit water, although are included in seep sampling.

Zinc tends to occur primarily as a dissolved metal at concentrations up to 20 mg/L when the pit is not being mined. This suggests that there is significant metal leaching occurring from wall rock and stockpiles. Geochemical equilibrium modelling similar to that discussed for the Faro Main pit (Chapter 4.1.5.1 Main Pit Water Chemistry and Chapter 6.2) suggests that metal concentrations are limited by solubility constraints. Carbonate alkalinity in the pit at values greater than 100

mg/L, combined with pH values greater than 7.5, are predicted to limit zinc concentrations to less than 15 mg/L by the formation of hydrated zinc carbonate or zinc oxyhydroxide complexes. Copper, cobalt, cadmium and nickel solubility can also by limited by the formation of hydroxide or carbonate complexes. However, solubility values for zinc are sensitive to pH and alkalinity within this range and can double with a decrease of 0.5 pH units in the neutral pH range. Thus, this control mechanism on water chemistry is most significant at pH values greater than 7.

Seep samples and extraction testing show that the major source of loading for these metals is the ore stockpile to the southeast of the pit. These loadings are discussed in more detail in Chapter 6.3.

Table 5.2
Annual Averages for Station V22

STATION: Vangorda pit water (V22)

Date	LAB pH pH unit	TOTAL ALKALINI mg/L	SUSPEN SOLIDS mg/L	TOTAL SULPHA mg/L	AMMONI NITROG mg/L	TOTAL COBALT mg/L	TOTAL COPPE mg/L	TOTAL NICKEL mg/L	TOTAL LEAD mg/L	TOTAL ZINC mg/L
JAN 1991 - DEC 1991										
Mean	7.69	173	568	273	8.9	0.642	0.186	0.13	1.438	9.13
Minimum Value	7.14	125	8	189	2.6	0.163	0.001	0.05	0.023	2.57
Maximum Value	8	207	3140	524	20	1.01	0.583	0.26	7.48	17.1
Num. Values	6	6	6	6	6	3	6	3	6	6
JAN 1992 - DEC 1992										
Mean	7.37	141	26	1152	27.6	2.03	28.25	0.73	0.111	99.9
Minimum Value	5.8	29	10	115	1.46	0.006	0.001	0.03	0.033	0.06
Maximum Value	7.99	235	60	3020	50.4	5.99	113	2.1	0.29	396
Num. Values	4	3	4	3	4	3	4	3	4	4
JAN 1993 - DEC 1993										
Mean	7.68	128	176	314	3.585	0.147	0.113	0.18	0.68	9.22
Minimum Value	7.38	107	8	230	3.06	0.136	0.007	0.17	0.06	7.37
Maximum Value	7.9	148	674	391	4.07	0.157	0.29	0.2	2.48	13.1
Num. Values	4	2	4	4	2	3	4	3	4	4
JAN 1994 - DEC 1994										
Mean	7.67	107	6.4	422	1.995	0.237	0.006	0.211	0.02	16.45
Minimum Value	7.6	97	4	387	1.3	0.229	0.002	0.197	0.01	11.7
Maximum Value	7.8	117	11	460	2.3	0.245	0.015	0.22	0.03	20.1
Num. Values	3	4	4	4	4	3	4	3	4	4
JAN 1995 - DEC 1995										
Mean	7.55	134	9	502	2.22	0.193	0.01	0.192	0.051	15.8
Minimum Value	7.2	92	1	426	1.5	0.193	0.002	0.192	0.02	9.33
Maximum Value	7.8	233	26	693	5.05	0.193	0.024	0.192	0.17	19.9
Num. Values	6	7	7	7	7	1	7	1	7	7
JAN 1990 - DEC 1995										
Mean	7.59	140	172	482	8.45	0.72	4.59	0.3035	0.489	26.7
Minimum Value	5.8	29	1	115	1.3	0.006	0.001	0.03	0.01	0.078
Maximum Value	8	235	3140	3020	50.4	5.99	113	2.1	7.48	396
Num. Values	23	22	25	24	23	13	25	13	25	25

5.6 Vangorda Creek Diversion

The development of the Vangorda Pit in 1991 required the construction of a 1000m long realignment of the Vangorda Creek around the ultimate perimeter of the pit. The diversion was built between January and April 1991 and consisted of a number of components including the headworks, a partially lined open channel, a plunge pool, culverts and drop structures. Design objectives for this facility required a system that would accommodate the 100 year storm event, would minimize seepage losses into the pit and would be stable.

The headworks of the diversion is constructed around an existing stream crossing on the road to the Swim and Dy deposits which consisted of an 1800 mm diameter CMP culvert embeded in a earth embankment. The work involved raising the embankment and extending the culvert using a smaller 1500 CMP. The diversion itself consists of an open channel lined with 2400mm diameter half-round culvert. Riprap is used to protect sideslopes of the channel from erosion.

At the downstream end of the open channel, flow discharges into a riprap lined plunge pool, which serves as an energy dissipater. Flow then enters a 2000 mm diameter culvert, which feeds a vertical, 3000mm diameter drop box structure located just north of the Vangorda haul road. The drop box redirects the flow into a 1600mm diameter pipe to the outfall on the south side of the haul road and back into Vangorda Creek.

Figure 5-11 shows the current alignment of the diversion including the 250m realigned section of the channel that was completed in early 1992 to accommodate an extension of the ultimate pit perimeter to the west. A typical section through the open channel is shown on Figure 8-12

A recent inspection of the facility (Steffen Robertson & Kirsten, 1996) showed no signs of settlement or cracking of the headworks embankment. There was no evidence of instability or surficial movement of the either the upstream or downstream faces. Some riprap has been lost above the culvert inlet.

The half-round culvert has not suffered any major structural damage during its 5-year life. However, periodic loss of fines in some sections of the channel have resulted in settlement of the riprap around the culvert. In one area, on the uphill side of the channel, subsidence of the rock cut slope has occurred on a number of occasions. The mine monitors this area regularly and removes any accumulation of debris when it occurs.

The sideslopes around the plunge pool show evidence of continual erosion and topping up of the riprap is required. The CMP's which make-up the drop box structure appear to be functioning satisfactorily.

The mine has also observed a small amount of movement on the pit wall immediately below the diversion channel which is likely due to seepage losses from the half-round culvert.

The decommissioning objective for the Vangorda Creek Diversion is to maintain the realignment around the perimeter of the pit and provide a system that will accommodate the 1 in 500 year storm event and ensure minimal long-term maintenance and monitoring requirements.

There are no chemical issues associated with this facility that need to be addressed.

5.7 Vangorda Rock Dump

5.7.1 Design and Development

The Vangorda rock dump was constructed beginning in May 1990 to contain mined waste rock from the Vangorda Pit. The two main rock types that were mined from the Vangorda pit include sulphides and phyllites. Geochemical analyses performed on the rock indicated that both rock types have the potential to generate acid rock drainage (ARD). The sulphide rock has a much higher potential for acid generation than the phyllite and therefore the original design called for the construction of separate cells to accommodate the two geochemically different rock types. The purpose of segregating the sulphides from the phyllite was primarily to allow for recovery and processing of the sulphides, should this prove economic at a later date. Segregation would also separate the seepage from each cell so that the effluent could be treated separately, if required, and seepage from the sulphides would report more directly to the collection system.

The original closure plan for the facility required the encapsulation of the mined rock with glacial till that would be stripped during development of the pit. The closure plan was to be implemented progressively as the rock pile expanded. The design required a starter dyke to be constructed from compacted glacial till, to El. 1135. Till berms would then be constructed as extensions to the starter dyke around the perimeter of the rock pile. The berms would be located to ensure an overall slope of 3:1 (Horizontal to Vertical). A till cap would then be placed over the top of the pile. The original ultimate elevation of the rock pile was 1155 metres. A till dyke would be constructed within the rock pile to segregate the two rock types. The design also required the installation of a seepage collection system.

Construction of the starter dyke, including the seepage collection system, was initiated in late May, 1990.

The design of the facility was modified in 1992 due to an expansion of the pit and a subsequent increase in the total volume of rock. The ultimate design height of the rock pile was increased from 30 metres to 67 metres. The original footprint of the rock pile was maintained.

In November 1993, Government Services of Canada (GSC) commissioned Pelly Construction Limited (PCL) to rehabilitate the Vangorda dump seepage collection system and initiate work on the resloping and capping of the waste within the dump. Steffen Robertson & Kirsten was retained by PCL to provide engineering consulting services for the work.

The work involved the upgrading of the existing seepage collection system located around the perimeter of the containment facility, recontouring rock slopes within a section of the rock pile, and providing instrumentation to monitor both the physical stability of the rock pile, and any impact on the groundwater quality. Upgrading of the seepage collection channel started in March 1994 and was completed by mid June 1994. The resloping of the waste rock and the placement of the till cover

began on May 6, 1994 and was completed by the middle of June, 1994. The configuration of the rock dump and collection facility has not been changed since then and is shown in Figure 5-1.

During construction of the seepage channel and the resloping of the rock pile, permanent instrumentation was installed to monitor seepage from the waste rock pile, potential seepage losses from the channel and piezometric heads that may develop in the existing till berm around the pile. The instrumentation included weirs, groundwater wells and piezometers.

The area of the Vangorda Dump contains material not originally expected to be in the dump. This material represents the partly oxidized cap of the ore deposit which proved to be difficult to process. The oxidized material contained cyanide soluble copper which interfered with the selective flotation of lead and zinc. It was discovered that this material contained most of its copper enrichment in the fine fraction thus the ore was screened, the coarse fraction milled and the fine fraction left in the Vangorda Dump. Unfortunately, the material was dumped in an area of shallow bedrock east of the extensive till blanket that underlies the bulk of the Vangorda Dump. The material also appears to have been placed directly on the drainage divide between the sulphide and phyllite portions of the dump. The phyllite dump has prograded over approximately half the oxide fines. The material is heavily loaded with soluble metal salts and appears to have significantly influenced the chemistry of Vangorda Dump Drains. Originally it was thought that this material primarily influenced the sulphide cell drains however it is now clear that Drains 2 and 3 are potentially affected as well.

5.7.2 Waste Rock Geochemistry

There are five dumps or stockpiles associated with the Vangorda open pit:

- Vangorda Main Dump (16×10^6 tonnes)
- low grade stockpile at open pit (510,000 tonnes at close of Curragh's mining, 439,000 tonnes now and currently being removed);
- oxide fines stockpile on Main Dump (225,000 tonnes);
- 2 small dumps at north and south end of pit, placed in 1995/96 (total <70,000 tonnes).

The estimated composition of the Vangorda Main Dump is shown in Table 5.3. Also shown in this table is the original prediction of the composition of waste rock, from the Vangorda Plateau Water Licence Application, for comparison.

In addition to the waste rock, there is a stockpile of 225,000 tonnes of oxide fines as described in section 5.2.1. The low-grade stockpile at the southeast end of the pit is currently being mined and milled, and will be completed in December 1996. Two small waste rock dumps were also recently developed near the Vangorda pit (and draining to the pit).

Table 5.3
Dump Composition

Dump	Composition	Tonnage As-built tonnes	% of total %	Tonnage IEE pred. tonnes	% of total %
Vangorda Main	Phyllites				
	phyllite including calcareous, non-calc. and chloritic	213,200	1%	88,560	1%
	carbonaceous phyllite	882,700	6%	366,660	5%
				455,220	
	Vangorda formation	1,095,900	7%	455,220	6%
	Mt. Mye non. calc. phyllite	7,295,600	46%	3,030,480	38%
	altered phyllites	4,608,500	29%	1,914,300	24%
	subtotal phyllites	13,000,000	81%	5,400,000	67%
	Sulphides				
	massive pyritic quartzite	422,577	0%	370,830	0%
	pyritic quartzites	845,155	1%	741,660	1%
	banded carbonaceous quartzites	1,732,268	1%	1,520,140	1%
	subtotal sulphides	3,000,000	19%	2,632,630	33%
	Total Main Dump	16,000,000	100%	8,032,630	100%
Vangorda north pit dump	Total	20,000	100%		
	sulphide	10,000	50%		
	phyllite	10,000	50%		
Vangorda south pit dump	Total	50,000	100%		
	sulphide	25,000	50%		
	phyllite	25,000	50%		
stockpile	Vangorda Pit Stockpile - sulphide	510,000	100%		
oxide fines	Vangorda Oxide Fines	225,000	100%		

The geochemistry of the waste rock was evaluated in some detail in the Vangorda Plateau IEE and the Water Licence Application (see section 2.4.5. of the IEE report). The testing included both static and kinetic testing; the kinetic work using 1 kg humidity cells. Additional rock sampling and testing was conducted for this closure plan document with the objective of characterizing rock types which were not addressed in detail in the earlier testing:

- the meta-intrusives which are now recognized as a potential source of nickel and cobalt;
- the Vangorda oxide fines and the low-grade stockpile material; and,
- additional rock samples from the Grum deposit which was not tested in as much detail as Vangorda in the earlier work.

Thus the results presented in this report for Vangorda and Grum in this report are not intended to represent the entire range of waste rock geochemistry for each deposit, but rather to complement the earlier sampling program for the IEE and Water Licence Development with more detailed sampling and testing of specific rock types. The discussion of rock geochemistry from the Water Licence Application is still considered applicable to the evaluation of waste rock geochemistry for both deposits and the sample data are included in this evaluation of dump and pit geochemistry.

Static testing indicates that the Vangorda sulphides, and to a lesser extent the phyllites, are potentially acid generating and release metals such as zinc into solution. Kinetic testing indicates that the zinc is released over a wide range of pH values and reports to drainage water even at near neutral conditions. These observations are supported by the 1995/96 testing program. The results of the static testing on the Vangorda rock samples are presented in detail in Appendix H.

The ABA and metals data for the baritic (oxide) fines material and for the low-grade stockpile samples are summarized below in Tables 5.4 and 5.5. These materials are potential acid generating and also contain sulphate; indicating that the material does oxidize and generate acidity.

Table 5.4

ICP Results on Selected Vangorda Samples (1995 program)

Sample	Description	As	BA	Ca	Cd	Co	Cr	Cu	Fe	Mg	Mn	Ni	Pb	Zn
UNITS		ppm	PPM	%	ppm	ppm	ppm	ppm	%	%	ppm	ppm	ppm	ppm
METHOD		ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP
Stockpile and Pit - Vangorda														
AR/VP/01	massive sulphide with phyllites	27	27	0.62	0.1	159	9	2505	>15.00	0.46	3619	70	7653	8821
AR/VP/010	massive sulphide (fresh)	3980	7	1.31	0.1	400	101	2880	15	0.44	1727	48	10000	9652
AR/VP/02	diss. sulphides	1310	33	0.48	23.3	54	43	658	8.41	0.51	1186	62	8504	22700
AR/VP/03	massive sulphides	1	66	2.59	0.1	112	1	2488	>15.00	0.70	5029	72	24300	23900
AVERAGE		1325	33.25	1.25	6	181	39	2128	>13.35	1	2890	63.00	18051	16081.66
Baritic Fines Samples - Vangorda														
AR/VP/96-1		529	53	0.32	0.1	100	62	5555	19.9	0.31	1704	54	33400	27500
AR/VP/96-2		348	56	0.18	0.1	89	62	4462	17.5	0.2	673	41	32500	10000
AR/VP/fines		315	51	0.39	0.1	90	18	4834	15.00	0.33	2392	28	>10000	>10000
AVERAGE		397	53.33	0.30	0.10	93	47	4950	17.47	0.28	1590	41	>16051	>16081

Table 5.5

Acid Base Accounts on Selected Vangorda Samples (1995 program)

Sample Label	Description	PASTE	S(T)	S(SO4)	AP	NP	NET NP	NP/AP
		pH	%	%	kg $CaCO_3$/tonne			
Stockpile and Pit - Vangorda								
AR/VP/01	massive S with phyllites	5.54	22.60	0.21	700	0.0	-700	0.0
AR/VP/02	diss. sulphides	5.34	8.16	0.29	246	0.0	-246	0.0
AR/VP/03	massive sulphides	6.81	27.80	0.32	859	120.6	-738	0.1
AVERAGE		5.60	19.52	0.27	601	40.2	-561	0.1
Baritic Fines Samples - Vangorda								
AR/VP/96-1		5.16	18.2	0.73	546	0.0	-546	0.0
AR/VP/96-2		3.37	17.5	1.39	503	0.0	-503	0.0
AVERAGE		3.66	17.85	1.06	525	0.0	-525	0.0

Short term extraction testing was done on samples of the baritic fines, as there is concern that this material is both acid generating and also a significant potential source of dissolved metals. These fines are stored in two smaller piles which are slightly different in appearance, with two samples from the more oxidized looking pile (with higher field conductivity values and lower paste pH) and one sample from the second pile (AR/VP/96-1). The samples for extraction testing were 100% minus -0.25".

Extraction testing indicates that there is a high stored soluble load of several metals in these fines, when compared to extraction testing on all other samples on the Vangorda Plateau. Results are provided in detail in Appendix H and I and are summarized in Table 5.6. For comparison, results are also shown for a sample on fresh sulphide from the Vangorda pit collected in 1996. Drainage from this material therefore could significantly affect the measured dump drainage chemistry for the Vangorda Dump, as discussed in the following section and quantified in the Site Water Chemistry discussions in Chapter 6.

Table 5.6

Calculated Soluble Extractions for Baritic Fines and Fresh Sulphides

Element	Acidity	Sulphate	Sb	As	Ba	Cd	Ca	Cr	Co	Cu	Fe	Pb	Mn	Ni	Zn
	mg/kg	mg/kg	mg/kg	g/k	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg
Fresh massive sulphide															
AR/VP/010			0.40	0.40	0.08	0.02	107	0.20	0.08	0.02	0.06	0.24	6.76	0.04	5.1
Baritic (oxide) fines															
AR/VP/96-1	1256	3908	0.80	0.80	0.04	5.96	498	0.40	8.90	21.20	3.14	4.40	226.00	2.54	850
AR/VP/96-2	6240	6844	4.00	4.00	0.20	5.60	218	2.00	5.20	117.00	690.00	1.00	210.00	1.80	2220
AR/VP/fines	7380	7380	4.00	4.00	0.20	21.16	840	0.20	16.40	127.20	185.20	0.80	796.00	6.80	4400

5.7.3 Surface Water Quality

The water chemistry of the Vangorda Dump is monitored in four ways; the series of six drains from the dump, the collection ditch to which all dump drainage reports, groundwater wells around the dump, and indirectly by the Little Creek Dam to which Drains 5 and 6 and the collection ditch report although other site water also reports here as discussed in the water balance. The chemistry of the Vangorda Dump drains has been compiled and reviewed in the company annual reports and reviewed with respect to specific technical interests by researchers including DIAND Water Resources (Mr. P. Roach) and MEND (Mr. S. Day and Mr. T. Higgs, personal communication).

Analysis of the data for both the Vangorda and Grum sites is somewhat different than for Faro in that the decision has been made to collect and treat all drainage for both dumps, and to cover at least a portion of each dump. These decisions were based on testing prior to start-up of the site which showed that oxidation, acid generation and metal leaching would be expected for the majority of the Vangorda waste rock, and for a lesser portion of the Grum waste rock (i.e. the phyllites at Vangorda are predicted to be acid generating however at Grum, some of the phyllites have significant carbonate content and potential to generate alkalinity). Thus, for closure planning, the critical aspects to the water quality are the concentrations and loadings of

contaminants currently issuing from the dump, the suite of contaminants to determine if the planned water treatment requirements are appropriate, and any indication of controls on dump drainage water chemistry that may change in the longer term.

Water chemistry data since 1988 are summarized in Appendix E. The dump drains 1 through 6 were completed in 1991 and thus water chemistry since 1992, with waste rock placed over and draining to these drains, is most relevant to the current evaluation.

The majority of the flow from the dumps is seen consistently in Drains #2 and #3 on the south side of dump and Drains #5 and #6 on the north side of the dump. The catchment for Drains 5 and 6 is primarily the sulphide cell area of the Vangorda Dump and includes the area where the oxide fines from the Vangorda pit are stored. Drains 2 and 3 collect drainage from phyllite and mixed phyllite/sulphide waste rock but also appear to receive drainage from the oxide fines.

Water from all of the drains show evidence of oxidation and metal leaching. The pH in Drains 3 and 4 is near neutral, but is decreasing from 1994 values of 7.6 and 6.5 to 6.7 and 6.1 respectively. Average sulphate concentrations have increased over the same period by about 300 mg/L to about 1600 mg/L in Drain 2 and 3057 mg/L in Drain 3. There is still considerable alkalinity in these drains, with average concentrations in 1995 of 252 mg/L in Drain 2 and 159 mg/L in Drain 3. In this pH range, below pH 7 and with elevated sulphate, such alkalinity values do not provide a significant control on dissolved zinc concentrations.

Metal concentrations in Drain 2 are lower than in Drain 3, but show an increase in 1995 (2 samples) compared to the sample in 1994. Zinc concentrations in the range of 65 mg/L are seen in Drain 2.

Metal concentrations in Drain 3 are elevated but have not changed significantly since 1994. Although the dissolved metal data are limited, generally at least 90% of the metal is present as dissolved metal. There is one questionable data point for zinc in 1995 that is reported as 19 mg/L but may in fact be 190 mg/L; consistent with the proportion of the total metals for all other samples. Dissolved concentrations of key metals are approximately 253 mg/L Zn, 0.01 mg/L Pb, 4.2 mg/L Ni, 2.7 mg/L Co and 0.002 mg/L Cu.

Drains 5 and 6, on the north or northwest side of the dump, drain directly to the Little Creek Dam. The chemistry in Drain 6 is similar to that of Drain 3, although with higher total concentrations of zinc, cobalt and sulphate. The drainage is near neutral at pH 6.3 with about 150 mg/L alkalinity. Dissolved concentrations of key metals are approximately 690 mg/L Zn, <0.25 mg/L Pb, 4.7 mg/L Ni, 8 mg/L Co and 0.008 mg/L Cu.

Drain 5 is the only drain which shows acidic pH values and very low alkalinity (averaging 11 mg/L). Sulphate concentrations are high, at 7140 mg/L. Dissolved metal concentrations are the highest measured on site with, for example, one sample at 1520 mg/L Zn, 12 mg/L Co and 51 mg/L Fe.

Calcium and magnesium concentrations are high in recent data for Drains 3, 5, and 6, ranging from 385 to 599 mg/L Ca. This suggests that there are neutralization reactions occurring within the dump, sufficient to maintain neutral conditions at Drains 3 and 6, but not at Drain 5. This could

result from the loading of stored acidity associated with the oxide fines stockpile, as discussed in section 5.7.2 and in the site water chemistry (Chapter 6).

Station V21 is located in the Vangorda Dump collector ditch, although a proposal has been made to move this station to the discharge end of the ditch rather than the centre. The flow at this station is intermittent and data are limited, particularly for the period during the shutdown. Drainage at the station includes both the drains from the dump, and runoff from the surface of the dump. Flow data for this station with intermittent flows are insufficient to allow calculation of the proportion of precipitation which reports as infiltration compared to runoff. In addition, it is expected that the flow and the chemistry data could be quite variable from this station due to the intermittent flows and the potential for evaporation and seepage within the collector ditch and the sump.

The data for this station are summarized in Appendix E. The water chemistry at this station clearly represents a combination of the loadings from the drains (except Drains 5 and 6) and dilution from runoff. Generally the drainage is near neutral with pH values ranging from 6.1 to 7.8. Alkalinity values are lower than in the drains, but still appreciable averaging 126 mg/L in 1995. The variability in the measured concentrations over the year suggests that there are varying proportions of infiltration versus runoff; alkalinity values are lowest in April (start of freshet) when the highest proportion of runoff would be expected. Highest concentrations of alkalinity and hardness ([Ca] + [Mg]) are seen in May, probably corresponding the highest infiltration and flushing of stored products from the winter (low infiltration) months. Sulphate concentrations are lower than in the drains, averaging 1030 mg/L. A similar seasonal pattern is suggested by the data for sulphate and for zinc. Total metal concentrations also appear to be related to the suspended solid load, although data are limited.

5.8 Grum Open Pit

5.8.1 General Description

The Grum deposit consists of several horizons which form a complex fold pattern as already discussed in section 5.2.2 and illustrated in Figure 3.4. Due to the local geometry of the deposit there are two separate zones that comprise the surface mineable Grum ore deposit. These are the "main" deposit to be mined in the "IV" pit, as outlined in the original project proposal, and an additional zone to the southeast known as the Champ Zone. The Champ Zone is so close to the main deposit that the two pits will actually merge and become one overall pit. In addition to the surface mines, there will be underground mining carried out from the bottom of the Grum pit in the final stages of the operation and in the years immediately following. The details of the potential underground follow-up to the surface mining phase are not worked out yet, but it is clear that some mining will be carried out as was the case at Faro and that it will largely follow the structure down plunge from near the pit bottom (see Figure 3.5).

The Grum open pit is a large pit being developed in a number of stages. Currently, Phase 1 is still several months from completion and the upper several benches of Phase 2 have been mined.

The ultimate pit is always under evaluation and is not yet finalized, however, for the purposes of this report, the Champ Zone extension of the Grum IV pit is included despite there not yet being a decision to mine this area. The implications of not mining this area are minimal as the closure issues for the Champ and no Champ pits are the same. There are no material changes to the closure plan that would result from not mining Champ.

The Grum pit will be 1100m. long, 700 m. wide and up to 900m. deep. A new access slot is being excavated at the southeast end of the pit (Figure 5-12) and ultimately the Champ zone would be mined southwest of that slot. The invert of the pit with the slot acess will be in the range 1230 to 1235 m. elevation, approximately 30 m. lower than the previous pit design.. The total pit size is unchanged from the original proposal since the extra stripping in the slot is compensated for by stripping avoided in the main pit. The height volume relationship for the Grum pit is shown in Table 5.7. The relationship with and without the Champ one is shown. The pit will be 4.4 million m^3 larger with Champ being mined. The mining of Champ is irrelevant to the elevation of the pit invert.

Table 5.7 shows the quantities of waste rock in the Grum IV pit as recalculated since the original mine plan in the 1989 I.E.E. report. In addition to the waste there will be 24.5 million tonnes of ore mined. This table does not include waste rock from the champ zone which amount to 4.4 million cu. M. (approximately 9 million tonnes) including approximately 1.3 million tonnes of ore.

Table 5.7
Grum Pit Rock Waste Quantities

	New 1992 Block model	Original Plan in 1988 IEE
A. Vangorda formation		
calcareous phyllite, non calcareous phyllite, and chloritic phyllite	99,209,680	116,105,785
carbonaceous phyllite	20,750,230	5,527,763
Sub-total Vangorda formation	119,959,910	121,633,548
B. Mt. Mye formation		
non calcareous phyllite,	24,146,190	19,465,589
C. Altered Phyllites	2,193,370	2,162,843
D. Sulphides		
massive pyritic sulphide	164,020	192,980
pyritic quartzites	688,850	1,146,030
banded carbonaceous quartzite	2,969,360	4,984,024
Sub-total sulphides	3,822,230	6,323,034
Total Rock Waste (no till)	150,121,700	149,585,014
Total treated as sulphide waste (C+D)	6,015,600	8,485,877

It is clear from Table 5.7 that the bulk of the rock waste to be excavated from the Grum Pit is phyllite, either calcareous or non-calcareous, and that there is appreciable but relatively little sulphide waste rock. The nature of these rocks is discussed in detail in section 5.2.2. Most of the rock to be excavated from the Champ pit will be calcareous phyllite along with a small amount of sulphide waste.

Figure 5-12 shows a plan of the geology of the Grum ultimate pit (including champ). The bulk of the sulphide in the pit walls is relatively deep in the pit and there will be no exposure of sulphides above flooding level assuming the pit floods to it's invert.

There is an existing set of underground workings at Grum which are largely below the pit now and eventually will be mined out by the pit. The portal for the access ramp to these workings is at approximately 1260 m. elevation thus there is no concern for water flow from these workings.

Future underground workings would also not provide a potential pathway out of the pit as the portal would be established in ore in the pit and follow the pit down plunge to the northwest.

The height capacity data for the Grum Pit at ultimate extent are shown in Table 5.8 which were obtained from pit bench plans (Anvil Range Mining Corporation, pers. comm.). It is not ceratin whether the Champ Zone to the south of the pit will be mined. The table lists the ultimate pit volumes with and without mining the Champ Zone. The total volume of the Grum open pit without the Champ Zone will be 42.6 million m^3 at closure. Excavation of the Champ Zone would increase the capacity to be excavated by approximately 4.4 million m^3.

Table 5.8.

Height-Capacity Data for Grum Pit - Ultimate Pit Design (Design IV with and without Champ Zone)

Bench (m AMSL)		Grum - IV Pit w/o Champ Zone		Grum - IV Pit w/ Champ Zone	
		Volume	Cum. Vol.	Volume	Cum. Vol.
toe	crest	m^3	m^3	m^3	m^3
1048	1054	70,760	70,760	70,760	70,760
1054	1060	168,830	239,590	168,830	239,590
1060	1066	237,970	477,560	237,970	477,560
1066	1072	309,710	787,270	309,710	787,270
1072	1078	367,470	1,154,740	367,470	1,154,740
1078	1084	431,580	1,586,320	431,580	1,586,320
1084	1090	493,160	2,079,480	493,160	2,079,480
1090	1096	551,210	2,630,690	551,210	2,630,690
1096	1102	631,900	3,262,590	631,900	3,262,590
1102	1108	703,980	3,966,570	703,980	3,966,570
1108	1114	788,950	4,755,520	788,950	4,755,520
1114	1120	853,000	5,608,520	853,000	5,608,520
1120	1126	947,380	6,555,900	947,380	6,555,900
1126	1132	1,011,440	7,567,340	1,011,440	7,567,340
1132	1138	1,098,760	8,666,100	1,098,760	8,666,100
1138	1144	1,162,690	9,828,790	1,162,690	9,828,790
1144	1150	1,257,870	11,086,660	1,257,870	11,086,660
1150	1156	1,333,460	12,420,120	1,342,640	12,429,300
1156	1162	1,432,150	13,852,270	1,450,400	13,879,700
1162	1168	1,513,180	15,365,450	1,553,600	15,433,300
1168	1174	1,626,950	16,992,400	1,788,820	17,222,120
1174	1180	1,704,680	18,697,080	1,907,380	19,129,500
1180	1186	1,821,330	20,518,410	2,053,310	21,182,810
1186	1192	1,907,440	22,425,850	2,166,680	23,349,490
1192	1198	2,041,940	24,467,790	2,304,010	25,653,500
1198	1204	2,140,400	26,608,190	2,436,140	28,089,640
1204	1210	2,299,910	28,908,100	2,595,880	30,685,520
1210	1216	2,405,470	31,313,570	2,734,650	33,420,170
1216	1222	2,606,850	33,920,420	2,945,960	36,366,130
1222	1228	2,721,650	36,642,070	3,100,550	39,466,680
1228	1234	2,950,230	39,592,300	3,319,020	42,785,700
1234	1240	3,047,250	42,639,550	3,430,130	46,215,830

5.8.2 Water Balance

The Grum Pit is located near a drainage divide and, accordingly, commands a total catchment area much less than the Vangorda Pit. The total catchment area is some 1.5 km^2 in extent. The Grum Northeast Interceptor Ditch controls about 0.5 km^2 of this total. Thus, the incremental catchment of the Grum Pit is 1.0 km^2.

Because of the aquifer dewatering program at the Grum site, it is more instructive to examine the water balance of the pit's incremental catchment than to focus exclusively on the pit (as was done for the Vangorda Pit). The inflows to this incremental catchment are:

- direct precipitation on the incremental catchment;
- groundwater originating from outside the incremental catchment; and,
- ditch leakage from the Grum Northeast Interceptor Ditch.

Outflows comprise:

- flows abstracted from the local aquifer using the Grum deep interceptor wells;
- groundwater seepage to Grum Creek;
- evapotranspiration from the incremental catchment; and,
- flows removed from Grum Pit and delivered to the Water Treatment Plant.

The change in storage within the local aquifer is also an important component of the water balance.

The available data are insufficient to accurately assess the groundwater, ditch leakage and aquifer storage components of the water balance. However reasonable estimates can be made for the other inflow and outflow components. During 1995, the average dewatering rate for the local aquifer was about 8 L/s. A further 2 L/s was removed from the bottom of the pit. Based on regional analysis, the net difference between the precipitation and evaporation components for the year 1995 was about 6 L/s (expressed as an average over the entire year).

5.8.3 Geochemistry

The water quality of the Grum open pit is monitored primarily at V23, which represents the open pit sumps. Groundwater is pumped from the aquifer prior to discharge into the pit. All water pumped from the open pit is treated prior to discharge.

Groundwater chemistry is discussed in section 5.4. Surface water chemistry data are summarized in Table 5.9, and shown graphically in Figures 5-7 and 5-8.

The annual average water chemistry data for key parameters is summarized in Table 5.9, for the period since stripping and dewatering began at Grum in 1991. Pit waters are consistently alkaline, with average pH values ranging from 7.2 in 1991 to pH 8 in 1995. Alkalinity values in the Grum pit have generally increased over the period of mining, as shown in Figure 5-7.

There is some evidence of sulphide oxidation in the sulphate concentrations measured in the pit sumps. The data in Table 5.9 and Figure 5-8 show that sulphate concentrations are reasonably consistent over time but are elevated compared to background waters and to groundwater flow into the pit. Sulphate concentrations averaged 217 mg/L from 1991 to 1995. Average annual calcium concentrations are generally about 100 mg/L over the same period indicating that there is no change evident in the rate or extent of acid generation and neutralization in the pit over this period as pit waters remain near neutral to alkaline. This reflects the lower pyrite content of much of the Grum pit wall rock compared to Vangorda and the less altered, strongly calcareous phyllites that tend to be found at Grum, but not at Vangorda.

Total metal concentrations appear to be variable over time but this is due primarily to the shutdown period when there was no mining and therefore very low suspended solids produced from the pit sumps. As shown in Table 5.9, metal concentrations in 1995 are higher than during the shutdown, averaging 0.2 mg/L. These values are still lower than in the early years of mining.

Dissolved metal concentrations are not typically measured but are useful to evaluate oxidation and metal leaching. The proportion of dissolved metals over time can estimated from data during a period of very low suspended solids. During the shutdown there would be increased settling time in the pit compared to during mining and thus most of the metals present in the water samples would be as dissolved metals or fine particulates which have poor settling characteristics. Total metal data (which comprises both particulate and dissolved metals) shows low concentrations of lead and zinc - averaging less than 0.02 and 0.04 mg/L respectively - thus dissolved concentrations are also low.

Table 5.9
Annual Averages for Station V23

STATION: Grum pit water (V23)

Date	LAB pH pH unit	TOTAL ALKALINI mg/L	SUSPEN SOLIDS mg/L	TOTAL SULPHA mg/L	AMMONI NITROG mg/L	TOTAL COBALT mg/L	TOTAL NICKEL mg/L	TOTAL LEAD mg/L	TOTAL ZINC mg/L
JAN 1991 - DEC 1991									
Mean	7.17	105	26	234	0.789	0.032	0.11	0.016	0.312
Minimum Value	5.25	12	5	188	0.01	0.032	0.11	0.004	0.026
Maximum Value	7.9	149	56	295	1.42	0.032	0.11	0.06	1.25
Num. Values	6	6	6	3	6	1	1	6	6
JAN 1992 - DEC 1992									
Mean	7.94	158	595	172	0.664	0.006	0.07	0.247	0.319
Minimum Value	7.86	108	4	133	0.01	0.006	0.07	0.002	0.022
Maximum Value	8.07	207	1160	210	1.35	0.006	0.07	0.45	0.571
Num. Values	3	2	3	2	3	2	2	3	3
JAN 1993 - DEC 1993									
Mean	8.16	258	50	232	4.525	0.022	0.09	0.024	0.0307
Minimum Value	7.9	214	4	157	3.45	0.006	0.01	0.003	0.01
Maximum Value	8.55	301	119	365	5.6	0.036	0.17	0.06	0.05
Num. Values	4	2	4	4	2	3	3	4	4
JAN 1994 - DEC 1994									
Mean	8.2	143.3	4	313	1.32	0.011	0.1057	0.02	0.043
Minimum Value	8.1	77.3	4	282	1	0.001	0.079	0.01	0.01
Maximum Value	8.3	192	5	330	1.67	0.017	0.12	0.03	0.075
Num. Values	4	4	4	3	4	3	3	4	4
JAN 1995 - DEC 1995									
Mean	8.02	283	25	166	10.66			0.027	0.206
Minimum Value	7.71	1	3	25	0.205			0.02	0.03
Maximum Value	8.3	485	68	298	35.3			0.05	0.45
Num. Values	5	5	5	6	6			6	6
JAN 1990 - DEC 1995									
Mean	7.84	181.5	104	217	4.05	0.016	0.093	0.051	0.19
Minimum Value	5.25	1	3	25	0.01	0.001	0.01	0.002	0.01
Maximum Value	8.55	485	1160	365	35.3	0.036	0.17	0.45	1.25
Num. Values	22	19	22	18	21	9	9	23	23

5.9 Grum Rock Dumps

There are three Grum Waste Dumps: the Till Dump, the Southwest Dump and the Main Dump. The Main and Till Dumps are being built on the moderate northwest slope of the Vangorda Creek valley and the Southwest Dump is in a relatively flat saddle on the crest of the ridge between the two branches of Vangorda Creek southwest of the Grum Pit. The Till Dump is currently complete but the other two dumps are only about 20% complete.

The Till Dump consists almost entirely of glacial till stripped from phase 1 of the Grum Pit. The dump has been built in five 15 m. lifts with setbacks resulting in gentle slopes suitable for re-sloping to 3H to 1V. A portion of the northeast side of the dump has already been re-sloped by Anvil Range. The top of the dump is nominally at 1305 m. elevation after placement of the last lift by Anvil Range in 1995. The Till Dump contains approximately 24 million tonnes of glacial till stripped from the buried valley in the northeast part of the pit.

The Southwest Dump consists of mainly calcareous phyllite of the Vangorda formation with about one third of the dump to be non-calcareous phyllite. The volume of the dump is approximately 20 million cu. m.. This dump mostly drains to the same area as the main dump and would be serviced by the same seepage collection system; the west edge of the dump extends into the drainage of the West Fork of Vangorda Creek however only rock from the pre-stripping of Phase 3 of the Grum Pit would be placed there; this material will be entirely calcareous phyllite and include no sulphide waste. The Southwest Dump could also contain some glacial till however it now appears that there will be no till placed there. The Southwest Dump has been enlarged from the original proposal by extending it 200 m. to the west and increasing the height to 1325 m. from 1315 m.. The Southwest Dump will have a low grade stockpile placed on a platform of calcareous phyllite in it's central area. Plans for a crusher and grinding plant in the central area as well have not materialized thus this low point in the dump could be filled with phyllites from the upper benches of the pit.

The largest dump is the main dump. This dump was to be a rock dump and was to contain a zone of segregated sulphide waste in a drainage controlled area. The dump has recently been reconfigured in response to necessary revisions to the size of the sulphide cell and the foundation conditions as well as it's construction history and revisions to the pit design.

The Main dump is being built on a moderate slope dipping 6 to 10 degrees southeast to south. There are local areas as steep as 12 degrees and some flat areas. Permafrost was not identified on this slope. The main dump will cover two minor areas of groundwater seepage. The most significant areas of groundwater discharge are in the Grum Creek valley and will be avoided by the dump footprint. The surface of this southwest slope is mantled by variable thickness of glacial till (0.1 to 6.7 m.). The till is generally overlain by fluvial sand and gravel which varies from 0.3 to 5.5 m. in thickness. A thin organic soil layer (0.2 to 0.7 m. deep) covers most of the area. Bedrock is generally within 7 or 8 m. of ground surface and in many places bedrock is immediately beneath the thin organic soil layer. Bedrock is generally easily excavated; test pits were dug 0.1 to 3.7 m. deep in bedrock without difficulty.

The proposed main dump will consist of seven lifts, mostly 30 m. high each (nominal crest elevations of 1150, 1180, 1210, 1240, 1270, 1300 and 1325 m. above sea level). Although dumping is currently concentrated in areas high on the hill side, the ultimate dump will be a series of overlapping lifts wedging out against the hill side, each providing a foundation and buttress for the lift above it. The dump will be built to this overall design slope of 2.5H to 1V or flatter. Each lift will be end dumped at the angle of repose and a setback will be provided for each lift to reach the overall design slope. Lift crest elevations shown on Figure 5-12 are nominal elevations; the dump tops will be graded to control runoff and minimize erosion on the dump face. A setback of 300 m. from Vangorda Creek will be maintained. A seepage collection ditch will be located approximately 50 m. downslope from the toe of the main dump.

Planned sulphide deposition in the main dump is delineated on Figure 5.13. Sulphides will be deposited on four lifts; 1240, 1270, 1300 and 1325 (crest elevations). Sulphides will be dumped on a minimum base of 10 m. of non-acid generating phyllite to isolate the sulphides from the original ground surface which is expected to be a locus of seepage flow and to provide buffering capacity for seepage through the sulphide cell. The sulphides will be dumped 45 m. back from the final dump face, so that there will be no direct exposure of sulphides to precipitation and runoff once the dump is completed. Internal till covers between sulphide lifts are currently under review by Anvil and government reviewers. In Anvil's opinion it seems unlikely that these covers would be effective in providing significant additional seepage control. This review will be completed by March 1997 and a final position developed. In the meantime there will be no overlapping of the sulphide cell lifts without the placement of a truck compacted till layer at least 1 m. thick.

The rationale of locating the new sulphide cell was to encompass the existing sulphides, adhere to the required setback from both original ground and dump face and lastly to keep the cell west of the western drainage divide for Grum Creek and east of the eastern drainage divide for the westerly seepage area below the dump. The volume of the sulphide cell allowed by these constraints is generous as shown in Table 5.10

Table 5.10
Volume of the sulphide cell

Lift Crest elev. (m.)	volume of cell (cu.m.)
1325	2,658,500
1300	3,796,929
1270	4,247,010
1240	2,861,640
Total cell	13,564,079

The ultimate size of the Main Dump is dependent on the ultimate size of the Grum pit but currently is planned to contain 108 million tonnes of rock, most (92 million tonnes) of which will be calcareous phyllite and about 5.2 million tonnes will be sulphides and altered phyllite.

5.9.1 Design and Development

5.9.2 Waste Rock Geochemistry

The estimated composition of the final Grum Dump, including the Southwest Dump, is shown in Table 5.11. This is based on the current design proposed in "Proposed Modifications to the Grum Waste Dump" submitted to the Yukon Territory Water Board by Anvil Range Mining Corporation in May, 1996.

Table 5.11

Grum Dump Composition

Dump	Composition	Tonnage tonnes	% of total
Grum Main	**Phyllites**		
	phyllite including calcareous, non-calc. and chloritic	76,053,018	
	carbonaceous phyllite	15,906,892	
	summation of Vangorda formation in Main Dump	91,959,910	85%
	Mt. Mye non. calc. phyllite	10,146,190	9%
	altered phyllites	2,193,370	2%
	subtotal phyllites	104,299,470	96%
	Sulphides		
	massive pyritic quartzite	164,020	0%
	pyritic quartzites	688,850	1%
	banded carbonaceous quartzites	2,969,360	3%
	subtotal sulphides	3,822,230	4%
	Total Main Dump	108,121,700	100%
Southwest Dump	calcareous phyllites	42,000,000	100%
	sulphides	-	

Waste rock samples for static and kinetic testing were collected from each of the four lifts under construction in August 1995. The results are summarized according to rock type assemblage in Tables 5.12 and 5.13, and are presented in detail in Appendix H. For comparison with the results presented in the Water Licence Application, an overall estimate of metal concentrations and acid generation and neutralization potential is provided in Tables 5.14 and 5.15.

Table 5.12

ICP Results on Waste Rock Samples (Summary)

	Al	As	BA	Ca	Cd	Co	Cr	Cu	Fe	Mg	Mn	Ni	Pb	Zn
UNITS	%	ppm	ppm	%	ppm	ppm	ppm	ppm	%	%	ppm	ppm	ppm	ppm
METHOD	ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP	ICP
NON-CALC. PHYLLITE														
Average	1.09	103	229.5	0.47	0.1	17	35	48	3.75	1.04	316	38	309	699
Minimum	0.24	1	173.0	0.44	0.1	14	29	31	3.71	0.82	218	37	73	150
Maximum	1.94	204	286.0	0.50	0.1	20	41	65	3.78	1.26	413	38	544	1247
CALCAREOUS PHYLLITE														
Average	0.86	82	154.8	3.35	0.10	13	33	30	2.95	1.57	454	31	96	151
Minimum	0.44	1	72.0	2.60	0.10	12	31	23	2.69	1.50	426	27	56	85
Maximum	1.16	155	300.0	3.96	0.10	14	36	37	3.24	1.70	477	34	141	235
CHLORITIC PHYLLITE														
Average	0.58	603	117.5	2.02	0.10	32	133	203	7.62	2.22	772	161	2844	11650
Minimum	0.56	589	81.0	1.99	0.10	31	132	129	5.57	1.81	673	142	2322	>10000
Maximum	0.60	617	154.0	2.04	0.10	33	134	277	9.66	2.63	870	180	3366	13300
GRAPHITIC PHYLLITE														
Average	1.29	146	230.3	1.36	0.10	18	54	79	4.24	1.37	366	53	672	2581
Minimum	0.31	1	57.0	0.49	0.10	17	35	26	3.52	0.75	168	38	56	95
Maximum	2.05	367	374.0	2.81	0.10	21	78	186	5.62	2.21	611	71	2510	>10000
SULPHIDE WASTE - GRUM														
Average	0.58	311	127.3	0.68	12.95	21	32	174	5.06	0.74	1187	62	5585	5911
Minimum	0.20	1	83.0	0.48	0.10	16	26	37	3.58	0.48	254	34	134	278
Maximum	1.67	798	230.0	0.77	26.80	25	38	355	6.43	1.10	2016	113	>10000	>10000
MIXED SULPHIDES & PHYLLITE WASTE														
Average	1.03	263	55.0	2.28	2.95	37	118	277	6.67	2.37	942	123	4393	7106
Minimum	0.32	10	20.0	1.62	0.10	27	15	44	4.22	0.95	629	51	130	643
Maximum	2.74	394	126.0	2.68	11.50	45	347	631	11.38	6.32	1307	204	>10000	>10000

Table 5.13

Acid Base Account Results on Grum Waste Rock (Summary)

Sample Label	Rock Type Description	PASTE pH	S(T) %	S(SO4) %	AP kg $CaCO_3$/tonne	NP kg $CaCO_3$/tonne	NET NP kg $CaCO_3$/tonne	NP/AP
NON-CALC. PHYLLITE								
AR/G3/11	5B6	7.67	0.40	0.00	13	9.9	-3	0.8
AR/G1/01	3G9	5.69	1.64	0.13	47	0.0	-47	0.0
Average		5.99	1.02	0.07	30	5.0	-25	0.2
Minimum		7.67	0.40	0.00	13	0.0	-47	0.0
Maximum		5.69	1.64	0.13	47	9.9	-3	0.8
CALCAREOUS PHYLLITE								
AR/G3/204-A	5B0	7.81	0.44	0.00	14	183.1	169	13.3
AR/G2/08	5B0	7.85	0.34	0.00	11	193.9	183	18.2
AR/G2/09	5B0	7.85	0.52	0.22	9	187.6	178	20.0
AR/G3/12	5B0	7.83	0.31	0.00	10	148.8	139	15.4
AR/G4/202-A	5B0	8.05	0.47	0.00	15	180.3	166	12.3
AR/G4/203	5B0	7.94	0.43	0.00	13	127.2	114	9.5
Average		7.88	0.42	0.04	12	170.1	158	14.3
Minimum		8.05	0.31	0.00	9	127.2	114	9.5
Maximum		7.81	0.52	0.22	15	193.9	183	20.0
CHLORITIC PHYLLITE								
AR/G1/02	5D4*	6.14	5.77	0.23	173	105.6	-68	0.6
AR/G4/201-C	5D4* + 4D	7.43	2.79	0.18	82	106.9	25	1.3
Average		6.42	4.28	0.21	127	106.3	-21	0.8
Minimum		7.43	2.79	0.18	82	105.6	-68	0.6
Maximum		6.14	5.77	0.23	173	106.9	25	1.3
GRAPHITIC PHYLLITE								
AR/G3/204-B	5A	7.79	0.54	0.12	13	10.8	-2	0.8
AR/G4/201-A	5A	7.68	0.79	0.12	21	13.4	-8	0.6
AR/G4/201-B	5A	7.92	0.63	0.19	14	154.6	141	11.2
AR/G2/10	5A + 5D4*	7.40	4.36	0.13	132	51.5	-81	0.4
Average		7.65	1.58	0.14	45	57.6	13	1.3
Minimum		7.92	0.54	0.12	13	10.8	-81	0.4
Maximum		7.40	4.36	0.19	132	154.6	141	11.2

Table 5.14

Range of Metal Concentrations for Grum Dumps

Dump	As ppm	Cu ppm	Cd ppm	Co ppm	Ni ppm	Pb ppm	Zn ppm
Main Grum							
Average	109	45	1	15	37	502	807
Min.	11	28	0	13	32	103	281
Max	211	66	2	16	42	847	1259
Southwest							
Average	82	30	0	13	31	96	151
Min.	1	23	0	12	27	56	85
Max	155	37	0	14	34	141	235

Table 5.15

Range of Acid/Base Characteristics for Grum Dumps

Dump	S(T)	S(SO4)	AP	NP	NET NP*	NP/AP*
	%	%	kg $CaCO_3$/tonne			
Main Grum						
Average	0.83	0.13	22.0	118	96	5.4
Min.	0.11	0.02	2.8	10	7	3.7
Max	1.22	0.32	31.8	136	104	4.3
Southwest						
Average	0.42	0.04	11.9	170	158	14.3
Min.	0.31	-	9.4	127	118	13.6
Max	0.52	0.22	14.7	194	179	13.2

5.9.3 Surface Water Quality

There are three locations at which seepage from the Grum dump is sampled; Stations V14, V15, and V16. There was no flow seen at station V14 during 1995. Station V15 is designed as a sump for seepage from the sulphide cell. Station V16 is the southeast sump which collects water from Grum Creek. In the past, Grum Creek water included some runoff from the haul road, but since 1995, the creek has received no input from that source.

The dump and the collection system for the dump is still under construction and thus it is too early to use these water chemistry data for any longer term evaluations of dump geochemistry. Measured water chemistry values will reflect not only the drainage from the dump, but also the influence of mining and construction activities, attenuation of flow within the dump, continued deposition of waste on the dump. The 1995 data for these stations is presented in the Anvil Range 1995 Annual Report to the Water Board, and discussed briefly below.

Generally, the water chemistry from the dump seepage is similar to the chemistry of the Grum pit water. Seepage pH values are alkaline, ranging from pH 7.2 to 8.1 at V15 and consistently around pH 8 and Station V16. This would be expected given the calcareous nature of much of the phyllite in the dump.

Metal concentrations are consistently low at both stations. Only iron shows any variability, with a maximum measurement of 0.23 mg/L total iron in one sample

5.10 Vangorda Plateau Haul Road

The Vangorda Plateau Haul Road is a heavy haul road developed for use by 154 tonne off highway trucks hauling ore from the Ore Transfer Pad to the Faro mill, a distance of 13 km. The road extends an additional 3 km. past the transfer pad along the south side of the Grum Pit to the pit entrance on the south side of the Vangorda Pit. The road was built by Curragh Resources

starting in October 1986 and was completed in1989. There has been significant upgrading of the road surface over the years.

The road surface is approximately 30 m. wide and there is a 2 m. high safety berm on either side of the road. The bulk of the road was built as a fill road; there are two minor cut areas, one on the east side of the West Fork of Vangorda and the other on the west side of the South Fork of Rose Creek. The road was built largely from mine waste rock selected for non-acid generating characteristics; the central 2 km. of the road was built from locally borrowed surficial deposits. Most of the rock on the Faro side of the haul road (8km.) is schist with lesser calc-silicate and minor intrusive, all from the Faro Pit. The rock on the Vangorda Plateau side is calcareous phyllite from the Grum Pit overlying glacial till locally.

The haul road crosses several major streams including the north and south Forks of Rose Creek as well as the West Fork and main stem of Vangorda Creek. The North Fork of Rose Creek crossing is a rock drain as described in section 4.5.8. A second, smaller rock drain crosses Reservoir Creek, a tributary to the fresh water reservoir. The other crossings are corrugated metal pipes of 600 to 1600 mm. diameter; 600 mm. overflow culverts exist at most crossings. Culvert crossings were sized for a 1:25 year return period flood and were not designed to allow for fish passage. The two largest fills over these culverts are the West Fork of Vangorda Creek and the main stem near the Vangorda pit. Side slopes are 2H:1V for sections built from overburden and 1.5H:1V for sections built from pit rock.

5.11 Little Creek Collection Facility

The Little Creek Collection facility is located immediately northwest of the Vangorda rock dump, at an approximate elevation of 1100 m a.s.l. in the side valley of Little Creek, a small tributary to Vangorda Creek (Figure 5-1). Upstream of the facility, Little Creek is intersected by the Vangorda Pit and by the access road to the Vangorda rock dump. The facility is located approximately 90 metres upstream of Vangorda Creek.

The Little Creek collection facility was constructed in 1990 to collect water pumped from the Vangorda Pit and seepage from the Vangorda rock pile. A detailed description of the facility is given in Steffen Robertson & Kirsten Report No. 160636/1 "Vangorda Plateau Development As-Built Construction Report For Little Creek Dam". The Little Creek collection facility consists of Little Creek Dam, an earth embankment built from local compacted till, and a storage pond ("Little Creek Pond") with a reservoir capacity of (~120,00 m^3) designed to store runoff over the winter months. Other associated collection facilities include a wet well, pump house and pipeline system to direct the water in Little Creek Pond to the water treatment plant near Grum Pit.

The crest elevation of Little Creek Dam varies from 1114.5 to 1120 m a.s.l., i.e. some 10 metres above natural ground. The side slopes are 2:1 (H:V) on the downstream side and 2.5:1 (H:V) on the upstream side. At present, the dam has no spillway and the water level in the pond is controlled by pumping to the water treatment plant. Drainage measures include a drain blanket at the base of the dam and collection ditches downstream of the dam which lead into a collection sump.

An estimated 66,500 m^3 of material were used for construction of the dam, 80% of which comprised low permeability material (glacial till) from stripping operations of the Vangorda Pit used in construction of the embankment and 20% comprising gravelly sand from within the limits of the collection pond for use in construction of the embankment shells and the drainage blankets.

5.12 Current Water Treatment Plant - closure

The current water treatment plant at Vangorda plateau is a straight lime neutralization plant (SLN) that uses a pond to settle solids, producing a low-density sludge (LDS). The capacity of the current treatment plant is 2000 Usgpm. The current treatment plant operates in compliance with the permit with some exceptions due to loss of suspended solids which can result in a minor exceedence in some metals. As a result, the discharge from the sludge settling pond is now diverted into two additional settling ponds (Sheep Pad Sediment Ponds,; see Figure 5-1). Recent monitoring suggests that these ponds are effective in lowering the suspended solids concentrations to permissible levels.

As outlined in chapter 9, the current treatment plant has sufficient capacity to treat all contaminated water from the Grum/Vangorda mine site on a seasonal basis. No significant changes to the deign of the existing facility are required to use it as a post-closure water treatment facility. The existing sludge settling ponds and clarifying ponds will be used past closure as well.

5.13 Infrastructure

In addition to the open pits and rock dumps there are various structures around the Vangorda Grum minesite that provide infrastructure support to the mining operations. These include fuel and power, de-watering facilities, buildings and light vehicle roads.

5.13.1 Roads, Borrow Areas and other disturbed land

In addition to the main Vangorda Pit to Faro Mill haul road there are numerous shorter heavy haul roads connecting the rock dumps to the open pits. These road ways are mostly on dump tops and have wide, heavily compacted surfaces. They are constructed of materials comparable to the surrounding dump material and rarely would be built of sulphide bearing material, only in the sulphide cells is this likely and there the top of the cell would be covered with till eventually.

There are also many smaller light vehicle roads around the minesite. These are un-surfaced roads generally 20 ft. or less wide and have not been strongly surface compacted by traffic. Many of these will continue to be used at closure to provide access to sampling or water management sites.

Adjacent to the pits there are several large cleared areas some of which have no known purpose other than having provided equipment passage or having served as a general work area at some time. These areas have been re-graded over the last few years and were mostly seeded in 1996 as a test of re-vegetation methods.

A large area was cleared for a contractors campsite near the office/dry building. The camp has now been removed but the area has not otherwise been reclaimed. Additional large cleared areas that have not yet been fully reclaimed are: an area near WTP cleared for future clarification ponds and related facilities, an area west of the nose of Vangorda dump which was graded but not planted during the last shutdown, cleared areas in the moose pond vicinity, a cleared borrow area above Groucho pond and the area of the "hot line" parking lot between the Grum pit and office.

The Ore Transfer Pad is at the west end of the Grum pit. The capacity of the pad is approximately 1.5 million tonnes. The pad was built on a calcareous phyllite base and is located on the drainage divide between the main stem of Vangorda Creek and the West Fork of Vangorda Creek. Ore is placed there by 154 tonne, diesel electric, pit haul trucks then rehandled to mechanical drive, articulated, 154 tonne, off highway, truck-trailer rigs for the 12 km. haul to the mill. The pad is a temporary storage site and any high grade ore is moved to the mill relatively quickly. Low grade ore is stockpiled at the north end of the pad. This material will be shipped to the mill for processing when spare mill capacity is available. A second low grade stockpile location is planned for the southwest dump. It will also be built on a calcareous phyllite base.

5.13.2 Buildings, Equipment and power

A 27 kv. main overhead power line along the haul road brings power from the Faro site transformers to transformer stations at Grum and Vangorda. Poles on this grid are single log poles. A distribution system of overhead 4167 v lines feeds power to various substations around the pit where temporary ground lines are used to connect to equipment.

There are relatively few buildings on the Grum Vangorda site. The major building is the office and dry complex. This is a two story building of approximately 1,500 square m per floor. It is of wood frame construction on a concrete slab. Exterior cladding is sheet steel.

There is a shop building of similar size and construction nearby. This building is not currently extensively used although a truck shop may be developed there later. There is also and a small water well house, an ambulance garage and ATCO trailer type building used for miscellaneous storage near the office building.

At the Water Treatment there is a small steel frame building on a concrete pad housing the plant, the lime bin and related structure and a few small temporary buildings or sheds used for storage, office and lunchroom space. A Butler building with a wooden extension is located at the old Grum exploration portal, this building was a shop for the underground project, then a truck maintenance shelter used during early days oft the Grum stripping and currently is unused.

In the ranch area (or lay down area) there are several small temporary wood frame buildings, some of which are modified ATCO trailers. In the same general area there are ten core racks and a wood frame core logging building of about 300 sq. m. floor space.

Explosives magazine for the pit area is a small wood frame building in a clearing on the ridge running along the northwest side of Vangorda Creek near tree-line There is also a small blasters shack near the water treatment plant used for minor maintenance and office purposes.

5.13.3 Shops, Fuel and Oil Storage

There are no major shop facilities at Grum, a truck shop may be developed as noted above but no new buildings would be constructed.

There are several large fuel tanks holding diesel fuel and gasoline used by the mining and support equipment. These tanks are in bermed areas but do not have impermeable liners beneath them. Over the years there have been several oil spills into these berms and it is likely that the surficial sand and gravel unit (normally 3 to 4 m. thick) is contaminated.

Fuel and lube distribution is via a small building housing pumps near the major fuel tanks at the main exit from the Grum pit. Over the years there has also been considerable diesel fuel spilled here and some contaminated soil can be expected though probably not as much as near the tank farm.

The contractors hauling ore from the Grum Transfer Pad to the Faro Mill use a shop and scale facility along the haul road a few km. West of the Grum Pit. These are wood frame / ATCO trailer type buildings on the order of 500 square m. in area

5.13.4 Pipelines and Water Management

Water management at the site has been carried out by a series of ditches and, for potentially metal contaminated waters, by pipelines. Pit water is pumped from temporary sumps on the active mining benches to the Little Creek Pond near the Vangorda Pit and the Groucho Pond near the Grum Pit. These two ponds feed the water treatment plant via buried and insulated sclar pipe. The line up from the little creek dam is 10" OD pipe with 2" walls to withstand high pressures, the other line is thin wall 10" OD sclar pipe. A temporary 20" OD Sclar pipe was used to carry water from the Water Treatment plant to Vangorda Creek but has been abandoned. This pipe lay on the surface and was un-insulated.

Pelly pond was a small sump used to settle solids in pit water pumped from the Vangorda Pit before it entered the Little Creek sump. The sump was no longer used after the de-watering of the Vangorda Pit in 1995 and there are no plans to use it again.

Major ditches include the Grum Interceptor and overburden dump ditch system which carried well water from the Grum interceptor wells, surface drainage from uphill of the pit and, from time to time, discharge from the Water Treatment Plant. This ditch leads to the Sheep Pad pond which serves as a sediment control pond. The out-fall from the sheep pad pond is a ditch mostly in rock but partly rip rap lined. All these ditches are approximately 1 to 2 m. wide at the base and have side slopes of 2H:1V, steep areas have been armored with rip-rap.

The Moose Pond is a natural depression on the northwest side of Vangorda Creek which was prepared for use as a settling pond for Grum Creek water. A ditch connecting the pond to Grum Creek has been built but the pond has never received any significant water as the flow has been low and the gravels in the ditch are porous.

The largest ditch on the property is the Vangorda Creek diversion which is described in section 5.6, Vangorda Plunge pool, is a rip-rap lined stilling basin at the foot of the Vangorda Creek Diversion. The outflow from the Sheep Pad Pond also enters Vangorda Creek in the Plunge pool. The Creek flows under the Vangorda Haul road in a 2 m. diameter corrugated metal culvert that is in two segments with a drop box.

The Vangorda interceptor is a ditch up hill to the northeast from the Vangorda pit. The ditch is excavated in till at its west end and in altered phyllite in its lower two thirds. The ditch is several metres wide at it's base and has steep sides, with no rip rap armoring. Station V20 is at the east end of the ditch and has detected no water quality anomalies.

The Vangorda Collector ditch is below the Vangorda rock dump, this ditch was built during the last shutdown by government using financial security seized from the previous owner. The ditch is approximately 2 m. wide at the base and has 2.5H:1V side slopes. The ditch walls and bottom are heavily armored with Rip-rap. The purpose of the ditch is to collect seepage from the drains under the Vangorda rock dump as well as surface runoff and convey it to the Little Creek Pond..

The Grum Collector ditch is not yet completely built and the design for the ditch is still in progress. The ditch is intended to collect seepage from the Grum dump and convey it to a sump where is will be pumped to the Little Creek Pond or directly to the Water Treatment Plant. Currently a road has been built along the proposed alignment of the collector ditch and a shallow starter ditch follows the road.

5.14 References

Piteau Associates, 1990. *Vangorda Pit - Hydrogeological Assessment and Development of Groundwater Control Strategy.*

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SRK -Robinson, 1994 *Construction Report - Rehabilitation PWGSC Project 760831, Vangorda Plateau, Faro Mine Yukon Territory.*

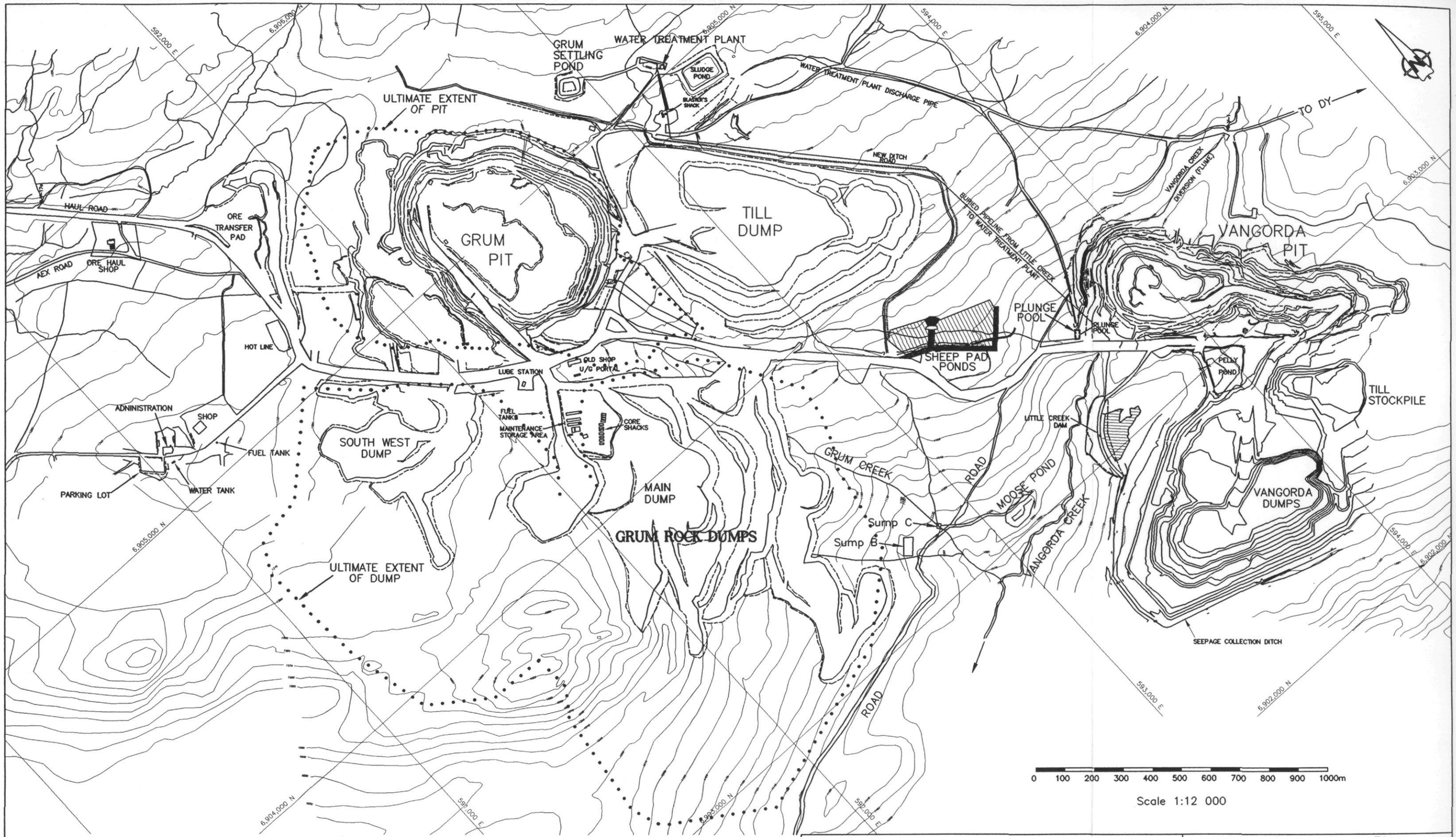
Steffen Robertson & Kirsten, 1989. *Vangorda Plateau Development - Initial Environmental Evaluation*

Steffen Robertson & Kirsten, 1991 *"Vangorda Plateau Development" As-Built Construction Report;* Steffen Robertson & Kirsten Report No. 160636/1.

Steffen Robertson & Kirsten, 1995. *Vangorda Plateau Development. As-Built Construction Report For Little Creek Dam.* Report No. 160636/1.

Steffen Robertson & Kirsten, 1996. *1996 Annual Inspection of Waste and Water Management Facilities.* Report No. A1141101.

FIGURES



Note:
Contours of Grum Pit and Dumps as of Spring 1996.
Ultimate extent of Pit and Dumps indicated as dotted lines.

Contour Interval 10 metres

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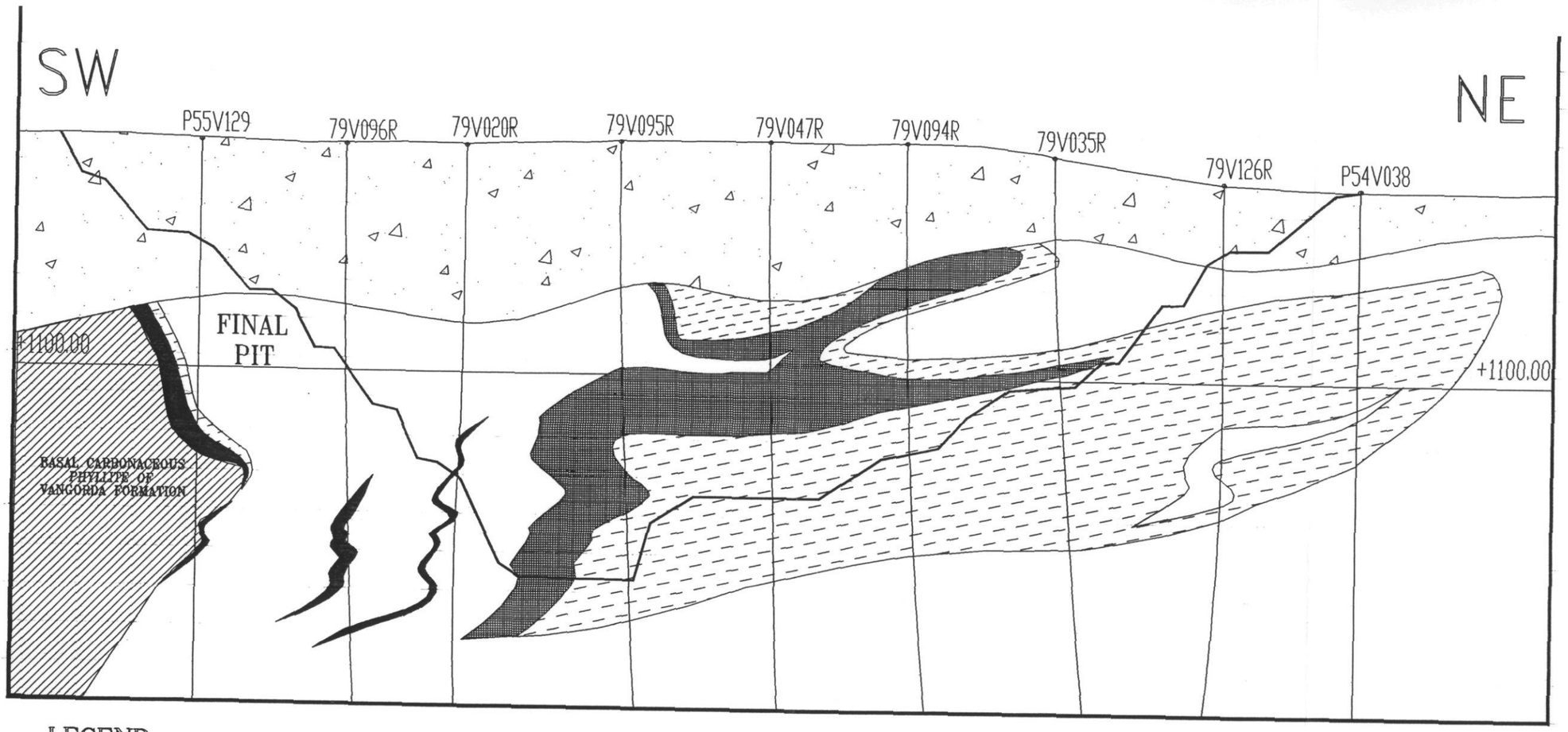
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Anvil Complex Closure Plan

Vangorda Plateau Mine Site Overview

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Oct. 1996		5-1

D:\Accumulated\Vanplateau Tue Nov 11 11:11:11 19



LEGEND

ORE ZONE ROCK TYPES

- Massive pyritic/baritic sulphides (high grade)
- Disseminated sulphides in quartzite (generally low grade)

WASTE ROCK TYPES

- Unconsolidated overburden
- Phyllite

20 0 20 40 60

metres

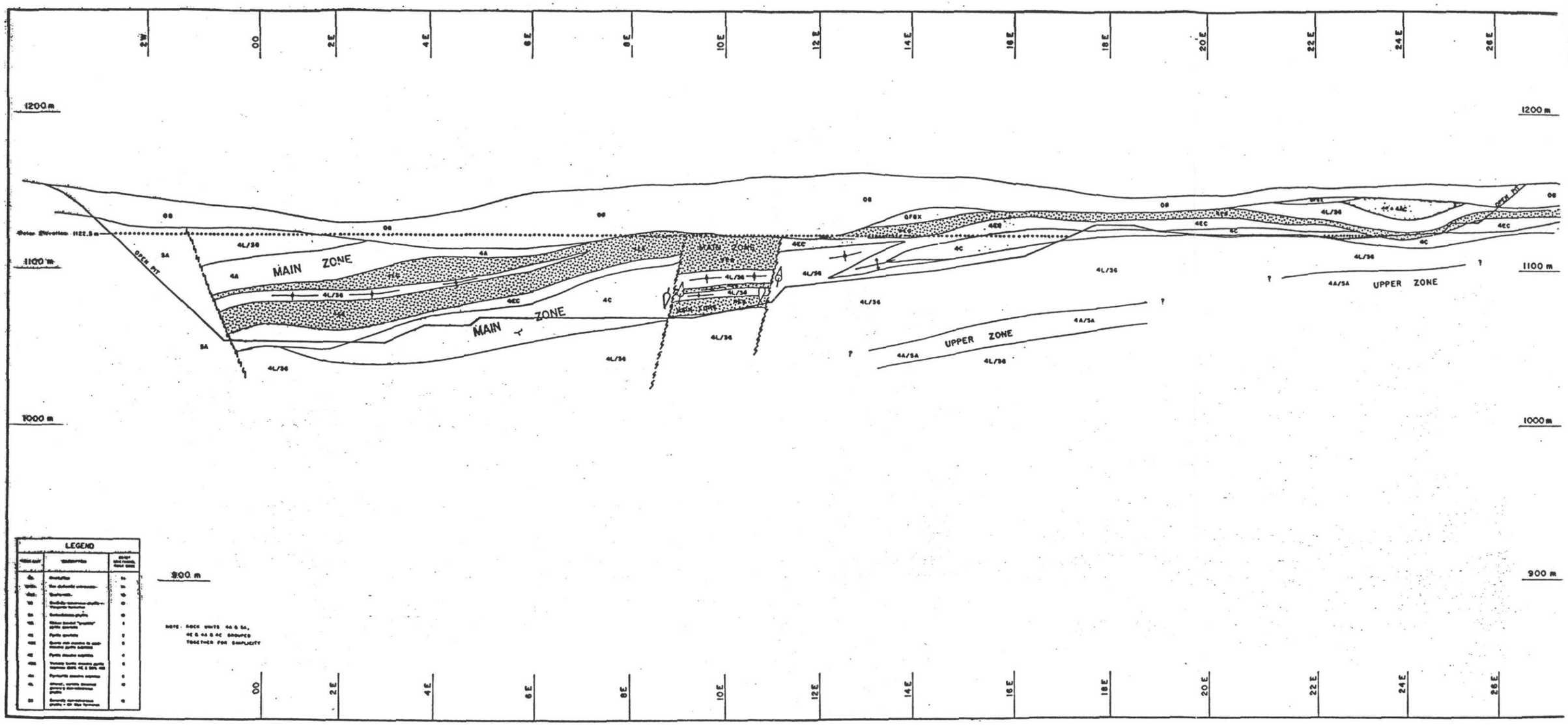
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VANGORDA DEPOSIT VERTICAL CROSS SECTION 06E

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		5-2



VANGORDA OPEN PIT

LONG SECTION 0+00

LOOKING NE

SCALE 1:2500

Note: Taken from the Initial Environmental Evaluation Vangorda Plateau Development, July 1989.

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VANGORDA DEPOSIT
LONGITUDINAL SECTION
0 + 00

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		5-3

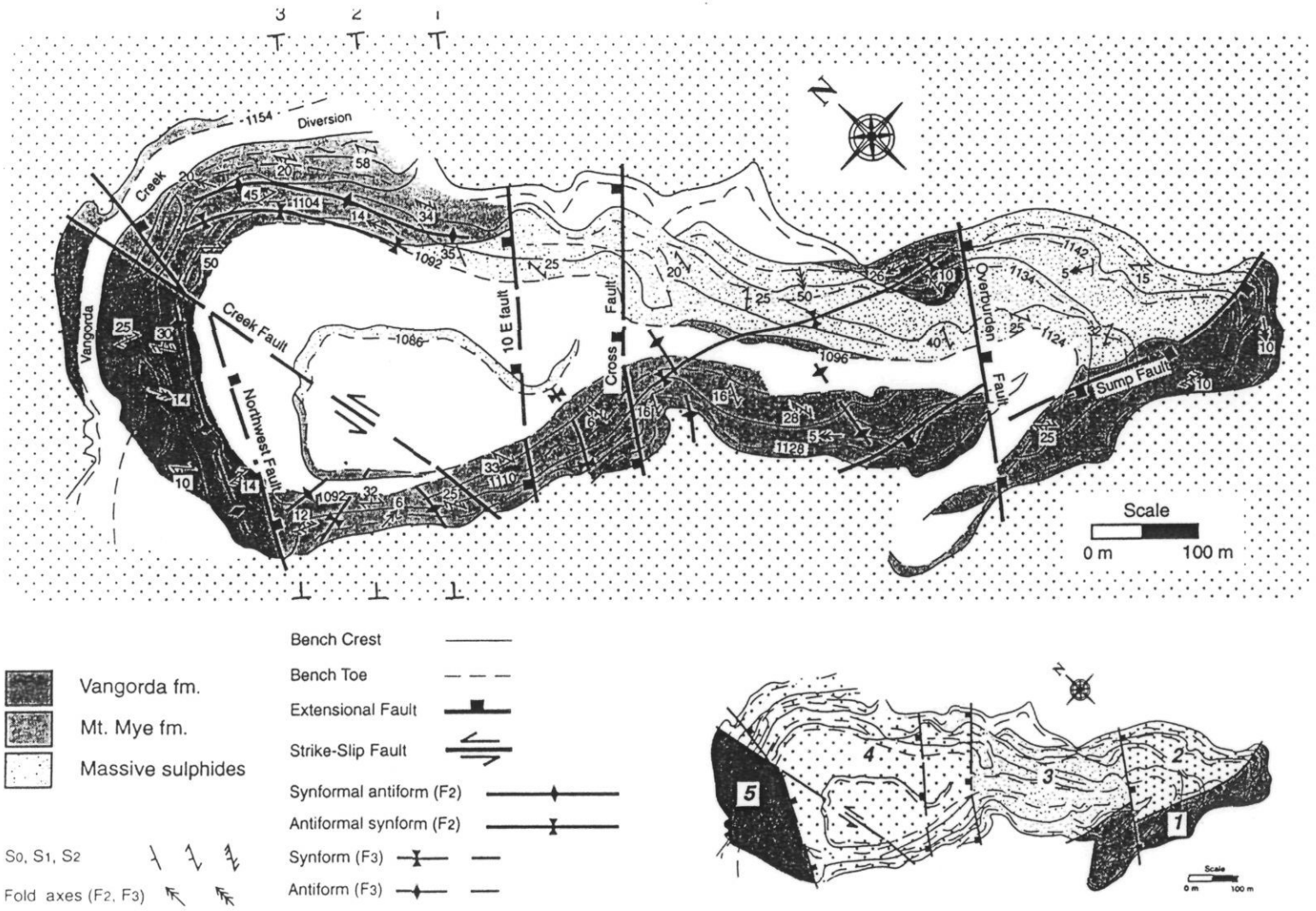


Figure 4.2. *Simplified geological map of the Vangorda open pit based on the 1 : 1000 - scale map (Figure 1.2). The inset shows the location of the various fault blocks discussed in the text.*

Figure 5-4. Vangorda pit geology (taken from D.L. Brown, "Deformation and metamorphism of the Vangorda Pb-Zn massive sulphide deposit, Yukon Territory, Canada", Ph. D. Thesis)

LEGEND

glacial till and glacio-fluvial sands

Vangorda Formation

unit 48 = 5D0, 5D4* mafic meta-igneous rocks, chlorite phyllite

unit 40 = 5B0 calcareous phyllite

unit 30 = 5A0 carbonaceous phyllite

Mt Mye Formation

unit 20 = 3G0

Ore Deposit

units 5 & 7 = 4E & 4G massive sulphides

unit 15 = 4EC undifferentiated massive and disseminated sulphides

unit 2 & 3 = 4A & 4C/4D disseminated sulphides in quartzite

unit 56 = 4L0 altered (bleached) phyllite

unit 77 = undifferentiated phyllite

water level of flooded pit at closure

Deep slope 15 overlies

Undefined 30 & 40

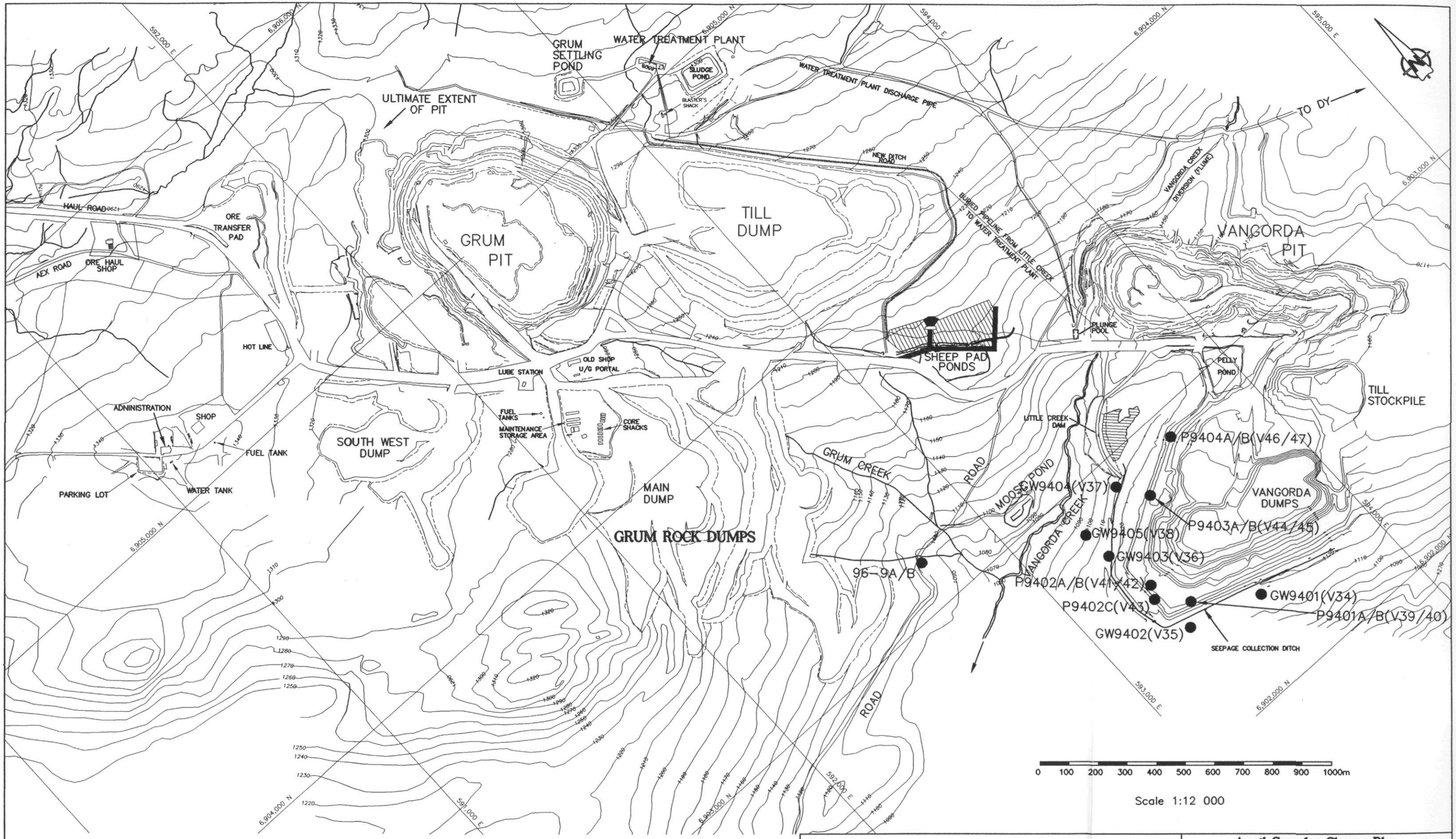
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GRUM OPEN PIT GEOLOGY

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		5-5



Scale 1:12 000

Note:
Contours of Grum Pit and Dumps as of early Spring 1996.
Coordinates are UTM in metres.
Base Map supplied by Anvil Range Mining Corporation.
Elevations are relative to Mean Sea Level (NAD27)

Contour Interval 10 metres

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Vangorda Plateau Ground Water Monitoring Stations

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		5-6

Figure 5-7 Open Pits - Total Alkalinity

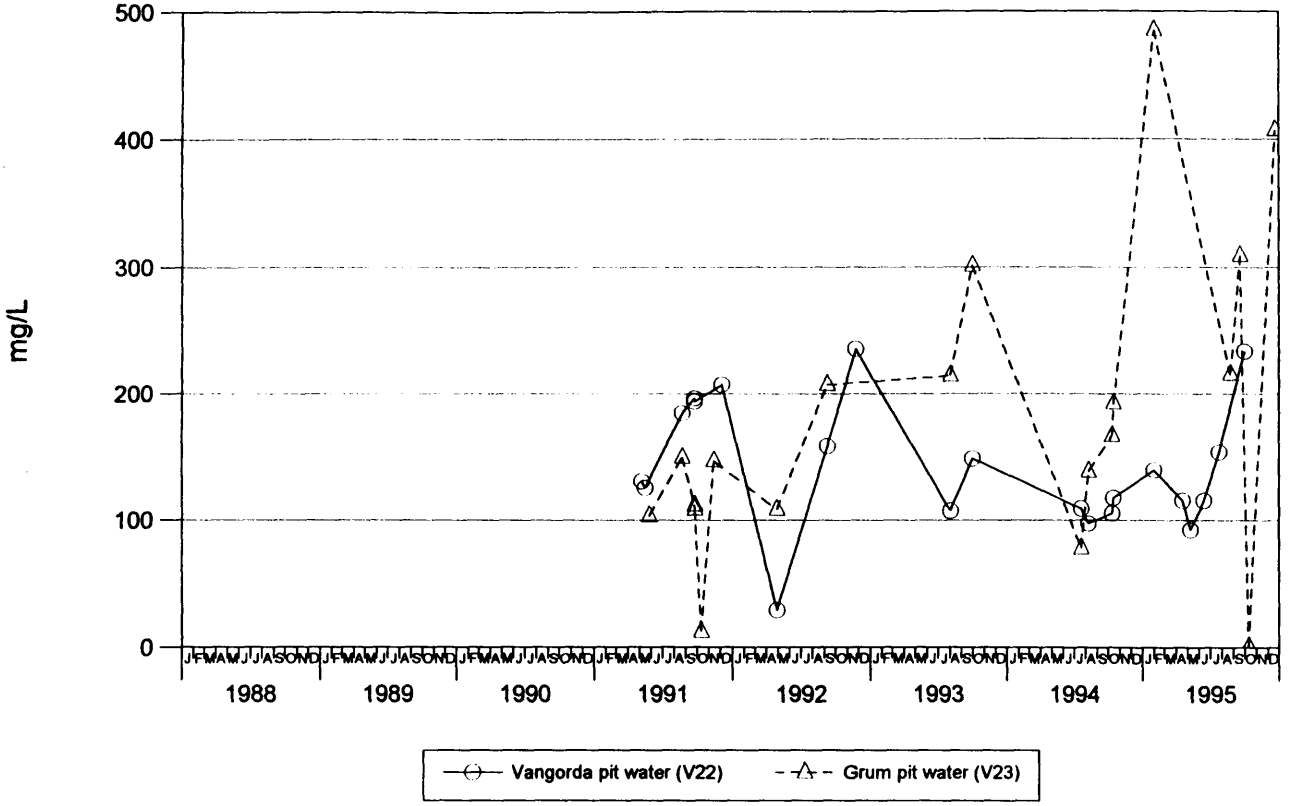


Figure 5-8 Open pit Water - Sulphate

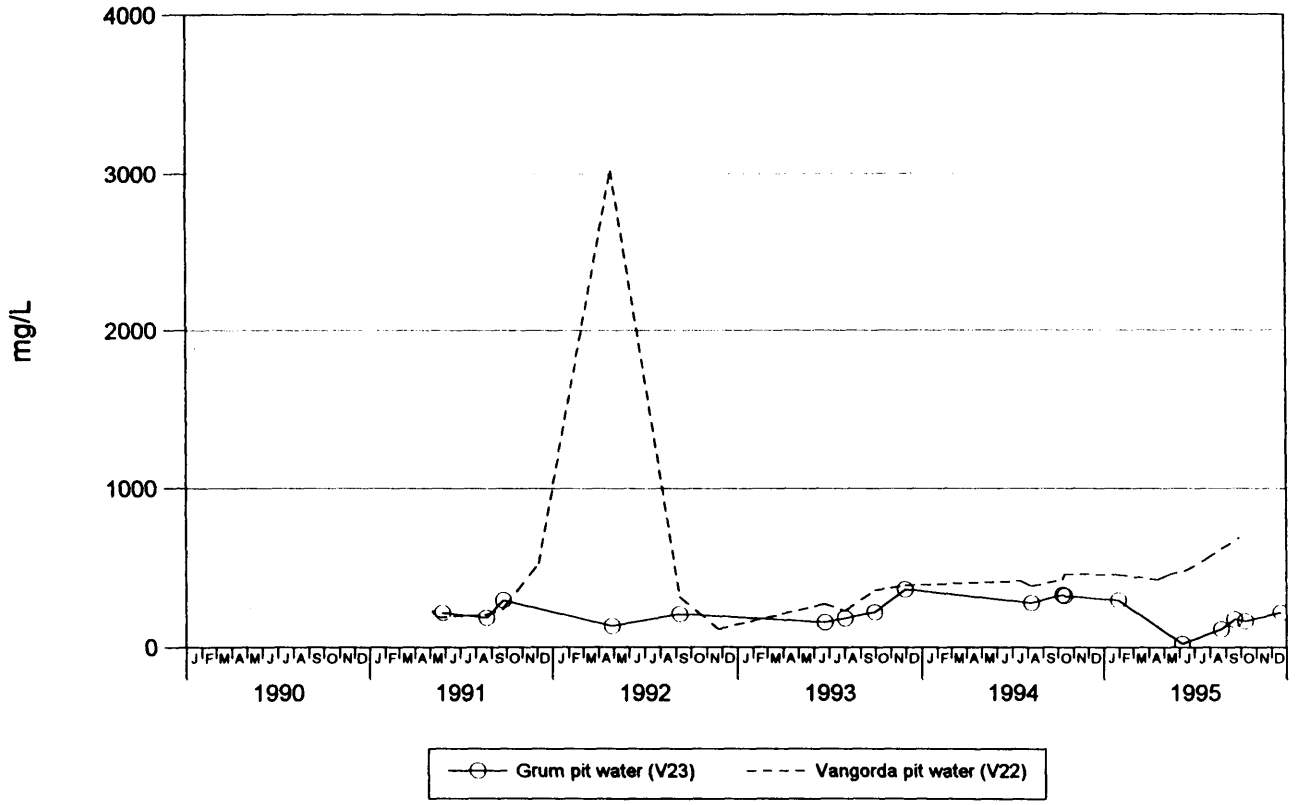


Figure 5-9 Vangorda Pit Water - Metals

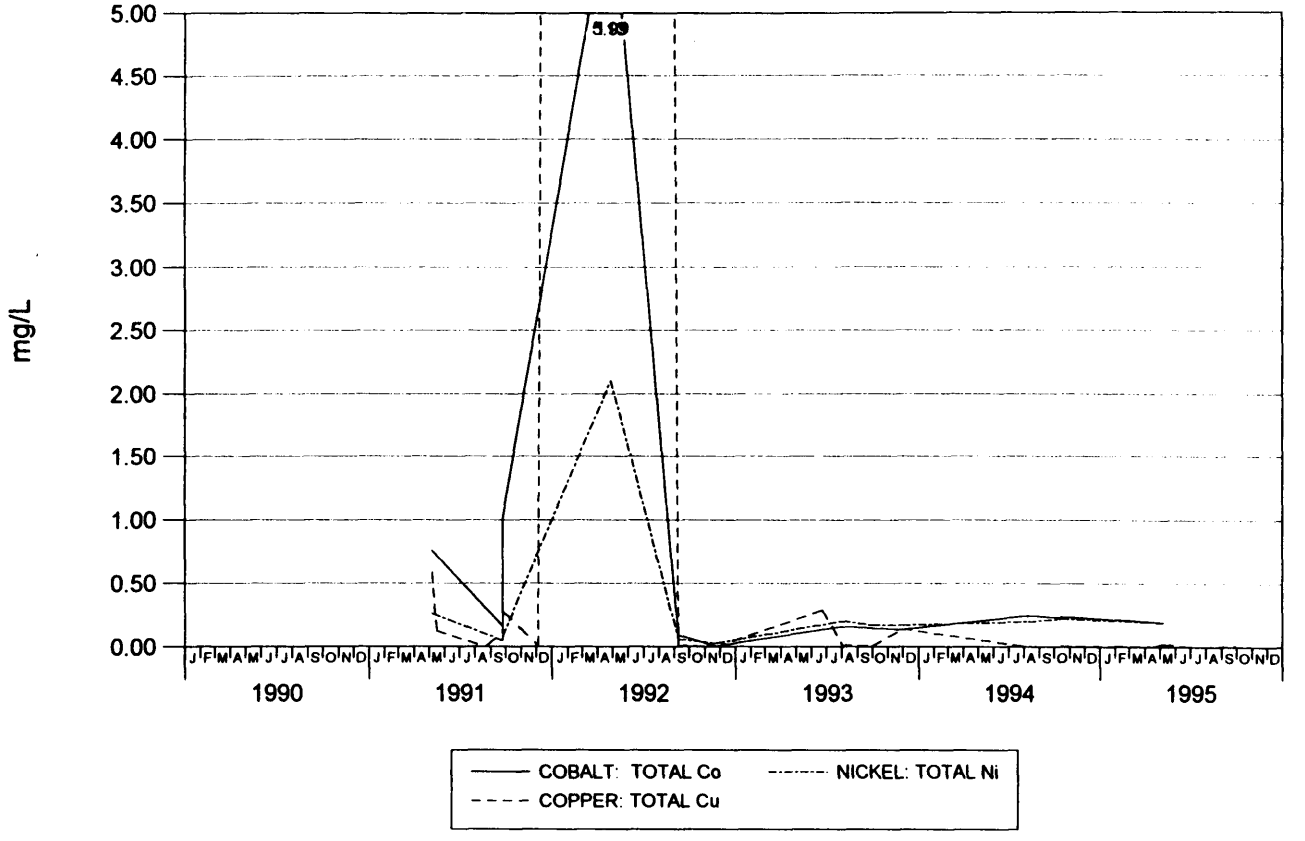
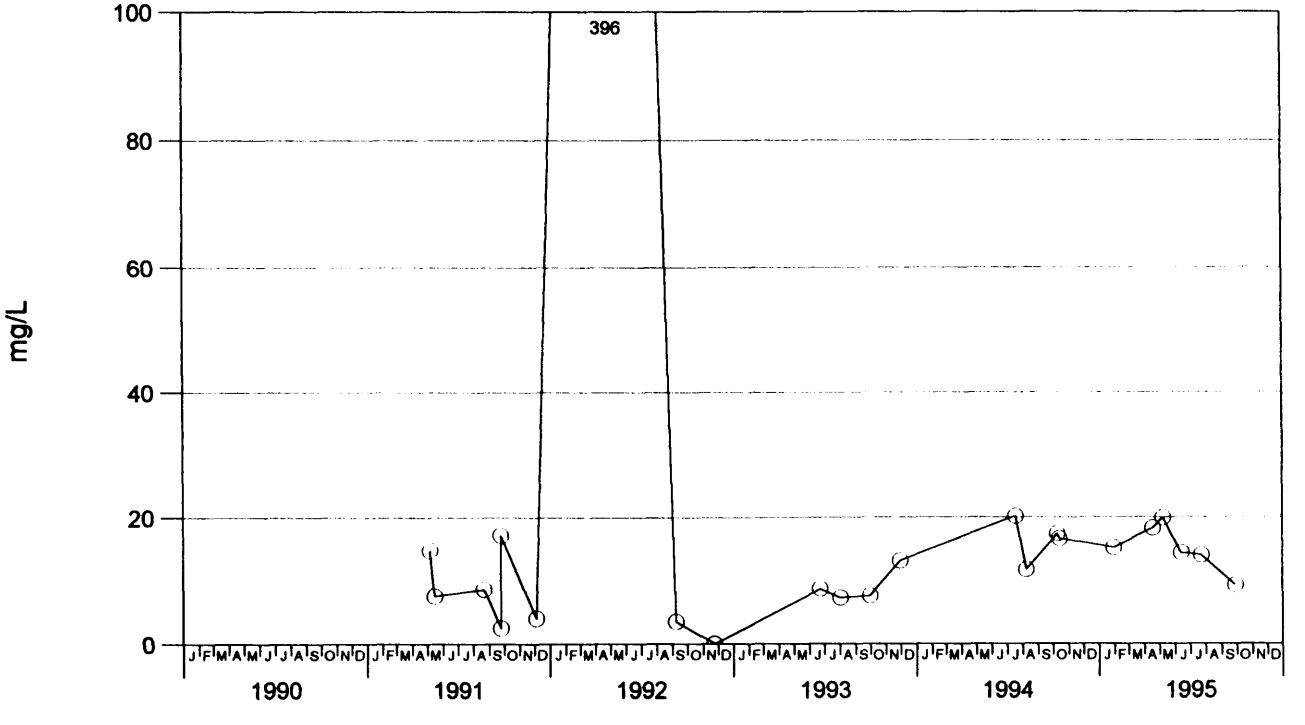
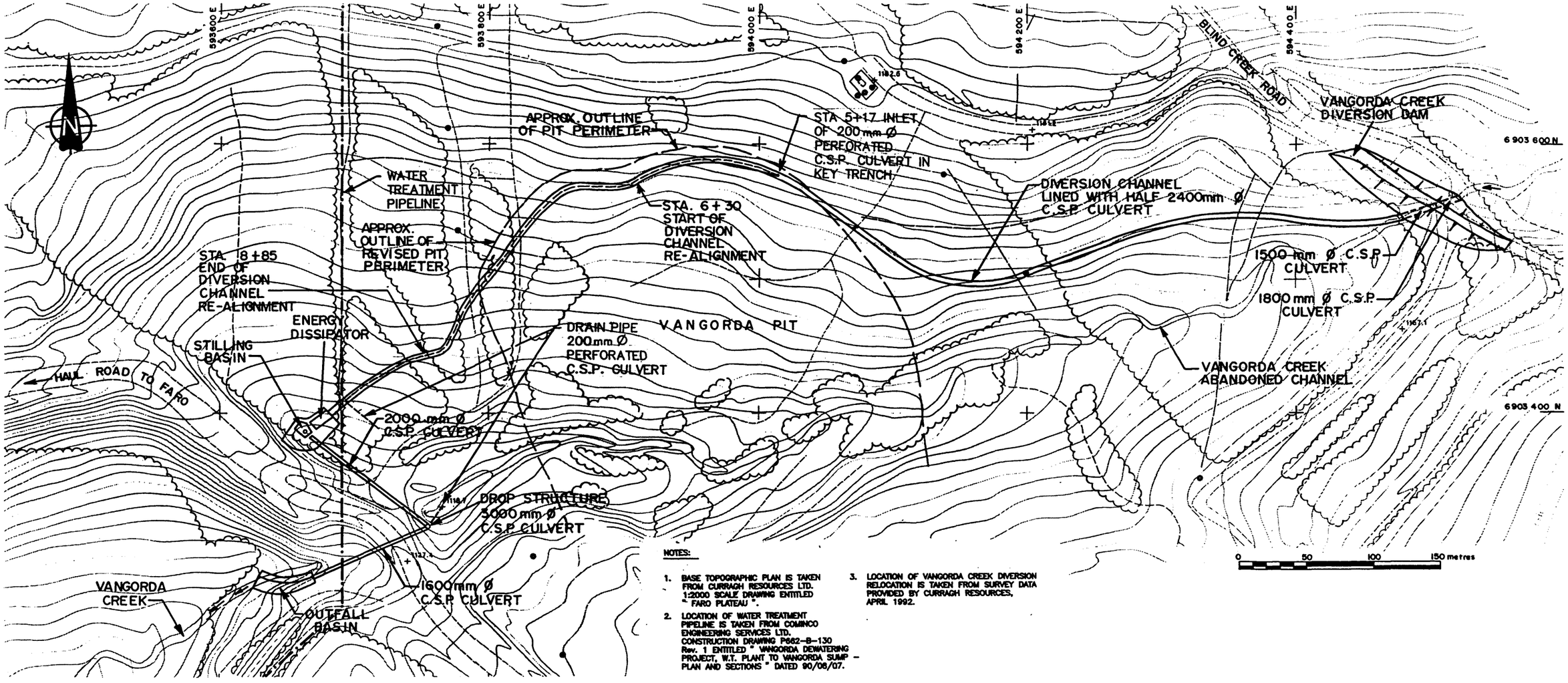


Figure 5-10 Vangorda Pit Water - Total Zinc





NOTES:

1. BASE TOPOGRAPHIC PLAN IS TAKEN FROM CURRAGH RESOURCES LTD. 1:2000 SCALE DRAWING ENTITLED " FARO PLATEAU ".
2. LOCATION OF WATER TREATMENT PIPELINE IS TAKEN FROM COMINCO ENGINEERING SERVICES LTD. CONSTRUCTION DRAWING P662-B-130 Rev. 1 ENTITLED " VANGORDA DEWATERING PROJECT, W.T. PLANT TO VANGORDA SUMP - PLAN AND SECTIONS " DATED 90/08/07.
3. LOCATION OF VANGORDA CREEK DIVERSION RELOCATION IS TAKEN FROM SURVEY DATA PROVIDED BY CURRAGH RESOURCES, APRIL 1992.

ROBERTSON GEOCONSULTANTS INC.
Consulting Geotechnical and Environmental Engineers

Anvil Range Mining Corporation

Anvil Complex Closure Plan

Vangorda Creek Diversion Current Alignment

PROJECT NO.	DATE	APPROVED	FIGURE
033001	NOV. 1996		5.11

6. Mine Water Chemistry

6.1 Site Water Balance

6.1.1 Introduction

An important prerequisite to developing any closure plan is to draw up the site water balance. This serves a variety of purposes, namely:

- to allow the investigators of the closure plan to develop a general understanding of the movement of water about the minesite;
- to quantify ungauged components of the water balance (e.g., leakage from a diversion ditch);
- to provide design flows for those proposed closure measures that involve either the storage of water or the treatment of water; and,
- to form the basis for establishing the minesite chemical load balances (as described later in Sections 6.2 and 6.3).

The first item mentioned above is perhaps the most important and deserves further explanation. Development of a minesite water balance is normally an iterative process. To begin the process, a draft version of the water balance is prepared. This draft is then tested to determine whether or not it satisfactorily reproduces the hydrological behaviour of the minesite (as measured by such items as an observed water level record for an open pit which is being filled or the discharge record of a local streamflow gauging station). If the match is unsatisfactory, refinements are introduced to the draft. These refinements may involve any number of activities, such as the discovery of an overlooked flow path, the infilling of missing data, or the rejection of suspect data. In the process of preparing the draft water balance and making any subsequent refinements, the investigators of the closure plan could develop a general understanding of the water-related aspects of the minesite. With this general understanding, the investigators are in a better position to develop practical closure measures for the minesite, particularly as they apply to the safeguarding of the local water resources.

- Development of a site water balance can be broken down into two broad steps, as follows:
- conceptualize the minesite as a system of elements and associated flow paths between the elements; and,
- quantify the amount of water generated by each element and the amount of water flowing along each flow path.

These steps, as they apply to the Faro and Vangorda Plateau minesites, are discussed below under separate headings. This is followed by a summary of the "tests" which were performed to

confirm that the developed site water balance provided a reasonable representation of the true conditions at the minesite.

6.1.2 Conceptual Representation

The first task in developing the conceptual representation of the water balance was to define its coverage, both in terms of time and space. The "time" coverage refers to the time period represented by the water balance (e.g., a particular calendar year, water year, or span of years). The "space" coverage is related to the overall area represented by the water balance (e.g., the area occupied by a tailings impoundment or by an entire watershed).

To aid in the selection of an appropriate "time" coverage, an examination was undertaken of the available records of flow and storage measurements made at the two minesites. This examination revealed that mine personnel significantly improved the monitoring of minesite flows in 1990 (both in the frequency of measurements and in the number of locations monitored). This improved level of monitoring continued while the mines were operational but was severely curtailed during the temporary shutdown. Since assuming the operations of the two minesites in late 1994, Anvil Range Mining Corporation has resurrected the full flow monitoring network established before the shutdown. Based on the review of the available flow data, the decision was made to reconstruct the water balance for the six-year period from January 1990 to December 1995. The further decision was made to develop the water balance using a monthly time increment so seasonal variations in the minesite flows could be characterized. The six-year "time" coverage was selected to allow full utilization of the flow records collected since 1990 (i.e., the period of enhanced flow monitoring). In addition, several advantages were realized by making the water balance extend over the period of temporary shutdown. The main advantage came in the estimation of the inflow rates for the open pits. By maintaining records of pit pond level over the shutdown period, the open pits effectively acted as accurate flow gauging stations (i.e., giant "bucket and stopwatch" measurements).

The selection of an appropriate "space" coverage for the water balance was based on ecological considerations. The prime objective of the closure plan for the two minesites is to protect the receiving environment. Accordingly, the water balance should not only represent the minesite proper but also the important receiving streams of the area. With this in mind, the coverage of the water balance was extended over the entire watersheds of Anvil Creek and Vangorda Creek.

Having defined the "coverage" of the water balance, the next task was to choose the basic "building blocks", or system elements, that would be used to construct the conceptual representation of the water balance. Three different types of system elements were adopted for this purpose. The first and most important type was the subcatchment. The Anvil Creek watershed was subdivided into a total of 20 of these for the purpose of reconstructing the water balance. The Vangorda Creek watershed, on the other hand, was subdivided into 16 subcatchments. Figures 6-1 through 6-3 were prepared to show the outlines of these subcatchments. Figure 6-1 is a small scale map showing the Anvil Creek and Vangorda Creek watersheds in their entirety. Figures 6-2 and 6-3 are larger scale maps which display the subcatchment boundaries in the immediate vicinity of, respectively, the Faro development and the Vangorda Plateau development. Each subcatchment on these three figures was assigned an unique identification number to allow cross-references to readily be made between the tables and

figures within this section of the report. The outlets of the subcatchments were dictated by the locations of water quality monitoring stations, tributary confluences, open pits, and dams. Portions of some of the catchment boundaries were defined by the location of diversion ditches (or, more specifically, the centrelines of the downslope embankments of the diversion ditches). Table 6.1 provides a description of each subcatchment, together with its measured drainage area and estimated mean annual runoff.

The second type of system element employed in the water balance was a storage element. The role of this type of element was to represent changes in the storage of water within important reservoirs, such as the Fresh Water Reservoir and the Main Pit. A total of ten storage elements were specified for the water balance, five in the each of the Anvil Creek and Vangorda Creek watersheds. Table 6.2 provides details of these storage elements.

Table 6.1

Details of Subcatchments for Historical Water Balance Analysis

Sub-Catchment ID No.	Subcatchment Description	Catchment Area (km^2)	Catchment Median Elevation (m)	MAR For Period 1990-95* (mm)
ANVIL CREEK WATERSHED				
1	Faro Creek at diversion inlet	13	1540	396
2	Faro Valley Interceptor Ditch at outlet	1.28	1420	342
3	Incremental catchment of Faro Creek Diversion Channel (FCD)	1.94	1370	320
4	Main Pit catchment (Zone I/III)	2.15	1260	270
5	Zone II Pit catchment (excluding area commanded by Zone II Interceptor Ditch)	0.33	1170	230
6	North Fork of Rose Creek above Faro Creek Diversion Channel (Station R7)	95	1470	365
7	North Fork catchment above X2 and below R7 and FCD	9.1	1220	252
8	Incremental catchment of North Fork Diversion Channel	1.04	1110	203
9	South Fork of Rose Creek at Fresh Water Reservoir Dam	67	1420	342
10	Incremental catchment of Pumphouse Reservoir	8.6	1230	257
11	Old Faro Creek channel above X23 (incremental)	0.9	1160	225
12	Old Faro Creek channel between X7 and X23	0.9	1160	225
13	Incremental catchment of Down Valley Tailings Impoundment	4.3	1050	176
14	Incremental catchment of Rose Creek Diversion Channel	17.6	1300	288
15	Guardhouse Creek below waste rock dumps	1.86	1480	369
16	Incremental catchment of North Wall Interceptor Ditch (NWID)	4.5	1190	239
17	Rose Creek above X14 and below NWID, X5 and X10	2.1	1130	212
18	Rose Creek between mouth and Station X14	105	1280	279
19	Anvil Creek above Rose Creek	322	1450	356
20	Anvil Creek between mouth and Rose Creek	321	1170	230
VANGORDA CREEK WATERSHED				
21	Vangorda Creek above Station V1	18.7	1590	419
22	Subcatchment of Blind Creek Road	1.4	1330	302
23	Subcatchment of Vangorda Northeast Interceptor Ditch	0.60	1200	243
24	Subcatchment of Vangorda Pit	0.80	1160	225
25	Subcatchment of Little Creek Dam	0.44	1130	212
26	Subcatchment of Vangorda Waste Dump	0.28	1130	212
27	Subcatchment of Grum Northeast Interceptor Ditch	1.8	1350	311
28	Subcatchment of Grum Pit	1.0	1300	288
29	Subcatchment of Overburden Dump	1.4	1240	261
30	Grum Creek between V2 and the haul road	1.2	1200	243
31	Vangorda Creek above V27 and below V2 and the plunge pool	3.7	1160	225
32	Shrimp Creek between V4 and V20	12.6	1150	221
33	AEX Creek above V6A	4.4	1360	315
34	West Fork Vangorda Creek between V5 and V6A	27	1190	239
35	Vangorda Creek above 29BC003 and below V27, V4 and V5	12.7	960	135
36	Vangorda Creek between V6 and 29BC003	1.2	700	18
PELLY RIVER WATERSHED				
37	Blind Creek above DIAND Station 29BC004	618	1180	231
38	Pelly River above WSC Station 09BC004 and below 29BC004 and V6	21400	1210	318
39	Pelly River between Anvil Creek and WSC Station 09BC004	454	800	63
40	Tay River above WSC Station 09BC005	3810	1160	228
41	MacMillan River above WSC Station 09BB002	13800	1130	336
42	Pelly River above 09BC001 and below Anvil Creek, 09BC005 and 09BB002	7860	880	109

Note: * MAR is an abbreviation for mean annual runoff. The period 1990 to 1995 was relatively wet. The MAR for this period was approximately 10% greater than the MAR experienced in the preceding 25 years.

Table 6.2

Details of Storage Elements for Historical Water Balance Analysis

Storage Element	Capacity (1000 m^3)	Comments
ANVIL CREEK WATERSHED		
Main Pit	51600 (to 3850 foot level)	Approximately 16 x 10^6 m^3 of water had accumulated in the Main Pit by the end of 1995. The volume occupied by the Southwest Pit Wall Dump should be removed from the stated capacity to estimate the volume available for storage of water and tailings.
Zone II Pit	330 (to 1124 m level)	The Zone II Pit is buried in waste rock. The stated capacity is a rough estimate of the volume of water required to fill voids of this waste rock.
Fresh Water Reservoir	4000 (to spillway crest)	This reservoir regulates the flow of the South Fork of Rose Creek to provide a supply of water for the mill. The capacity can be increased by about 0.6 million cubic metres by adding stoplogs to the spillway.
Aquifer in area of Pumphouse Reservoir	unknown	The mill supply is supplemented during the winter by abstracting water from the aquifer in the Rose Creek valley (using Wells PW3 and PW6). This storage element crudely keeps track of the change in storage within this aquifer.
Combined storage of Cross Valley Dam and Intermediate Dam (above tailings surface)	4000 (approximately)	The Cross Valley Dam has a capacity of 1.7 million cubic metres. Based on adjustments to an outdated topographic map of the tailings surface, the Intermediate Dam was estimated to have a capacity of about 2 million cubic metres.
VANGORDA CREEK WATERSHED		
Vangorda Pit	4600 (to 1122.5 m level)	The stated number represents the estimated capacity of the ultimate pit. During the temporary shutdown, the capacity was somewhat less at 4.0 million cubic metres.
Little Creek Dam	76 (at the maximum operating level of 1112.6 m)	This capacity was derived from a pre-mining topographic map. The actual capacity may be larger if significant quantities of soil were borrowed from the reservoir area.
Combined storage of Groucho Pond and Clarification Pond	47 (approximately)	The combined storage of these two ponds is small and, therefore, has minimal influence on the site hydrology.
Aquifer in area of Grum Pit	2000 (approximately)	This element was used to keep track of the change in storage within the aquifer. Over the period 1990 to 1995, the net amount of water removed from this aquifer was roughly 1.0 x 10^6 m^3 (i.e., difference between dewatering volume and recharge volume).
Grum Pit	40600 (at 1230 m level)	The stated number is the projected storage capacity of the ultimate pit. The capacity during the temporary shutdown was considerably less than this figure.

Finally, the third type of element was related to ore processing and water treatment facilities located within the two watersheds. For the period 1990 to 1995, two facilities were of importance, namely: the Faro Concentrator, which abstracted water from Rose Creek; and, the Vangorda Water Treatment Plant, which treated drainage from the Vangorda Plateau minesite.

All of the discussion to this point has focussed on the essential attributes of the water balance (i.e., its time period, its areal coverage, and the type of elements it would be represented by). With this preliminary information laid out, the next task was to set up the framework for the conceptual representation. This was facilitated by creating a set of figures known as "box and stick" diagrams. The "boxes" represent system elements while the "sticks" represent the flow paths between the elements.

Figures 6-4 and 6-5 present the "box and stick" diagrams for, respectively, the Anvil Creek watershed and the Vangorda Creek watershed. Figure 6-6 shows the "box and stick" diagram for the major drainage basin which contains both of these watersheds, namely the Pelly River basin. A variety of symbols are used on these three figures to differentiate between the various types of system elements and flow paths. Rectangular boxes with single outlines represent subcatchment areas. The assigned identification numbers for the subcatchments appear in the upper, left-hand corners of the boxes (see Figures 6-1 through 6-3 for the locations and boundaries of the subcatchments). Rectangular boxes with double outlines depict ore processing and water treatment facilities. Circles designate important storage elements within the two watersheds. Lines drawn on the figures are used to show the interactions of the subcatchments, mine facilities, and storage elements. Solid lines represent generic flow paths (i.e., these lines signify the movement of water from one system element to another but do not suggest the mode of movement, be it surface flow, interflow, groundwater, or some combination thereof). Dashed lines, on the other hand, signify that the water moves exclusively via the ground (as regional groundwater, ditch leakage, or embankment seepage). Figure 6-7 presents a detailed legend for the "box and stick" diagrams.

With completion of the conceptual representation of the water balance, the next step was to quantify the flows generated by the elements and the flows moving along the interconnecting flow paths.

6.1.3 Estimation of Flows

A number of sources of information were assembled for the purpose of estimating the water balance flows. These are:

- the last eleven volumes of the Annual Reports prepared for the Faro Mine (i.e., the reports for years 1985 to 1995);
- all Annual Reports prepared to date for the Vangorda Plateau minesite (i.e., years 1991 to 1995);
- the correspondence prepared by Peat Marwick Thorne Inc. (the interim receiver) which documented environmental activities during the most recent mine shutdown (April 1993 to November 1994);

- a number of environmental and geotechnical reports prepared by various consulting firms (Steffen, Robertson and Kirsten Inc., Piteau Associates Engineering Ltd., Acres Consulting Services, Golder Associates Ltd., and Laberge Environmental Services);
- the 1:50,000 and 1:250,000 scale topographical maps of the region produced by the Department of Energy, Mines, and Resources;
- regional climatic data collected by the Atmospheric Environment Service of Environment Canada;
- regional streamflow data monitored by the Water Survey of Canada and DIAND;
- spot flow measurements made by DIAND at several key locations around the two minesites; and,
- other unpublished data collected by mine personnel, the most notable of these being the surface water level records for the open pits and the stage records for the automatic streamflow gauging stations.

The Annual Reports proved to be the single most valuable source of information for reconstructing the historical water balance. This was true for two reasons. Firstly, these reports assembled, in one place, most of the flow measurements made at and nearby the two minesites. Secondly, these reports documented the history of water management at the minesites, providing a wide range of details on the mine activities that directly, or indirectly, affected the site water balance. A few examples of these details are: the schedule of pit dewatering, the as-built plans for diversion ditches, and the date on which the Main Pit was commissioned as a repository for tailings.

An examination of the Annual Reports, together with the other sources of data mentioned above, revealed that the flow database for the minesite was quite extensive. This database could not be efficiently processed by manual means. Accordingly, all of the available flow data were assembled and entered into one of two EXCEL spreadsheets. One spreadsheet was dedicated to data collected at sites within the Anvil Creek watershed while the other spreadsheet served the same role for the data collected in the Vangorda Creek watershed.

Once the raw flow data had been entered, work proceeded on organizing the spreadsheets into a numerical representation of the water balance. The rows within the two spreadsheets were used to keep track of specific months within the 72-month period from January 1990 to December 1995. The columns, on the other hand, were used to represent the individual system elements and flow paths which, together, made up the conceptual representation of the water balance (see Figures 6-4 and 6-5).

The monthly flow record for each element and each flow path was reconstructed using one of the following four approaches:

- computed averages of spot flow measurements;
- regional analysis;
- groundwater modelling; and,
- application of the continuity equation.

Priority was given to the use of the first approach because it was recognized as being, potentially, the most accurate. In the final tally, this approach was used to reconstruct the monthly flow records for a significant number of flow paths at the two minesites. It was particularly applicable to flow paths subject to a high monitoring frequency and/or having reasonably uniform discharge rates throughout the year. Listed below are some of the flow paths which were quantified using the first approach:

- abstractions from the Pumphouse Reservoir;
- releases from the low-level outlet of the Fresh Water Reservoir;
- dewatering flows from the Zone II Pit;
- seepage emanating from the toe of the Cross Valley Dam; and,
- flows pumped from the Little Creek Dam to the Water Treatment Plant.

For many elements and flow paths, spot flow measurements were not available and therefore the first approach could simply not be used. For still others, the frequency of monitoring was insufficient to accurately assess the true monthly flows and, again, the first approach was not suitable. In the latter case, the assembled spot flow measurements were not discarded but rather used at a later stage to validate the flow values provided by the alternative estimation techniques.

The second approach to flow estimation (i.e., regional analysis) was used to quantify the runoff generated by the subcatchment areas. In broad terms, regional analysis involved transposing the monthly flow record of a regional streamflow gauging station to the minesite subcatchments. This was accomplished by scaling the regional streamflow record according to two different factors. One factor was equal to the ratio of areas (i.e., the area of the minesite subcatchment divided by the area of the gauging station catchment). The other factor was used to account for differences in the average depth of precipitation falling on these two different areas. The reader is referred to section 3.7 for a more detailed description of regional analysis, as it was applied to the two minesites.

The third approach, groundwater modelling, was employed to estimate the seepage rates from the Main Pit and the Zone II Pit. The modelling procedure is detailed in Appendix J.

The final technique for estimating flows entailed applying the continuity equation (i.e., inflow - outflow = change in storage). This technique was used throughout the water balance to estimate flows that could not be assessed using any of the other approaches. The continuity equation was applied by examining the "local" water balance of each subcatchment. For example, the flow path between Subcatchments Nos. 7 and 8 on Figure 6-4 was assessed by examining the "local" water balance of Subcatchment No. 7. As can be seen by reference to Figure 6-4, the water balance for this subcatchment comprises a total of six components, five are inflows while the remainder is an outflow component, and also the focus of this example. Based on the continuity equation, the magnitude of this outflow component was computed by summing the magnitudes of the five inflow components. A similar application of the continuity equation was repeated for almost all of the subcatchments, although the added complication of water storage sometimes had to be considered (e.g., the accumulation of water in the Main Pit).

Derivation of the historical water balance for the Faro and Vangorda Plateau minesites involved a lengthy process. Details of this process have been fully documented within the water balance spreadsheets. The documentation was facilitated by a special feature of EXCEL known as a "text note", which allows descriptive notes to be attached to cells within the spreadsheet. These "text notes" were used to provide such information as:

- the basis for all flow estimates made using regional analysis, including drainage areas and estimated unit runoffs for the subcatchments;
- sources of observed flow and water level measurements;
- methods and assumptions used to patch missing data within the observed flow records;
- sources of groundwater flow estimates; and,
- details on the application of the continuity equation to estimate many of the components of the historical site water balance.

The EXCEL spreadsheets used to reconstruct the historical water balances for the Anvil Creek watershed and the Vangorda Creek watershed were entitled, respectively, "FAROWATB.XLS" and "VANGWATB.XLS". Printouts of these two spreadsheets, including the text notes, are presented in separate appendices (Appendix P on "Faro 1990-1995 Historical Water Balance" and Appendix Q on "Vangorda Plateau 1990-1995 Historical Water Balance").

While details have been reserved for the appendices, important aspects of the historical water balance are presented below.

6.1.4 Temporal Variations in Historical Water Balance

The purpose of the "box and stick" diagrams is to provide a general overview of the water balance for the Anvil Range Mining Complex. Useful information can be gleaned at a glance from these diagrams, such as the relative magnitude of different flow streams, or the general way in which water is moved about and stored within the mine complex.

In effect, the "box and stick" diagrams provide a summary of the entire water balance on just a few pieces of paper. To achieve this level of conciseness, certain detail had to be sacrificed. In particular, these diagrams do not reveal any information about variations in the water balance flows from year to year, and from month to month. Without this information, some of the flow rates on these diagrams may be misinterpreted, particulary those attached to flowlines which were not actually active, or indeed not even in existence, over the full six-year period from 1990 to 1995. For example, Figure 6-4 indicates that the Faro Concentrator discharged water to three different locations during the period 1990 to 1995, namely the Down Valley Tailings Impoundment, the Main Pit, and the North Wall Interceptor Ditch. None of these three flowlines were ever used coincidently. In fact, some periods existed when all three were idle. The numbers on these particular flowlines do not represent the flow rate during just the operational periods but rather the flow rate averaged over the entire 6-year period from 1990 to 1995. (In other words, the stated numbers equal the total volume of water conveyed along the flowline in m3 from 1990 to 1995 divided by the number of seconds in the six-year period.)

To allow the reader to achieve an appreciation for the annual and seasonal variations which occurred in the water balance flows, a series of 12 hydrographs are presented in this section. All of these hydrographs were originally prepared to aid in the development of the water balance spreadsheet. Table 6.3 provides an index to these hydrographs which describes what role each hydrograph played in developing the water balance spreadsheet and what portion of the "box and stick" diagrams each is applicable to. The reader is referred to Appendices P and Q for more detail on the development of the site water balance.

6.1.5 Extreme Flood Estimation

6.1.5.1 Extreme Flood Discharge Rates

Flood estimates were required at several key points around the minesite to provide the basis for the conceptual design of spillways and diversion channels. Two extreme flood events were examined for each point of interest, namely the 500-year peak instantaneous flood and the Probable Maximum Flood (PMF). The former is a rare event which is expected to only be exceeded once every 500 years on average (or roughly once in six lifetimes). The latter event is defined as "the flood that may be expected from the most severe combination of meteorologic and hydrologic conditions that are considered to be *reasonably possible* in the geographical region encompassing the basin under study" (USACE, 1980). The frequency of occurrence of the PMF is difficult to assess because of a lack of historical streamflow record. Based on extrapolation from traditional estimates of storm-induced floods which have more frequent occurrence, the PMF for the study area has a likely return period in excess of several thousand years (or perhaps even tens of thousands of years). A different estimation technique was adopted for each type of event. The Rational Method was used to estimate the magnitude of the 500-year floods while the Creager Curve was employed to estimate PMF events.

Rational Method

This is a well-established technique for estimating the instantaneous peak of a flood generated by an intense rainfall storm. It is essentially a modelling technique which simulates the response of a catchment to a given rainfall input. This "model" is expressed in equation form as:

$$Q = CIA/3.6$$

where: Q is the peak instantaneous discharge of the flood (m^3/s);

C is a runoff coefficient (dimensionless);

I is the average rainfall intensity which causes the flood (mm/h); and,

A is the catchment area (km^2).

Table 6.3

Index to Hydrographs for Historical Water Balance Analysis

Figure No.	Purpose of Hydrograph	Corresponding Element on "Box and Stick" Diagrams
6-8	Estimate inflow rate to Main Pit. (The inflow rate was adjusted until the simulated and observed records of pit storage matched reasonably well.)	"Main Pit Change in Storage" on Figure 6-4
6-9	Estimate net volume of water abstracted from Zone II Pit between January 1990 and December 1995. (See water balance spreadsheet for explanation of less than satisfactory fit between simulated and observed levels.)	"Zone II Pit Change in Storage" on Figure 6-4
6-10	Validate water balance flows for Rose Creek at Station X14	Subcatchment No. 17 on Figure 6-4
6-11	Validate water balance flows for North Fork of Rose Creek at Station R7	Subcatchment No. 6 on Figure 6-4
6-12	Validate water balance flows for Faro Creek above the diversion channel	Subcatchment No. 1 on Figure 6-4
6-13	Validate water balance flows for Rose Creek Diversion Channel at Station X10	Subcatchment No. 14 on Figure 6-4
6-14	Assist in estimating inflow volume originating from Vangorda Pit.	"LCD Change in Storage" on Figure 6-5
6-15	Estimate inflow rate into Vangorda Pit.	"Vangorda Pit Change in Storage" on Figure 6-5
6-16	Validate water balance flows for Vangorda Creek at DIAND Station 29BC003	Subcatchment No. 35 on Figure 6-5
6-17	Validate water balance flows for Vangorda Creek at Station V1	Subcatchment No. 21 on Figure 6-5
6-18	Validate water balance flows for Grum Northeast Interceptor Ditch at Station V18	Subcatchment No. 27 on Figure 6-5
6-19	Validate water balance flows for AEX Creek at Station V6A	Subcatchment No. 33 on Figure 6-5

The runoff coefficient specifies the proportion of the rainfall that quickly runs off the catchment to form the flood hydrograph. The remainder of the rainfall is retained on the catchment for subsequent evaporation or slow release to the catchment's streams. The value of the runoff coefficient varies from storm to storm depending on the initial moisture content of the soils and the extent of frozen soils within the catchment. For estimation of the 500-year floods, a value of 0.8 was selected to represent a catchment with minimal capacity to retain rainfall.

In order to apply the Rational Method, it was necessary to estimate a characteristic of each study catchment known as the "time of concentration". This is a measure of the response time of the catchment. It may be interpreted as the time it takes for the most remote portion of the catchment to contribute to the flow at the outlet of the catchment. The time of concentration was estimated using empirical relationships that depend on the physical characteristics of the catchment (Leopold, 1991 and McCuen, 1982).

Information on design storm rainfalls likely to be experienced at the minesite was obtained from the "Rainfall Frequency Atlas for Canada" (Hogg and Carr, 1985).

The estimated flood peaks computed using the Rational Method are provided in Table 6.4. Note that the basic premise of the Rational Method is the largest flood will occur when the selected rainfall intensity has a duration exactly equal to the time of concentration. Furthermore, the computed flood peak is assumed to have the same frequency of occurrence as the causative rainfall intensity.

Creager Curve

An empirical relationship between flood discharge and catchment area was used in this study to approximate the magnitude of the PMF at various locations around the minesite. Two different types of information were used to establish this relationship. The first was the results of previous PMF studies conducted for the Faro minesite. At least three consulting firms have undertaken such studies, namely: Klohn Leonoff Ltd. (KL) for the 1981 Faro Abandonment Plan; Acres Consulting Services Ltd. for a 1985 hydrologic investigation of the Fresh Water Reservoir; and, Steffen Robertson and Kirsten Inc. (SRK) for a decommissioning study of the Down Valley Tailings Impoundment. All three consulting firms used a hydrologic model to simulate the flood hydrograph associated with the PMF. Various techniques were used by the consultants to assess the magnitude of the precipitation event believed to cause the PMF (i.e., the so-called Probable Maximum Precipitation). From the three earlier investigations, PMF estimates were extracted for the spillways of the Intermediate Dam and the Fresh Water Reservoir.

The second type of information assembled for the study was a listing of some of the largest known floods to have ever been experienced in Canada. The most up-to-date and comprehensive listing located in the literature was the one contained in the publication entitled "Hydrology of Floods in Canada: A Guide to Planning and Design" (NRCC, 1989). The authors of this publication assembled information on 22 of the highest known floods to have occurred to that time.

Figure 6-20 was prepared to facilitate comparison of the available information. It is a logarithmic graph of unit flood discharge versus catchment area. Unit discharge is computed by dividing the absolute flood discharge by the contributing catchment area. As can be seen, the maximum

observed floods exhibit a trend of decreasing unit flood discharge with increasing catchment area. Furthermore, the data follow a concave-downward pattern on the logarithmic graph.

Superimposed on this graph are two curves known as Creager Curves. The equations for these curves are double exponentials which allow the curves to closely match the pattern observed in the extreme flood values. A Creager Curve is an empirical relationship that provides a simple means of comparing floods in different regions and on different sizes of catchment. The positioning of a Creager Curve on a logarithmic plot is dictated by a single constant known as the Creager Constant. The magnitude of the Creager Constant may be used as a measure of the flood-producing characteristics of a region. The higher the constant, the more severe the flood regime is. A Creager Curve can be made to pass through the SRK and Acres PMF estimates by specifying a Creager Constant of 26. As can be observed on Figure 6-20, such a curve passes through the band of maximum observed floods in Canada. To make a Creager Curve intersect the KL PMF estimate, the constant must be increased to 54. This higher curve envelopes all of the observed maximum floods.

For the purpose of this study, the Creager Curve with a constant of 26 was adopted to provide PMF estimates for the minesite. The rationale for this decision was largely based on an examination of the 22 unusual floods plotted on Figure 6-20. The following observations were made about these floods:

- Not one of the 22 unusual floods occurred in the Yukon.
- The most extreme floods in the data set occurred on either the coast of British Columbia or the eastern slopes of the Rocky Mountains. These areas are likely subject to more severe storms than experienced in the Yukon. This is because no significant topographic barrier lies between these areas and their prime moisture sources (i.e., the Pacific Ocean for the coast of B.C. and the Gulf of Mexico and the Atlantic Ocean for the eastern slopes of the Rocky Mountains). In contrast, the interior of the Yukon is well protected from major moisture sources by mountain barriers.
- The magnitude of some of the 22 flood events may be overestimated, particulary those which occurred on small catchments. For example, the flood for the smallest catchment (i.e., Harvey Creek on the coast of B.C.) was computed using the slope-area method. The computations were performed assuming clear water but, in reality, the flood waters likely carried a significant sediment load (both bedload and suspended load). This neglect of the sediment load likely led to an overestimation of the true flood peak.
- Some of the more extreme floods were generated in the wettest regions of the country. For example, the flood on the fourth smallest catchment (i.e., Zeballos River) occurred on the western coast of Vancouver Island, an area which receives in excess of 3000 mm of precipitation per year. This is an order of magnitude more precipitation than experienced at the Anvil Range Mining Complex.

Table 6.4 provides the PMF values estimated using the Creager Curve.

Table 6.4

Estimated Peak Instantaneous Flood Discharges for Various Decommissioning Structures

Decommissioning Structure	Catchment Area (km^2)	500-Year Return Period (m^3/s)	Probable Maximum Flood (m^3/s)
Faro Creek Diversion Channel	15.3	27	150
Intermediate Dam Spillway	221	190	840
Vangorda Creek Diversion Channel	20.1	31	180

6.1.5.2 Extreme Flood Volumes

For some of the closure measures examined in this closure plan, the volume of the PMF event was of more importance than its actual peak. As will be discussed later in this report, the proposed closure measures for the Vangorda and Main Pits include a provision for an emergency flood storage. This storage would prevent the escape of contaminated drainage from the open pit in the event of a diversion breach.

It is proposed to design these storages based on either of two events, namely:

- the maximum possible volume of runoff which could pour through the diversion breach in a one week period (as would result from an extreme rainfall event); or,
- half this volume.

The first step in assessing these volumes was to estimate the maximum possible depth of rainfall that could fall at the minesite in a one week period. An estimate was obtained by referencing the flood study prepared for the Down Valley Tailings Impoundment Abandonment Plan (SRK, 1991). In this study a statistical approach was used to estimate the magnitude of the Probable Maximum Precipitation (PMP) for a range of durations from 1 to 24 hours. These values were plotted on a logarithmic plot and extrapolated to a duration of 7 days. In following this procedure, the maximum depth of rainfall that could fall in a week was estimated to be 370 mm. Roughly half of this depth could fall within a 24-hour period within that week.

The volume of runoff generated was set equal to the product of the rainfall depth, the diversion catchment area, and a runoff coefficient of 0.8. For the Main Pit, it is proposed to provide an emergency surcharge storage which could contain the full flow generated by the 370 mm rainfall event. This would be a volume of 4.5 million cubic metres. For the Vangorda Pit, the surcharge would be equal to 3.1 million cubic metres, or half the flow generated by the extreme event.

6.2 Faro - Mine Water Chemistry

6.2.1 Technical Issues

There are three main questions to address in the prediction of post-closure drainage water chemistry from waste dumps:

- the potential for metal loads to increase or decrease in the longer term;
- the potential for acid generation or acidity to increase or decrease in the longer term; and,
- the presence of other metals of concern that are not the drainage currently, but may appear in the longer term.

These questions were addressed through waste rock sampling testing programs which included "conventional" static and kinetic testing, and specific testing to address questions of longer-term potential for acid and metal generation. The approach to prediction of long-term loads from the waste dumps was empirical, based on laboratory and field testing. Results of the laboratory tests were scaled to field conditions, using a series of simple spreadsheet models, to calculate the current and potential annual average loadings from the dumps.

There are number of inherent difficulties or weaknesses in predictive modelling of field geochemistry from a testing database the most significant being:

- characterization of the composition of the dumps and the variability; and,
- determination of the controls on drainage chemistry including infiltration, flowpaths within the dump, solubility controls, effect of evaporation, and migration time through the dump.

While each has been considered in a qualitative manner, and some addressed in a quantitative manner, quantification of most of these variables is beyond the scope of this investigation and may, in fact, be considered to still be in the realm of research.

Thus, while these calculations are based on our best understanding of dump sampling and testing, and of the controls of dump water quality, they are still only estimates of the potential for contaminant loading from the waste rock, stockpiles, and open pits. Such estimates are required for evaluation and for design of closure measures (e.g. water treatment requirements) and are therefore provided in the following section of this chapter. Results must be calibrated to field measurements. These "predictions" or estimates of contaminant loading and the resultant receiving water chemistry are therefore presented as the range of reasonable values that could be expected over time from the site.

The importance of these preditions is in the use to which they are put to the need for reclamation, water treatment and water management at closure. The results indicate that, at present, there are mechanisms which are reducing the potential loadings to surface waters from many of the site components. The waste rock and the open pits clearly do not represent an unanticipated

environmental problem. However, it is recnogized from this work that control measures for dump drainage waters and for pit water would be needed in the long term to protect the receiving environment.

6.2.2 Conclusions from Laboratory Testing

The static and kinetic testing programs indicate the following major conclusions about the characteristics of the waste rock relevant to the predictions of longer-term water chemistry for the site, and address the issues posed in section 6.2:

- the sulphide assemblages are clearly acid generating and the major source of dissolved metals;
- although the sulphides represent a minor portion of the total waste rock, the sulphides are the major current and longer term source of metals and acidity;
- the calc. silicates and some schists release carbonate alkalinity to drainage;
- zinc is the major metal of concern and is released from the waste rock over a wide range of pH conditions and initial metal concentrations;
- oxidation processes are well established in older and younger sulphides;
- the "peak" in oxidation rate has been reached or, in some cases, passed;
- there is storage of soluble products within the waste rock samples; and,
- many of the rock assemblages in older dumps have been significantly depleted in oxidation potential, and continued production is dominated by flushing of stored products more so than the rate of production.

With respect to the Faro site the results of the rock analyses, extraction testing, and long term kinetic testing indicate that zinc, sulphate and acidity are the primary "contaminants" released from the rock. It is particularly evident from the kinetic test results that while other metals can be released in varying quantities, few are released in the absence of zinc. Other metals would include iron, manganese, lead, nickel, and cobalt all of which can be treated in the same lime neutralization and precipitation circuit that would remove zinc. Zinc is therefore used as the basis for most of the predictions and water quality comparisons. Sulphate is also used as an indicator of oxidation rate and extent and a comparatively conservative chemical constituent.

The only exception would be calcium, which is released from the schist and from the calc. silicate in the absence any appreciable zinc release. There are no appreciable quantities of metals such as arsenic which would require special treatment processes. Total sulphur and zinc concentrations do not appear to affect oxidation rates within the range of expected concentrations for each rock type assemblage; sulphides or mixed sulphides with comparatively low sulphide content oxidize and release metals at similar rates to higher sulphide samples.

The data indicate that while oxidation reactions will continue within the dumps, the rate of oxidation and rate of metal release will not change markedly over time. Over 25 years of exposure the older sulphides and other rock assemblages appear to have reached a "stable"

oxidation rate. The younger samples appear to be approaching similar rates although are at an earlier stage of oxidation.

Enhanced oxidation tests support the observation that oxidation and leaching rates are relatively well established in the field, as shown in Appendix I for selected samples. The procedure of the test was to rapidly oxidize the sample by the addition of different amounts of hydrogen peroxide and then analyse the resulting solution for the same parameters as a standard shake flask. The objective was to determine change in the release of sulphate, acidity, and metals as the rocks are progressively more oxidized i.e. as time and oxidation proceed. Samples from older dumps samples show relatively small increase in sulphate, acidity, metals and relatively small decrease in pH with further oxidation suggesting that these samples are approaching the end of the oxidation process. The total release corresponds to the extent rather than the rate of oxidation. Younger samples tend to show more marked change in the total release with increasing strength of oxidant, but similar proportions to the older samples. While these are quite simple, qualitative tests they support the conclusions that the contaminant loadings seen in the field should not change dramatically with time.

Column leach testing on different rock types supports also shows that the "pure" sulphide samples, and schists or diorites mixed with sulphides are the primary sources of metal release. The extraction and the kinetic testing also show that the zinc is readily released into solution over a range of pH values and release is not limited to, nor accelerated, by increasing acidity. However, the majority of the sulphide samples in the kinetic testing released acidic drainage from the start of testing, even from "young" samples. For the longer term predictions, it was assumed that all sulphides in the dumps would be acidic. Depletion calculations support this observation for majority of the sulphides and sulphides mixed with both intrusives and schists. Schist samples however show neutral paste pH values - even from older samples with little remaining AP (acid potential) or NP neutralization potential. The calc. silicate material is clearly acid consuming.

The oxidation rate however does not appear to be critical to dump drainage chemistry. The primary control is not oxidation rate but rather the hydrology of the dump and the extent of flushing of the soluble products.

Short-term extraction testing, using distilled water, indicates that there is a significant load of soluble metals, sulphate and acidity associated with the sulphides and, to a lesser extent, with the mixed schists or intrusives and sulphides. The un-mixed or "pure" schists do not show a significant stored load by comparison, and a moderate release of alkalinity. Intrusives mixed with sulphides reflect the loadings of the sulphide component, as a pure intrusive sample showed very low metal and sulphate release. Calc. silicates show very low stored metal loads and a consistent release of alkalinity from all samples.

Given that the samples tested in this program are from relatively shallow depths and are therefore comparatively well flushed compared to the overall dump, it appears that there may be a considerable soluble load stored within the waste rock, and particularly the sulphide waste rock, within a dump. This in turn suggests that the control on the drainage water chemistry from the dump is controlled primarily by hydrology (i.e. the amount of water infiltrating through the dump) and also by attenuation reactions within the dump (either physical or chemical attenuation mechanisms). Overall estimates of the stored load in each dump were provided in Chapter 4.

The kinetic test data also show the effect of the stored load on the water chemistry. Despite intensive irrigation during the early testing period, the release of soluble products continues for several testing cycles and appears to dominate water chemistry. In all columns, metal release rates in the later stages of testing are less than in the early stages (after the flushing period). This suggests both that the rate of oxidation does not control the production of contaminants from the dump and that, for the majority of the rock, the rate of oxidation and acid production has reached a constant (and perhaps maximum) rate.

6.2.3 Method

The steps and assumptions in the annual dump loading calculations for each of the dumps are discussed in Appendix R and summarized in the following:

1. Calculate tonnage of each rock type, in each of the dumps shown in Figure 4-10
2. Classify each dump as "old" i.e. constructed prior to 1980, or "young" i.e. constructed since 1980.
3. Calculate production of metals, acidity and alkalinity from laboratory kinetic tests in terms of mg "contaminant" per kg of rock per week for each rock type or combined rock type and for the different ages of kinetic testing samples.
4. Estimate the annual production from each dump, using the production rates from laboratory testing and the rock type composition for each dump.
5. Adjust predictions of loading for temperature, surface area/particle size, effect of channelling.
6. Using annual production rate and MAR (see section 3.6), calculate average annual concentration in drainage from each dump.
7. Compare predicted ranges of concentrations with MINTEQA2 for equilibrium solubility constraints; and,
8. Allocate the loadings from each dump to the appropriate drainage area or receiving water, as discussed in Chapter ?? and shown in Table 6.5, for use in loading calculations.

Table 6.5
Dump Drainage

Symbol	Name	Proportion of dump drainage reporting to:						
		Main Pit	Zone II Pit	X23	Rose Creek Valley	North Fork Rose Creek	North Fork Rose Creek, downstream of rock drain	Guardhouse Cr.
NWU	Upper Northwest Dump			50%				50%
NWM	Middle Northwest Dump			50%				50%
NWL	Lower Northwest Dump			50%				50%
LPL	Lower Parking Lot Dump							100%
UPL	Upper Parking Lot Dump							100%
FVN	Faro Valley North	100%						
FVS	Faro Valley South	100%						
MDW	Main Dump West			100%				
MDE	Main Dump East			70%			30%	
ID	Intermediate Dump					0%	100%	
NEU	Upper Northeast Dump	50%	5%			45%		
NEL	Lower Northeast Dump		60%			40%		
NEO	Outer Northeast Dump		100%					
ZIIW	Zone II West					100%		
ZIIE	Zone II East		90%			10%		
RZD	Ramp Zone Dump					100%		
RD	Ranch Dump					100%		
SWPWD	Southwest Pit Wall Dump	100%						
LGSPA	Low Grade Stockpile A	30%		70%				
LGSPC	Low Grade Stockpile C	30%		70%				
FTW	Fuel Tank Dump W			100%				
FTE	Fuel Tank Dump E			100%				
MME	Mt. Mungly East			100%				
MMW	Mt. Mungly West			100%				
MGSP	Medium Grade Stockpile			100%				
OXSP	Oxide Fines Stockpile			100%				
SPB	Stock Piles Base			100%				
CHSP	Crusher Stockpile			100%				
OHRW	Outer Haul Road West			20%	50%		30%	
OHRE	Outer Haul Road East					100%		
NFRD	North Fork Rock Drain					100%		

Two comparisons were done to calibrate the predictions. The geochemical equilibrium model MINTEQA2 was used to consider the chemical controls on dump drainage and the potential for limiting maximum concentrations of metals of interest. Predicted and measured total annual loadings for sub-catchments were compared to consider the "field" controls on water chemistry and determine a reasonable range for predicted values (see appendix R).

6.2.4 Faro Waste Rock Dumps

The contaminant loading predictions for each major dump and smaller dumps are summarized in Table 6.6 for selected metals. These are the maximum predicted longer term loadings as there is assumed to be no attenuation by physical or chemical reactions in the dump, and no reduction of metals by solubility controls. These values could reasonably be the maximum predicted load release since it has been shown that the rate of metal release from a dump is not limited by the rate of oxidation, that there is storage of oxidation products within each rock assemblage, that there is oxidation and metal release occurring in all materials with reactive sulphide, and that the development of the oxidation process has peaked or is declining.

Table 6.6

Summary of Calculated Annual Average Loadings from Dumps

Zone II Dumps Estimated Maximum Total Annual Load (tonnes)

Parameter	SO_4	Fe	Mn	Ni	Pb	Zn
Zone II West	158	2.8	1.7	0.0	0.1	16.7
Zone II East	378	1.1	0.2	0.0	0.1	4.0

Upper Northwest Dumps Estimated Maximum Total Annual Load (tonnes)

Parameter	SO_4	Fe	Mn	Ni	Pb	Zn
Upper Northwest Dump	107	22.3	0.1	0.0	0.1	9.4
Middle Northwest Dump	298	62.6	0.1	0.0	0.2	7.4
Lower Northwest Dump	300	57.6	0.1	0.0	0.2	7.7

Faro Valley Dumps Estimated Maximum Total Annual Load (tonnes)

Parameter	SO_4	Fe	Mn	Ni	Pb	Zn
Faro Valley North	269	57.3	0.1	0.0	0.2	5.5
Faro Valley South	14	0.2	0.0	0.0	0.0	0.9

Northeast Dumps Estimated Maximum Total Annual Load (tonnes)

Parameter	SO_4	Fe	Mn	Ni	Pb	Zn
Lower Northeast Dump	362	6.7	3.3	0.1	0.2	36.7
Outer Northeast Dump	3	0.0	0.0	0.0	0.0	0.1
Upper Northeast Dump	231	4.7	2.3	0.0	0.2	25.7

Table 6.6 con't
Summary of Calculated Annual Average Loadings from Dumps

Main and Intermediate Dumps Estimated Maximum Total Annual Load (tonnes)

Parameter	SO_4	Fe	Mn	Ni	Pb	Zn
Main Dump East	1926	38.0	28.5	0.3	1.8	248.8
Main Dump West	805	8.3	7.1	0.1	0.5	57.6
Intermediate Dump	1463	35.2	29.0	0.2	1.8	242.2

Parking Lot Dumps Estimated Maximum Total Annual Load (tonnes)

Parameter	SO_4	Fe	Mn	Ni	Pb	Zn
Upper	67	11.9	0.0	0.0	0.0	0.6
Lower	44	7.3	0.0	0.0	0.0	0.4

Stockpiles & Dumps Estimated Maximum Total Annual Load (tonnes)

Parameter	SO_4	Fe	Mn	Ni	Pb	Zn
Mt. Mungly East	34	3.6	0.3	0.0	0.0	5.5
Mt. Mungly West	11	1.7	0.1	0.0	0.0	2.4
Fuel Tank East	52	0.8	0.1	0.0	0.0	1.3
Fuel Tank West	7	1.2	0.1	0.0	0.0	1.7
Oxide Fines Stockpile	27	4.3	0.4	0.0	0.1	6.2
Stock Piles Base	59	0.0	0.0	0.0	0.0	0.0
Low Grade Stockpile A	76	12.1	1.1	0.0	0.2	17.6
Low Grade Stockpile C	65	10.4	0.9	0.0	0.1	15.2
Medium Grade Stockpile	0	0.0	0.0	0.0	0.0	0.0

Table 6.6 con't

Summary of Calculated Annual Average Loadings from Dumps

Haul Road Dumps Estimated Maximum Total Annual Load (tonnes)

Parameter	SO_4	Fe	Mn	Ni	Pb	Zn
Outer Haul Road East	138	1.8	1.3	0.0	0.1	11.3
Outer Haul Road West	329	2.5	0.8	0.0	0.1	11.8

Near Pit Dumps Estimated Maximum Total Annual Load (tonnes)

Parameter	SO_4	Fe	Mn	Ni	Pb	Zn
Ranch Dump	16	0.2	0.1	0.0	0.0	0.9
Ramp Zone Dump	30	0.1	0.1	0.0	0.0	1.0
Southwest Pit Wall Dump	96	2.5	2.0	0.0	0.1	17.1

Comparison with field data suggest that the current loadings measured in the surface waters of the receiving environment are significantly less than would be predicted from mass balance calculations which do not consider attenuation processes.

The predicted loadings for each dump are attributed to the appropriate catchments in Table 6.7 (for zinc) and 6.8 (for sulphate). These loadings to various sub-catchments and monitoring stations are then compared to the measured loadings in 1995 in Table 6.9 for zinc, and Table 6.10 for sulphate. These two parameters were selected to provide a comparison between physical attenuation mechanisms (that could affect both parameters) and chemical attenuation mechanisms (i.e. that could affect zinc concentrations).

Table 6.7

Predicted Maximum Diss. Zinc Loadings to Subcatchments

Symbol	Name	Annual production and loading of zinc (tonnes)						
		Main Pit	Zone II Pit	X23	Rose Creek Valley	North Fork Rose Creek	North Fork Rose Creek, downstream of rock drain	Guardhouse Cr.
NWU	Upper Northwest Dump	-	-	5.2	-	-	-	4.2
NWM	Middle Northwest Dump	-	-	5.3	-	-	-	2.1
NWL	Lower Northwest Dump	-	-	5.3	-	-	-	2.4
LPL	Lower Parking Lot Dump	-	-	-	-	-	-	0.4
UPL	Upper Parking Lot Dump	-	-	-	-	-	-	0.6
FVN	Faro Valley North	5.5	-	-	-	-	-	-
FVS	Faro Valley South	0.9	-	-	-	-	-	-
MDW	Main Dump West	-	-	57.6	-	-	-	-
MDE	Main Dump East	-	-	174.2	-	-	74.6	-
ID	Intermediate Dump	-	-	-	-	-	242.2	-
NEU	Upper Northeast Dump	12.8	1.3	-	-	11.6	-	-
NEL	Lower Northeast Dump	-	32.3	-	-	4.4	-	-
NEO	Outer Northeast Dump	-	0.1	-	-	-	-	-
ZIIW	Zone II West	-	-	-	-	16.7	-	-
ZIIE	Zone II East	-	3.6	-	-	0.4	-	-
RZD	Ramp Zone Dump	-	-	-	-	1.0	-	-
RD	Ranch Dump	-	-	-	-	0.9	-	-
SWPWD	Southwest Pit Wall Dump	17.1	-	-	-	-	-	-
LGSPA	Low Grade Stockpile A	5.3	-	12.3	-	-	-	-
LGSPC	Low Grade Stockpile C	4.5	-	10.6	-	-	-	-
FTW	Fuel Tank Dump W	-	-	1.7	-	-	-	-
FTE	Fuel Tank Dump E	-	-	1.3	-	-	-	-
MME	Mt. Mungly East	-	-	5.5	-	-	-	-
MMW	Mt. Mungly West	-	-	2.4	-	-	-	-
MGSP	Medium Grade Stockpile	-	-	0.0	-	-	-	-
OXSP	Oxide Fines Stockpile	-	-	6.2	-	-	-	-
SPB	Stock Piles Base	-	-	0.0	-	-	-	-
CHSP	Crusher Stockpile	-	-	0.0	-	-	-	-
OHRW	Outer Haul Road West	-	-	2.4	5.9	-	3.5	-
OHRE	Outer Haul Road East	-	-	-	-	11.3	-	-
NFRD	North Fork Rock Drain	-	-	-	-	-	-	-
	Total by Station	46.1	37.3	290.0	5.9	46.2	320.4	9.6

Table 6.8
Predicted Maximum Sulphate Loadings to Subcatchments

Symbol	Name	Annual production and loading of sulphate (tonnes)						
		Main Pit	Zone II Pit	X23	Rose Creek Valley	North Fork Rose Creek	North Fork Rose Creek, downstream of rock drain	Guardhouse Cr.
NWU	Upper Northwest Dump	-	-	91	-	-	-	16
NWM	Middle Northwest Dump	-	-	263	-	-	-	35
NWL	Lower Northwest Dump	-	-	255	-	-	-	45
LPL	Lower Parking Lot Dump	-	-	-	-	-	-	44
UPL	Upper Parking Lot Dump	-	-	-	-	-	-	67
FVN	Faro Valley North	269	-	-	-	-	-	-
FVS	Faro Valley South	14	-	-	-	-	-	-
MDW	Main Dump West	-	-	805	-	-	-	-
MDE	Main Dump East	-	-	1,348	-	-	578	-
ID	Intermediate Dump	-	-	-	-	-	1,463	-
NEU	Upper Northeast Dump	115	12	-	-	104	-	-
NEL	Lower Northeast Dump	-	262	-	-	100	-	-
NEO	Outer Northeast Dump	-	3	-	-	-	-	-
ZIIW	Zone II West	-	-	-	-	158	-	-
ZIIE	Zone II East	-	340	-	-	38	-	-
RZD	Ramp Zone Dump	-	-	-	-	30	-	-
RD	Ranch Dump	-	-	-	-	16	-	-
SWPWD	Southwest Pit Wall Dump	96	-	-	-	-	-	-
LGSPA	Low Grade Stockpile A	23	-	53	-	-	-	-
LGSPC	Low Grade Stockpile C	20	-	46	-	-	-	-
FTW	Fuel Tank Dump W	-	-	7	-	-	-	-
FTE	Fuel Tank Dump E	-	-	52	-	-	-	-
MME	Mt. Mungly East	-	-	34	-	-	-	-
MMW	Mt. Mungly West	-	-	11	-	-	-	-
MGSP	Medium Grade Stockpile	-	-	-	-	-	-	-
OXSP	Oxide Fines Stockpile	-	-	27	-	-	-	-
SPB	Stock Piles Base	-	-	59	-	-	-	-
CHSP	Crusher Stockpile	-	-	-	-	-	-	-
OHRW	Outer Haul Road West	-	-	66	164	-	99	-
OHRE	Outer Haul Road East	-	-	-	-	138	-	-
NFRD	North Fork Rock Drain	-	-	-	-	-	-	-
	Total by Station	537	616	3,116	164	584	2,139	207

Table 6.9
Comparison of Zn Loadings in Dump Drainage

Subcatchment	Station	Predicted Zn Load tonnes per year	Field data** tonnes	Factor	Potential Influences
Main Pit from external dumps	X22B	46.1	30.0	65%	1991, 1992 field data, flow/storage in dumps main control
Zone II Pit	X26	37.3	5.5	15%	neutralization
X23 (measured)	X23*	290	5	2%	~50% of flow in groundwater, attenuation along flowpath, alkalinity in dump
X23 (1990 estimate)	X23*	290	40 (1990 est.)	14%	prediction in 1990 including groundwater
X7	X7*	290	30	10%	attenuation along flowpath, load reporting to groundwater, additional loading between X23 and X7
North Fork Rose Creek	X2* + RCV*	372	—	—	load reporting to groundwater, attenuation along flowpath
Rose Creek Valley	(X2*+X7*+RCV*)	662	31	5%	
-loss to groundwater (50%)	(X2+X7+RCV)	331	31	9%	adjusted for seepage calc., load reporting to groundwater
Upper Guardhouse Cr.	NWI	8.6	0.01	0%	attenuation in dump, in 1995 also included water from mill periodically
- loss to groundwater (50%)		-4.3			est. seepage to groundwater and surface ponding
Lower Guardhouse Cr.	GHSCR2	1.0	0.04	4%	attenuation in dump, input from mill area, sparse data for variable flow
Guardhouse Cr. to Int. Pond	(NWI+GHSCR2)	5.3	0.05	1%	attenuation, load to groundwater

* groundwater component not quantified

** 1995 field data used unless not applicable, and otherwise noted

Table 6.10
Comparison of Sulphate Loadings in Dump Drainage

Subcatchment	Station	Predicted SO_4 Load tonnes per year	1995 field data tonnes	Factor	Potential Influences
Main Pit from external dumps	X22B	537	412	65%	1991, 1992 field data, flow/storage in dumps main control
Zone II Pit	X26	616	145	15%	
X23	X23*	3116	324	2%	~50% of flow in groundwater, attenuation along flowpath, alkalinity in dump
X7	X7*	3116	1254	40%	attenuation along flowpath, load reporting to groundwater, additional loading between X23 and X7
North Fork Rose Creek	X2* + RCV*	2887	394	10%	load reporting to groundwater, attenuation along flowpath
Rose Creek Valley	(X2*+X7*+RCV*)	6003	—	—	
-loss to groundwater (50%)	(X2+X7+RCV)	3002	1648	5%	adjusted for seepage calc., load reporting to groundwater
Upper Guardhouse Cr.	NWI	96	13	9%	flow attenuation in dump, in 1995 also included water from mill periodically
- loss to groundwater (50%)		-48	—	—	est. seepage to groundwater and surface ponding
Lower Guardhouse Cr.	GHSCR2	111	186	0%	sparse data for variable flow, input from mill area
Guardhouse Cr. to Int. Pond	(NWI+GHSCR2)	159	199	4%	

* groundwater component not quantified

Generally, the predictions are higher than the field estimates for sulphate but are in the same order of magnitude. The comparison between the predicted and measured values for sulphate is surprisingly consistent for such a complex set of predictions and field conditions. The loadings for both zinc and sulphate to the Faro Main Pit, which are dominated by the Faro Valley Dumps, are predicted to be only slightly larger (1.3 times) the values measured in 1991/92 for the seepage into the pit (as represented by Station X22A). This comparison is considered to be the most "accurate", as these dumps which drain to the pit would the least affected by loading to groundwater (all seepage reports to the pit) and the least affected by attenuation reactions with the subsurface. It must be noted that these comparisons were completed after the geochemical predictions - there was no attempt to "calibrate" the predicted loadings to the measured loadings.

Greater differences between the predicted and measured zinc loading values are evident for stations where there is believed to be a significant component of groundwater flow, such as the Rose Creek Valley at the toe of the dumps. Future groundwater monitoring with the 1996 groundwater well installations (see section 4.2.13) will be useful to calibrate these predictions with a similar comparison between predicted and measured flows.

In addition to the separation of flow and loading to the groundwater system, the potential for chemical and physical attenuation of flow and loading within the dumps and the subsurface was considered. The factor difference between the predicted and measured loads is much greater for zinc than for sulphate, suggesting that there are both chemical and physical mechanisms with the dumps and within the subsurface controlling the release of contaminants to the receiving environment. Maximum predicted values in the mass balance model and closure water quality predictions do not include chemical attenuation controlling mechanisms and have not been adjusted to reflect current field measurements. Thus, it is believed that these predictions would represent a "worst case" situation with no residual capacity for attenuation.

The potential mechanisms of attenuation that contribute to the differences between predicted and measured loadings include:

1. Neutralization and precipitation of metals by contact with alkaline drainage from calc. silicates and, to a lesser degree, schists. While this mechanism is only significant at alkaline pH values, the field measurements and the kinetic testing indicate that pH values of 8 and alkalinity values of 200 mg/L do occur from these rock assemblages. Table 6.11 shows equilibrated concentrations of zinc predicted using MINTEQA2 for a range of pH, zinc and alkalinity values. Details of the calculations are provided in Appendix R. The MINTEQA2 results suggest that considerable reductions in zinc concentrations could occur in dumps where there is layering (of sulphides above the alkali source) or mixing with alkali sources such as the sulphide cells of the Main and Intermediate Dumps. Field seepage and groundwater measurements indicate pH values of greater than 7 for several dumps. Sulphide and NP depletion calculations for the kinetic testing are shown in Appendix I. Estimated total NP values for each dump were discussed in section 4.2.

2. Co-precipitation or surface exchange mechanisms which are difficult to quantify using existing geochemical modelling. There are field indications of precipitation or surface exchange seen in the higher soluble metal content (based on extraction testing) of calc. silicate, schist and intrusive samples from areas of infiltration of drainage from

sulphides compared to "clean" samples of each rock type, as seen in Appendix I , for example samples AR/NW/26 A, AR/DR/12 and AR/SC/08. The "scavenging" effect or co-precipitation of metals with, for example, hydrous iron oxides is well recognized in the literature however may not be as significant a mechanism in the dump drainage.

3. Loss of loading via seepage to groundwater, particularly in the Rose Creek Valley in the area of Station X23, X7 to X2. There can also be loss to shallow groudnwater or surface water ponding, such as at the toe of the Northwest Dump and areas of the Northeast Dump. Seepage estimates show that up to 50 mm infiltration equivalent may report to groundwater in areas such as the Rose Creek Valley (section 4.2.13).
4. Storage of loading within the dump as a result of storage of water (residual moisture content, flowpath transit time, and evaporation or precipitation).

Tables 6.9 and 6.10 also list mechanisms which are believed to be contributing to the differences in predicted and measured loads for different dumps. Considering these mechanisms, the factors comparing measured to predicted are reasonably consistent being larger for dumps with significant potential for neutralization, long flowpaths in the dump, and high potential loss to groundwater (e.g. Main and Intermediate Dumps). The possible influence of seepage in groundwater may be seen in the difference between the values at X23 and further along the flowpath at X7; more of a groundwater discharge area although there are additional potential loading sources between the two.

Table 6.11

Predicted Equilibrated Zn Concentration in Dump Drainage

Conditions	Baseline	Slightly Acidic	Neutral	Slightly Alkaline
Input				
Data for Column No.	9	5	8,12	8,12
pH	3	6	7	7.5
Alkalinity (mg/L $CaCO_3$)	0	30	150	150
[Zn] (mg/L)	2094	2200	2200	2200
MINTEQA2 Equilibrated				
[Zn] (mg/L)	—	2100	510	96

Lower factors of different are seen in older and/or lower height dumps where the transit time for water flowing through the dump is less or there is less potential loss to groundwater (e.g. Faro Valley Dumps). The Zone II pit waters could be affected by neutralization not only from waste rock but also from Main Pit water.

6.2.5 Faro Main Open Pit

The effect of the addition loading to the pit water is predicted to be comparatively small, given the existing loads of sulphate and zinc (and other metals) currently in the open pit. Prior to deposition of tailings to the pit, approximately 80% of the zinc and sulphate loadings were contributed by the sulphate wastes deposited in the pit. The remaining 20% of the load was contributed by drainage from the pit walls and external dumps.

After pumping from the pit ceased in July of 1992, there were two major changes to the water management that would affect water chemistry in the pit; the addition of fresh, alkaline tailings from the pit, and the periodic pumping of Zone II pit water to the Main pit (see sections 4.1.4.3 and 4.4).

Deposition of tailings in the pit provides alkalinity to the pit water. The layer of tailings over the sulphide wastes also provides a lower permeability layer between the sulphide waste and the pit water. This may limit the circulation of water through these sulphide wastes, thereby limiting the longer term release of additional soluble contaminants from the pit. The extent to which the loadings from the in-pit dumps will contribute to the overlying pit water can be quantified by continued monitoring, particularly after the sulphide wastes are covered with tailings.

A comparison of the measured and predicted flows from the external dumps to the pit was provided in Tables 6.9 and 6.10. To address the release of contaminants from the sulphide wastes and pit walls not included in the previous calculations, a simple mass balance was completed as summarized in Table 6.12. This table includes the comparison for the drainage from external dumps, as these represent a portion of the loading to the Main Pit. The Zone II loadings have not been included.

The alkalinity in the pit provides a measure of control for long term zinc concentrations in the pit, although not to levels that would be considered acceptable for discharge. A comparison of predicted equilibrated zinc concentrations is summarized in Table 6.13 with details of the calculations included in Appendix R. The comparisons use the MINTEQA2 geochemical equilibrium model. Measured pit water chemistry from 1995 samplings was first equilibrated and minerals near saturation noted. Zinc and sulphate concentrations were increased to reflect approximately 10 years of loading to the pit. Alkalinity and pH values were then adjusted to consider the possible controls on zinc solubility. While this mechanism would not in itself provide treatment in the pit, there is a marked effect of alkalinity (carbonate) on zinc solubility that would limit concentrations as shown in Table 6.13.

Table 6.12

Comparison of Measured and Predicted Loadings to Main Pit Mass Loading Balance for the Main Pit

Station	Description	Data Period	Loading per year		Calc. total load to 1996	
			SO_4 (tpa)	Zn (tpa)	SO_4 (tonnes)	Zn (tonnes)
X22	Water pumped from bottom of pit during dewatering, including in-pit dump drainage.	1991/1992	1166	178	4664	712
X22A	Water from A30 sump (pit walls) during dewatering, comprising Faro Valley Dump seepage and north wall seepage.	1991/1992	412	30	1648	120
	- contribution from tailings	1995 & 1996			-800	=
Pit water	Calculated loading to pit water (not including Zone II) for period of pit filling to July 1996				5512	832
Pit water	Pit sampling 1995/96.	1995/1996			1120	125
	Calculated factor for pits (predicted/measured)				4.9	6.7
Pit water	Predicted loading from external dumps.		537	46		
	Calculated factor for dumps (predicted/measured)		1.3	1.5		

Table 6.13
Predicted Equilibrated Zn Concentration in Pit Water

Conditions	Pit Water	Short term (continued loadings)	Med. term (with tailings deposition)	Longer term
	1996	Increase [Zn]	Increase [Zn] and [SO_4]	Decrease alkalinity with high zinc
Input*				
pH	7.5	7.5	7.5	7.5
Alkalinity (mg/L CaCO3)	50	50	50	25
[SO4] (mg/L)	700	700	1400	1400
[Zn] (mg/L)	7.8	16	50	50
MINTEQA2 Equilibrated				
[Zn] (mg/L)	—	12	27.6	44
Predicted Controlling Mineral Phases		$ZnCO_3 \cdot H_2O$	$ZnCO_3 \cdot H_2O$ ZnO	$ZnCO_3 \cdot H_2O$

* A full suite of metals from 1995 and 1996 pit sampling was used for modelling.

Key variables are listed here.

6.3 Vangorda Plateau - Mine water Chemistry

6.3.1 Method

The methods for estimation of longer term water chemistry from the Vangorda and the Grum dumps are the same as discussed in section 6.2.3 for Faro. Briefly, column and humidity test results were used to calculate a production rate for each "contaminant" in terms of "mg/kg/week" for each of the sulphides and phyllites. These results were multiplied by the field tonnage, factor for particle size or surface area of rock in the dump compared to the lab, factor for temperature, factor for extent of flushing (channelling of flow) and by weeks to calculated an annual production rate. It is recognized that the laboratory test, which uses more water per unit weight or area of rock than in the field, will achieve more complete flushing than in the field.

Also, it is recognized that with increasing dump height there is an increasing likelihood of flow being concentrated in specific flow channels (i.e. larger void spaces). Different scenarios were run with changing degree of flushing. It was decided to use a relatively conservative estimate of 50% of the surface area flushed per year. The extent of flushing determines the extent of accumulation of stored soluble products within the dump. During intensive flow events, or with

any disturbance of the dump or the cover on the dump, some portion of this stored load could be flushed resulting in a high concentration in the drainage combined with high flow. While it is difficult to quantify the extent to which dump settling, loss of integrity of a cover, or an intensive infiltration period, it is important to estimate the accumulation to determine if the above events represent a risk in the longer term.

For the Vangorda and the Grum dumps, one of the closure options is the placement of a till cover to reduce infiltration on all or part of the dump. Placement of a cover is effective in reducing the total infiltration through a dump, however it does not necessarily reduce the total loading. If the reduction in flow changes the surface area of the rock being flushed, i.e. changes the distribution of water such that there is less total loading, then a cover can reduce the total loading. However, if the flow patterns remain the same but the volume of water is reduced, the total loadings could remain essentially the same and only the concentration in the drainage would increase unless there are geochemical saturation controls on the drainage water chemistry.

Quantifying these unsaturated flow processes for each dump would require more sophisticated modelling and extensive field investigation that is not warranted for the questions at hand. Rather, different flushing and infiltration regimes were simulated and a "high" and "low" predicted loading determined. For the purposes of these simulations, the annual average infiltration of 30% of MAR and 70% of MAR were used, to represent covered and not covered conditions. 70% was based on seasonal distributions of flow and assuming that only during the high flow spring period is there runoff from the dumps, for the rest of the year the infiltration would be 100%. 30% is based on a somewhat conservative estimate of the effectiveness of a till cover to reduce infiltration.

The contaminant loading estimates were done for three conditions:

- for the "current" geochemical conditions based on laboratory testing of comparatively fresh samples collected in 1995/96 and in the Water Licence Application testwork;
- for the longer term conditions for which the primary concern was the development of acidic conditions in samples which are currently generating neutral to alkaline drainage; and,
- to address the difference in infiltration and flushing with covers.

The estimates for the current conditions were then compared to field data where available to see if the predictions appear reasonable. No changes or "calibration factors" were made to the model. It is believed that the field data are affected by a variety of conditions that cannot be directly quantified, as discussed earlier in this Chapter.

At Vangorda, the most significant of these influences is the placement of the baritic fines on the waste dump which may be influencing the chemistry at three of the four drains that carry water from the dump. At Grum, the relatively short time period for water migration through the pile and the continued construction of the dump would have more influence on the water chemistry at the very low concentrations currently seen in dump seepage or pit waters. At both sites, and in some laboratory testing, the alkaline pH conditions and low sulphate concentrations can induce solubility constraints that would reduce the total load measured in the water sample compared to the prediction.

The predicted loads for current conditions at Vangorda are similar to the measured loads, indicating that there are no major flaws in the modelling approach. The same calculations were used for longer term conditions, and for the Grum site for which field comparison data are not available. These predictions therefore provide a range of predicted loadings sufficient to address closure options, treatment requirements and longer term changes in water chemistry.

6.3.2 Vangorda Dumps

Results of the modelling show, for Vangorda:

- the drainage from the sulphides at Vangorda would not be considered acceptable for discharge as concentrations are predicted to exceed 10 mg/L for zinc with elevated values for other metals and increasing acidity;
- the presence of sulphides mixed with phyllites and the deposition of baritic fines within the phyllite cell at Vangorda would also result in elevated loadings to the receiving environment;
- drainage from the baritic fines and probably the sulphide cell is at pH 5 or less; and,
- the phyllite cell contains sulphides and possibly baritic fines and therefore may eventually generate acidic drainage.

The estimations of loadings for the Vangorda waste dumps were based on kinetic testing results from both the Water Licence Application (1989) and from the 1995/96 test program. The more recent geochemical testing program for this report focused on sulphide material, particularly the low-grade ore and the baritic fines material. In the earlier testing program, tests were done in duplicate with one humidity cell operated under "standard" conditions and the other pre-treated with acidity and inoculated with bacterial cultures to accelerate the oxidation process. There are a number of concerns with the geochemical changes that occur in the sample as a result of the acidification but, in the case of "high grade" sample wherein acidification would deplete only NP or an acidic sample, the results may provide an indication of longer term metal production. Results of simulations including these tests are provided for reference in Appendix R, containing contaminant loading concentration calculations.

Metal production rates for both sulphides and phyllites from the IEE were compared to the metal production rates from a low grade ore sample from the Vangorda pit to estimate the range of potential loadings for current and future conditions. The samples tested in the 1989 program (for the Water Licence Application) all generated acidic drainage and thus would be expected to represent "worst case". Zinc production rates are about twice the values in the alkaline samples from the 1995/96 program.

In the field, drainage from the oxide fines and at least some of the sulphide cell is at pH 5 or less (based on the Vangorda dump drain water chemistry discussed earlier), similar to the earlier kinetic testing samples. However most of the drains still show near neutral pH values and alkalinity indicating that much of the waste rock is not (yet) acidic. In the longer term, net acidity could be anticipated from the Vangorda sulphides and possibly phyllites mixed with either sulphides or baritic fines. Thus, the predictions using the IEE values and the 1995/96 values probably represent a realistic range of longer term water quality.

Details of the predictions are provided in Appendix I.

Predictions for the "current conditions" are summarized in Table 6.14, with details of the calculations in Appendix R. The estimates of concentrations provided in this table are based solely on dilution of the loadings with the flows estimated from the water balance and do not include any geochemical equilibrium modelling. These values are provided for "order of magnitude" comparisons with the field data; showing that the basis for the predictive calculations is reasonable. They are also useful in the water treatment estimates.

Table 6.14

Estimation of Short-term Concentrations from Vangorda Waste Rock

Production based on:	Sulphides 1995/96 column mg/L	Sulphides IEE cell mg/L	Phyllites IEE cell mg/L	Fines Extractions mg/L
SO4	38024	4335	24192	8133
Alkalinity	795	20	1419	0
Acidity	3456	1146	645	6673
Co	0.4	3	4	13
Ni	0.5	3	3	5
Zn	244	449	71	3351
Dump Area (ha)	25	25	26	0.88

Table 6.15
1995 Vangorda Dump Water Chemistry

Parameter	Drain 2 mg/L	Drain 3 mg/L	Drain 5 mg/L	Drain 6 mg/L	Little Creek Sump mg/L
SO4	1590	3057	7140	4757	1030
Alkalinity	252	159	11.1	147	126
Acidity					
Co	0.615	2.67	12	8	
Ni	2.045	4.23	7.9	4.7	
Zn	64.05	253	1500	689	54.4

Table 6.15 shows the 1995 water chemistry for drainage from the dump and in the Little Creek Sump. The predictions were presented by rock type to indicate the relative loadings from the different waste rock types. The predicted values for the dump drainage from the areas of sulphides, phyllites, and fines respectively are similar to the chemistry from the relevant drain (see section 5.7.3). Drains 5 and 6 primarily drain the sulphide cell with some contribution to Drain 6 anticipated from the baritic fines. Drainage from these fines also reports in part to Drain 2 and possibly Drain 3. Drain 3 is located in an area of the dump with both phyllites and sulphides. The Little Creek sump collects drainage from the Vangorda Dump, but also from other areas of the site including runoff from the rest of the catchment. The catchment for the Vangorda Dump is approximately 70% of the total catchment of Little Creek Sump, thus would provide dilution of the predicted load. The field values therefore are not directly comparable to a prediction but do provide some confidence in the basis in the methods used herein for the current and therefore the longer term predictions for both Vangorda and Grum.

Table 6.16 shows the predicted loadings for Vangorda Dump for the three conditions listed earlier:

- loadings in the short term, prior to implementation of any further closure measures;
- loadings in the medium term, after implementation of covers; and,
- longer term loadings should net acidic conditions develop in the dump.

Table 6.16
Vangorda Main

Total production of:	Short-term, prior to covering		Medium term, covered		Long term, covered	
	Loading tonnes	Est. Max. Conc. mg/L	Loading tonnes	Est. Max. Conc. mg/L	Loading tonnes	Est. Max. Conc. mg/L
SO4	2,114	31173	932	26082	915	25598
Alkalinity	76	1113	423	11824	394	11023
Acidity	148	2188	192	5360	57	1607
Co	0.2	2.34	0.1	3.80	0.2	4.79
Ni	0.1	2.01	0.2	4.44	0.2	4.40
Zn	16	239	4	125	11	313

The predicted loads for current conditions are about 1.5 to 3 times the estimated current loading for most parameters, suggesting the prediction method is reasonable, if slightly conservative (in terms of overestimation). One major difference is seen in the sulphate predictions due in part to the use of a high sulphide sample for kinetic testing with a stored oxidation product load. Geochemical modelling using MINTEQA2 on the column testing samples indicates that the waters are at gypsum saturation and close to saturation for some cadmium, lead, aluminium and zinc minerals. Similar controls would exist within the dump and the drainage collection system, limiting sulphate concentrations.

The current loadings from the dumps are collected and pumped to the treatment plant via the Little Creek Sump. Longer terms loadings (also shown in Table 6.16) are predicted to be up to 50% higher than the current loadings, assuming that acidic conditions develop throughout the dump - in both the sulphides and phyllites. Unfortunately there are no data yet available for the influence of the existing cover on drainage water chemistry. In predicting the influence of a cover on the longer term loadings, it was assumed that the cover would reduce both the total volume of water flushing the rock and to a lesser degree, also the surface area flushed. The results show a lower loading, but similar concentrations in the drainage from the dump. Since the dump drainage and the surface runoff is collected in the same ditch, the overall effect would be to reduce concentrations to the treatment plant from the predicted values without covers.

6.3.3 Grum Dumps

Contaminant loading estimates for the Grum Dumps are summarized in Table 6.17. The results show that:

- drainage from Grum should be alkaline with concentrations of most metals and acidity about 15% of those at Vangorda;

- concentrations of nickel however are predicted to be higher at Grum than at Vangorda, and cobalt levels similar, due to the presence of the meta-intrusives at Grum. This prediction however arises from the use of this material in kinetic testing and subsequently in the metal production estimates. It is probably therefore a somewhat conservative prediction.

Table 6.17

Estimated Contaminant Loadings for Grum Dump, Under Alkaline and Acidic Conditions

Table 6.17a Predicted Conditions

Total production* of:	Short term, prior to cover		Longer term, cell cover	
	Loading tonnes	Est. Max. Conc. mg/L	Loading tonnes	Est. Max. Conc. mg/L
SO4	1,066	5850	930	5,891
Alkalinity	466	2558	423	2,679
Acidity	205	1125	189	1,200
Co	0.1	0.79	0.1	0.8
Ni	0.2	1.23	0.2	1.0
Zn	5	25	3	21

* Based on 1995/96 Col. 13 testing and IEE values for phyllites

Table 6.17b "Worse Case" conditions for comparison

Total production* of:	Acidic S drainage, no cover		Acidic S drainage, cell cover	
	Loading tonnes	Est. Max. Conc. mg/L	Loading tonnes	Est. Max. Conc. mg/L
SO4	1,023	5612	912	5782
Alkalinity	395	2166	394	2497
Acidity	84	463	55	350
Co	0.2	1.28	0.2	1.06
Ni	0.2	1.21	0.2	0.99
Zn	21	118	10	64

* Based on IEE testing (acidic drainage from sulphides)

Table 6.16 shows the estimated range of contaminant concentrations for the Grum Dump. It is believed that these predictions may be biased to the maximum (i.e. overestimate) possible values for four reasons:

- the Grum waste rock is generally of lower metal and sulphide content and significantly higher alkali content than the Vangorda, for which most of the historic database is available;
- net acidic conditions are not predicted for the Grum dump based on current testing;
- there is more opportunity for neutralization and attenuation reactions to occur within the Grum dump (compared to the Vangorda Dump) due to the higher alkali content and to the greater dump high and layering of the sulphides. A similar mechanism was postulated for the Faro Main and Intermediate Dumps to explain the reduction in loading to 10% of predictions; and,
- the dumps are higher, providing a longer flowpath for drainage and therefore increased potential for attenuation within the dump, particularly chemical saturation and precipitation given the predicted equilibrium controls (see Appendix R).

The potential for attenuation and the reactions within the dump require field data for calibration. Field data, particularly seep data from different areas of the dump, will continue to be collected to calibrate these predictions and evaluate acidity and alkalinity production. The loading estimates herein should be regarded as conservative, semi-empirical estimates.

In terms of closure planning and the estimation of water treatment requirements, the water from the Grum dumps is less than 10% of the total contaminant loading for alkaline drainage. These conditions are expected to persist in the dumps, and net acidic drainage is not anticipated. This is based on the estimated alkali content of the dumps, and on kinetic testing results which indicate that, in the laboratory, there is at least 20 years before the remaining NP would be depleted. Recognizing that the colder temperatures in the field (estimated range of 1 to 5 °C) within the dumps based on reported results can reduce reaction rates in the field to approximately 20% of the laboratory rates at 20 to 23 ° C (Loki Gold Corporation, Brewery Creek Project, Initial Environmental Evaluation, 1994), it may be 100 years before acidic drainage could be anticipated from within the sulphide cell. At this point, the majority of the sulphide has also been oxidized. The underlying lift of calcareous phyllites provides additional buffering capacity within the dump for the longer term, as do calcareous phyllites deposited with the sulphides in the shorter term.

Predicted loadings from the Grum Southwest Dump are low, at less than 1 tonne per year dissolved zinc and about 340 tonnes of sulphate. These values are still believed to be overestimates as they are based on testing of "average" phyllites including all types of phyllites rather than the "clean" phyllites that are being placed in this dump. Field monitoring for this dump, as for the Main Dump, will be necessary to calibrate predictions, particularly at such low concentrations.

6.3.4 Vangorda Open Pit

A contaminant loading balance for the open pit was calculated over the period of the shutdown, approximately two years, to quantify the relative contribution of loadings from the different sources. The balance shows that at least 90% of the sulphate loading is contributed by the predicted drainage from the low grade stockpile at the south end of the pit. Zinc concentrations are more difficult to balance, as there are solubility controls which limit the concentration of the seepage in the field, but which are difficult to explicitly quantify in geochemical modelling. However, the stockpile is predicted to contribute between 25 and 90% of the total zinc loading to the pit. Cobalt and nickel loadings are predicted to be significantly less than measured in the pit waters.

The total contaminant load in the pit in spring of 1995 was approximately 410 tonnes of sulphate, 13 tonnes of zinc, and 0.2 tonnes each of cobalt and nickel. Predicted loadings of sulphate to the pit from the stockpiles range from 49 tonnes based on the kinetic testing of the Water Licence Application and assuming 50 channelling (or 50% flushing of the produced load each year) to 430 tonnes based on the 1995/96 testing program of a stockpile sample. Predictions for dissolved zinc loadings are 5 and 3 tonnes respectively. Predicted cobalt and nickel loadings are an order of magnitude less than the measured loadings.

Three possible explanations for the higher measured than predicted loading are considered:

- the quantity of sulphides (low grade) remaining on the upper benches during the shutdown was underestimated as some of this material was removed for processing during 1995 and 1996;
- the laboratory sample showed slightly higher than average sulphide, cobalt and nickel contents but lower nickel, cobalt and zinc loadings in extraction tests than the average of the stockpile samples. Therefore the sample had a lower stored load than the average in the field. If the loading were increased proportional to the difference in sample and overall average grade, the predicted loading from the stockpile would be about 90% of the total stored load; and/or,
- there may be leaching occurring from submerged wall rock that was not accounted for in this modelling.

Since the majority of the remaining low grade material in the pit and all of the surface stockpile will be removed in 1996, these loadings are not relevant at closure.

Seep sampling in the pit shows that the next most significant source of loading is the sulphide exposures in the pit walls. Values of 8 mg/L dissolved zinc and over 230 mg/L sulphate have been measured in seeps in the areas of sulphide in the northeast pit wall. Sulphides represent approximately 30% of the pit wall area and therefore 30% of the drainage and seepage from the walls could be at concentrations of 8 mg/L dissolved zinc.

The relative contributions of loading from other rock types are too small compared to the stockpile loadings to be calculated from the pit water chemistry data and, realistically, are very low concentrations to be "modelled" from laboratory testing alone. These predictions are also complicated by the potential geochemical equilibrium controls on metal concentrations at the

alkaline pH values anticipated for any net acid consuming materials. Field contaminant production rates for wall rock were estimated from both kinetic testing and extraction testing results, scaled based on surface area (production rate x F_{SA}, where the production per unit weight in the laboratory is calculated in terms of particle surface area and then scaled to the field surface area exposed). The concentrations estimated from these calculations range <0.5 to 5 mg/L.

The total loading to the pit water therefore is primarily from seepage into the pit, and runoff over pit walls. The proposed till covers on sulphides in the upper benches would reduce the loadings from runoff, but not from groundwater seepage through the pit walls. It has been estimated that approximately 25% to 50% of the flow into the pit occurs as groundwater seepage. Therefore, assuming worst case of 50% groundwater inflow, 30% of which is at 8 mg/L Zn (from sulphides) and the remaining 70% at a estimated 2.5 mg/L (an approximate average including loadings from carbonaceous and other phyllites), half of the water to the pit could be at 4 mg/L dissolved Zn. Even if diluted with "clean" runoff from the catchment (assuming till covers in place) it does not appear that effluent water quality could be anticipated from the Vangorda pit at least within the flooding period while oxidation and leaching reactions occur in pit walls and thereafter until sufficient "clean" dilution has occurred. Unless sulphate reduction reactions were initiated, there are no natural geochemical controls in the pit water that could be predicted to reduce zinc concentrations to effluent quality levels.

6.3.5 Grum Open Pit

The geology of the Grum pit is such that the potential for acid generation and metal leaching is much lower than at Vangorda in terms of rock geochemistry, and is further reduced by the configuration of the pit with little or no sulphide exposed in the pit walls above the flooded elevation. Kinetic testing however has shown that while the sulphides are the primary source of oxidation and metal loadings, there can be metal leaching from non-calcareous phyllites although net acid generation is not anticipated. Extraction tests also show that there is considerable soluble alkalinity associated with the calcareous phyllites - perhaps sufficient to maintain pit water pH at neutral to alkaline values, as have been measured in the field from waste rock drainage, where there can be geochemical controls on metal solubility (cobalt, nickel, copper, iron and to a degree zinc). The relatively low sulphate concentrations in the laboratory and the field would contribute to a geochemical control on zinc, as aqueous zinc sulphate species contribute to higher zinc solubility (see Appendix R).

Groundwater flow into the pit at closure would occur primarily through the aquifer, and through the upper non-sulphide pit walls.

It is anticipated that, in the short term, the pit water chemistry would be similar to the current pit water chemistry as measured at station V23 and diluted by the clean groundwater that is currently recovered by pumping before discharge into the pit.

However, there are two potential sources of metal loading although net acid generation is not anticipated in the short term; from sulphides exposed in the lower pit wall and from sulphide waste which is deposited in the pit (i.e. not hauled to waste dumps). The definition of "short-term" depends on the eventual location of the waste within the pit in terms of flooding, and on the rate of

flooding of the pit. Kinetic testing of Grum samples indicates that, at a minimum, it could be 20 years before acidic conditions develop in the laboratory for the sulphides and the sulphides mixed with phyllites - and much longer under field conditions.

The second potential source of metals and sulphate is the sulphide waste left in the pit. Currently the design is for approximately 1.7 million tonnes of sulphide waste to be placed in the in-pit dump. Annual contaminant production from this rock is estimated to be about 200 tonnes sulphate, from 2 to 17 tonnes zinc, about 0.1 tonnes nickel and about 0.05 tonnes cobalt. Loadings from pit walls are orders of magnitude less than these loadings, due to the relatively low surface area of rock exposed in pit wall compared to broken rock.

If control measures are implemented to remove or attenuate this load (i.e. chemical addition) and/or water treatment immediately upon flooding, and if the sulphide exposed in pit walls are flooded, it may be reasonable to anticipate very low metals in the pit water. Concentrations could be anticipated to be lower than currently seen in pit waters.

6.4 References

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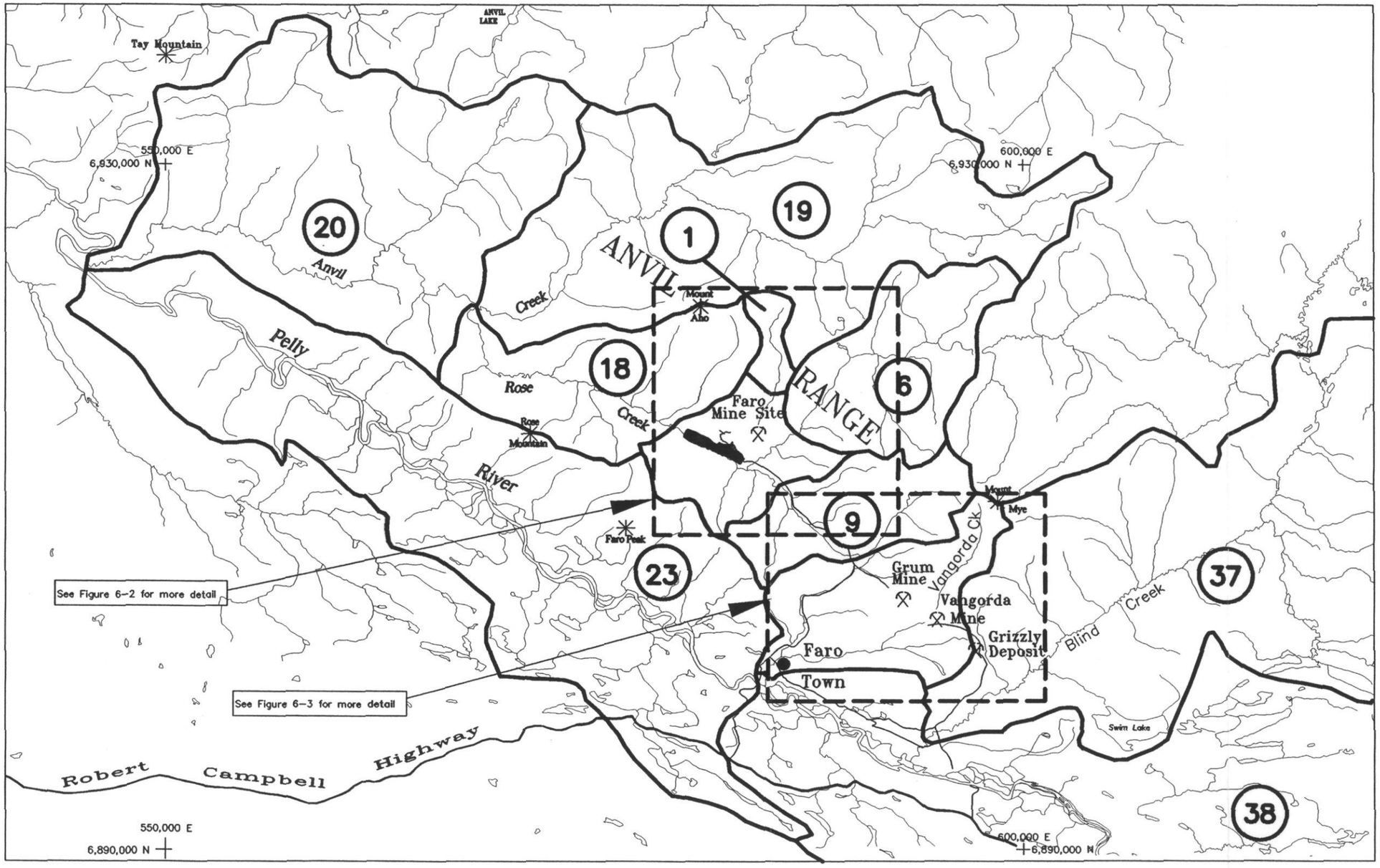
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FIGURES



Notes: Map derived from 1:250,000 scale
NTS Mapsheet 105 K
North American Datum 1927
Transverse Mercator Projection

0 2500 5000 7500 10,000 12,500 METRES
SCALE: 1: 250 000

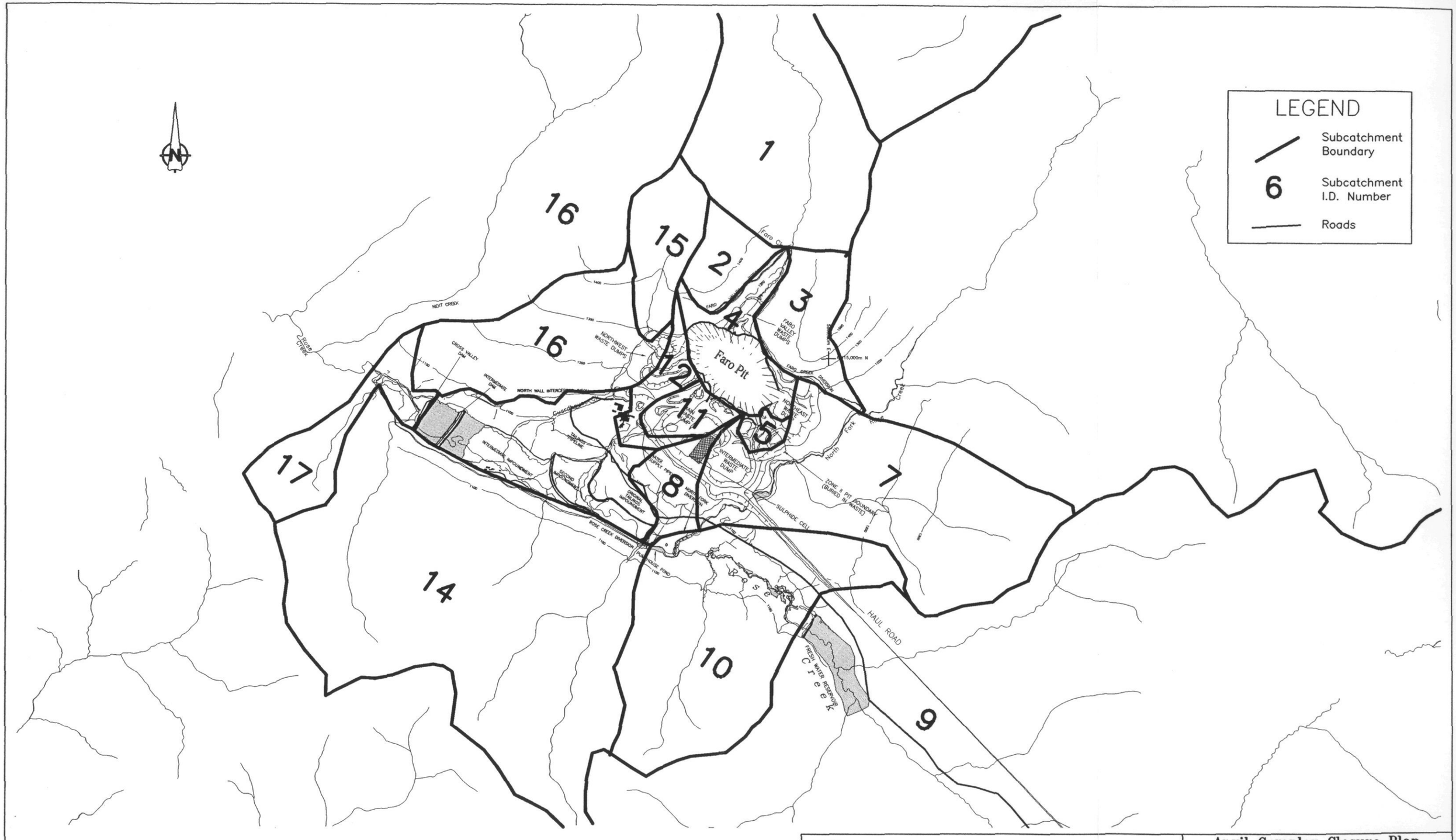
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Anvil Complex Closure Plan

MINESITE CATCHMENTS

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		6-1



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Anvil Complex Closure Plan

Minesite Subcatchments in Vicinity of Faro Development

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		6-2



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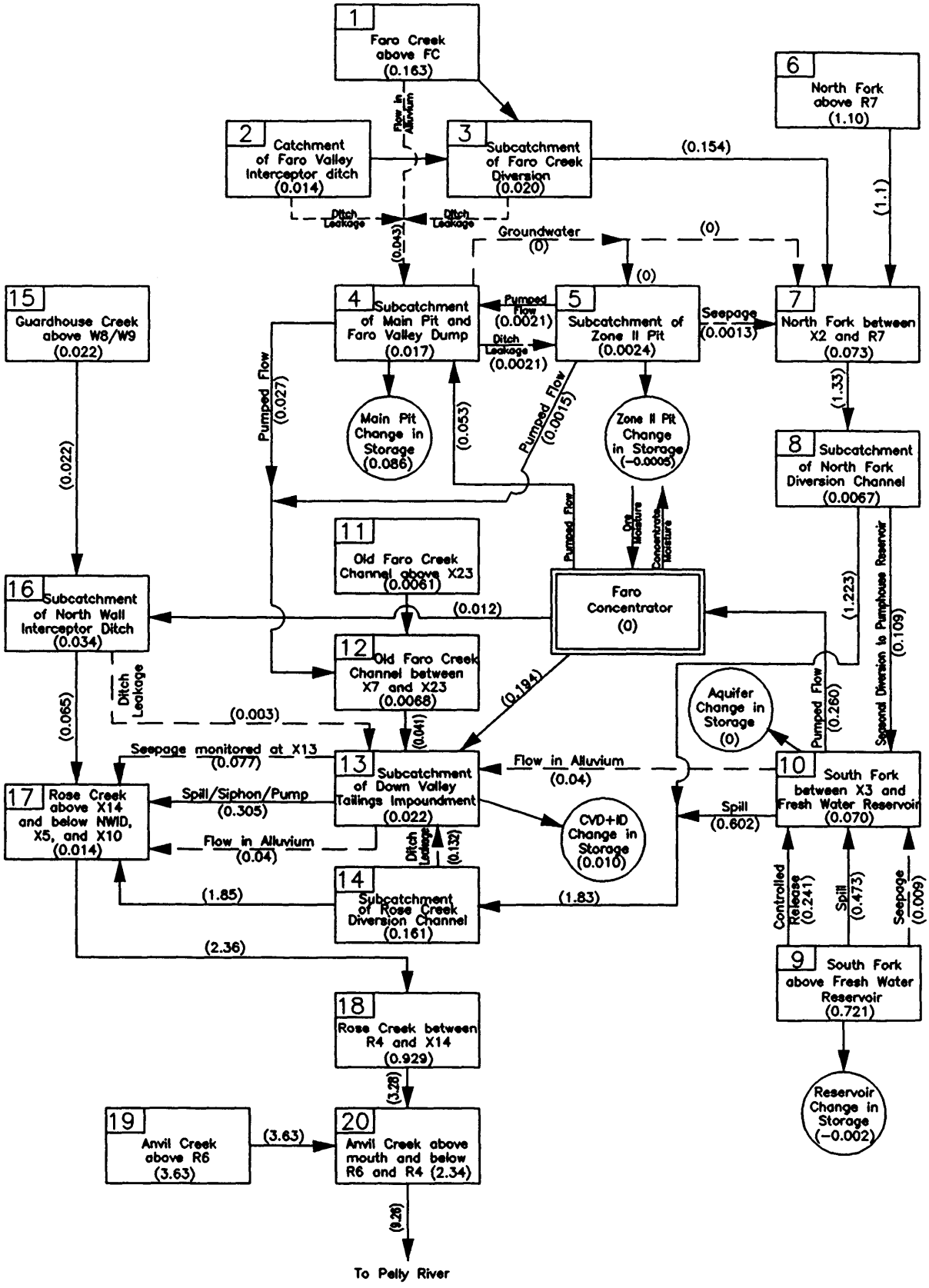
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Anvil Complex Closure Plan

Minesite Subcatchments in the Vicinity of Vangorda Plateau Development

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Nov. 1996		6-3



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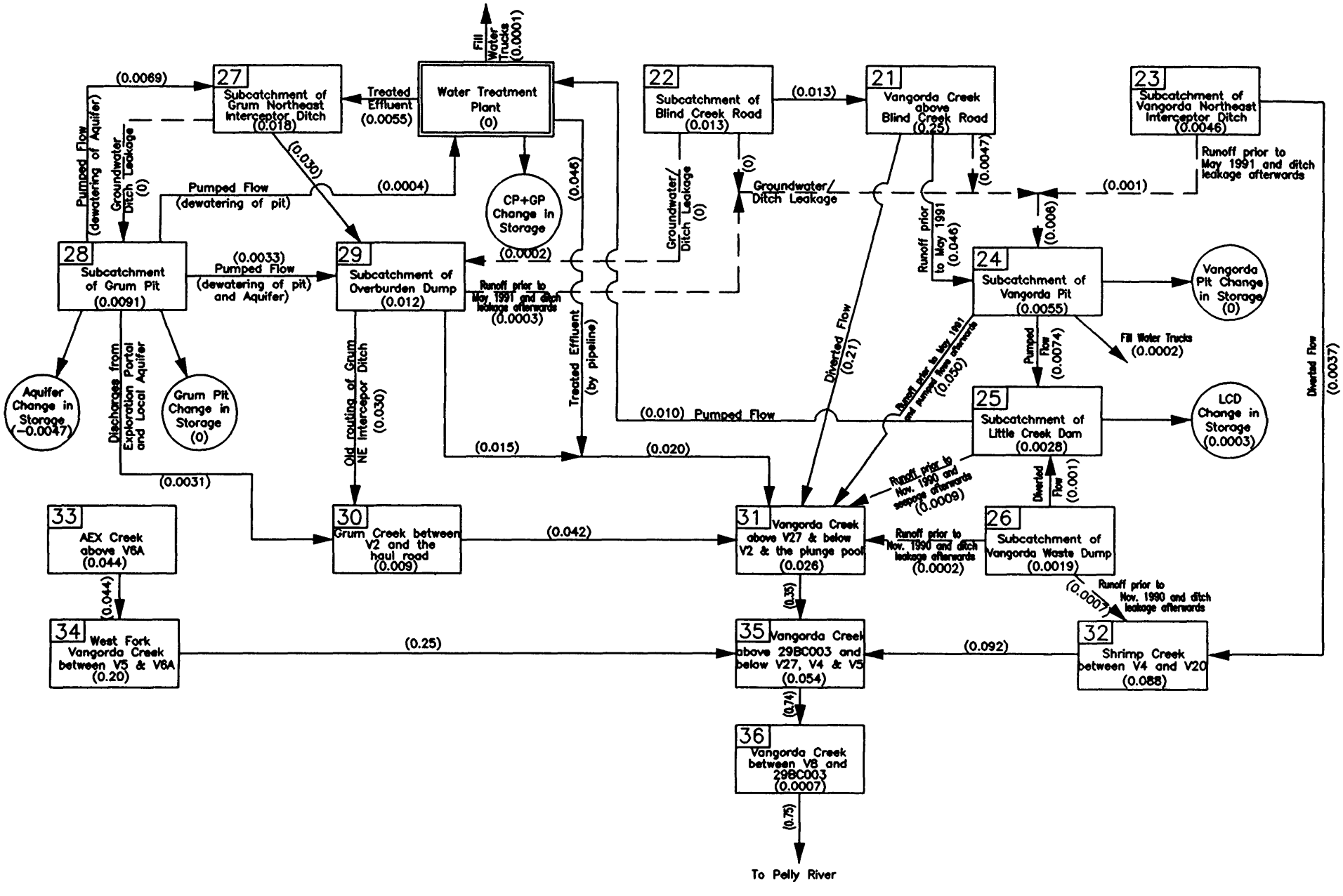
Anvil Range Mining Corporation

Anvil Complex Closure Plan

Anvil Creek Watershed Average Annual Water Balance for Period 1990 to 1995

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Oct. 1996		6-4

.\FARO\completed\6-4Water-Balance-updated Fri Nov 29 01:40:39 1996 Drawn: L. du Toit



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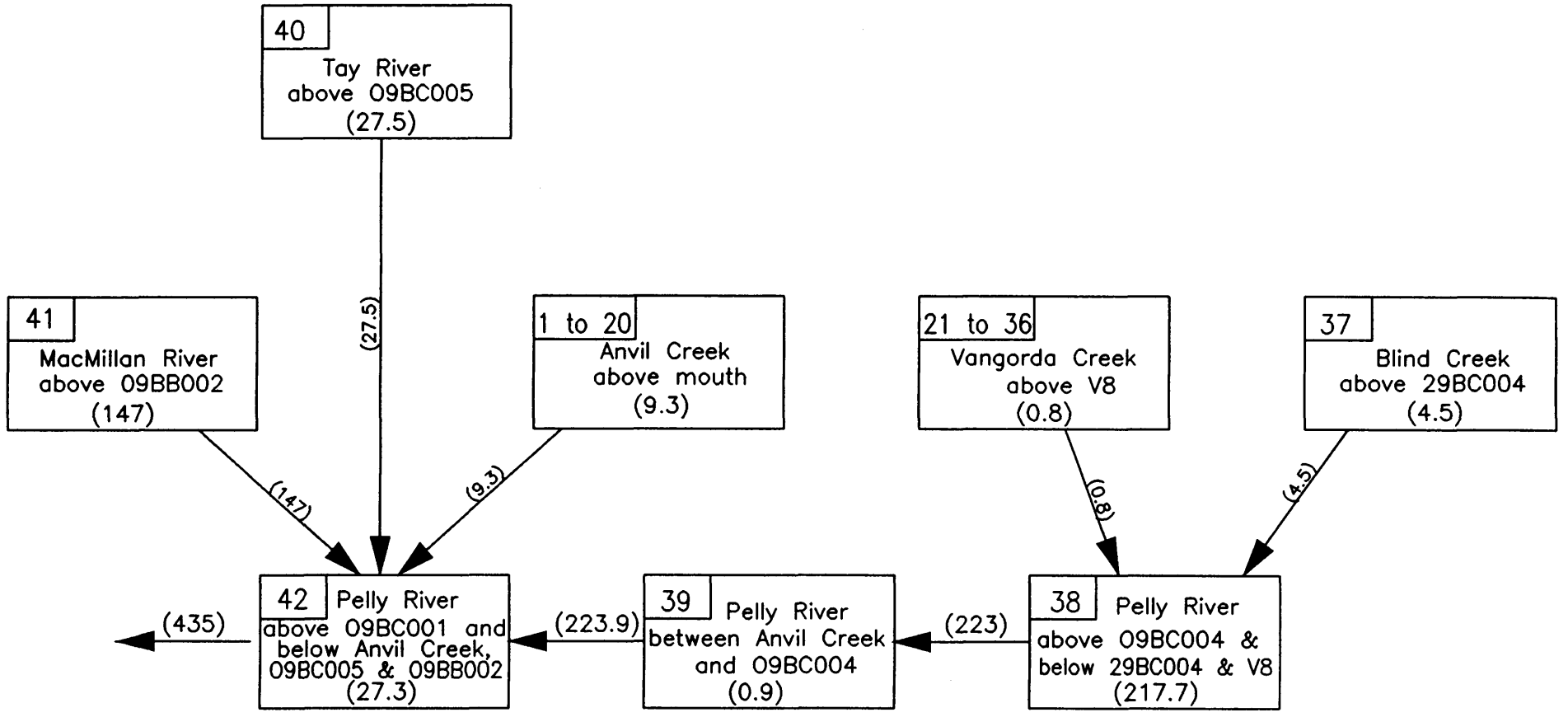
Anvil Range Mining Corporation

Anvil Complex Closure Plan

Vangorda Creek Watershed Average Annual Water Balance for Period 1990 to 1995

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Oct 1996		6-5

\\PAHU\completed\b-bwater-balance-Vangorda Fri Nov 29 02:31:09 1996 L. du Toit



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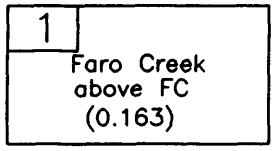
Anvil Range Mining Corporation

Anvil Complex Closure Plan

Pelly River Watershed Average Annual Water Balance for Period 1990 to 1995

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Oct. 1996		6-6

\FARO\completed\b-bwater-balance-PellyRiver Fri Nov 29 02:36:56 1996 L. du Toit



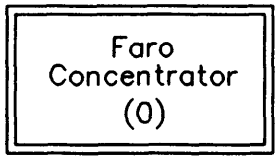
Subcatchment Area

- box in upper, left-hand corner contains the identification number for the subcatchment (see Figures and Table).
- bracketed number denotes average annual runoff generated by subcatchment in m^3/s for period 1990 to 1995.
- no distinction is made as to how the runoff flows through the subcatchment, be it overland flow, interflow or groundwater.



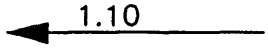
Storage Element

- bracketed number denotes average rate in m^3/s at which the storage element was filled during the period 1990 to 1995.
- a negative number indicates a net amount of water was removed from the storage element between January 1990 and December 1995.



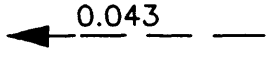
Water Treatment Facility or Faro Concentrator

- bracketed number denotes net gain of water within facility (assumed negligible in all cases).



Generic Flow Line

- a solid line is used to represent surface flows, pumped flows, or combined surface and groundwater flows.
- number denotes average flow rate in m^3/s for period 1990 to 1995.



Groundwater Flow Line

- a dashed line is used to represent flows which pass exclusively through the ground (e.g., ditch leakage and regional groundwater flux)
- number denotes average flow rate in m^3/s for period 1990 to 1995.

NWID

Abbreviations for Dam and Ditch Names

- NWID = North Wall Interceptor Ditch
- CVD = Cross Valley Dam
- ID = Intermediate Dam
- CP = Clarification Pond (for the Water Treatment Pond)
- GP = Groucho Pond
- LCD = Little Creek dam

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Anvil Complex Closure Plan

Legend for 1990-1995 Historical Water Balances

PROJECT NO.	DATE	APPROVED	FIGURE
033001	Oct. 1996		6-7

Figure 6-8
Comparison of Simulated and Observed Storage Levels
Within the Main Pit for the Period 1990 to 1995

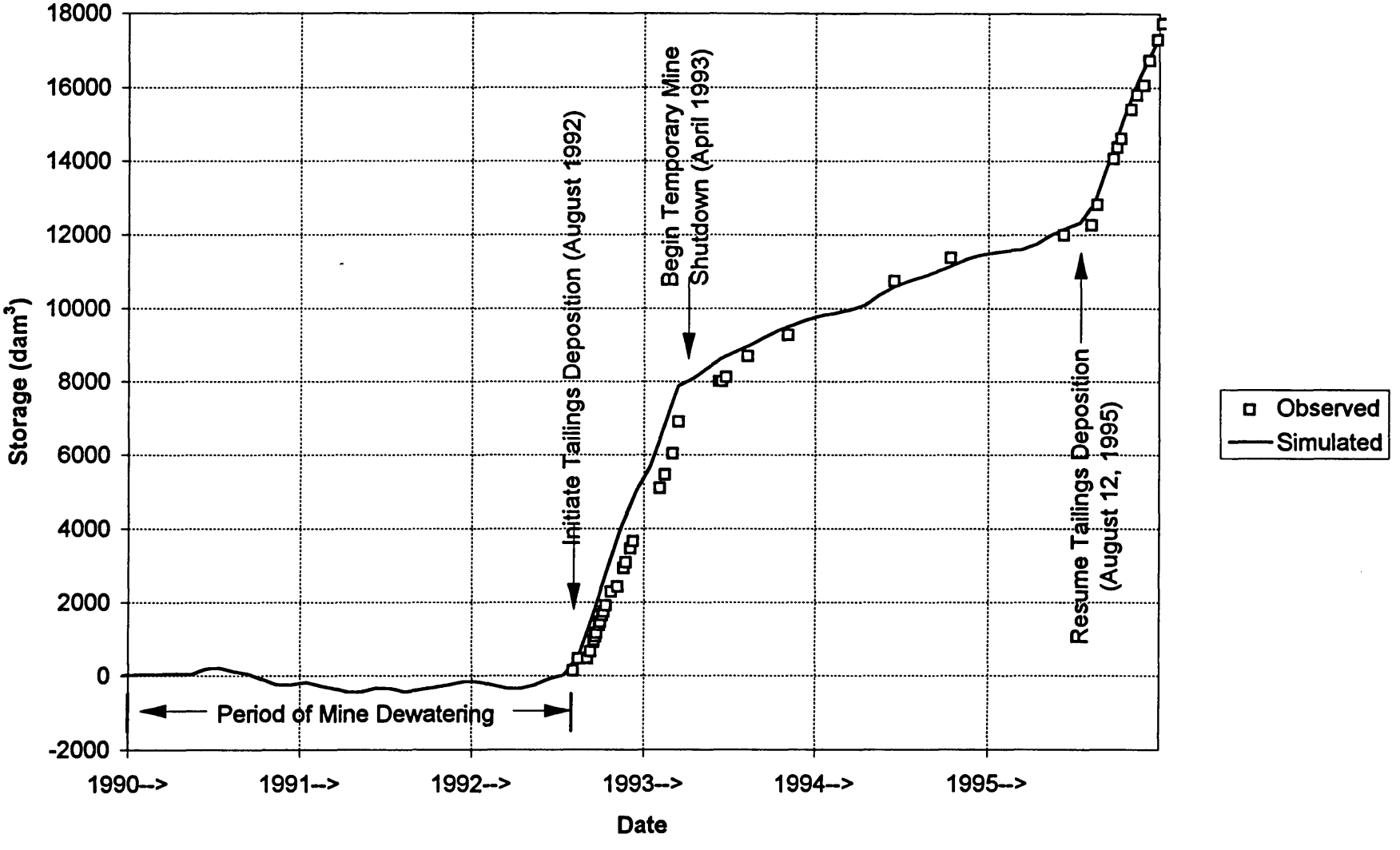


Figure 6-9
Comparison of Simulated and Observed Water Levels
for Zone II Pit for Period 1989 to 1995

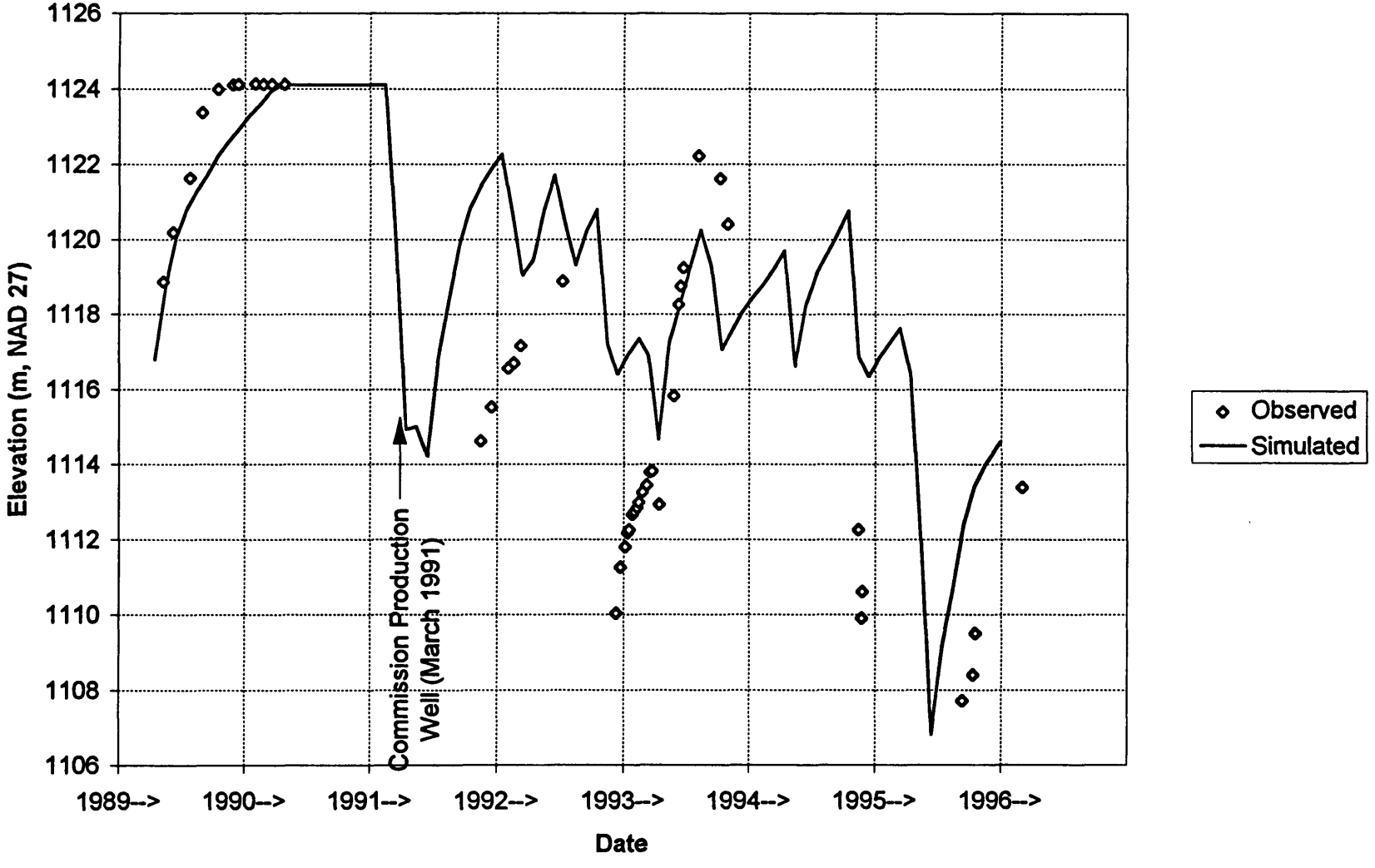


Figure 6-10
Comparison of Observed and Simulated Flows
for Station X14 for Period 1990 to 1995

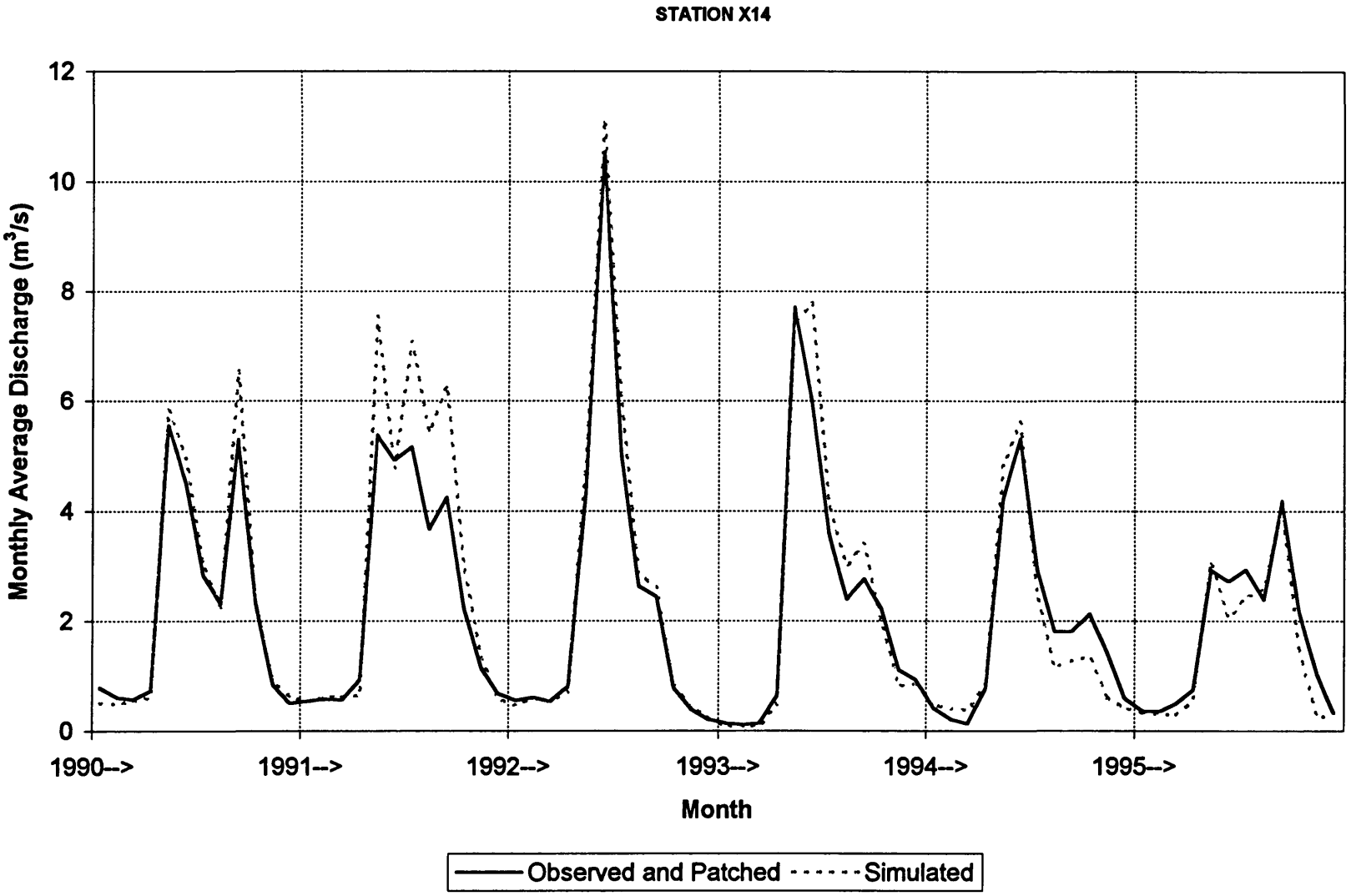


Figure 6-11
Comparison of Observed and Simulated Flows
for Station R7 for Period 1990 to 1995

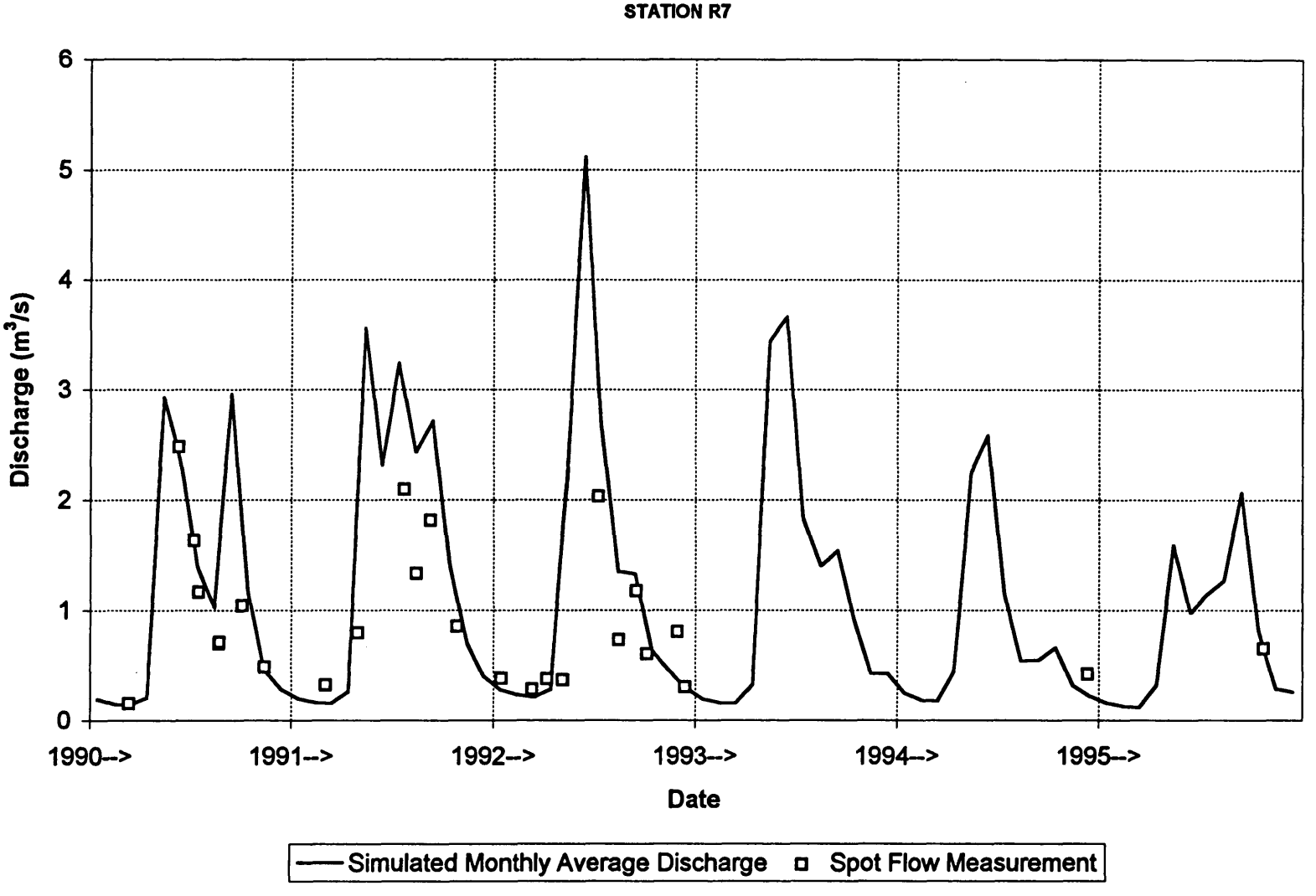


Figure 6-12
Comparison of Observed and Simulated Flows
for Station FC for Period 1990 to 1995

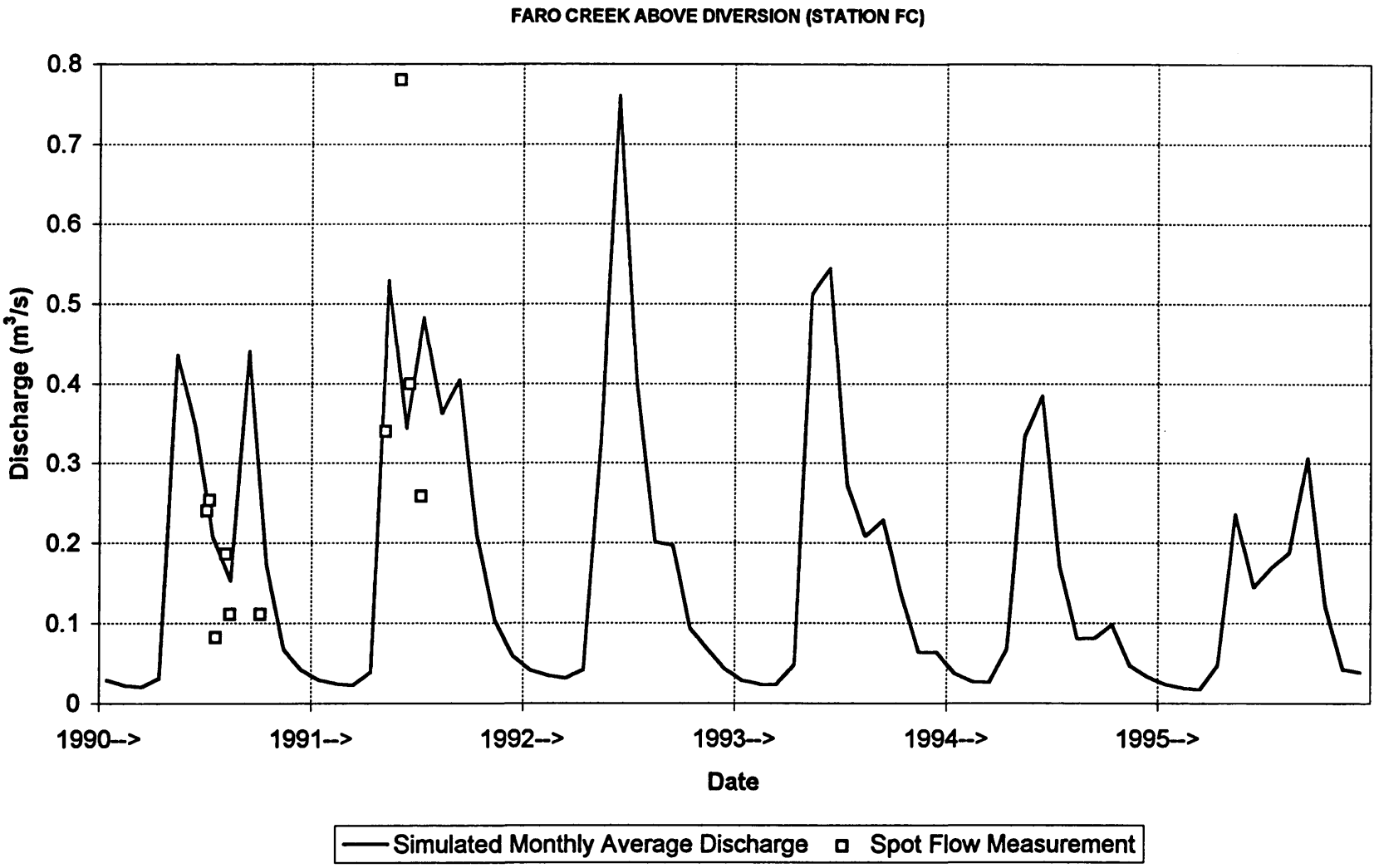


Figure 6-13
Comparison of Observed and Simulated Flows
for Station X10 for Period 1990 to 1995

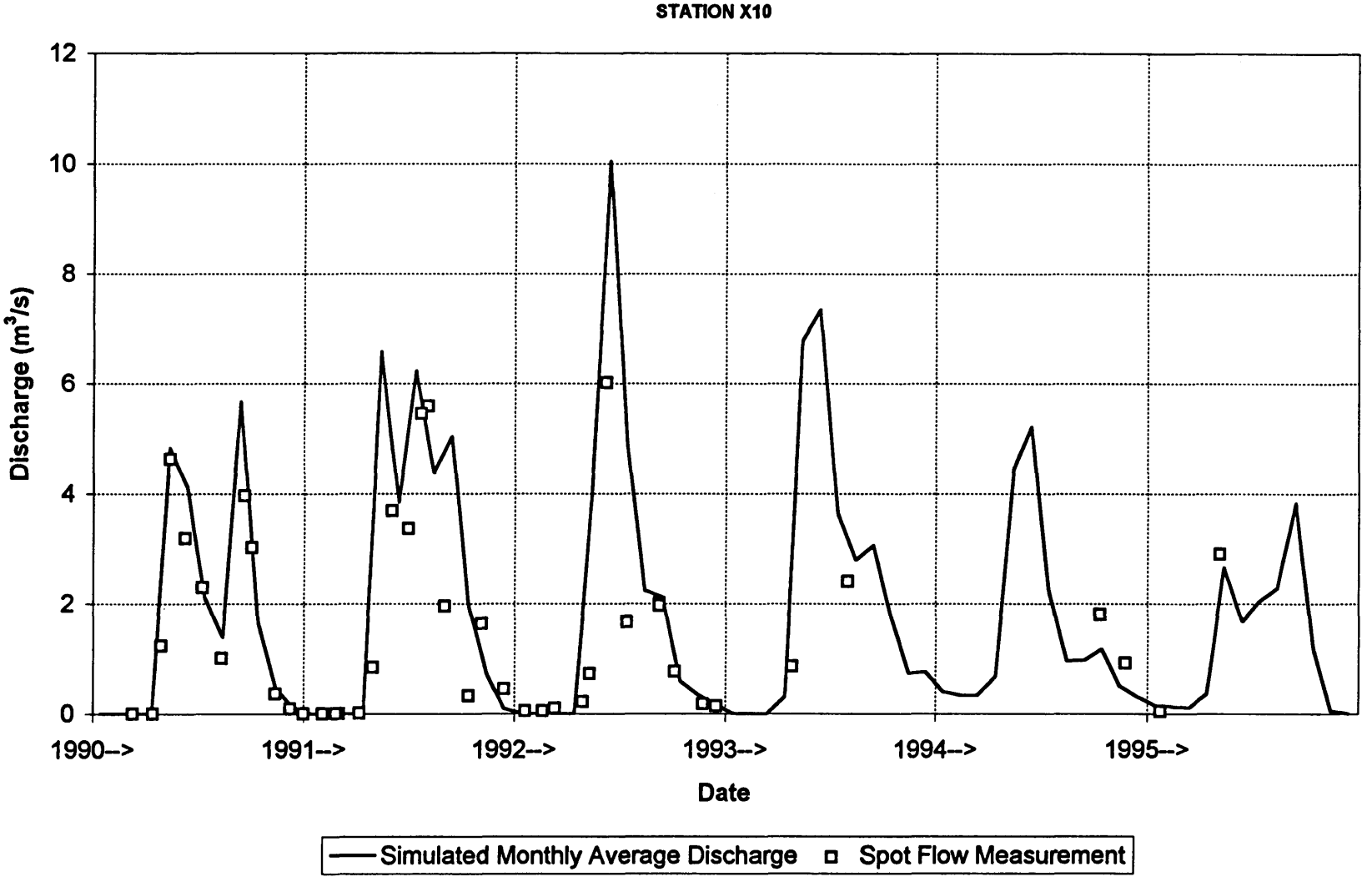


Figure 6-14
Comparison of Simulated and Observed Water Levels
for Little Creek Dam for Period 1990 to 1995

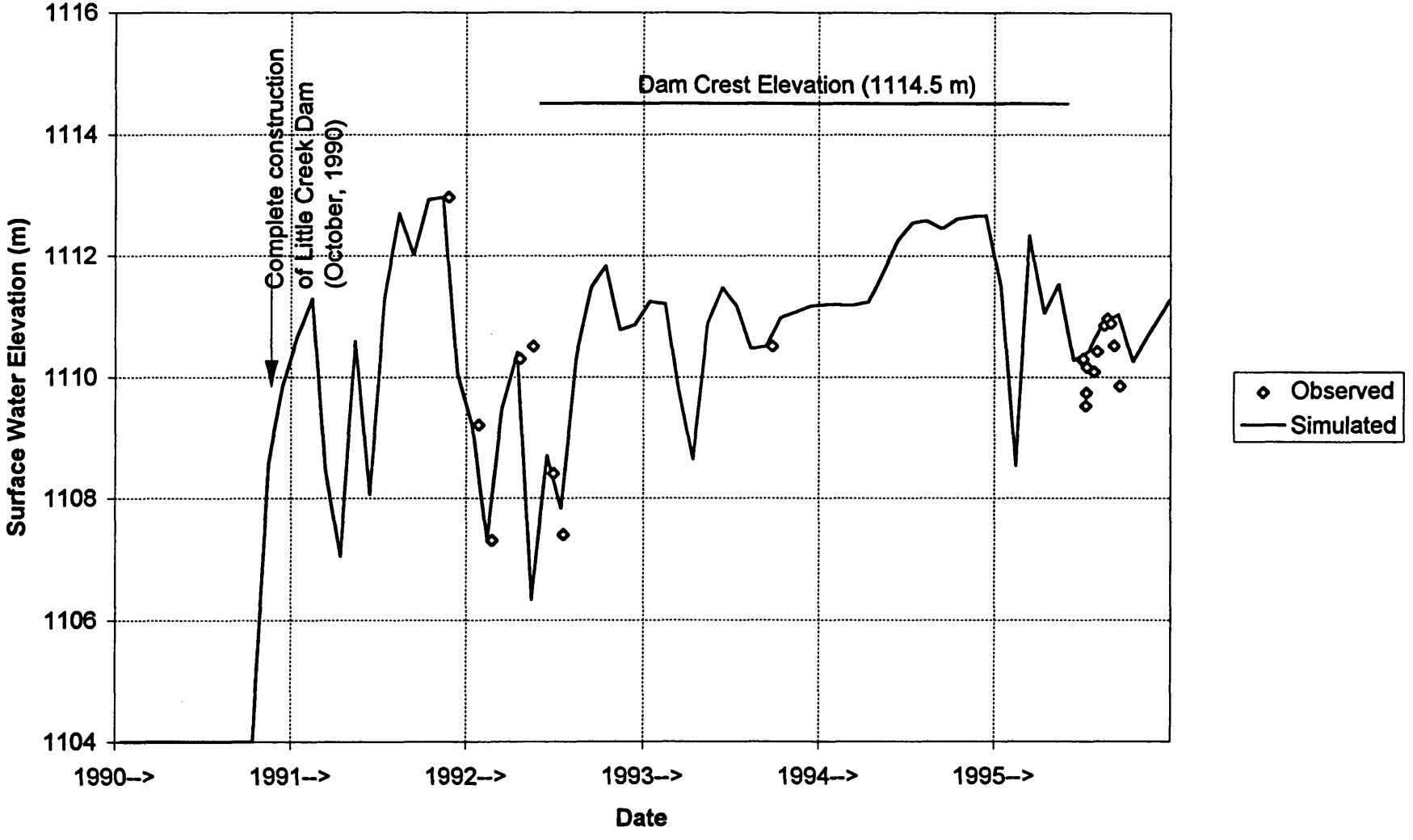


Figure 6-15
Comparison of Simulated and Observed Storage Levels
Within the Vangorda Pit for the Period 1990 to 1995

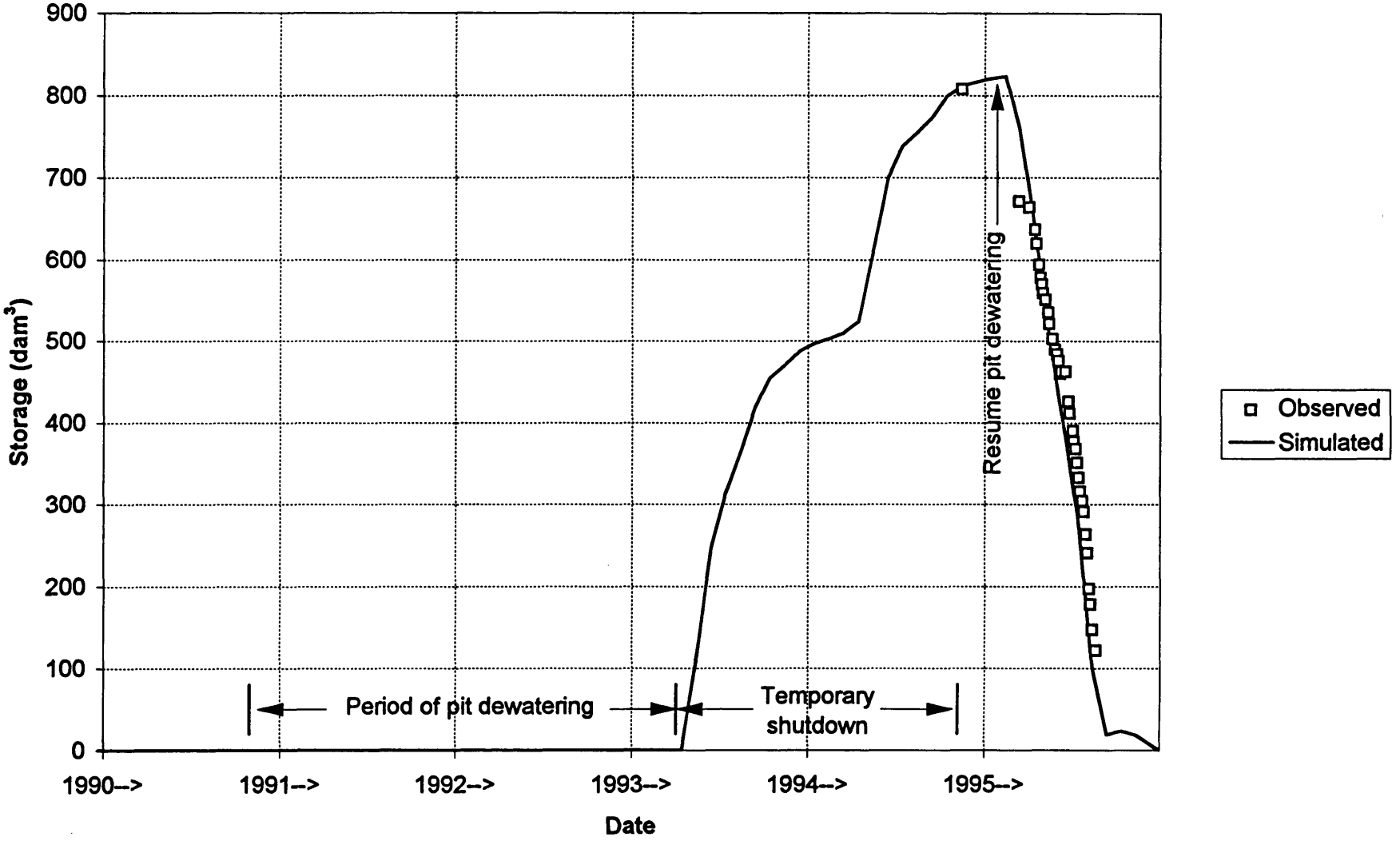


Figure 6-16
Comparison of Observed and Simulated Flows
for Station 29BC003 for Period 1990 to 1995

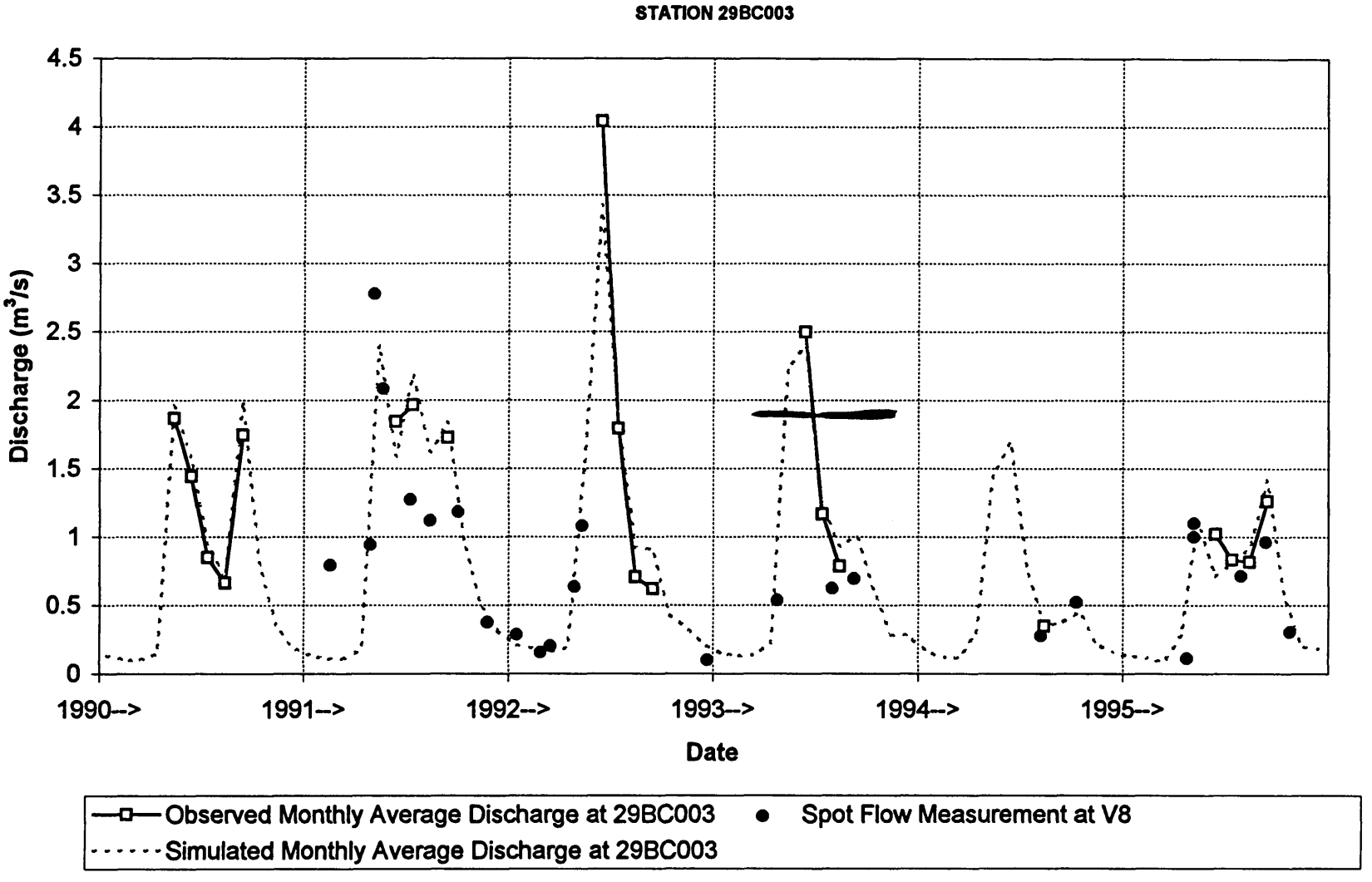


Figure 6-17
Comparison of Observed and Simulated Flows
for Station V1 for Period 1990 to 1995

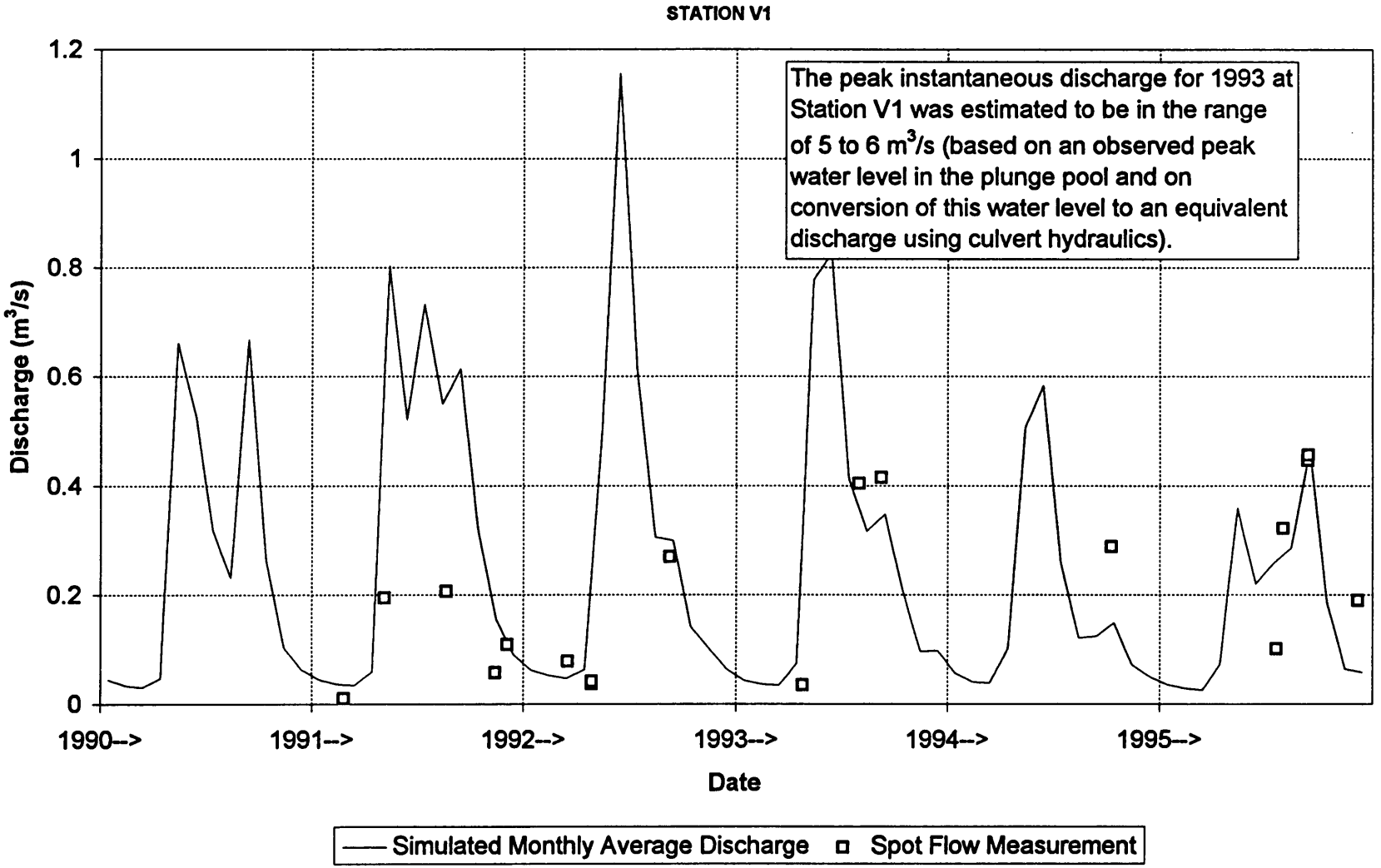


Figure 6-18
Comparison of Observed and Simulated Flows
for Station V18 for Period 1990 to 1995

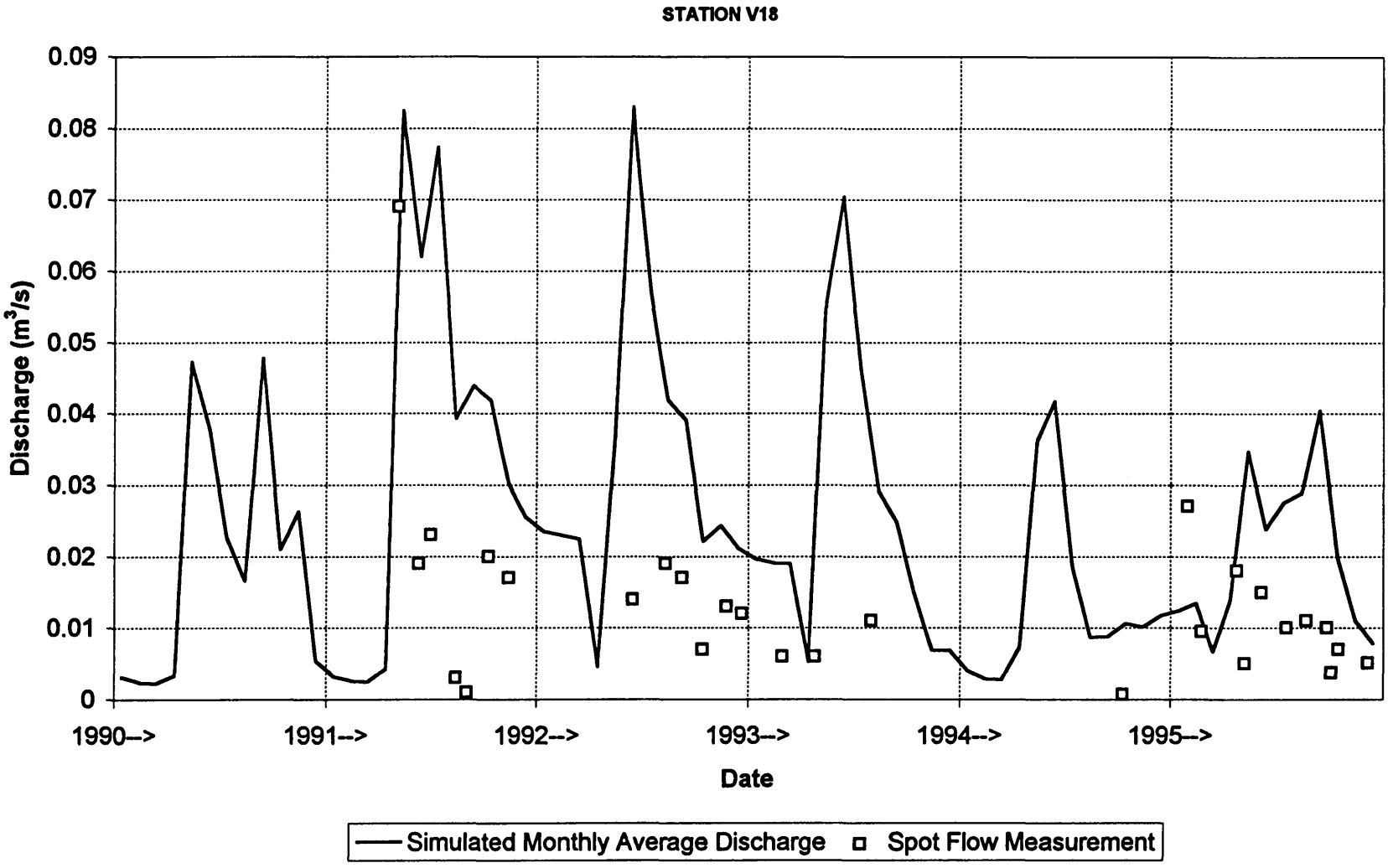


Figure 6-19
Comparison of Observed and Simulated Flows
for Station V6A for Period 1990 to 1995

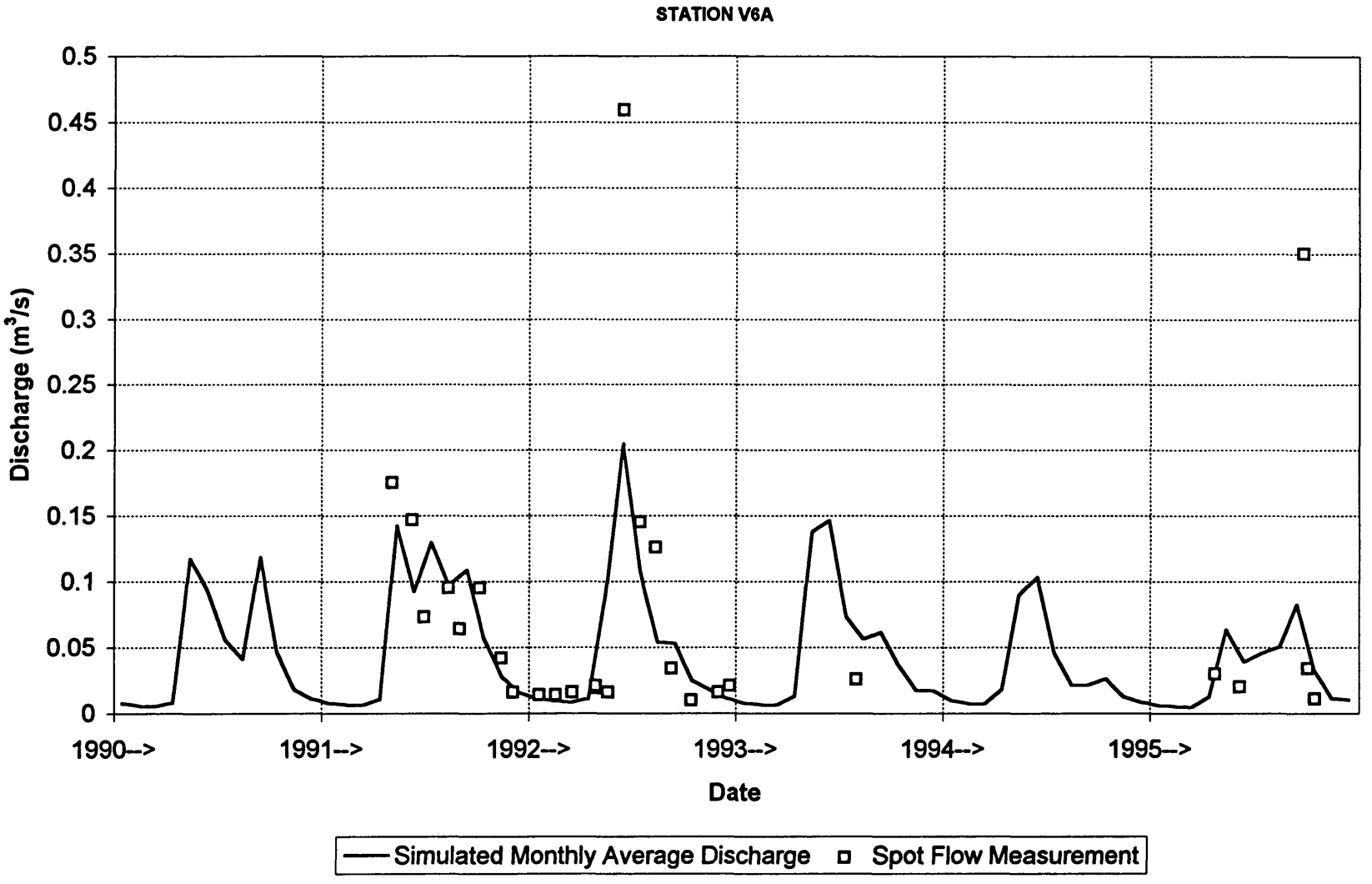


Figure 6-20
Envelope Curve of Maximum Observed Floods in Canada

