

**EVALUATION OF ARD FROM  
BRITANNIA MINE AND THE  
OPTIONS FOR LONG TERM  
REMEDICATION OF THE IMPACT  
ON HOWE SOUND**

**BCAMD 3.31**

This project was prepared for  
Ministry of Energy, Mines and Petroleum Resources B.C. Acid Mine Drainage Task Force  
By  
Stephen, Robertson and Kirsten (B.C.) Inc.

**November 1991**

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**Prepared for:**

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B.C. Acid Mine Drainage Task Force  
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**November 1991**

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**EVALUATION OF ACID MINE DRAINAGE (AMD) FROM BRITANNIA MINE  
AND THE OPTIONS FOR LONG-TERM REMEDIATION  
OF THE IMPACT ON HOWE SOUND**

**PREFACE**

Prepared by

Reclamation Section  
Resource Management Branch  
Ministry of Energy, Mines and Petroleum Resources

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The Britannia Mine is located at Britannia Beach, on the east shore of Howe Sound, approximately 48 km north of Vancouver. Operated from 1905 to 1963 by the Britannia Mining and Smelting Company Ltd., and from 1963 to 1974 by Anaconda Mining Company, in its prime the mine was one of the biggest copper producers in the British Empire. The mill/mine complex finally shut down in 1974, and nowadays is used as a museum. The area containing the mine workings and the mill is currently owned by Copper Beach Estates.

The major environmental concern at Britannia Mine is with acid mine drainage, and associated high copper and zinc levels emanating from old adits and waste dumps<sup>1</sup>. While dissolved copper concentrations are an order of magnitude less than they were during mine operation, they still have an impact on the invertebrate and fish in Howe Sound. This study was carried out to assist the present owner in dealing with this problem. Funding was granted by the Sustainable Environment Fund (SEF), with management provided by the British Columbia Acid Mine Task Force. The objectives of the study were to review all available information, critique the present system of control, and to recommend possible long-term solutions to reduce or eliminate acid mine drainage.

Mining at Britannia was carried out in seven ore bodies, all located east of Britannia Beach, in the ridge between Britannia and Furry Creeks. Over the seventy years of operation, 80 km of underground workings - the source of 95% of the ore<sup>2</sup> - and five open

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<sup>1</sup> **Waste Dump:** Barren or submarginal rock or ore which has been mined but is not of sufficient value to warrant treatment, and is therefore removed ahead of the milling process.

pits were created. Access to the underground workings was primarily provided through portals at 200, 1550, and 2100 ft above sea level. Surface pit excavation occurred 5 km inland, largely between 3000 and 4500 ft above sea level. In addition to portals and open pits, surface disturbance associated with mining included three small waste dumps, the road network, a series of dams on Britannia Creek, and subsidence associated with the underground workings.

Ore generated by mining was processed in milling facilities at Britannia Beach. During operation, approximately 45 million tonnes of ore were processed, predominantly copper, but lesser amounts of silver, zinc, and gold were also recovered. Tailings<sup>3</sup>, the main waste product from the mill, were either discharged into Howe Sound or used underground as backfill. Concerns regarding the fate of the tailings were addressed in a recent study of Howe Sound, paid for in part by the Ministry in conjunction with the British Columbia Acid Mine Drainage Task Force.

Acid generation results from the oxidation of sulphide minerals contained in rock exposed to air and water. Water draining through areas of sulphide oxidation can transport the reaction products. While this phenomenon is often referred to as acid mine drainage, it is not confined to lands disturbed by mining; for example, natural springs are often acidic in the vicinity of outcrops of sulphide-bearing rock.

The principal ingredients in the AMD process are reactive sulphide minerals, oxygen and water. The oxidation reactions are often accelerated by microbiological activity. Not all mining operations that expose sulphide-bearing rock result in acid mine drainage. For example, AMD will not occur if the sulphide minerals are non-reactive or if the rock is not leached. If alkaline material is available in the rock waste, pH changes may be buffered by neutralizing reactions as the drainage passes through the waste.

The main impact of acid mine drainage is caused by the dissolution of heavy metals. Typically, mined rock contains abnormally high concentrations of a number of heavy metals that will be released by weathering reactions occurring concurrently with sulphur oxidation. In the usual pH range of soils and water, heavy metals released by weathering normally precipitate in immobile compounds. However, under highly acidic conditions, the heavy metals dissolve and are entrained in runoff and seepage, and can cause a detrimental impact on water quality in the receiving environment.

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<sup>2</sup> **Ore:** A mineral, or mineral aggregate, containing precious or useful metals or metalloids, and which occurs in such quantity, grade, and chemical combination as to make extraction commercially profitable.

<sup>3</sup> **Tailings:** The portion of the ore which after grinding and washing is regarded as too poor to be treated further; used especially of the debris from stamp mills or other ore-dressing machinery, as distinguished from material (concentrates) that is to be smelted.



The geological strata of the ore bodies mined at Britannia consist of massive sulphides, widely disseminated or concentrated in stringers and along bedding planes. Prior to mining, sulphide oxidation will have been limited to a thin surface layer, a few small outcrops, and areas of landslide debris. Exposure, cracking and comminution of the sulphide bedrock in the underground workings, pits, and rock dumps have significantly increased the amount of sulphide oxidation, which in turn increases the rates of acid generation and metal dissolution. The open pits and gloryholes collect drainage, funnelling rainwater through the extensive underground workings, flushing out the oxidation products.

Under the precepts of sustainable development, mining is regarded as a temporary land use. This objective is enforced by current mine reclamation legislation requiring that land and water resources be reclaimed to a level of productivity not less than that which existed previously, and that water released from a minesite meet long-term water quality standards.

Acid mine drainage is a major impediment to successful reclamation for mining operations throughout the world. In British Columbia, AMD presently occurs at six operating mines. Four other metal mines and two coal mines are considered potentially acid generating, as are several proposed mines. Under the British Columbia Mine Development Assessment Process, all mines with a potential to generate acid mine drainage must develop plans to prevent it.

To enable the British Columbia mining industry to comply with the present comprehensive reclamation requirements, the Ministry of Energy, Mines and Petroleum Resources is participating in a program of research aimed at developing cost-effective solutions to major environmental problems. A major part of this program focuses on acid mine drainage. As part of this, the Ministry coordinates and provides funding for the British Columbia Acid Mine Drainage Task Force, and strongly participates in activities of the national acid mine drainage program (Mine Environmental Neutral Drainage Program - MEND). In 1992, the proposed research program for the British Columbia Acid Mine Drainage Task Force is \$1.5 million, with funding shared by industry and the Provincial and Federal Governments.

The problems of acid mine drainage will not easily be solved. However, work carried out during the last five years has done much to enhance our knowledge and ability to predict, prevent, control, and monitor AMD. The cooperative efforts between our Ministry and the British Columbia mining industry are widely regarded with envy by the rest of Canada, and we are recognized by many as world leaders in acid mine drainage technology. As a result, the British Columbia consulting industry is currently selling its services abroad, an important spin-off of the ongoing research activity.

At present, six of the mine sites abandoned prior to the enactment of reclamation legislation are acid generating. Fortunately, many of these older sites are small

underground operations and the impact is manageable. The administration and control of acid mine drainage from historic mine sites is shared between Environment Canada and two British Columbia Ministries: Environment, Lands and Parks and Energy, Mines and Petroleum Resources.

Our Ministry has assumed responsibility for reclamation of several closed mine sites, including the Mount Washington Mine on Vancouver Island. Work carried out at Mount Washington since 1988 has included the testing of various cover types and spoil configurations, and developing monitoring technology. Results to date have included the development of a cost effective asphalt emulsion/geotextile cover designed to seal the surface, thereby reducing the removal of oxidation products. The program of AMD research at Britannia Beach is being done in concert with that at Mount Washington.

While the Britannia Mine closed prior to the creation of AMD guidelines, it still had to obtain a waste disposal permit under the Pollution Control Act. This permit ordered that:

- all mine drainage water be directed to the 200 ft portal;
- mine drainage collected at the 200 ft portal be analyzed monthly for copper and three times per year for other elements;
- when dissolved copper levels in mine drainage exceed 15 mg/l, water be treated in copper recovery (cementation) plant; and
- mine drainage collected at the 200 ft portal be discharged to Howe Sound via submerged outfall.

Recent investigations suggest that contaminated mine water from the 2100 ft level is bypassing the treatment area, and draining through Jane Creek and then Britannia Creek, directly into Howe Sound. To prevent this, the property owner is required to develop an abatement plan, including a cost estimates and a schedule of implementation. This study, carried out by Steffen, Robertson and Kirsten (B.C.) Inc. (SRK) in association with Gormley Process Engineering (GPE), was undertaken to ensure there are no other sources of Britannia AMD, and to suggest alternative mitigative measures to prevent any impact. Its detailed objectives were as follows:

1. To conduct a literature review of all information already available on mining, acid generation and drainage contamination.
2. To identify the major sources of contaminated loading.
3. To critically review the present system of diverting mine water, and to report on the effectiveness of the present treatment facility.
4. To review the present Ministry of Environment, lands and Parks permit conditions.

5. To research and report on the current technologies and estimated costs for the removal of copper from water.
6. To recommend mitigation measures to reduce AMD and remove copper loading and present a timetable with estimated costs.

The management strategy to control acid generation and the works to treat mine drainage should be based on these results. A museum already exists on the site, thereby providing an opportunity for any works constructed to serve an educational function, in addition to the research and treatment purposes.

## REPORT 166004

# EVALUATION OF ARD FROM BRITANNIA MINE AND THE OPTIONS FOR LONG TERM REMEDIATION OF THE IMPACT ON HOWE SOUND

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**REPORT 166004****EVALUATION OF ARD FROM BRITANNIA MINE  
AND THE OPTIONS FOR LONG TERM REMEDIATION  
OF THE IMPACT ON HOWE SOUND****EXECUTIVE SUMMARY**

The decommissioned Britannia Mine is located at Britannia Beach approximately 48 km north of the city of Vancouver, on the east shore of Howe Sound. The underground and open pit mine was operated by the Britannia Mining and Smelting Company Ltd. from 1905 to 1963 at which time it was purchased and operated by Anaconda Mining Company until shutdown in 1974. The area is currently owned by Copper Beach Estates. During operation, approximately 45 million tonnes of ore were processed for recovery of copper and lesser amounts of silver, zinc and gold. Tailings were deposited directly into Howe Sound through two intertidal outfalls located near the mouth of Britannia Creek.

Acid rock drainage containing elevated acidity and metal levels has issued from the Britannia Site since the operational period, discharging into Britannia Creek and Howe Sound. During mine operation, drainage waters from the 2200 and 4100 levels were directed through two copper precipitation plants containing scrap iron, before discharging into Britannia Creek. In 1972, in an attempt to improve drainage quality from the site, acidic mine water was diverted within the mine workings to the 4100' level for treatment in a copper cementation plant prior to discharge. Since 1978, mine water has been collected, treated and discharged at depth to Howe Sound. Subsequent investigations however have indicated that contaminated water is again draining from the 2200' level directly into Jane Creek and then into Britannia Creek and ultimately into Howe Sound.

Steffen, Robertson and Kirsten (B.C.) Inc. (SRK) in association with Gormely Process Engineering (GPE) were retained by the Ministry of Energy, Mines and Petroleum Resources to conduct the initial phase of this study. This phase comprised a review of the available information, assessment of acid generation and contaminant drainage from the site based on this data, identification of the major sources of contaminant loading, and a critical review of the present system of control, and to identify recommendations for possible long term solutions to reduce or eliminate acid drainage. This study is a preliminary review to identify the major contributing factors to contamination from acidic drainage to Britannia Creek and identify options for remediation. Further detailed investigation have been recommended where the available information is insufficient to define or evaluate an option with confidence.

## REPORT 166004

# EVALUATION OF ARD FROM BRITANNIA MINE AND THE OPTIONS FOR LONG TERM REMEDIATION OF THE IMPACT ON HOWE SOUND

## 1.0 INTRODUCTION

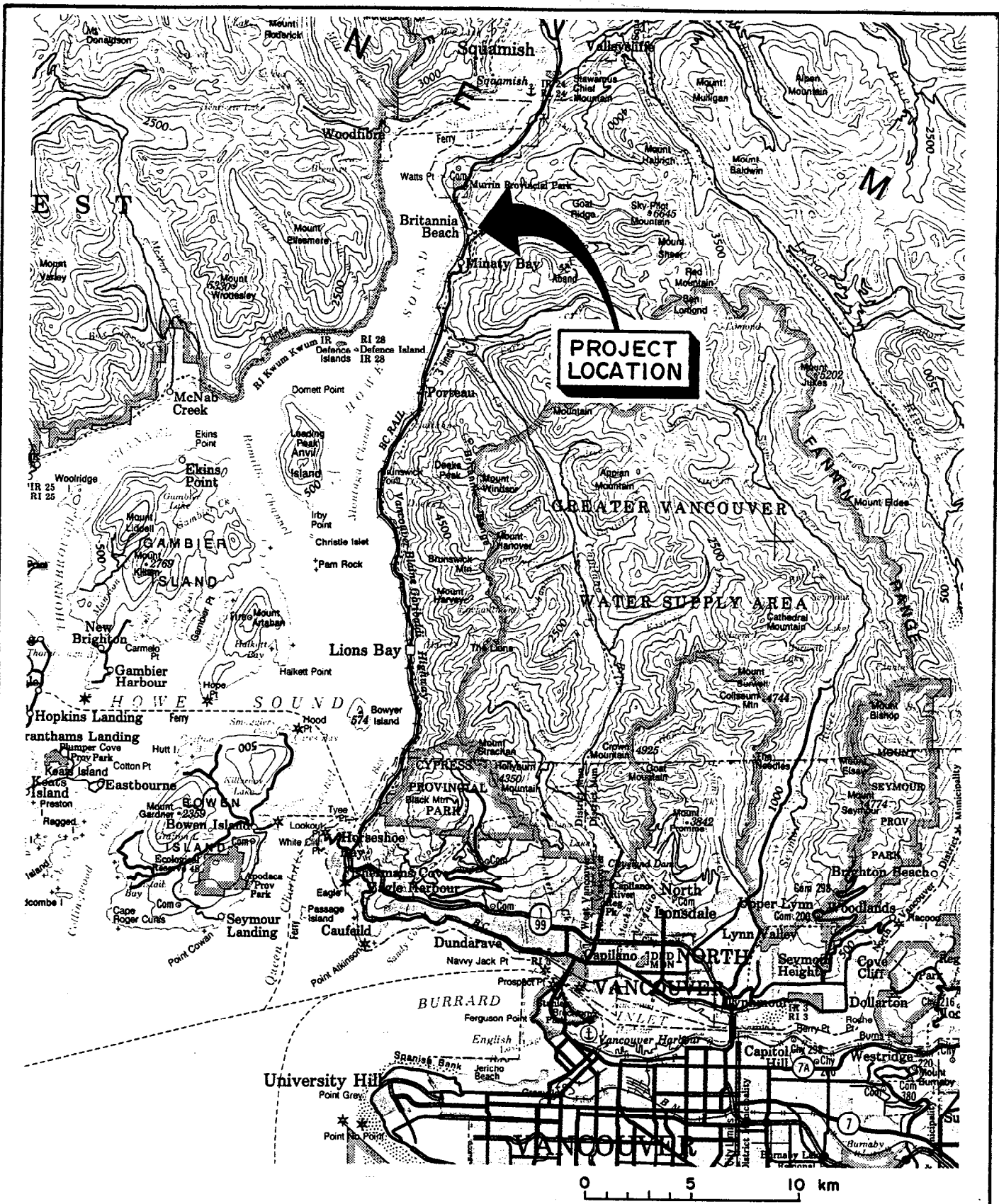
### 1.1 Background

The decommissioned Britannia Mine is located at Britannia Beach approximately 48 km north of the city of Vancouver, on the east shore of Howe Sound, as shown in Figure 1.1. The underground and open pit mine was operated by the Britannia Mining and Smelting Company Ltd. from 1905 to 1963 at which time it was purchased and operated by Anaconda Mining Company until shutdown in 1974. The area is currently owned by Copper Beach Estates. A general site location plan is presented in Figure 1.2.

During operation, approximately 45 million tonnes of ore were processed for recovery of copper and lesser amounts of silver, zinc and gold. Tailings were deposited directly into Howe Sound through two intertidal outfalls located near the mouth of Britannia Creek.

Mining at Britannia Mountain has disturbed the surface and the precipitation falling on this area finds its way into the mine rather than draining into the valley bottoms. In passing through the mine, it contacts sulphide surfaces where bacterial and chemical action have solubilized metals and generated acid. Acidic drainage containing elevated dissolved metal values, particularly copper, has been issuing from the mine site since 1905 and discharging into Howe Sound.

During mine operation, drainage waters from the 2200 and 4100 levels were directed through two copper precipitation plants containing scrap iron, before discharging into Britannia Creek. In 1972, in an attempt to improve drainage quality from the site, acidic mine water was diverted within the mine workings to the 4100 level for treatment in a copper cementation plant prior to discharge. Since 1978, mine water has been collected, treated and discharged at depth to Howe Sound. Subsequent investigations however have indicated that contaminated water is again draining from the 2200' level directly into Jane Creek and then into Britannia Creek and ultimately into Howe Sound.



MINISTRY OF ENERGY MINES AND PETROLEUM RESOURCES

DATE JAN. 1991

LOCATION PLAN

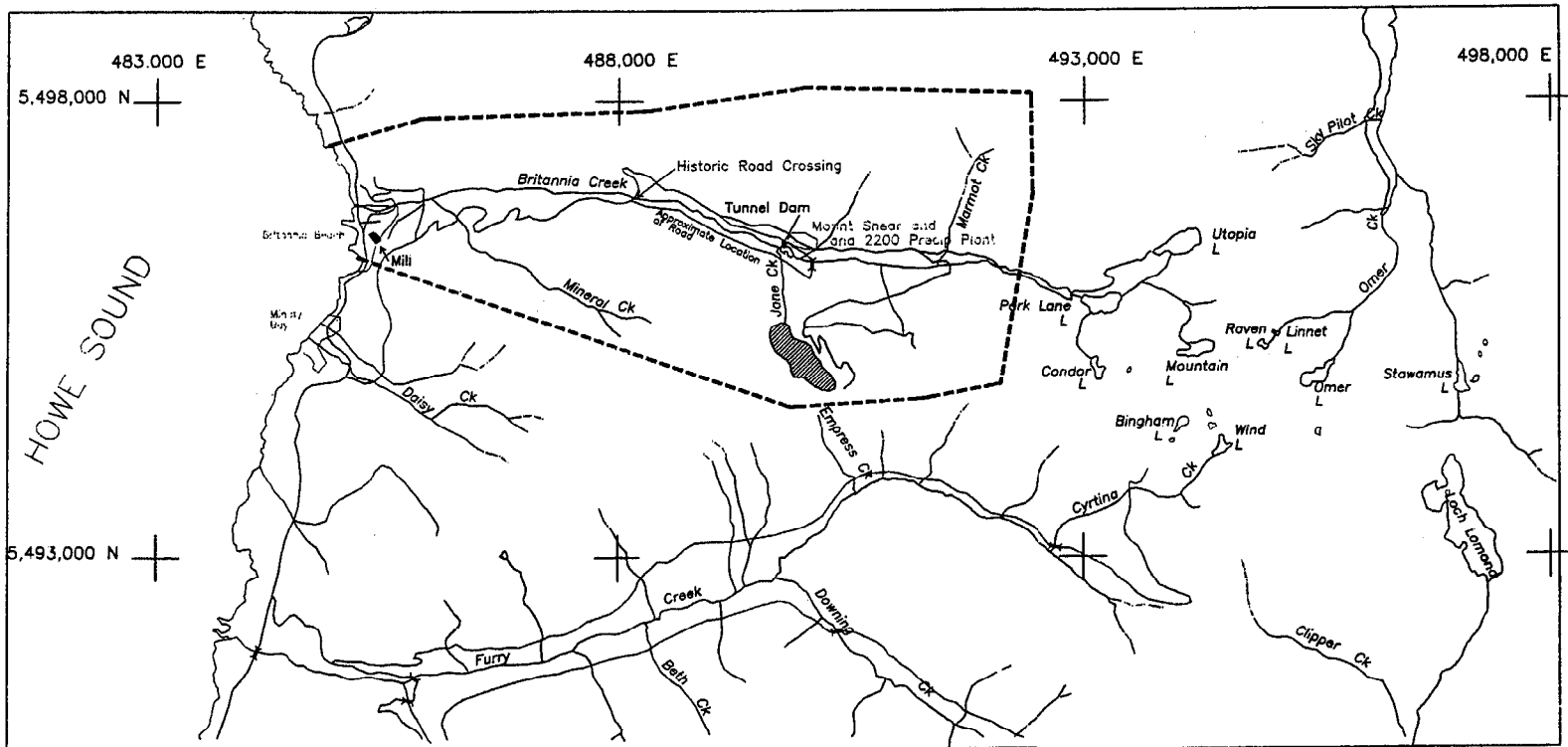
PROJ. NO. 66004

APPROVED

NO.

1.1

STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers



- LEGEND**
- Stream, definite
  - - - Stream, indefinite
  - Gravel Roads
  - Paved Roads
  - - - Study Area
  - ▨ Open Pit and Main Waste Dump Area
  - Townsite

Note: Base map derived from 1:50,000 scale  
 NTS mapsheet 92 G/11  
 North American Datum 1927  
 Transverse Mercator Projection



Ministry of Energy, Mines and Petroleum Resources		DATE March, 1991
<b>BRITANNIA MINE SITE LOCATION PLAN</b>		FILE NO. 66004
		APPROVED
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers		REV. 1.2

Although copper concentrations in the drainage are about 10 percent of the discharge concentrations during operation, this drainage still poses a serious threat to invertebrate and fish populations in Howe Sound. Investigations by Environment Canada, Environmental Protection Service and the provincial Waste Management Branch have shown that the drainage from the mine workings and the water of Britannia Creek are toxic to aquatic life. The adverse impact on the marine environment has been noted up to 18 km from the mine site, with elevated metal levels, particularly copper and zinc, detected in molluscan bivalves, such as mussels and oysters (Errington and Ferguson, 1987), in Howe Sound.

The Ministry of Environment, Waste Management Branch, provided the following discussion submitted by Brent More, Biologist, (B.C.M.O.E., Environmental Protection Group), of the environmental impacts on Howe Sound from the drainage at the Britannia Site.

On February 19, 1991, the undersigned [Brent Moore] was asked by the Industrial Section to provide a brief summary of identified environmental impacts at the Britannia Mines AMD site.

From 1983 to 1986, the Environmental Section studied the impact of the site's acid mine drainage on receiving water organisms. It was found that the discharge flowing out of the 2200 Portal into Britannia Creek - by way of Jane Creek- was of greatest concern. This concern was largely due to the layering of the creek's contaminated fresh water discharge over the denser marine water in Howe Sound. Therefore, the effluent was not as readily dispersed throughout the water column, and was allowed to remain in the more biologically productive surface waters in the Sound. Of lesser, but still significant, importance is the contamination of the Jane Creek headwaters by leachate from waste rock piled in the Jane Basin area.

Even if the creek contamination concerns noted above are effectively addressed, it appears that the submerged 4100 level outfall to Howe Sound could still significantly impact on Howe Sound. A survey conducted by the Environmental Section in 1990 has shown that the near copper complexing capacity of the marine receiving waters is exceeded even more significantly near the depth of the diffuser than at surface, near the creek plume.

To eliminate significant contamination of the receiving environment, both the 2200 Portal and waste rock discharges to the Britannia Creek system would have to be diverted to the 4100 discharge system, and the collected effluent would have to be effectively treated to levels noted in the relevant Pollution Control Objectives. A significant data base has now been gathered to support these conclusions. [End of Summary].

The present Ministry of Environment (MOE) pollution abatement order limits dissolved copper levels at Britannia to 15 mg/L. A system is in place to direct drainage from the site through a deep outfall into

Howe Sound. When the copper level is less than 15 mg/L, the seepage is routed directly to Howe Sound through a deep water outfall. When the copper level is above 15 mg/L, seepage is routed through a scrap iron cementation process. The cementation process does not remove any appreciable amount of acidity and cannot reduce copper concentrations to levels achieved with conventional treatment systems based on neutralization and precipitation of copper hydroxide. The discharge from the cementation plant is directed to the deep outfall.

## 1.2 Terms of Reference

The site was identified as an area of study by the B.C. Acid Mine Drainage Task Force. The overall objective of this study is to evaluate the current extent of acid mine drainage from the Britannia Mine site and develop a long term remediation plan to mitigate the impact of drainage from the mine site on Howe Sound.

Steffen, Robertson and Kirsten (B.C.) Inc. (SRK) in association with Gormely Process Engineering (GPE) were retained by the Ministry of Energy, Mines and Petroleum Resources, Dr. John Errington, P. Eng. to conduct the initial phase of this study. This phase comprised a review of the available information, assessment of acid generation and contaminant drainage from the site based on this data, identification of the major sources of contaminant loading, a critical review of the present system of control, and identification of possible long term solutions to reduce or eliminate acid drainage. This study is a preliminary review to identify the major contributing factors to contamination from acidic drainage to Britannia Creek and Howe Sound and to identify options for remediation. It was found during the study however that the available information about the physical and geotechnical stability of the site, and the nature of the underground workings is limited. It was therefore decided jointly by SRK and the Review Committee representing the B.C. AMD Task Force, to focus efforts on defining further detailed investigations where the available information is insufficient to define or evaluate an option with confidence. The focus of the study was on the Britannia Creek drainage. The adjacent Furry Creek catchment has not been identified as an the acid drainage problem, and was beyond the scope of this investigation. Where available, information on Furry Creek has been provided for a more complete picture of the Britannia site.

## 1.3 Approach

With the above objectives in mind, the approach to this investigation has been to develop an overall understanding of the site from available information and limited site investigations, to identify the major sources of environmental impact and develop potential strategies for remediation, and define further investigations.

A number of key issues must be addressed to effectively evaluate the problem and potential remediation options:

- Sources of metal loads and drainage locations
- Drainage collection to a central treatment facility
- Water treatment

These were addressed in the following tasks:

- Task 1: compilation and review of data;
- Task 2: identification of the sources of loading and drainage discharge locations;
- Task 3: evaluation of the site hydrology and hydrogeology, and identification of the sources of water entry;
- Task 4: identification of drainage collection systems;
- Task 5: evaluation of treatment options
- Task 6: review of order and monitoring requirements
- Task 7: identification of long term remediation options, and identification of further studies.

The premining conditions were briefly assessed to understand the nature and potential extent of the environmental impacts of acid generation, and provide a reference for site remediation objectives. Mine development and water quality records during operation and since closure were reviewed and the environmental impacts defined. Three site visits by SRK and/or GPE personnel were conducted to assess the current site conditions and environmental impacts of these developments with reference to acid drainage. Potential remediation options were then developed on a preliminary basis and evaluated in terms of cost and benefit to identify preferred options. A staged approach to remediation is recommended to allow rapid implementation of initial and possibly short term measures, with an evaluation of the effectiveness of each stage, and allow further investigation as required.



## **2.0 PRE-MINING ENVIRONMENT**

### **2.1 Surface Topography**

The Britannia Mine is located in the Howe Sound, Coastal Region of B.C. The terrain is mountainous with elevations ranging from sea level to 4,500 ft. Exploration activity was recorded as early as 1888 and mine operations began in 1905 (Ramsey, 1967). Records and maps reflecting this early period in the mine life are consequently limited. All of the available maps, data and site information are presented and referenced to the Imperial system and are thus presented, as such, in this report.

### **2.2 Climate**

A climate station was operated at the township of Britannia Beach during the period 1913 to 1974. This station was located at elevation 49 m and recorded daily temperature, rainfall and snowfall. No official climate stations were operated at higher elevations within the study area.

The total annual precipitation at the Britannia Beach station averages 2164 mm, according to the 1951-80 climate normals published by the Atmospheric Environment service. About 96% of this amount falls as rain and the remainder as snow. The months of greatest precipitation are October to March. On average, the driest month is July. The average number of days in the year with measurable precipitation is 174.

The area of surface mining is situated at high elevation and therefore experiences significantly higher precipitation than measured at the coastal climate station. Within the general region, the average annual precipitation gradient is about 200 to 300 mm per 100 m of ascent. The proportion of precipitation that falls as snow also increases with elevation.

The 1951-80 normals for Britannia Beach show a mean annual temperature of 9.8°C. July is the warmest month, with a mean daily temperature of 18.2°C, and January is the coldest month, with a mean daily temperature of 2.0°C. The extreme maximum and minimum temperatures on record are 39.4 and -21.7°C, respectively.

### **2.3 Site Hydrology and Hydrogeology**

#### **2.3.1 Mine Site Catchments**

The mine development is situated in the drainage basins of Britannia Creek and Furry Creek. These streams drain comparable catchment areas (i.e., 29 and 54 km<sup>2</sup>, respectively) and both flow westward to

discharge into Howe Sound. Figure 2.1 shows the location of the mine development in relation to the coastline and the major streams in the region.

Within the drainage basin of Britannia Creek, the mining activity was primarily confined to two south bank tributaries known as Jane and Mineral Creeks. The catchments of both tributaries served as access points to the underground workings. In addition, the Jane Creek catchment contained most of the surface mining that occurred during the life of the Britannia Mine.

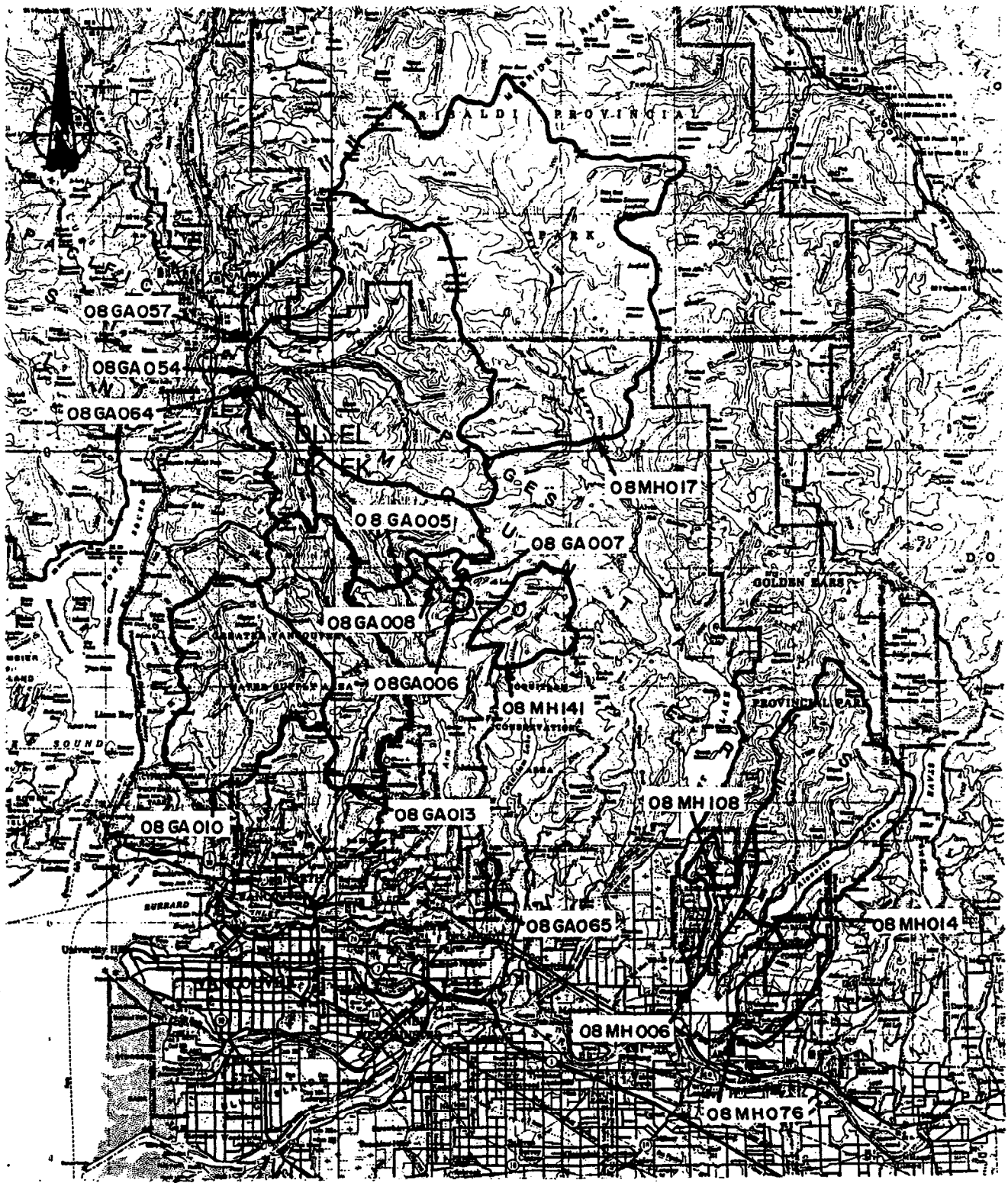
The area disturbed by mining within the Furry Creek catchment is drained by a number of small, steep-gradient streams, one of which has been named Empress Creek. These streams share a common drainage divide with Jane Creek. Figure 2.2 shows the catchment areas, and locations of the watercourses of the Britannia area prior to mine development.

### 2.3.2 Available Data

No long-term records of streamflow are available for Britannia and Furry Creeks. However, the southern coast of British Columbia around Vancouver is well served by a reasonably dense network of streamflow gauging stations. A total of sixteen of these regional stations, all operated by the Water Survey of Canada (WSC), were used to assist in characterization of the pre-mining hydrology. The positions of the selected WSC stations, in relation to the mine site, are plotted on Figure 2.1 together with outlines of their catchment boundaries. Reference to this figure shows that the assembled stations command catchments having a wide range of physiographic characteristics.

Table 2.1 is a list of the selected streamflow stations showing the period of record and the mean annual runoff (MAR). The MAR is expressed as a unit runoff (i.e., ratio of measured flow rate to contributing catchment area) in order to facilitate comparison between catchments. All of the selected stations measure unregulated flows, or flows that have been minimally influenced by man's activities.

The hydrology of the mine site streams was estimated using a technique known as regional analysis. This technique makes use of general relationships between streamflow and the characteristics of the catchment that generated the streamflow. In order to develop these general relationships, it was necessary to measure various characteristics of the catchments from topographic maps. Two suitable and easily determined characteristics are drainage area and median elevation. Tables 2.1 and 2.2 provide these measured characteristics for the catchments of the regional WSC stations and of the ungauged project streams, respectively.



Scale - 1:500,000

08MH04 — I.D. NO. FOR WSC STREAMFLOW GAUGING STATION

—— CATCHMENT BOUNDARY

MINISTRY OF ENERGY MINES AND PETROLEUM RESOURCES

DATE JAN. 1991

LOCATIONS OF REGIONAL WSC STREAMFLOW GAUGING STATIONS

PROJ. NO. 66004

APPROVED

NO.

2.1

STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers



**TABLE 2.1  
DETAILS OF REGIONAL WSC STREAMFLOW GAUGING STATIONS**

STREAMFLOW GAUGING STATION		PERIOD OF RECORD	MEAN ANNUAL RUNOFF (mm)	CATCHMENT AREA (km <sup>2</sup> )	CATCHMENT MEDIAN ELEVATION (m)
ID No.	Name				
08MH014	Alouette River at outlet of Alouette Lake	1916 - 1925	3522	203	690
08GA008	Brandt Creek near Vancouver (upper station)	1913 - 1921	3798	3.37	1050
08GA010	Capilano River above intake	1914 - 1988	3670	172	870
08MH141	Coquitlam River above Coquitlam Lake	1982 - 1988	3693	54.7	1090
08GA005	Indian River near Vancouver (upper station)	1912 - 1921	4100	90	950
08MH108	Jacobs Creek above Jacobs Lake	1965 - 1980	2492	12.2	530
08MH076	Kanaka Creek near Webster Corners	1960 - 1988	1860	47.7	280
08GA054	Mamquam River above Mashiter Creek	1966 - 1986	2410	334	1140
08GA057	Mashiter Creek near Squamish	1966 - 1981	2110	38.9	1160
08GA065	Noons Creek at Meridian Substation Road	1976 - 1988	3004	2.59	760
08MH006	North Alouette River at 232nd Street	1960 - 1988	2437	37.3	500
08GA006	Norton Creek near Vancouver	1912 - 1921	3576	2.36	830
08MH017	Pitt River near Alvin	1952 - 1965	3301	515	1450
08GA013	Seymour River above intake	1914 - 1926	3770	153	860
08GA064	Stawamus River below Ray Creek	1972 - 1988	2850	40.4	1090
08GA007	Young Creek near Vancouver	1912 - 1921	3360	6.22	1010

**TABLE 2.2**  
**PHYSIOGRAPHIC CHARACTERISTICS OF MINE SITE CATCHMENTS**

LOCATION	CATCHMENT AREA (km <sup>2</sup> )	CATCHMENT MEDIAN ELEVATION (m)	CATCHMENT MEDIAN ELEVATION (ft)	MINE LEVEL (ft)
Britannia Creek at mouth	29	1150	3770	530
Empress Creek at mouth	0.41	1210	3970	330
Furry Creek at mouth	54	1070	3510	790
Jane Creek at mouth	1.7	1210	3970	330
Mineral Creek at mouth	2.9	1040	3410	890

\* As described in Section 3.1.

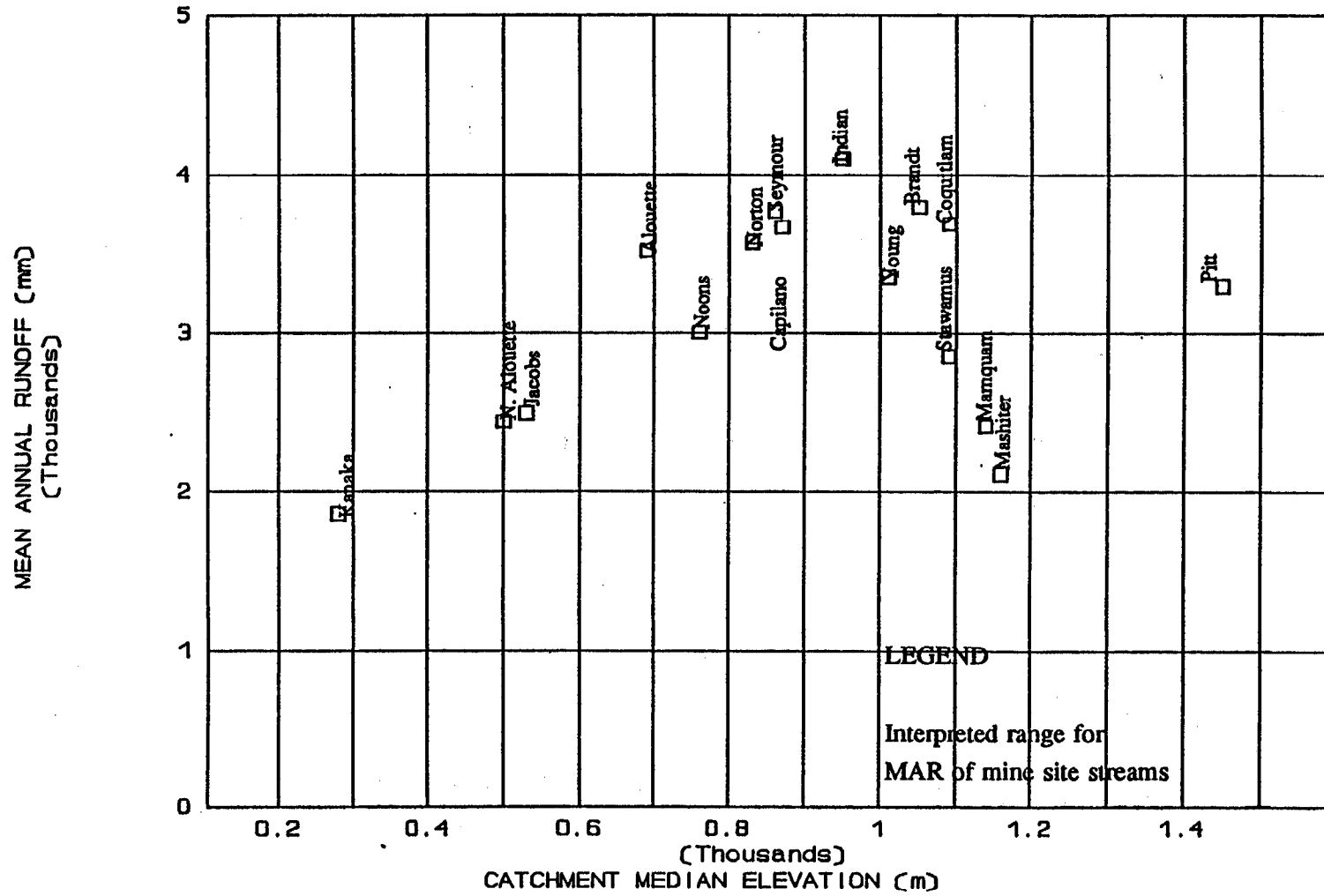
### 2.3.3 Natural Hydrology

Median elevation generally accounts for a large proportion of the variation in MAR between catchments. This is illustrated in Figure 2.3 based on the data from the assembled WSC stations. The data follow the expected trend of increasing MAR with increasing catchment median elevation. A second trend is also evident in which unit runoff decreases as one moves inland.

Based on the relationship indicated in Figure 2.3, estimates were made of MAR at various key points on the mine site drainages (see Table 2.3). Because of the scatter of data used to develop the relationship, a high and low estimate were provided at each key point that was believed to bound the actual MAR. The mine site streams experience relatively wet conditions with estimates of MAR ranging from 3000 to 4500 mm.

The seasonal runoff distributions for the mine site streams were approximated using normalized average monthly hydrographs from the regional WSC stations. Figure 2.4 graphically presents the monthly distributions, expressed as percentages of MAR, for the four highest elevation WSC stations. As with MAR, catchment median elevation is a good predictor of the seasonal runoff distribution experienced at a given location. At median elevations of about 1090 m, the normalized distributions for the Coquitlam and Stawamus Rivers match reasonably closely and exhibit a bimodal shape with peaks in May and November. The former peak is due primarily to snowmelt while the latter is caused by fall rains. The Mamquam and Pitt Rivers have higher median elevations (i.e., 1140 and 1450 m, respectively) and accordingly have distributions with later snowmelt peaks and diminished fall peaks.

Figure 2.3 Regional Relationship Between Mean Annual Runoff and Catchment Median Elevation



**TABLE 2.3**  
**ESTIMATED NATURAL MEAN ANNUAL AND MONTHLY STREAMFLOWS FOR MINE SITE**

NATURAL MEAN MONTHLY STREAMFLOW (m <sup>3</sup> /s)										
MONTH	Britannia Creek at mouth		Empress Creek at mouth		Furry Creek at mouth		Jane Creek at mouth		Mineral Creek at mouth	
	Low Estimate	High Estimate	Low Estimate	High Estimate	Low Estimate	High Estimate	Low Estimate	High Estimate	Low Estimate	High Estimate
January	2.2	2.8	0.031	0.040	3.5	4.7	0.13	0.17	0.19	0.25
February	2.2	2.8	0.031	0.040	3.5	4.7	0.13	0.17	0.19	0.25
March	2.6	3.4	0.037	0.048	4.2	5.6	0.15	0.20	0.23	0.30
April	2.8	3.6	0.039	0.051	4.4	5.9	0.16	0.21	0.24	0.32
May	5.9	7.6	0.084	0.108	9.5	12.7	0.35	0.45	0.51	0.68
June	5.7	7.3	0.081	0.104	9.1	12.1	0.33	0.43	0.49	0.65
July	3.5	4.5	0.050	0.064	5.6	7.5	0.21	0.27	0.30	0.40
August	1.5	1.9	0.021	0.027	2.4	3.1	0.09	0.11	0.13	0.17
September	1.8	2.3	0.025	0.032	2.8	3.7	0.10	0.13	0.15	0.20
October	3.3	4.2	0.047	0.060	5.2	7.0	0.19	0.25	0.28	0.38
November	3.6	4.7	0.051	0.066	5.8	7.7	0.21	0.27	0.31	0.42
December	3.4	4.3	0.048	0.061	5.4	7.2	0.20	0.25	0.29	0.39
<b>AVERAGE</b>	<b>3.2</b>	<b>4.1</b>	<b>0.045</b>	<b>0.058</b>	<b>5.1</b>	<b>6.8</b>	<b>0.19</b>	<b>0.24</b>	<b>0.28</b>	<b>0.37</b>
<b>MAR (mm)</b>	<b>3500</b>	<b>4500</b>	<b>3500</b>	<b>4500</b>	<b>3000</b>	<b>4000</b>	<b>3500</b>	<b>4500</b>	<b>3000</b>	<b>4000</b>



# Mean Monthly Distribution of Runoff at Regional WSC Streamflow Gauging Stations

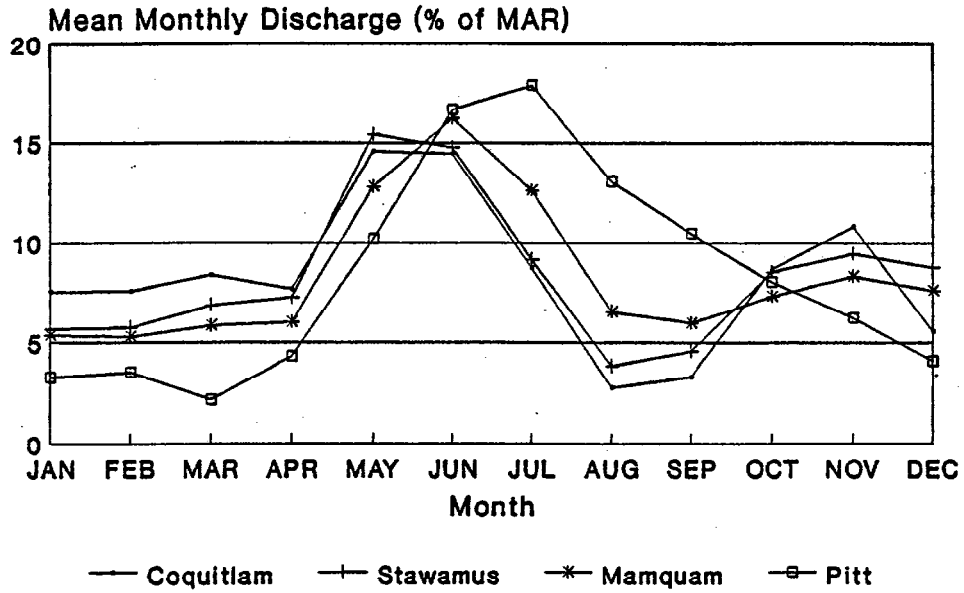


Figure 2.4

Based on similarities in median elevation, the mine site streams were assumed to have the same average monthly distribution as observed at Stawamus River. With this distribution, the natural flows, or flows that existed prior to mine development, were estimated at key points within the Britannia Creek and Furry Creek catchments (see Table 2.3). For reasons discussed above, upper and lower estimates were provided for each month. These estimates provide a bound on the long-term average streamflow but not on the natural variation of streamflows from year to year (i.e., the upper and lower estimates should not be used as indications of dry and wet years).

### 2.3.4 Site Hydrogeology

There is no data on the hydrogeology of the pre-mining environment. It is probable that the original water table was a subdued image of topography. Along Mineral Ridge and in the area of Jane Basin the groundwater table was probably in the order of 75 m below the ground surface (James, 1929). Recharge of the groundwater system would have occurred along this ridge. The major flow paths would have been from the ridge area towards the north and south, to Britannia and Furry creeks, respectively. There may have been a small component of flow parallel to Mineral Ridge through the fracture system of the Britannia shear zone. The porosity of the rocks in Mineral Ridge are probably in the order of 1% or less, therefore, groundwater flow and storage is not considered a major component of the system.

## 2.4 Geochemistry

The geology and geochemistry of the ore and waste rock provides an indication of the sources and the potential for acid generation and metal leaching, potential impacts on water quality, and the future pattern for acid generation. The geology of the area has been described in detail by James, 1929; GSB, 1990; McCullough, 1968.

The principle economic mineralization in the area is associated with quartz mineralization, and consists of pyrite and chalcopyrite, with important concentrations of sphalerite in certain areas of the deposits (Jane and No.8). "The seven orebodies are located within a sheared band of apparently steeply dipping metamorphosed sedimentary and volcanic rocks of varying competency that form a roof pendant two miles wide and seven miles long, surrounded by granitic rocks of the coast range intrusive complex ... Fracturing occurred in the firmer rock units permitting ore solutions to penetrate until conditions were favourable for ore deposition. These favourable conditions varied in the different sections of the mine and resulted in different types of orebodies" (The Anaconda Company (Canada) Ltd., 1965).

Generally the economic mineralization was contained in massive sulphide orebodies, as stringer type deposits, and as disseminated sulphide and concentrations along bedding planes. The MINFILE report described the sulphide orebodies at Britannia as "...highly heterogeneous mixtures of sulphides, remnant

altered host rocks, and discrete veins. The main mineralogy of orebodies is simple and fairly constant. Pyrite is by far the most abundant mineral, with less chalcopyrite and sphalerite and minor erratically distributed galena, tennantite, tetrahedrite and pyrrhotite. The main nonmetallic minerals include quartz and muscovite (chlorite), anhydrite and siderite."

Zinc mineralization tended to occur in the upper central parts of massive orebodies, as in the Fairview Zinc vein. The main massive orebodies with rich chalcopyrite ores were the Bluff, East Bluff, No. 5, No. 8, and 040. Several of the orebodies showed variations in sulphide distribution; No.8 massive graded from zinc to copper although the No.8. and 8A contained higher zinc than 8B. In the Bluff deposit, sphalerite was found to be abundant only above the 1800 level.

## 2.5 Water Quality

The pre-mining surface water quality from the Britannia site has not been established from the available data. There are indications however that metal leaching and iron staining was naturally occurring from the site as early as the late 1800's (B.C. Dept. of E.M.P.R., 1890 - 1900). This would suggest that the sulphide mineralization is naturally reactive, and oxidation and acid generation could result in some degree of natural elevation of metal and sulphate concentrations in surface waters prior to mining development.

The presence of gypsum and anhydrite throughout the orebodies could tend to contribute elevated sulphate concentrations to natural waters. Sulphide mineralization varied over the property from copper rich through copper and zinc to zinc rich. Copper mineralization dominated, however, and thus elevated copper, iron, and sulphate concentrations and loadings could be expected in the long term, with lesser zinc based on relative abundance. Zinc is, however, soluble over a greater pH range than copper, and this elevated zinc might have been apparent in drainage from areas not undergoing rapid sulphide oxidation and metal leaching.

It would be expected that natural drainage waters from the area could contain elevated sulphate concentrations. Exposed mineralization from outcrops and local disturbances and natural erosion appears to be readily oxidized contributing dissolved and total metals to the natural drainage. Biological data are not available and it is not apparent to what degree the natural water quality affected utilization of Britannia Creek. It is beyond the scope of this report to determine the extent of fish utilization in Britannia Creek prior to mining.

### 3.0 MINE DEVELOPMENT

#### 3.1 Mining Operations

Exploration in the Britannia area was first recorded in the late 1800s and the Britannia deposits were discovered in 1888. There was essentially no development, however, until 1900 when approximately 67 m of underground headings were advanced in the Jane deposit, near the 1000 level. Production from the mine began in 1905. Up to 1911, a total of 203,000 tonnes of ore was mined, of which 69,000 tonnes was milled. During these early years there was some open pit mining although the main source of ore was the underground production. After 1911, all ore production was from the underground mine, and the production rate jumped to over 100,000 tonnes per year and rose to nearly 2,000,000 tonnes per year by 1931. In 1921, however there was no ore milled as the mill was destroyed by fire. During the depression years to 1935 production dropped by approximately 50 percent after which it jumped back to nearly 2,000,000 tonnes per year. Production again fell by 50 percent during the war years. After the war, Britannia Mine steadily produced in the order of 700,000 tonnes per year until the late 1950s when production began to fall. In the 1960s, production was about 730,000 tonnes per year with copper grades ranging from 1.1 to 1.4 percent and with zinc grades of about 1.0 percent. Ore production was boosted after 1966 when open pit mining commenced in Jane Basin. This continued until 1974 when the mine was shut down permanently. Over the mine life a total of  $47.9 \times 10^6$  tonnes of ore was mined, of which  $47.4 \times 10^6$  tonnes was milled.

All ore was transported to the milling facilities at Britannia Beach. Ore was transported from the various ore zones to the concentrator using overhead tramway, inclined railway, and an electric tram line. The Britannia mill produced copper, zinc, and pyrite concentrates. Gold and silver were recovered primarily in the copper and pyrite concentrates. In the latter years, tailings were discharged directly into Howe Sound primarily through two intertidal outfalls located near the mouth of Britannia Creek.

Seven orebodies were mined through open pit, underground and gloryhole developments. Approximately 80 km of underground workings and four open pits were excavated to extract the ore. The access to the underground workings from the Britannia Creek basin is provided primarily through portals at the 2200 level, 2750 level and 4100 level. (Mean Sea Level corresponds to approximately the 4300 level for each 100 foot gain in elevation, 100 feet should be subtracted from 4300 feet.) The reference levels for the underground workings at the Britannia Mine were measured from a datum near the maximum elevations of the property and, therefore, the 2200 level is at a higher elevation than the 4100 level.

During the operational phase, flows in the Britannia Creek catchment were regulated for the purposes of power generation. A total of 9 storage or diversion dams were constructed within the drainage basin of this stream, most of which supplied the requirements of two power plants. A single diversion dam was

also constructed on Furry Creek. Operation of the hydroelectric development was discontinued in the same year as the permanent closure of the Britannia Mine. At present, only three of the original dams remain, namely, Tunnel, Utopia and Park Lane. The latter two dams are located in the headwaters of Britannia Creek. Due to a potential for failure of the dams and subsequent flooding downstream, the Water Management Branch recently (May, 1989) blasted drain holes in Utopia and Park Lane dams. Tunnel dam is located approximately midway along the length of Britannia Creek and now serves as the water supply for the high-pressure system for Britannia Beach township. The reservoir for this dam is almost full of debris, much of which was derived from breaching the Park Lane dam.

At closure of the mine, a number of steps were undertaken to limit the impact of contaminated mine drainage on the surface water resources of the area. Diversion dams were constructed to force the mine water to drain internally through the underground workings down to the lowest haulage tunnel, known as the 4100 level. The most important diversion dam was constructed in 1974 in the 2200 level tunnel near the portal. A submerged outfall was constructed at the shoreline of Howe Sound for the disposal of mine water into the ocean at depth.

The dam had been installed in an effort to retain 2200 level mine water in the mine, and direct the flow down through the workings to report with the effluent from the 4100 portal. In 1983 drainage was noted again from the 2200 level. In 1985, Klohn Leonoff Ltd. was retained by Copper Beach Estates to determine the integrity of the dam and to provide recommendations on improving the internal drainage. The report prepared by Klohn Leonoff (1985) was essentially a site visit report which concluded that the accessible areas of the 2200 tunnel were generally stable. Klohn Leonoff indicated that they had been unable to inspect the dam in detail because it was covered by formwork. There did not seem to be sufficient flow capacity within the mine to handle all the drainage, and thus, the dam leaked and overflowed, resulting in continued discharge at the 2200 level. A rock slough on the 2200 level near the No. 3 shaft also limited the potential flow downwards by this means, and pipes that had been installed within the slough were inadequate for the flow. Timber and debris were also present which limited the flows. In addition to a cleanup of the slough and debris, Klohn Leonoff recommended that diversions or control structures should be considered to help limit runoff entering the mine.

In summary, since the mine shut down, most of the surface water enters the mine workings through the open pits and gloryholes above the 1050 level. Drainage is routed through the workings to discharge at approximately the 4100 portal, but actually exits underground at the 4150 level. At the present time, particularly during periods of high water flow, drainage also issues from the 2200 level and discharges into Jane Creek which empties into Britannia Creek. If flow rates are excessively high, or copper concentration is less than 15 mg/L, the drainage is routed through a 100 foot deep outfall into Howe Sound. If copper concentrations in the drainage exceed 15 mg/L, all drainage from the 4150 discharge

can be routed through the cementation treatment plant, prior to discharge to Howe Sound through the deep outfall.

### 3.2 Geochemistry

Prediction of acid generation potential was not conducted for the Britannia site either before mining or during operation. A recent survey, by the B.C. Acid Mine Drainage Task Force of mines, in the province included the Britannia Beach operation. No data were provided in response to the questionnaire on the quantities or acid generation characteristics of the waste. A test program was conducted by Environment Canada in 1983 and 1984 (Ferguson, 1985) to investigate static and kinetic prediction test techniques and included a waste rock sample from Britannia. It was noted, however, that this sample was not necessarily representative of all of the waste rock from the site.

Acid base accounting on the sample indicated a neutralization potential of +29 tonnes  $\text{CaCO}_3$  equivalent/1000 tonnes of material at 16.2 percent sulphur content, yielding a net neutralization potential of -477 tonnes  $\text{CaCO}_3$  equivalent/1000 tonnes of material. Kinetic testing, using humidity cells, confirmed that the material was acid producing, generating a pH in the drainage water of 4.8 after 1 week of operation and pH 3.0 after 10 weeks of operation. Approximately 300 mg of sulphate were produced after 10 weeks, from a 200 g sample of waste material (1.5 mg/g). Acidity and dissolved iron production indicated that the sample neutralization capacity was exceeded during the test. Dissolved metal concentrations in the leachate averaged 5.41 mg/L Cu, 2.34 mg/L Zn, 2.2.5 mg/L Al and 26.5 mg/L Fe (equivalent to milligrams of dissolved metal per gram of waste material).

### 3.3 Water Quality

The water quality of drainage from the mine workings was a concern for most of the mine life. Dissolved copper was noted in mine waters in the 1920s (Hooper, 1970). Generally elevated copper was detected with increased production from a mining area. Water quality data were compiled for the period from 1923 to 1965 (Hooper, 1970) and correlated with production from the Fairview zone. The data showed maximum copper concentrations of approximately 1300 mg/L copper (up to 1934) decreasing to about 700 mg/L in 1938. Production from the Fairview pit had decreased to less than 100,000 tons per year just prior to this period (1934-1936). Another peak in copper concentration was evident in 1944, subsequently followed by decreasing concentrations down to less than 400 mg/L in 1965. Mining from the Fairview area reached a maximum of 1.1 million tons per year from 1937 to 1940. Thus copper concentrations tended to correspond to mining rates with a time lag of approximately five years, probably reflecting the rate of oxidation and metal leaching from the exposed rock.

From a description of each of the main orebodies and the wall rocks presented by James (1929) both the ore and wall rocks have abundant pyrite and other sulphides which are potentially acid generating. It would appear that oxidation was noted in several areas throughout the workings although the description was somewhat unclear, "...Pyrite is more resistant than other minerals, but it is invariably corroded by quartz, and is not stable..." (James, 1929).

The contribution of surface disturbances to contaminant loadings was also noted during operations. Correspondence from Anaconda (1974) indicated that two surface sources, rather than the mine workings, were responsible for the elevated metal levels in Jane Creek.

- i) an old mine waste dump located "... below and to the south of Jane Basin, on the steep slide area of the west side of the valley." The waste rock was discarded from the nearby 1200 portal, from which there was no drainage at the time of observation. Jane Creek at that time did however flow by this dump before entering Britannia Creek. The dump was thought to contain both pyrite and chalcopyrite;
- ii) the original mining townsite area was located in the "...general Jane Basin area...". Mining activity from the underground workings ceased in 1957 but surface mining continued periodically until 1974. Within the basin area therefore there are reportedly both exposed workings and extensive slides. One slide was described as extending from the Basin down to Britannia Creek with a significant quantity of mineralized rock distributed over the mountainside.

A sampling program in 1974 included Jane Creek immediately above the confluence with Britannia Creek. The results indicated minor concentrations of heavy metals and near neutral pH, however the sampling was conducted in January and March. Flows in the two sampling periods were 600 and 200 IGPM (38 and 13 L/s). Metal concentrations were <0.6 mg/L Cu, <1.3 mg/L Zn, 0.05 mg/L Fe, and 52 mg/L  $\text{SO}_4^{2-}$ . It is not clear whether these are dissolved or total metal concentrations; total solids in solution were 10.4 mg/L.

An excellent overview of the more recent history of the Britannia acid mine drainage problem, together with a summary of the impacts on Howe Sound is provided in the review by Goyette and Ferguson (1985). Two internal reports for the Ministry of Environment (Moore, 1985; Moore and van Aggelen, 1986) provide detailed analytical results of investigations with respect to water quality and environmental impact. These reports provided useful analytical and flow data that were used in treatment plant design for this study.

The review by Goyette and Ferguson has been summarized below, with respect to acid generation and water quality:

### Water quality

- all minewater was diverted to the "Beach" treatment plant in 1972 and then discharged to Britannia Creek, until the deep outfall was constructed in 1978;
- flows of 65 m<sup>3</sup>/d with low dissolved copper concentrations (0.43 mg/L) were found from the 2200 level in 1980;
- at this time the copper concentration in the drainage from the 4100 level was 14 mg/L, indicating contaminated minewater was still diverted from the 2200 level. In 1983 however significant flows of contaminated mine water had emerged from the 2200 level;
- elevated treatment plant copper concentrations corresponded more to high influent concentrations than to large flows, however both are contributing factors. Overall copper removal efficiency was 19 percent during the early 1980s;
- similar water quality was noted in Britannia Creek to 1976 with 1.6 mg/L dissolved Cu, 37 mg/L dissolved Zn, and 40 mg/L dissolved Fe recorded at the mouth. Concentrations were reduced by an order of magnitude in 1980 with the diversion of minewater to the deep outfall discharge, although metal concentrations in the creek increased again in 1983/1984 with the reintroduction of the 2200 portal drainage;
- sampling in 1983/1984 of the minewater and selected sites along Britannia Creek indicated the water was acutely toxic to fish. Britannia Creek was probably acutely toxic to fish over most of the mine life;
- copper concentrations in the drainage from the 4100 portal have decreased from about 66 mg/L in 1966 to 17 mg/L in 1984.

### Contaminant loading

- high flow periods correspond to high copper loadings and concentrations indicating a large store of soluble oxidation products are contained in the workings;
- historically water quality in Britannia Creek has been impacted by the high metal loadings from contaminated drainage. Surveys in 1972 indicated that the 2200 portal drainage was the only significant source of copper to Britannia Creek several hundred metres upstream of its confluence with Mineral Creek. There was an unidentified source, however, between this station and below Mineral Creek. Treated minewater from the plant contributed 61 percent of the copper (253 kg/d) to the Creek with unidentified sources contributing the rest;
- pH, sulphate and metal loadings to Britannia Creek were lower in 1984 than 1972;
- the mean copper loading from the 2200 portal in 1984 was 124 kg/d;
- estimated mean dissolved copper load to Howe Sound via Britannia Creek in 1972 was 417 kg/d;
- loadings from the treated minewater and plant bypasses from 1980 to 1984 varied from 50 to 700 kg/d with peaks occurring during June/July and November/December, corresponding to high



minewater flows. These were less than the 1972 loadings in the Creek, however, the mine water in 1980 to 1984 did not include the 2200 portal drainage, which could be a significant component.

### 3.4 Water Treatment

In 1924 a recovery process was developed to precipitate dissolved copper onto scrap iron. A plant was installed in the Jane Flats area along with a number of small plants. During most of the mine life, acidic drainage from the mine workings was collected for treatment in two treatment plants; one at the Townsite or 2200 level in a plant constructed in 1928, and the other at the 4150 or "Beach" level constructed in 1955. The 2200 level plant was shut down in early 1972. The drainage from the 2200 level portal, previously treated in this plant, was diverted internally to the 4100 level.

Copper production from this process reached one million pounds (453,600 kg) a year during the early 1930s. The predicted recovery in 1970 was 262,000 pounds (118,840 kg). This decline in production would suggest a decrease in copper leaching from the mine drainage. It might also reflect changes in the water flow patterns, flushing of the workings, and effectiveness of the cementation plant.

The treatment process is based on the electrochemical reduction of copper in solution to copper metal with simultaneous oxidation and release of iron from scrap iron. Copper precipitates on the iron scraps and can ultimately be recovered by smelting if desired. The discharge from the cementation process is routed to the deep water outfall. The cementation process does not remove any appreciable amount of acidity and cannot reduce copper to levels achieved with conventional treatment systems based on neutralization and precipitation of copper hydroxide.

The cementation process is particularly suitable for small waste flows since long contact times are required, but does not effectively treat high flows at low copper concentrations which are encountered at certain times of the year. The recovery of copper from the mine drainage waters varied over the year, with the highest copper recovery during the highest precipitation periods: fall rains and spring runoff. This reflects the increased area of the workings flushed during the high precipitation events, which remain dry or infrequently flushed during the dryer seasons. Goyette and Ferguson (1985) reported that more than 70 percent of the copper was recovered in the treatment plant when influent concentrations exceeded 80 mg/L, however removal efficiencies are less than 50 percent at concentrations less than 40 mg/L.

Investigations were conducted in the early 1970s to evaluate removal of the iron and copper from acidic mine water and prevent precipitation of metal hydroxides upon discharge into Howe Sound (Drake and Robertson, 1973; Anaconda correspondence, 1974). A neutralization circuit was considered for treatment of the mine effluent after the cementation plant, or dilution with mill water. This work was not pursued as the mine closed shortly thereafter.

TABLE 3.1  
SUMMARY OF FLOWS, COPPER CONCENTRATIONS, AND COPPER LOADING  
FROM THE 4100 PORTAL, 1983-1989

Year	Average Flow m <sup>3</sup> /h	Average [Cu] mg/L	Cu Loading kg/d
1983	664	19.7	313
1984	561	18.4	247
1985	396	16.1	153
1986	494	19.6	233
1987	597	17.8	256
1988	475	18.5	211
1989	442	15.3	162
AVERAGE	518	18.0	225

**TABLE 3.2**  
**SUMMARY OF ASSAY INFORMATION ON 4100 AND 2200 PORTAL FLOWS.**

Parameter (mg/L)	2200 21/06/83	2200 06/12/83	2200 21/04/84	4100 21/06/83	4100 13/07/83	4100 15/08/83	4100 08/09/83	4100 06/12/83	4100 21/04/84	4100 21/05/85	4100 20/06/85
pH	2.9	3.5	2.7	3.4				3.5	3.2	3.8	3.9
Acidity (pH 8.3)	445	109	494	367				306	277	239	258
Sulphate	880	293	895	1620				1700	1620		
Dissolved metals											
Al	31.3	8.05	36.4	39	37.6		35.3	30.6			
B	0.02	0.01		0.17	0.17	0.31	0.34	0.25		0.28	0.20
Cd	0.17	0.06	0.18	0.19	0.14	0.14	0.11	0.14	0.0058	0.12	0.16
Ca	129	53.2	121	441	411	418	478	419		450	387
Co	0.17	< .1	0.22	0.22	0.14	0.3	0.29	0.35		0.1	0.1
Cu	51.4	12.3	43	31.7	30.3	23.6	22.5	18.3	14	11.1	18.1
Fe	36.8	3.16	45.1	11.3	17.1	50.6	56.1	17.1	14	10.8	4.74
Pb	0.12	< .1	0.16	0.22	0.14	0.22	0.09	0.23		0.2	0.2
Mg	38.5	11.3	39.4	88.6	82.4	100	115	83.4		94.3	81.7
Mn	2.6	0.9	3.12	5.47	6.21	8.4	9.2	7.52		7.66	5.98
Ni	0.07	< .05	0.05	0.11	0.08	0.15	0.09	0.07		0.07	0.06
Zn	26.1	8.21	31.6	41.9	30.9	27.1	27.3	28.1	0.21	28.1	30.8
Flow estimate	210										

\* Acidity in mg/L equiv. CaCO<sub>3</sub>, flow in m<sup>3</sup>/h

The available treatment plant data have been compiled to indicate typical concentration ranges and flow rates and estimate required plant capacity (Moore, 1985; Moore and van Aggelen, 1986; Goyette and Ferguson, 1985; MOE treatment plant performance records). These data are summarized in Table 3.1 and Table 3.2. They are further discussed in Chapter 6, and presented graphically as Figure 6.11 - 6.13.

While there is a downward trend in copper tenor of the solutions since the mine closed in 1974, Figure 6.11 shows that the average of the copper tenors over the period 1983 - 1989 provides an adequate estimate of copper production for study of the potential for metal recovery. Similarly, Figure 6.12 shows that average hourly flows vary from 400 to 675 m<sup>3</sup>/h, there is no significant trend in flow and an average flow based on this period is adequate for this phase of study.

It is noted from this data Figure 6.13 that there appears to be an increase in copper tenor with flow rate; probably due to greater flushing and better wetting of sulphide surfaces during wetter years. The regression coefficient for the line shown is not statistically different from zero however, so the statement cannot be made in certainty. This correlation was also noted by Goyette and Ferguson (1985).

While there is a downward trend in copper tenor of the solutions since the mine closed, Figure 6.13 shows that the averaging of the copper tenors over the period 1983 - 1989 provides an adequate estimate of copper production for study of the potential for metal recovery.

### **3.5 Existing Environmental "Regulation"**

The Britannia Mine was permanently shutdown in November 1974. Anaconda, the previous operator, applied for permit under the Provincial Pollution Control Act in May 1973 that included provisions for treatment of waste water from a property using a lime treatment process. The permit Discharge criteria applied for May 1973 conformed to the Level "C" Objectives for Marine Discharge listed in "Pollution Control Objectives for the Mining, Mine-Milling Smelting and Associated Industries" published by the Waste Management Branch in March 1973. Acidic minewater from the 2200' Level was diverted within the working to the copper recovery plant at 4100 level after 1972. The plans for treatment of acidic drainage from the 4100 level portal included with the permit application were not implemented since the mine was shutdown in 1974 before the application approval process was completed.

The operation of the treatment plant and minewater discharge quality is controlled by an order issued January 29th, 1981, from the director of the Waste Management Branch (B.C.M.O.E.) to Copper Beach Estates. The order requires that all mine water is collected and directed to the 4100 level, and then to the "treatment" plant (copper recovery) if the dissolved copper is greater than 15 mg/L. The order also provides for bypass of minewater if the dissolved copper is less than 15 mg/L.

The 4100 level copper recovery plant continued in operation after shutdown. After 1978 all minewater was collected and discharged through a 100 ft deep outfall into Howe Sound. Responsibility for operation of the pollution control works was transferred to Copper Beach Estates Ltd. (CBE) and is controlled by an order issued by the Director of the Waste Management Branch to CBE in January, 1981. This order requires that all mine drainage be directed to the 4100 level copper recovery plant if the dissolved copper concentration exceeds 15 mg/L. Bypass is allowed if dissolved copper concentration is less than 15 mg/L.

Surveys conducted by both Environment Canada and Ministry of Environment have resulted in the following basic observations regarding the level of compliance,

- i) recorded bypasses occur regularly when the copper concentrations exceed 15 mg/L, (bypasses are used to protect the plant from washout during periods of excessive mine flows);
- ii) overall copper removal in the plant is low;
- iii) contaminated minewater is discharged to Britannia Creek from the 2200 portal during high flow periods; and,
- iv) operation of 'the treatment system' is erratically controlled.

## **4.0 EXISTING ENVIRONMENT**

### **4.1 General**

An initial site visit was conducted August 30, 1990 by SRK and GPE personnel to develop an appreciation of the site, identify the scope of the field investigations and define sampling requirements and locations. Mr. Eric Beresford (MEMPR), Dr. John Errington (MEMPR) and Mr. Dan Cumming (Copper Beach Estates) conducted a tour of the site, including surface workings up to the East Bluff and Jane pits, and underground from the 2200 portal. Access to the upper pit areas (Empress and No. 5 pits) was not possible due to road conditions and time constraints.

Two field investigations of the Britannia mine site were undertaken on September 21 and December 6, 1990 with the objectives of:

- identifying sources of acidic drainage;
- sampling and chemically characterizing mine discharges, seeps and receiving water;
- measuring or estimating discharge flows for an assessment of site hydrology and the determination of contaminant loadings;
- providing a basis to compare the present (1990) situation to historical conditions.

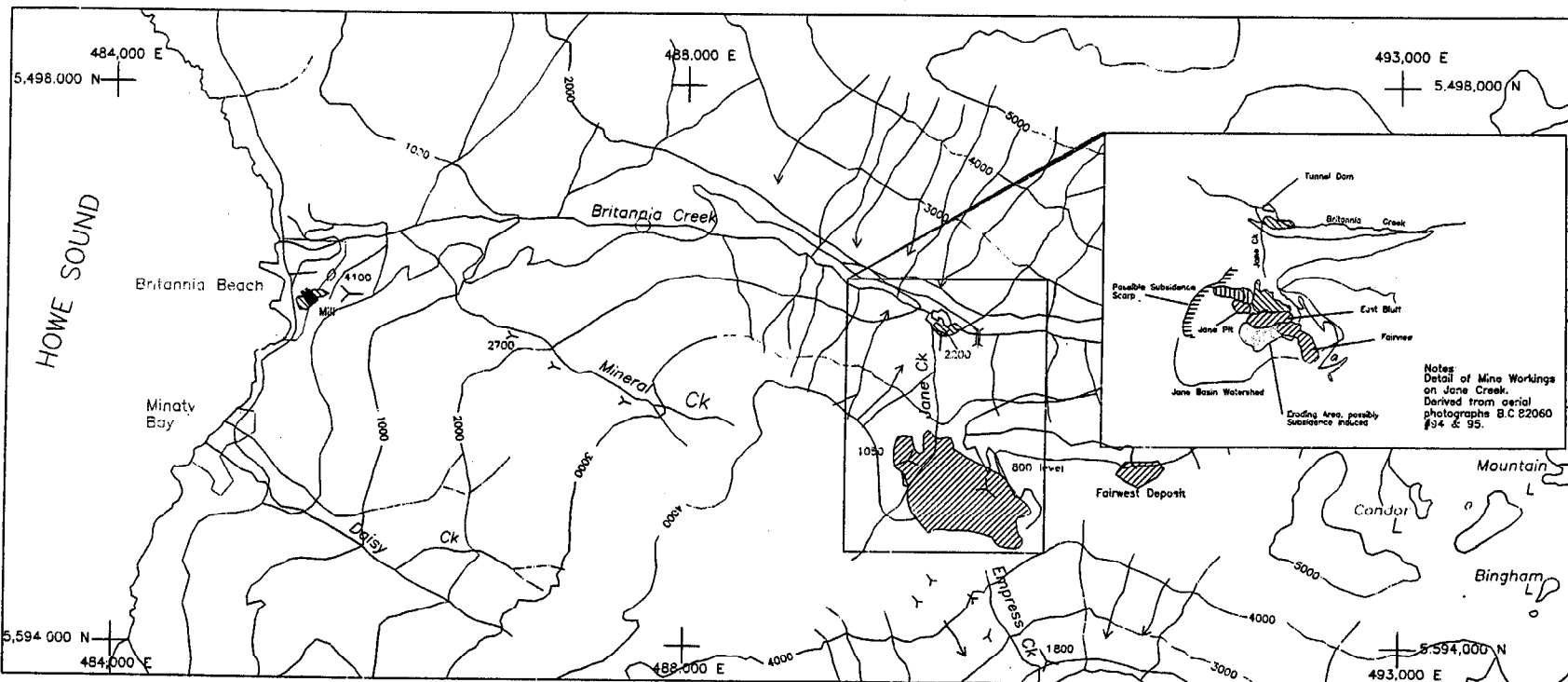
The focus of the investigation was the Britannia Creek drainage and contributing watercourses in accordance with the terms of reference for the project. Access to the upper areas of the site, above 2200 level, was not possible due to road construction and snow in the second survey.

The sampling locations and results of these investigations are discussed in Section 4.2 through 4.4, and presented in Figures 4.1 through 4.6. Detailed procedures of investigation and results are presented in Appendices 1 and 2.

## **4.2 Mine Development**

### **4.2.1 Pits**

Open pit mining was conducted in five pits, as shown on Figure 4.1. The quantity of material removed from each pit is not known, however the majority, probably in the order of 95 percent, was taken from the East Bluff and Fairview pits, Photograph 4.1. There was some initial open pit mining during the early

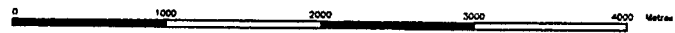


Map Scale 2:1

**LEGEND**

- Stream, definite
- - - Stream, indefinite
- Gravel Road
- Paved Road
- Trails
- Townsite
- ▨ Waste Dump Area
- ▩ Waste Dumps
- ▧ Open Pit and Main
- ▦ Slides
- Eroding Area
- Y Adits, principle mine access
- Y Adits, mine access unknown
- ▬ Subsidence Scarp
- Mill

North American Datum 1927  
 Contour Interval 1000 Feet  
 Elevations in feet above mean sea level  
 Transverse Magnetometer Program  
 Additional Elevations derived from 1:20,000  
 scale planimetric and contour map 822 068



Ministry of Energy, Mines and Petroleum Resources		DATE March, 1991
<b>BRITANNIA MINE</b>		PROJECT NO. 66004
<b>SURFICIAL MINE WORKINGS</b>		DRAWN BY [blank]
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers		NO. 4.1





years of the mine, however most of the open pit mining occurred after 1966. Some gloryholes, subsidence and early open pit mining had disrupted the topography prior to open pit mining in 1966. Glory holes were developed under all of the pits except the Empress, however, underground mine development has effectively reached the surface in this area. Subsidence and erosion has since filled all of the glory holes leaving steep sided cone shaped depressions, except for the Fairview pit which still has an open connection to the underground mine.

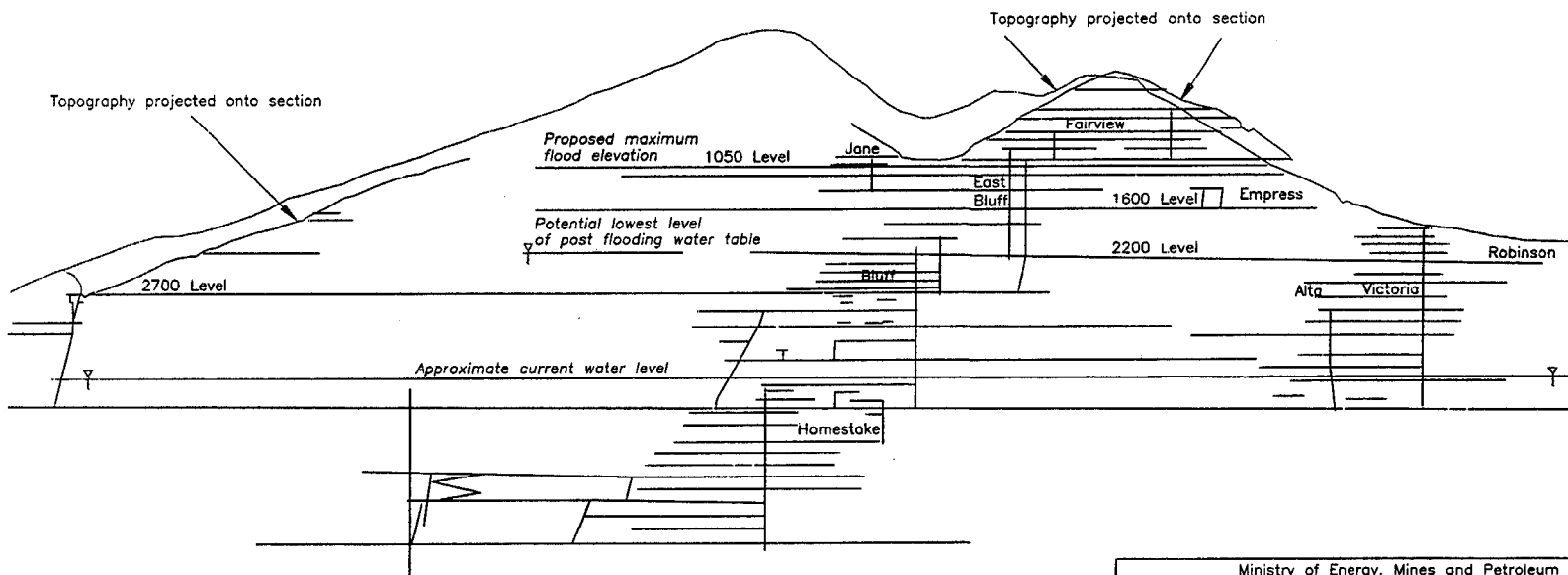
The 1050 level heading is exposed in the Lower Jane Pit, and the 850 level heading is exposed in the East Bluff Pit. (Note: Elevation 4300 ft = 0 level in the mine, mine level increases by 100 foot with each 100 drop in elevation). Essentially all of the disturbed areas within the pits have continuing slope instability as patchy vegetation has re-established in only a few areas. Above the Jane pit, and in the area of the 1915 landslide, there appears to be a scarp which may be indicative of ongoing subsidence. Essentially all of the pit walls appear to be acid generating, based on observed mineralogy and the presence of oxidation products.

#### 4.2.2 Underground

Underground mining has been the main source of ore at the Britannia Mine, accounting for in the order of 95 percent of the ore, and has exploited seven orebodies. Ore extraction used several methods, principally: timber supported open stopes, cut and fill, and sub-level caving. It is believed that most of the fill was cycloned tailings. Other mining methods may have been used. The proportion or location of the stopes which are backfilled is not known at this time. The mine is shown in a composite section in Figure 4.2. It is understood that there are many vent raises to surface, none of which are shown on this figure, however, it is expected that most of them are near the area of the open pits except for any associated with the Victoria camp. These vent raises would be located in the Furry Creek valley.

It is understood that in many areas of the mine that the timbers have rotted and the openings are collapsed. Some of the ore passes are also believed to be partially collapsed (Klohn Leonoff, 1984). The access drifts in the mine are believed to be approximately three by three metres in section. The condition of these headings and the accessibility throughout the mine is not known. It is understood that at mine closure all mine entrances were caved, although this does not seem to apply to the 2200 level portal in Jane Creek, where only a partial water diversion exists. An examination of the detailed mine plans will be required to identify the location of all portals and vent raises.

Acidic waste water currently drains from the mine. The principle sources of the acidity from the mine workings are probably the sublevel cave stopes and any stopes where backfill was used.



**LEGEND**

- Levels
- ▽ Water Table
- Topography

SECTION LOOKING NORTH

Notes: Figure derived from  
Copper Beach Estates  
Section is a simplified  
projection of the mine  
workings

Ministry of Energy, Mines and Petroleum Resources

**BRITANNIA MINES  
UNDERGROUND MINE WORKINGS**

STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers

DATE  
March, 1991

PROJ. NO.  
66004

APPROVED

NO.

4.2

### 4.2.3 Waste Dumps

There are three main areas where waste rock has apparently been dumped:

- in Jane Creek adjacent to the East Bluff pit,
- at the 4100 level portal,
- 2200 level portal.

These areas are shown on Figure 4.1. The quantity of material at each site is not known. A rough estimate of the waste dump geometry adjacent to the East Bluff pit indicates about 700,000 tonnes of waste. Estimates of the quantity of waste dumped at the 2200 level and 4100 level have not been made, however these quantities appear to be in the order of 30 and 10 percent respectively of the estimated 700,000 tonnes located at the Bluff Pit. These estimates are based on interpretation of high level airphotos, the original topographic data and mine records. Field measurements will be necessary to obtain more accurate estimates.

Most of the waste rock is contained in the dump adjacent to the East Bluff pit. A small portion of the waste at East Bluff pit was produced from the underground mine development and early open pit mining and may be up to 50 years old. The balance of the waste was produced between 1966 and 1973. The waste appears to be an acid generating material based on observations of its mineralogy and the presence of oxidation products. ABA tests have not been conducted on this material. Based on airphoto interpretation this waste dump partially overlies the debris of the 1915 landslide.

Waste has been dumped at the 4100 level portal area including the area south of the mill and the dock area. Most of the waste appears to be contained in the area south of the mill and probably consists of non-acid generating material produced during development of the 4100 level and an overlying layer of spilled ore and low grade material, which appears to be acid generating based on the iron oxide staining. The waste dumped at the 2200 level portal is believed to be development waste only and is probably acid generating, although ABA testing has not been conducted. Historical references (Ramsey, 1967) describe some landslides which travelled from the Jane Basin to Britannia Creek. Although these could not be identified in the airphotos, it is probable that they underlie the waste at the 2200 level portal. Britannia Creek has been subject to periodic flooding (1921, 1989) and these floods have probably eroded some of the waste deposited at this location.

Based on photographic evidence (Ramsey, 1967), waste has also been dumped at the 1800 portal on the Furry Creek side near the Victoria Camp, however, it has not been identified in the air photos as the area has naturally revegetated.

#### **4.2.4 Tailings**

Essentially all of the tailings were either discharged directly into Howe Sound or were cycloned to yield a coarse fraction for use in backfill with the fines being discharged into Howe Sound. However, some material, possibly tailings, was discharged to a tailings pond above the treatment plant at 4100 level, as shown on Figure 4.1 and Photograph 4.2. This was apparently an emergency pond, or settling pond. This pond may also have received effluent from the 2100 treatment plant during operation, and served as a settling pond.

The quantity of material contained here is not known, but based on plan dimensions of 20 x 40 m and an estimated depth of 5 m, the pond contains approximately 8000 tonnes of waste. Recent road construction has resulted in road fill material overlying a portion of the tailings pond. Placement of the road fill material over the tailings has not resulted in slope failures which implies that the tailings are highly consolidated and/or self cemented with iron oxide. These tailings are also believed to be acid generating from the oxidized surface appearance.

Unconfirmed reports suggest there may also be some tailings deposition in the beach area on the north side of Britannia Creek.

#### **4.2.5 Roads**

Many roads have been developed in the Britannia Creek valley. It appears that all of the roads have been constructed using cut and fill techniques and that imported mine waste has not been used in road construction, except for roads within the pits and on the waste dumps. Only those roads near to the pits are likely to have been constructed of acid generating material. It does not appear that any of the other roads involved cuts deeper than a few metres and are therefore unlikely to have exposed fresh acid generating material.

#### **4.2.6 Landslides**

There have been at least two instances of landslides in the Jane Creek area. The first slide occurred on March 22, 1915. Based on the records (Ramsey, 1967) and airphoto interpretation, this slide originated near the crest of the west side of Jane Basin along a shear zone oriented roughly north-south. It is not clear how large this slide was, however based on photographs of the damage, the reported size of several million tonnes is probably high by a factor of ten. The acid generation characteristics value of this material is not known, however slide debris was suspected historically to contribute dissolved metals to Jane Creek as described in Section 3.4.

Photograph 4.2  
Pond at 4200 level  
above treatment plant



The second landslide was probably due to the failure of a waste dumps near the Jane Portal at the 1050 level produced during the early underground mine development and open pit mining. The records are unclear regarding the date or the exact source or quantity of material involved. According to the records, the debris from the landslide forms part of the acid generating material deposited near the 2200 level portal.

### **4.3 Hydrology and Hydrogeology**

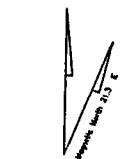
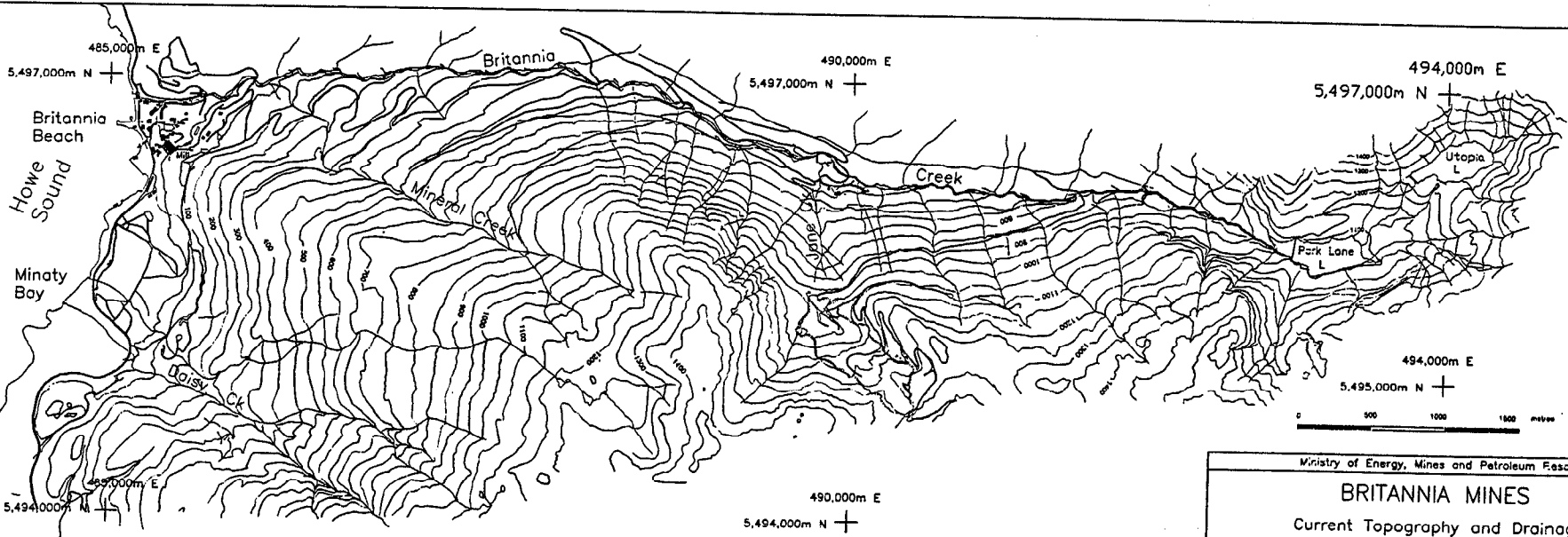
#### **4.3.1 Catchment Developments**

Developments within the study area that have significant effect on the existing hydrology comprise:

- i) alterations of surface drainage patterns,
- ii) discharges of mine water from portals and
- iii) abstractions for water supply.

Figures 4.1 and 4.3 show current hydrological regime and locations of these developments.

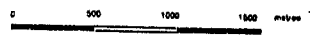
The alteration of surface drainage patterns is due to the existence of both the surface and underground workings. Most of this alteration is confined to an area known as Jane Basin, which comprises the upper reaches of Jane Creek and a small area along the drainage divide shared by Jane and Furry Creeks. The existing topography and catchments of Jane Basin are shown in Figure 4.4. The obvious impact of surface mining on the drainage patterns was to alter topography and thereby change stream courses and drainage divides. However, the impact is accentuated by the existence of well-drained underground workings directly beneath the open pits. Due to caving in the underground, the rock at the bottom of the open pits is probably highly fractured and therefore has high infiltration capacity. Also, vent raises probably exist that increase the effective infiltration. The coupled influence of the surface and underground mining probably causes most of the runoff from Jane Basin to be funnelled into the underground workings through the pervious bases of the open pits. This makes the Jane Basin essentially an enclosed drainage basin that likely contributes to the streamflow in the lower reaches of Jane Creek and to Furry Creek. The contribution, however is not considered to be significant.



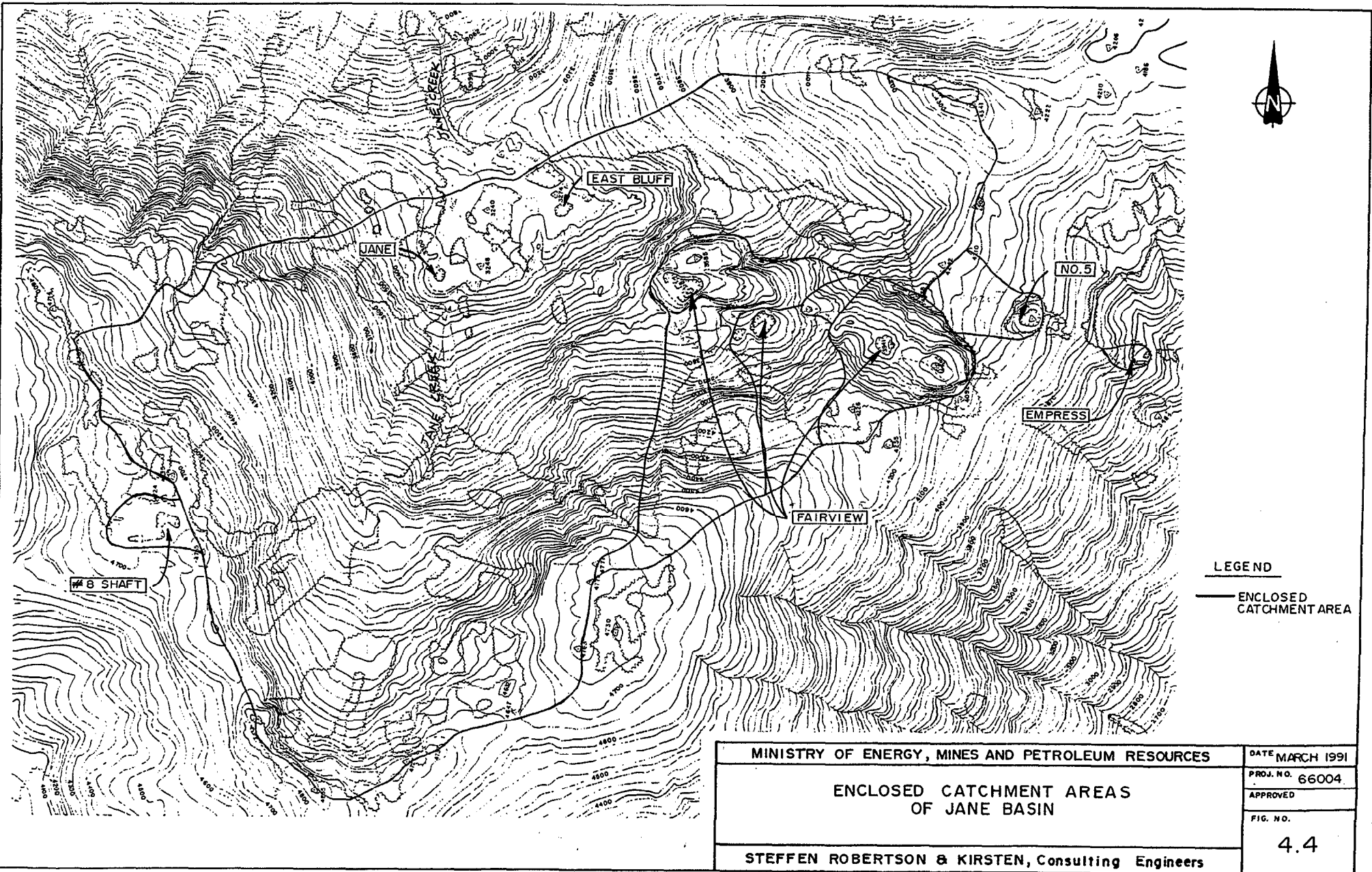
**Legend**

- Drainage
- Gravel Roads
- Paved Roads
- Trails
- Minesite

Notes: Map derived from 1:5,000 base maps supplied by Richmond Plywood, North American Datum 1927  
 Contour interval 50 metres  
 Transverse Mercator Projection



Ministry of Energy, Mines and Petroleum Resources		March, 1991
<b>BRITANNIA MINES</b>		66004
Current Topography and Drainage		
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**LEGEND**  
 — ENCLOSED CATCHMENT AREA

MINISTRY OF ENERGY, MINES AND PETROLEUM RESOURCES	DATE MARCH 1991
ENCLOSED CATCHMENT AREAS OF JANE BASIN	PROJ. NO. 66004
	APPROVED
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	FIG. NO. <b>4.4</b>



The drainage of mine water from the underground workings represents a second impact on the mine site hydrology. As explained in Section 3.1, most of the water from the Britannia underground drains at the 4150 foot level where it is subsequently disposed of through a submerged outfall to the depths of Howe Sound. The next largest source of mine water is believed to flow from the 2200 portal which discharges to Jane Creek. The flow of mine water from this portal has been observed to be seasonal, occurring only during periods of high flow. Mine water is known to discharge from at least two other sources, namely the 2700 portal in the catchment of Mineral Creek and a portal in the area of Empress Creek.

Abstractions are made from at least two locations within the catchment area of Britannia Creek, both of which are related to the water supply of the township at Britannia. The high pressure water system for the township of Britannia is supplied from Tunnel Dam in Britannia Creek. A penstock, once used for generation of hydroelectric power, diverts water from Tunnel Dam to a reservoir for the high pressure system. Overflow from this reservoir makes its way to Thistle Creek, a small stream that discharges directly into Howe Sound. The potable water supply is obtained from a small dam on Mineral Creek. This dam is located above the inflows from the 2700 portal.

An additional influence on the mine site hydrology could be drawdown of groundwater caused by underground workings. This could reduce base flows below natural levels for streams that drain the mine area.

#### 4.3.2 Available Data

During the field trips conducted on September 21, 1990 and December 6, 1990, spot measurements of streamflow were made at several locations within the drainage basin of Britannia Creek. Spot flow measurements were also made by others in earlier field investigations of the Britannia Mine area. These data are valuable for the assessment of contaminant loadings carried by the mine site streams however, they are too sparse for accurately characterizing the present-day hydrology of the study area streams. The data are discussed in detail in Section 4.4.

Although few streamflow measurements are available an excellent flow record has been kept on mine drainage from the 4150 level discharge. The flows are gauged by two Parshall flumes, one located on a channel leading to the treatment plant and the other on a bypass channel leading directly to the submerged outfall. Stage measurements are made on a weekly basis. The flow record at the two flumes commences in March 1978 and is complete up to the present.

A record of infrequent flow measurements has also been made of the mine drainage discharging from the 2200 level portal.

### 4.3.3 Present-day Hydrology

Section 2.2.3 provides estimates of the natural hydrology at key points within the study area. The following discussion outlines the extent the abandoned mine development has altered the natural hydrology at those same key points.

Within the study area, the greatest disturbance to the surface hydrology has occurred on Jane Creek. As explained above, it is believed that most of the runoff from the upper reaches of Jane Creek is trapped by and then funnelled through the open pits, or glory holes, into the underground workings. Figure 4.4 shows the outline of the enclosed drainage basin commanded by each of the open pits within the Jane Basin. The total area covered by enclosed drainage basins, including the small area within the Furry Creek catchment, amounts to 1.19 km<sup>2</sup>.

The total runoff from the enclosed drainage basins was estimated using regional analysis. At a median elevation of about 1250 m, the unit MAR was estimated to fall within the range of 3500 to 4500 mm. Figure 4.5 shows a high and low estimate for the average monthly distribution of runoff generated by the enclosed drainage basins. For comparison sakes, the average monthly flows measured at the 4150 portal are superimposed on this figure. The measured mine flows compare remarkably closely to the estimated runoff from the Jane Basin. Although not all of the mine drainage exists at the 4150 portal, the correlation suggests that surface runoff from Jane Basin may be the single largest source of mine water.

With the flow from the upper reaches essentially cut off, the streamflows in Jane Creek at its mouth have been drastically reduced from the pre-mining condition. The effective catchment area for Jane Creek at its mouth is now about 0.59 km<sup>2</sup>, or 35 percent of the original size. Thus, the flows in Jane Creek have probably been reduced by a similar proportion. Mine drainage from the 2200 portal only partially compensates for the lost contribution of runoff from the upper reaches of Jane Creek.

The hydrological regime of Mineral Creek, another tributary of Britannia Creek, has experienced disturbances due to mine development. Several tunnels daylight within the catchment of Mineral Creek, and at least one of these, the 2700 or 2750 portal, is a source of mine water. The 2700 portal was not visited during the 1990 field trips as access to this area was not possible, however, it is understood from site personnel that the flows are relatively small. The other impact on the hydrology is the abstraction for the Britannia Beach water supply. Further data would be required to quantify the difference in flows between present-day and natural conditions.

# Mean Monthly Runoff and Mine Drainage Distribution

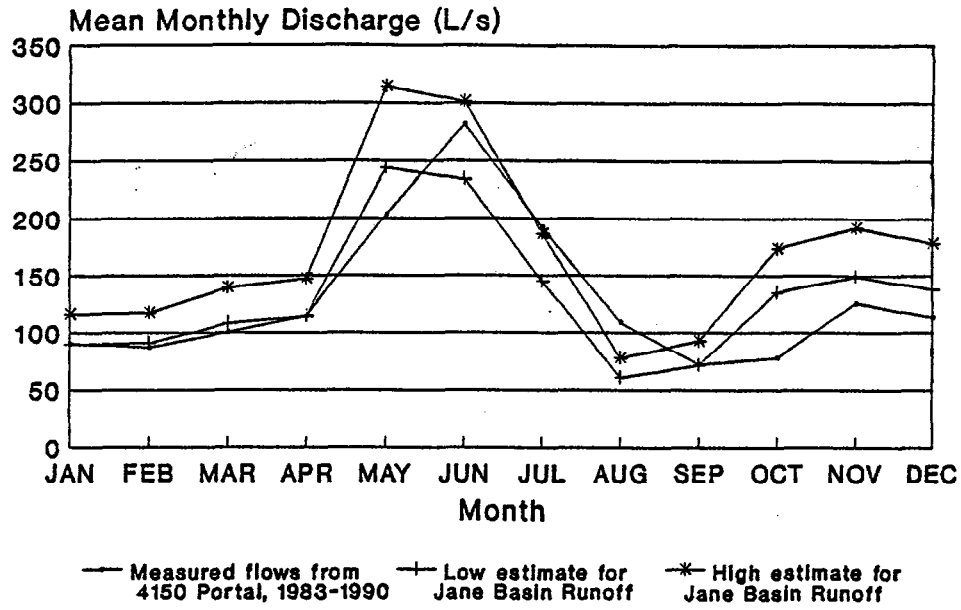


Figure 4.5

During the operational phase of the Britannia Mine, the flow regime of Britannia Creek at its mouth was probably significantly altered due to the operation of the hydroelectric development. This is no longer the case as present-day hydrology probably resembles natural hydrology fairly closely. The alterations of flow on the two tributaries of Mineral and Jane Creeks are relatively small compared to the runoff generated by Britannia Creek as a whole. Also, the dams remaining from the hydroelectric scheme have minimal influence on the flows of Britannia Creek at its mouth. The abstraction from Tunnel Dam, effectively an interbasin transfer to Thistle Creek, is also relatively small.

Within the catchment area of Furry Creek, the mine development has caused minor disturbance to the hydrology of Empress Creek and some neighbouring unnamed streams. One of the open pits in the Jane Basin straddles the divide between Britannia and Furry Creeks. Another two open pits No. 5 and Empress are located in the Empress Creek drainage and may act as enclosed drainage basins. The resulting reduction of catchment area of Furry Creek is small. As with Mineral Creek, the flow of Empress Creek and its neighbouring streams has been supplemented by the discharge of at least one portal. More information would be required to assess the impact of man's activities on the hydrological regimes of the Furry Creek tributaries that drain the mine area. At the mouth of Furry Creek, the alteration to the streamflows by the mine development is expected to be imperceptible.

#### **4.4 Water Quality**

##### **4.4.1 Sampling Program**

Two water quality sampling investigations were conducted by SRK to identify the major sources of contaminant loading to Britannia Creek. The two sampling periods represent extreme flow events: the September sampling during a low flow period and the December during a high flow period. Access to the site was limited during the fall period by landslides and resultant highway closures between Horseshoe Bay and Britannia Beach. Precipitation data for the Squamish Airport is provided in Table 4.1.

Measured flows at the mouth of Britannia Creek were 97 and 1770 L/s during the September and December sampling, respectively. For the determination of all potential sources of contaminated drainage, the high flow sampling period probably indicates more accurately potential drainage sources. The results from the December investigation will be presented in the following discussions with the results of the September investigation following in parentheses.

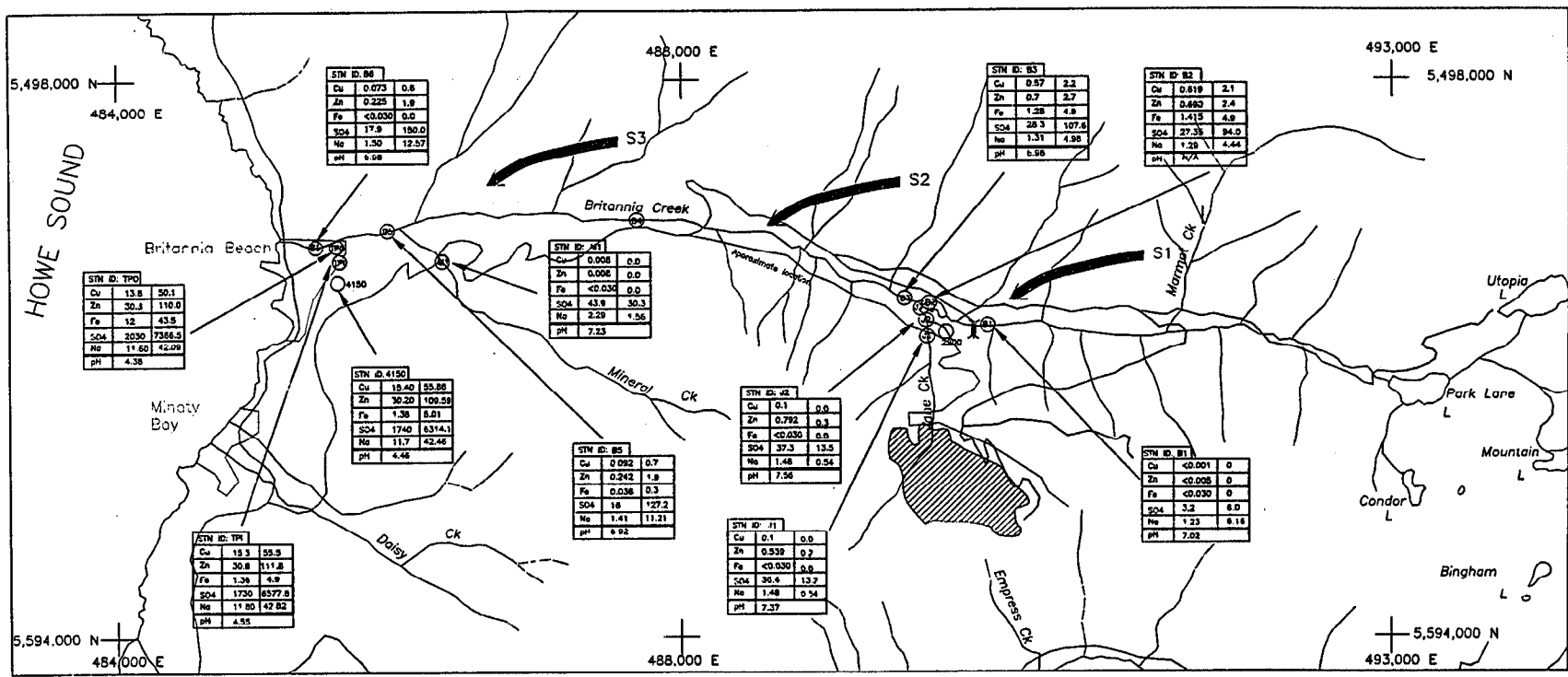
**TABLE 4.1**  
**MONTHLY PRECIPITATION AT**  
**SQUAMISH AIRPORT (mm)**

MONTH	1951-80 NORMAL	1990	% OF NORMAL
JAN	313.2	415.9	133
FEB	234.3	386.8	165
MAR	188.9	136.9	72
APR	149.1	67.1	45
MAY	77.2	56.5	73
JUN	68.5	118.2	173
JUL	52.3	34.0	65
AUG	73.0	51.5	71
SEPT	127.0	29.8	23
OCT	301.3	329.0	109
NOV	314.4	929.1	296
DEC	347.8	396.5	114
YEAR	2247.0	2951.3	131

Sampling sites were selected along Britannia Creek and on the contributing water courses such that source areas could be isolated and the loading contributions from specific areas could be calculated. The locations of the sampling sites are shown in Figure 4.6a and 4.6b. The results of the water quality sampling program are summarized and compared with historical data in Tables 4.2 through 4.7 and provided in detail in Appendix A. The water quality and waterflow data has been compiled in a Geographic Information System (GIS) and is displayed graphically in Figure 4.6a and 4.6b. Compilation of the data in this format allows graphical presentation of any individual parameter, and with input of historical data, comparison of trends with respect to place or time.

Britannia Creek was sampled at six locations. Background water quality, that is upstream of the influence of mining areas, was determined at site B1 at approximately 2200 ft elevation. During the September sampling road work was being done in this area and access to the north side of the creek was limited. In December, the Creek crossing at the 2200 level was washed out.

Sampling was conducted along the length of the creek to identify sources of loading along the Creek and specifically to address known and suspected sources:



**LEGEND**

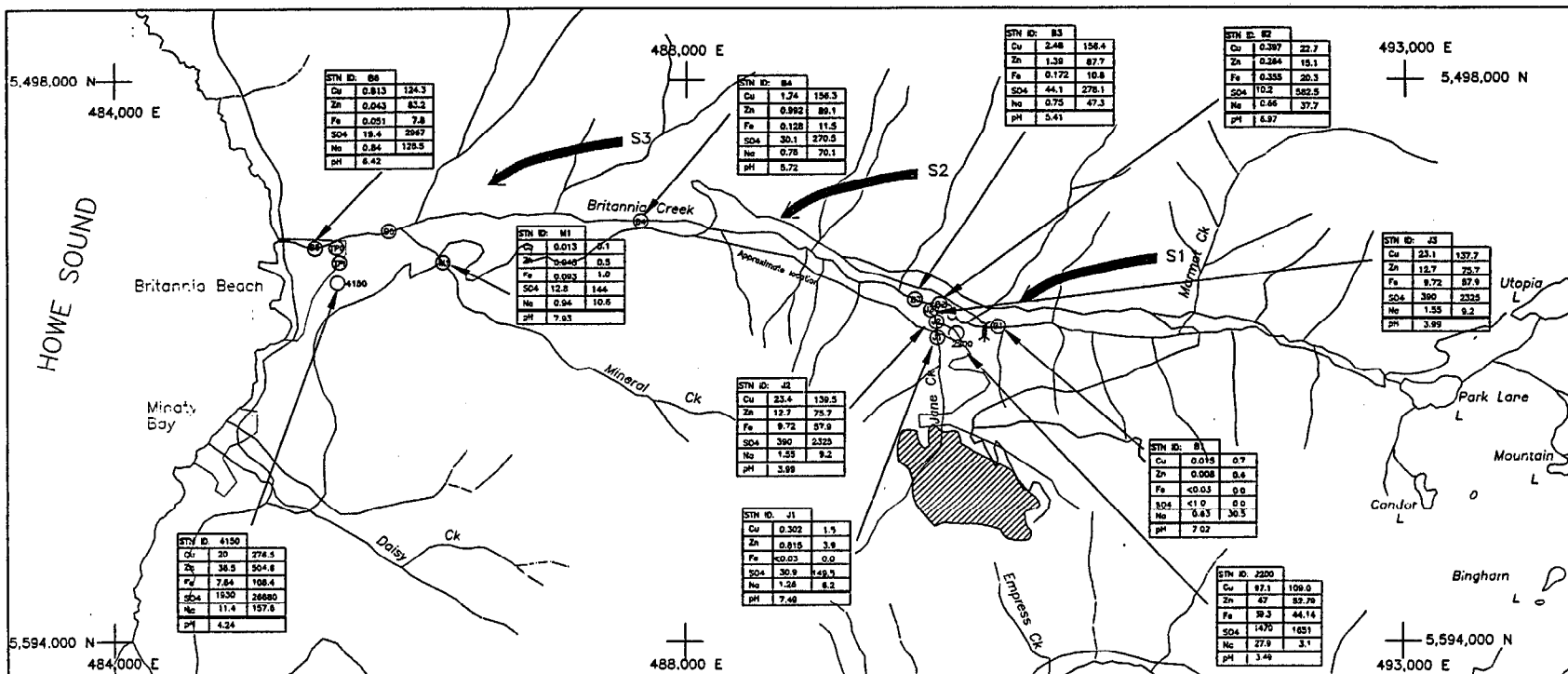
- Stream, definite
- Stream, indefinite
- Gravel Road
- Paved Road
- Trails
- Minesite
- Townsite
- Water Quality Monitoring Station

Parameter	Class mg/L	Loading kg/day
Cu	<0.001	0
Zn	<0.005	0
Fe	<0.030	0
SO4	3.2	0.0
Na	1.23	0.18
pH	7.02	

North American Datum 1987  
 Contour Interval: 100 Feet  
 Elevation in Feet above mean sea level  
 Unconsolidated Mineral Potential  
 and/or mineral resources inferred from 1:50,000  
 scale photometric and radiometric maps, 922,088  
 Station F1 located at highway crossing of Furry Creek.



Ministry of Energy, Mines and Petroleum Resources		Date: March, 1991
<b>BRITANNIA MINE</b> <b>WATER QUALITY RESULTS for SEPTEMBER</b>		File No: 66004
		4.6A
STEVEN ROBERTSON & KIRSTEN, Consulting Engineers		



STN ID: 86
Cu 0.813 124.3
Zn 0.043 83.2
Fe 0.051 7.8
SD4 18.4 2947
Na 0.84 126.5
pH 8.42

STN ID: 84
Cu 1.74 156.3
Zn 0.992 89.1
Fe 0.128 11.5
SD4 30.1 270.5
Na 0.78 70.1
pH 5.72

STN ID: 83
Cu 2.48 156.4
Zn 1.38 87.7
Fe 0.172 10.8
SD4 44.1 278.1
Na 0.75 47.3
pH 5.41

STN ID: 82
Cu 0.397 22.7
Zn 0.284 15.1
Fe 0.335 20.3
SD4 10.2 582.5
Na 0.66 37.7
pH 6.97

STN ID: M1
Cu 0.013 2.1
Zn <0.045 0.5
Fe 0.093 1.0
SD4 12.8 144
Na 0.94 10.5
pH 7.93

STN ID: J2
Cu 23.4 139.5
Zn 12.7 75.7
Fe 9.72 57.9
SD4 390 2323
Na 1.53 9.2
pH 3.99

STN ID: B1
Cu 0.015 0.7
Zn <0.008 0.6
Fe <0.03 0.0
SD4 <1.0 0.0
Na 9.63 30.5
pH 7.02

STN ID: J3
Cu 23.1 137.7
Zn 12.7 75.7
Fe 9.72 57.9
SD4 390 2323
Na 1.53 9.2
pH 3.99

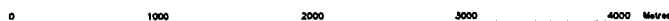
STN ID: 4150
Cu 20 278.5
Zn 36.5 504.8
Fe 7.84 108.4
SD4 1930 28820
Na 11.4 157.6
pH 4.34

STN ID: J1
Cu 0.302 1.5
Zn 0.815 3.9
Fe <0.03 0.0
SD4 30.9 149.5
Na 1.28 8.2
pH 7.49

STN ID: 2200
Cu 87.1 109.0
Zn 47 82.79
Fe 39.3 44.14
SD4 1470 1651
Na 27.9 3.1
pH 3.48

STN ID: J3		
Parameters	Conc. mg/L	Loading kg/day
Cu	23.1	137.7
Zn	12.7	75.7
Fe	9.72	57.9
SD4	390	2323
Na	1.53	9.2
pH	3.99	

North American Datum 1927  
 Contour Interval: 1000 Feet  
 Contours in Pink where there are local  
 drainage basins  
 Additional contours derived from 1:25,000  
 scale planimetry and aerial map 92E/068  
 Station F1 located at Highway crossing of Furry Creek.



Ministry of Energy, Mines and Petroleum Resources		File No. 66004
BRITANNIA MINE WATER QUALITY RESULTS for DECEMBER		March, 1991
		4.6B
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers		

TABLE 4.2 WATER QUALITY DATA, SEPTEMBER 1990

		Britannia Creek B1	Jane Creek J1	2200 Portal	Jane Creek J2	Britannia Creek B2	Britannia Creek B3	Mineral Creek	Britannia Creek B5	4100 Portal	Treatment Plant In	Treatment Plant Out	Britannia Creek B6
		7.02	7.37	4.29	7.56	NA	6.98	7.23	6.92	4.46	4.55	4.38	6.98
pH						NA							
Acidity	CaCO3	1	4	142	3	NA	15	2	3	266	254	242	2
Sulphate	SO4	3	36	559	37	27	28	44	16	1740	1730	2030	18
Dissolved Metals													
Copper	D-Cu	<0.001	0.100	13.5	0.100	0.620	0.570	0.008	0.092	15.4	15.3	13.8	0.073
Iron	D-Fe	<0.030	<0.030	3.82	<0.030	1.42	1.28	<0.030	0.036	1.38	1.36	12.0	<0.030
Sodium	D-Na	1.23	1.48	11.10	1.48	1.29	1.31	2.29	1.41	11.70	11.80	11.60	1.50
Zinc	D-Zn	<0.005	0.54	11.70	0.79	0.69	0.70	0.05	0.24	30.20	30.80	30.30	0.23
Total Metals													
Copper	T-Cu	<0.001	0.14	13.70	0.31	0.80	0.75	0.01	0.11	15.60	15.40	14.00	0.11
Iron	T-Fe	<0.030	<0.030	6.45	0.13	1.90	1.73	<0.030	0.13	3.64	4.13	13.40	0.13
Zinc	T-Zn	<0.005	0.559	11.7	0.858	0.714	0.728	0.048	0.242	30.2	32.4	30.3	0.232

Results are expressed as milligrams per litre except where indicated otherwise.  
 < = Less than detection limit

TABLE 4.3 WATER QUALITY DATA, DECEMBER 1990

		Britannia Creek B1	Jane Creek J1	2200 Portal	Jane Creek J2	Jane Creek J3	Britannia Creek B2	Britannia Creek B3	Britannia Creek B4	Mineral Creek M1	Britannia Creek B6	4100 Portal	Furry Creek
		7.02	7.49	3.49	3.99	3.97	6.97	5.41	5.72	7.93	6.42	4.24	8.36
pH													
Acidity	CaCO3	2.7	3	764	184	181	5.3	20	10.4	2	5	332	<1
Sulphate	SO4	<1.0	30.9	1470	390	391	10.2	44.1	30.1	12.8	19.4	1930	<1
Dissolved Metals													
Copper	D-Cu	0.015	0.302	97.1	23.4	23.1	0.397	2.48	1.74	0.013	0.813	20	0.011
Iron	D-Fe	<0.03	<0.03	39.3	9.72	8.46	0.355	0.172	0.128	0.093	0.051	7.84	<0.03
Sodium	D-Na	0.63	1.28	2.79	1.55	1.59	0.66	0.75	0.78	0.94	0.84	11.4	0.84
Zinc	D-Zn	0.008	0.815	47	12.7	12.7	0.264	1.39	0.992	0.043	0.544	36.5	0.006
Total Metals													
Copper	T-Cu	0.015	0.455	97.3	23.5	23.1	0.453	2.49	1.75	0.017	0.838	20.9	0.011
Iron	T-Fe	<0.03	0.042	42.6	10.5	9.13	0.414	1.02	0.702	0.114	0.212	9.32	0.033
Zinc	T-Zn	0.008	0.85	49.1	13	12.7	0.264	1.44	1.02	0.045	0.546	39.1	0.006

Results are expressed as milligrams per litre except where indicated otherwise.  
 < = Less than detection limit



The copper concentrations in the September sampling were similar to recent data as summarized by Goyette and Ferguson (1985), and shown in Table 4.4 and 4.5. The December sampling however showed high copper concentrations and flow from the 2200' portal, similar in loading to historical spring discharge data. Zinc concentrations were also elevated and are also of concern with respect to water quality.

TABLE 4.4  
PORTAL DRAINAGE

SAMPLING DATE	2200 LEVEL DISCHARGE		.4100 LEVEL DISCHARGE	
	Diss. Cu mg/L	Flow m <sup>3</sup> /d	Diss. Cu mg/L	Flow m <sup>3</sup> /d
June 21, 1983	51.4	6000	31.8	--
December 6, 1983	12.3	39	18.3	--
March 21, 1984	43	6600	14.0	--
May 24, 1984	23.8	2530	19.7	--
September 27, 1984	5.12	--	--	--
September 21, 1990	13.5	--	15.4	3628
December 6, 1990	97.1	1123	20.0	13,824

Total and dissolved metal (Cu, Zn) concentrations are similar, indicating most of the metal load is dissolved. Total iron values, however, are higher than the dissolved in Britannia Creek at sites B3 and B4; where Jane Creek joins Britannia. Metal precipitation was evident along the creek in this area, and pH values are higher than in the upstream drainage, ranging from 5.4 to 7.0. The total iron load is probably due in part to physical mobilization of some of the iron oxide and hydroxide precipitates. Sodium concentrations were typically less than 1.5 mg/L but were elevated in the drainage from the portals (up to 11.4 mg/L from the 4100 level) and slightly elevated (2.3 mg/L) in Mineral Creek suggesting drainage from the 2700 portal was contributing Britannia Creek in December.

In September, total and dissolved copper values in the treatment plant influent were 15 mg/L, approaching the lower design limit for this plant. Some of the flow through the treatment plant was diverted into the settling pond below the plant, at various stages along the course of the cementation launders. Copper concentrations in the treatment plant effluent were similar at 14 mg/L. Iron concentrations however increased from 1.4 to 12 mg/L Fe (diss.) from influent to effluent, indicating cementation reactions were occurring.

The initial flushing of oxidation zones after a prolonged dry period tends to result in high concentrations and metal loadings due to stored oxidation products. Sampling was not possible at the Britannia site in the initial weeks of heavy rainfall due to road closures along the Squamish highway which limited access from Vancouver. The December sampling trip was conducted after approximately two months of rainfall.

**TABLE 4.5**  
**COMPARISON OF BRITANNIA CREEK AND MAJOR DISCHARGES**  
**1972, 1976, 1980, 1983/84, 1990**

Parameter	Year	Flow	pH	SO <sub>4</sub>	T.Cu	T.Zn	T.Fe	D.Cu	D.Zn	D.Fe
Creek U/S	72	60.1	6.7	1.6	0.047	0.03	0.07	-	-	-
	80		-	-	0.03	0.02	<0.01	-	-	-
	83/84		6.6	2.0	-	-	-	0.02	0.02	0.03
2200' Minewater	72	2.6	3.0	1598	28.4	62.4	223	-	-	-
	80		-	-	0.43	0.47	0.66	-	-	-
	83/84		3.0	757	-	-	-	27.1	20.1	26.3
	Dec 6	3.8	-	-	-	-	-	12.3	8.21	-
	Mar 21	3.0	-	-	-	-	-	4.3	31.6	-
Creek Below 2200'	72	39.1	4.4	123	2.1	4.5	26.8	-	-	-
	80		-	-	0.17	0.17	0.06	-	-	-
	83/84		6.3	10.3	-	-	-	0.36	0.28	0.05
	Dec	-	7.2	-	-	-	-	0.18	0.21	-
Mineral Creek	72	0.70	7.7	1229	0.09	1.3	1.2	-	-	-
	80		-	-	0.02	0.04	-	-	-	-
	83/84		6.5	9.2	-	-	-	0.03	0.06	0.04
4100' Minewater	72	5.1	3.0	2197	20	66.1	104.8	-	-	-
	80	-	3.7	-	-	-	-	18.3	28.1	-
	83/84		3.6	-	-	-	-	1.4	0.26	-
Creek at Mouth	72	111.4	4.0	235	2.6	7.1	9.3	-	-	-
	76	-	-	-	-	-	-	1.6	37	40
	83/84		6.5	11.7	-	-	-	0.14	0.15	0.16
	Dec 6		7.6	-	-	-	-	0.03	0.13	-

\* units are mg/l except pH - pH units  
T. = Total (mg/l)  
D. = Dissolved (mg/l)

It would be expected that dissolved metal concentrations would be somewhat less than in the initial flush. As shown graphically in Figures 4.7 and 4.8 comparing the September and December sampling, the December sampling program demonstrated high concentrations associated with high flow events, corresponding with results determined in previous sampling programs (Goyette and Ferguson, 1985).

The variation in flow rates and concentration from each of the drainage sources evident in this and previous surveys indicates the need to address seasonal fluctuations in drainage water quality. Such variations must be understood for the design of any treatment process. The objective of the sampling program described herein was to identify and characterize the major sources of loading to Britannia Creek. In order to accurately quantify the total annual contributions of each source, and identify sources which may not have been distinct during these periods, a more extensive survey on at least a monthly basis would be required over at least a one year period, including two spring runoff periods.

#### 4.4.5 Contaminant Loadings

Flow data were used to calculate chemical mass loadings for each of the sampling programs. These results are summarized in Table 4.6 and 4.7 and shown graphically for key parameters in Figure 4.6a and 4.6b. The water quality sampling data have been incorporated into a Geographic Information System (GIS). Detailed results of the loading calculations are presented in Appendix A.

These results indicate the major source of contaminant loading is the drainage from the underground workings. In December, 88 % of the total copper loading in Britannia Creek was contributed by the 2200 portal drainage. This drainage also had the highest copper concentration at 97 mg/L, and the flow from the portal represented less than 1 % of the total flow of Britannia Creek at the mouth. It must be recognized however that flow from the underground workings which discharges at the 4150 level is discharged to Howe Sound either directly, or through the treatment plant.

Flow was not issuing at a measurable rate from the 2200 portal during the September survey however the drainage had low pH (4.29) with elevated acidity at 142 mg/L. Dissolved metal concentrations were; 13.5 mg/L Cu, 3.8 mg/L Fe, and 11.7 mg/L Zn. It would be expected however that, when flowing, this drainage would contribute significantly to the metal loadings in Jane and Britannia Creeks.

The total copper loading to Howe Sound from Britannia Creek calculated for each of the two sampling periods was 0.9 kg/day (Sept.) and 128 kg/day (Dec.). Discharge directly to Howe Sound from the 4100 level and treatment plant discharge was 51 kg/d Cu (Sept.) and 289 kg/d (Dec. bypass of the treatment plant due to high flows). These data are summarized in Table 4.8.

### Britannia Mine Site Sulphate And Acidity Levels, Sept & Dec

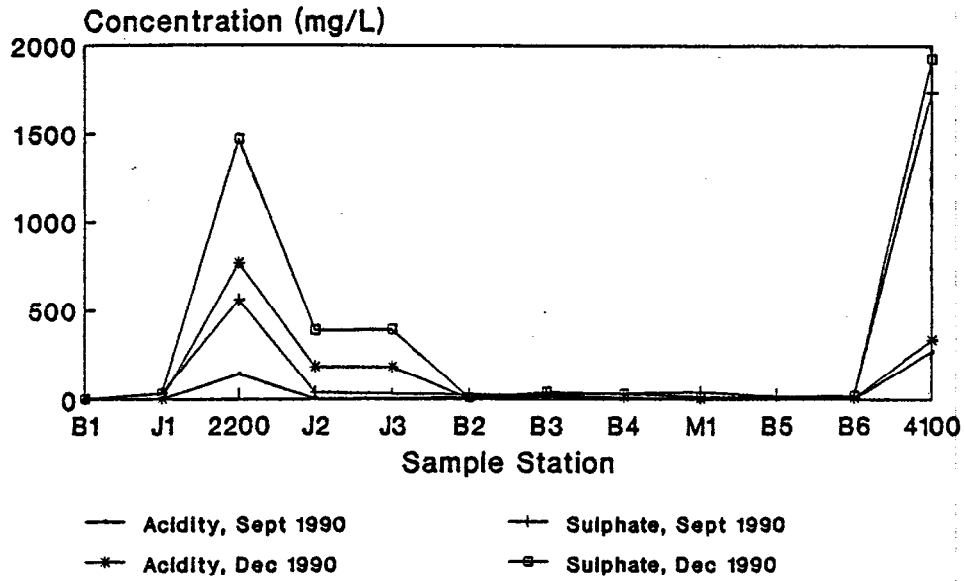


Figure 4.7

### Britannia Mine Site Copper and Zinc Concentrations

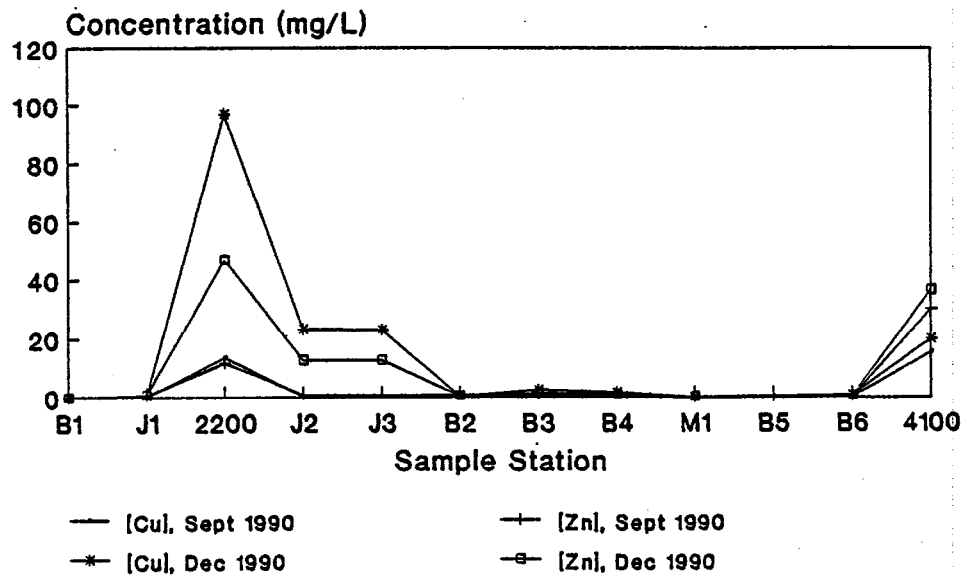


Figure 4.8

TABLE 4.6 CALCULATED LOADINGS, SEPTEMBER 1990

	Britannia Creek B1	Jane Creek J1	2200 Portal	Jane Creek J2	Britannia Creek B2	Britannia Creek B3	Mineral Creek	Britannia Creek B5	4100 Portal	Treatment Plant In	Treatment Plant Out	Britannia Creek B6
	CALC											
Estimated Flow L/s	58	4.2	0	4.2	39.8	44	8	92	42	42	42	97
pH	7.02	7.37	4.29	7.56	NA	6.98	7.23	6.92	4.46	4.55	4.38	6.98
Sulphate SO4	16.0	13.2	0.0	13.5	94.0	107.6	30.3	127.2	6314.1	6277.8	7366.5	150.0
Dissolved Metals												
Copper D-Cu	0.00	0.04	0.00	0.04	2.1	2.2	0.01	0.73	55.9	55.5	50.1	0.61
Iron D-Fe	0.00	0.00	0.00	0.00	4.9	4.9	0.00	0.29	5.0	4.9	43.5	0.00
Sodium D-Na	6.2	0.54	0.00	0.54	4.4	5.0	1.6	11.2	42.5	42.8	42.1	12.6
Zinc D-Zn	0.00	0.20	0.00	0.29	2.4	2.7	0.03	1.9	110	112	110	1.9
Total Metals												
Copper T-Cu	0.00	0.05	0.00	0.11	2.7	2.9	0.01	0.87	56.6	55.9	50.8	0.92
Iron T-Fe	0.00	0.00	0.00	0.05	6.5	6.6	0.00	1.0	13.2	15.0	48.6	1.1
Zinc T-Zn	0.00	0.20	0.00	0.31	2.5	2.8	0.03	1.9	110	118	110	1.9

Results are expressed as kilograms per day except where indicated otherwise.  
 < = Less than detection limit

TABLE 4.7 CALCULATED LOADINGS, DECEMBER 1990

	Britannia Creek B1	Jane Creek J1	2200 Portal	Jane Creek J2	Jane Creek J3	Britannia Creek B2	Britannia Creek B3	Britannia Creek B4	Mineral Creek M1	Britannia Creek B6	4100 Portal	Furry Creek
Estimated Flow L/s	560	56	13	69	69	661	730	1040	130	1770	160	NA
pH	7.02	7.49	3.49	3.99	3.97	6.97	5.41	5.72	7.93	6.42	4.24	8.36
Sulphate SO4	0.0	150	1651	2325	2331	583	2781	2705	144	2967	26680	NA
Dissolved Metals												
Copper D-Cu	0.73	1.5	109	140	138	23	156	156	0.15	124	276	NA
Iron D-Fe	0.00	0.00	44.1	57.9	50.4	20.3	10.8	11.5	1.0	7.8	108	NA
Sodium D-Na	30.5	6.2	3.1	9.2	9.5	37.7	47.3	70.1	10.6	128	158	NA
Zinc D-Zn	0.39	3.9	52.8	75.7	75.7	15.1	87.7	89.1	0.48	83.2	505	NA
Total Metals												
Copper T-Cu	0.73	2.20	109	140	138	25.9	157	157	0.2	128	289	NA
Iron T-Fe	0.00	0.20	47.8	62.6	54.4	23.6	64.3	63.1	1.3	32.4	129	NA
Zinc T-Zn	0.39	4.1	55.1	77.5	75.7	15.1	90.8	91.7	0.51	83.5	541	NA

Results are expressed as kilograms per day except where indicated otherwise.  
 < = Less than detection limit

**TABLE 4.8  
CALCULATED LOADINGS TO HOWE SOUND**

	BRITANNIA CREEK	4100 LEVEL	
		Treatment Plant Discharge	Direct Discharge
<b>Total Copper Loading (kg/day)</b>			
September	0.9	51	--
December	128	--	289
<b>Total Zinc Loading (kg/day)</b>			
September	1.9	110	---
December	84	---	541

Zinc loadings to Howe Sound are significant, and appear to be higher than in previous surveys. Copper loadings to Britannia Creek are less than historical values (during operations) however are similar to, or slightly higher than, copper loadings in the early 1980s (Section 3.1.3).

The relatively high concentrations and loadings from the portal drainage samples in the December survey after prolonged high precipitation indicates there is a significant store of dissolved or soluble oxidation products in the underground workings. A high precipitation event tends to flush areas which normally remain "dry" and store oxidation products. It would be expected that the initial drainage contained higher concentrations than this sampling, after approximately two months of high precipitation. At a conservative estimate therefore, the combined drainage from Britannia Creek and the 4100 level contributed over 24 tonnes of copper to Howe Sound during the two month period preceding the sampling. The stored oxidation products would tend to provide a "slug" of contaminants in runoff and drainage immediately following precipitation periods.

Metal precipitate formation was evident along Jane and Britannia Creeks and drainage channels from the portals. Iron staining was evident along the upper reaches of Britannia Creek to approximately station B4. As shown in Photograph 4.8 at station B3, there was apparently poor mixing of the two streams, and the characteristic orange-yellow precipitate was evident along the banks but a white precipitate, possibly sulphate salts, dominated in the centre of the stream. These metal oxide and hydroxide precipitates represent a store of potentially mobile products which could be flushed during high flow periods, or leached with acidic flow and reported as a "slug" of elevated dissolved or total metal contamination.

Three "other" sources of drainage contributing to Britannia Creek were identified from the hydrological balance. Based on the chemical mass loading calculated for the Creek, approximate metal and concentrations were calculated for each of these drainage sources.

Source S1 (see Section 4.4.3) was the most contaminated source contributing 101 L/s of flow with dissolved metal loadings of 22 kg/day Cu, 20.3 kg/day Fe, 14.7 kg/day Zn, and 583 kg/day  $\text{SO}_4^{2-}$  (2.1 kg/d Cu, 4.9 kg/d Fe, 2.4 kg/d Zn and 78 kg/d  $\text{SO}_4^{2-}$  in the September sampling despite no apparent increase in flow). These increases in loading were noted between B1 and B2, upstream of the confluence of Jane Creek with Britannia Creek.

Possible sources of contaminants at this level include:

- drainage from the waste rock material around the old core shack and powerhouse;
- waste rock dumps on which these facilities were constructed;
- seepage from the 2200 level portal or above directly into Britannia Creek, or through the waste at the 2200 level into the Creek;
- acid generation below the 2200 level site and above or within the debris accumulated at Tunnel Dam.

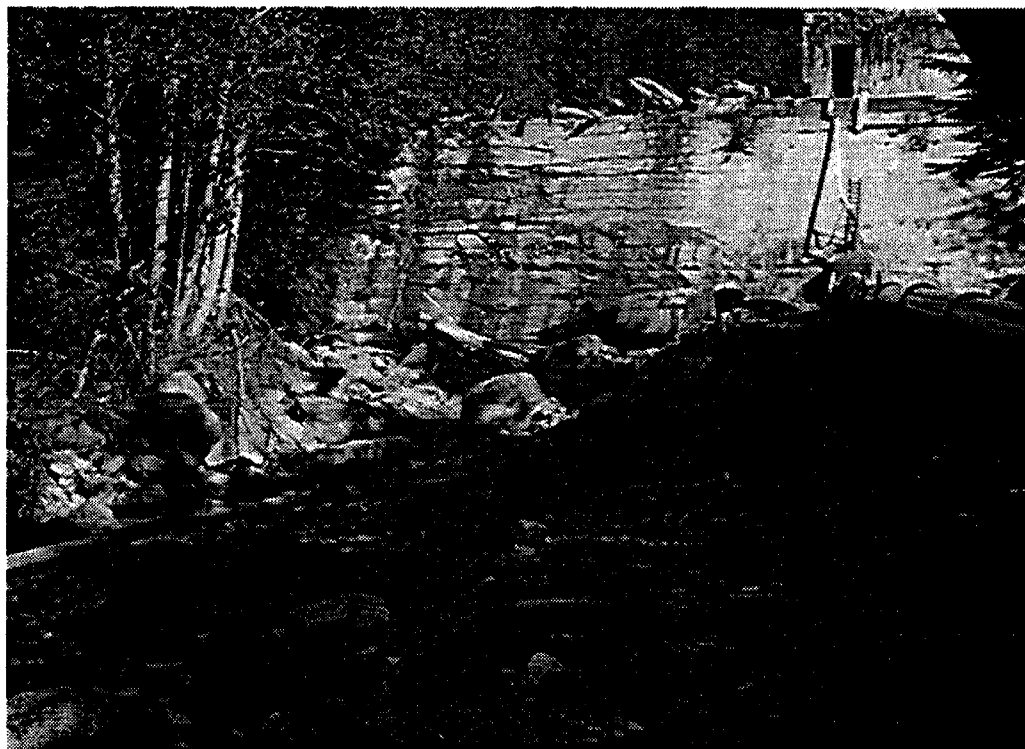
The flow rate and concentrations at this site were estimated from sites B3 and J2.

Source S2 appeared to be "fresh" water and did not contribute contaminants to the Creek. Solution pH values increased slightly from Station B3 above the source to B4 below the source, and sulphate decreased slightly.

Source S3 did not appear to be severely contaminated although sulphate loading increased slightly from B4 to B6. Solution pH increased from 5.7 to 6.4 and dissolved Cu and Fe loadings decreased slightly, possibly due to precipitation of metals at more alkaline pH values. Sulphate was however contributed from this source; approximately 120 kg/d (23 kg/d).



**Photograph 4.8** Britannia Creek  
Sampling Station B3



**Photograph 4.9** Tunnel Dam above  
Station B2, B3 and J3



## **5.0 ENVIRONMENTAL CONSIDERATIONS**

### **5.1 Physical/Geotechnical**

Current and future physical impacts at the Britannia Mine will occur primarily in the Jane Basin. These will consist of subsidence of the area above the underground workings, failures on the pit walls, and ravelling of the waste dumps. A glory hole is present in the bottom of the Fairview pit and the area below the East Bluff pit is the zone where most of the underground mining was conducted, therefore continued subsidence is likely in these areas, especially as underground timbers continue to rot. A scarp appears to be present above the Jane pit indicating the possibility of ongoing subsidence in this area. This should be verified by a ground traverse. The limits of subsidence to date cannot be accurately estimated because only high level airphotos have been examined.

Portions of the pit walls are still actively eroding and because subsidence is continuing it is likely that these areas will not stabilize within the next decade. Trees are not likely to establish on these slopes because of the lack of soil and active slope ravelling. Vegetation is beginning to establish on the waste dumps and should help to reduce future erosion.

Other areas where physical impacts may occur are the waste dump at the 2200 level portal and the roads in the Jane Basin. Erosion of the waste dump may occur as a result of a flood event or change in the creek channel of Britannia Creek. Any eroded material would be deposited further downstream. Some of the roads in the pit area are eroding as the fill ravel and the ditches become blocked. This is a relatively minor impact relative to the others described above.

### **5.2 Drainage Water**

#### **5.2.1 Sources of Water Entry**

The majority of the flow into the Britannia underground workings is believed to discharge at the 4150 portal. Smaller amounts of mine water are known to flow from at least three other portals (namely, 2200 level, 2700 level and at least one within the Furry Creek drainage).

A detailed record of flow measurements has been maintained of the mine drainage emerging from the 4150 portal. The data reveal a long-term average flow rate of about 130 L/s. Only spot measurements or visual estimates are available for assessing the discharges from the other three portals. Based on this sparse data set, the combined flow from the three portals probably amounts to something less than 10 percent of the discharge from the 4150 portal.

The primary source of water entry to the underground workings is considered to be the area known as Jane Basin. The effects of surface and underground workings in this general area have combined to produce conditions that maximize infiltration to the underground. The bottom of the open pits in Jane Basin may be sufficiently permeable that all runoff flowing into the open pits subsequently seeps to the underground workings.

Indirect evidence for the above was found by comparing the estimated runoff into the open pits with the measured flows out of the 4150 portal. The total catchment area commanded by the open pits is about 1.2 km<sup>2</sup>. For this size area, the mean annual runoff was estimated to lie within the range 130 to 170 L/s, or values very similar to the measured mine drainage flow. As shown on Figure 4.1 this comparison is also valid on a monthly basis.

### **5.2.2 Sources of Loading**

The major sources of contaminant loading to Britannia Creek were identified by review of historical data, discussion with those familiar with the area, observation of the site and metal staining along active and dry creek beds and seeps, and two site sampling surveys. These sources are discussed in detail in Sections 3.1 (historical) and 4.3 (current).

Drainage through the open pits, exposed surface workings, underground workings and water courses is contaminated with acidity and metals from three major sources:

- i) oxidation products generated by continuing acid generation and metal leaching reactions;
- ii) flushing of stored oxidation products;
- iii) flushing and dissolution of salts which precipitated as a result of dilution/neutralization reactions along the drainage path;

The relative contribution of each is difficult to quantify with the existing data however:

- i) it would appear that the rate or extent of oxidation has decreased since the mine was operational by reduced concentrations in the portal drainage. This is probably due both to a reduction of disturbed areas which reduces exposed fresh mineral surfaces for oxidation, and, a reduction in fine particles which would contribute to elevated total metal levels during operation. The consumption of available sulphides, and therefore long-term potential for contaminant generation cannot be predicted from available data;

- ii) there is a considerable store of oxidized products within the underground workings based on the elevated concentrations and loading after an extended precipitation period (fall of 1990);
- iii) iron and copper precipitates were evident along both Jane and Britannia Creeks, but not Mineral Creek in the area of the road crossing;
- iv) precipitation of metal oxides and hydroxides along drainage flowpaths could serve to coat the flowpath and limit flushing of potentially oxidizing areas. Such precipitation was evident along the 2200 portal drainage channel.

The major sources of metal and acidity loading from the mine workings which were identified in this survey include:

- i) Extensively acidized open pit and glory hole walls which drain primarily into the underground workings;
- ii) 2200 portal drainage;
- iii) 4100 portal drainage.

Suspected or potential sources of contaminated drainage from the mine workings which were not sampled or not contributing significant loadings during these sampling periods include:

- i) drainage from the 2700 portal into Mineral Creek;
- ii) drainage from the 1800 portal into the Furry Creek drainage basin.

Sources of loading from the wastes which were not specifically identified from seeps, but appeared to contribute contaminants currently or historically;

- i) waste dumps at the East Bluff pit area. Waste in the mill area (Beach level) may be generating contaminants but would probably drain directly by groundwater flow into Howe Sound and not be apparent in sampling of Britannia Creek;
- ii) mine waste at the 2200 level including rock, timbers and structural material at the 2200' level on which the old powerhouse, core shack and road are located;

- iii) landslides were not specifically identified and major seeps were not evident from areas of slide debris in this program. Historical records however suggest that at one time surface drainage from the Jane Basin area contained contaminants from the slide debris.

Potential sources of contaminants from facilities and wastes have been identified from a review of the site, and understanding of geochemistry. These sites are classified as "potential" as they were either not:

- i) contributing drainage to Britannia Creek at the time of the study, or were beyond the scope of this investigation, i.e. were not in the Britannia Creek watershed, or
- ii) contributing to Howe Sound directly or requiring detailed hydrogeological investigations.

These areas should be addressed if further investigations are conducted at the Britannia site, and include.

- i) settling ponds and facilities at the former water treatment plant at the 2100 level. It would be expected that drainage from this area would discharge into the Creek near site B4. From the results of the mass loading calculations for the Creek, it did not appear to contribute contaminants from surface or subsurface runoff at the time of sampling;
- ii) settling pond at the Beach level treatment plant;
- iii) pond at the 4150 level which was apparently an emergency storage or tailings pond, receiving contributions from the mill and the 2200 level cementation plant. Surface water from the pond would decant into the treatment plant;
- iv) tailings deposition on the beach near the mouth of Britannia Creek (unconfirmed);
- v) portal drainage to the Furry Creek drainage basin.

## 6.0 REMEDIATION OPTIONS

There are three approaches to the control of acid rock drainage:

- i) control of acid generation reactions;
- ii) control of migration of contaminants;
- iii) collection and treatment of acid drainage.

Control of acid generation reactions is the most desirable approach: preventing the onset of rapid acid generation by early identification and control of potentially acid generating materials. Once the acid generation reactions are established the approach must be to control the movement of the oxidation products and minimize the discharge of these products to the receiving environment. In some cases this can be accomplished by the use of covers and seals, and diversion of surface and groundwater to minimize the movement of these contaminants. Alternatively, for some sites the most practical alternative is to collect the contaminated drainage for treatment prior to discharge. These three approaches are not exclusive however, and a combination of these measures is generally the most effective and economically acceptable solution.

The data contained within this review of the Britannia site indicate that rapid acid generation has been established both on surface and within the underground workings throughout the site. Control of further acid generation may be feasible in some areas, however there appears to be a significant store of oxidation products. The approach to remediation of the site is therefore seen to be a combination of the latter two approaches. Where possible, the migration of contaminants would be limited by controlling the flushing and infiltration of surface water by covers or surface and ground water diversions. It is recognized that there is a significant store of oxidation products contained within the surface waste and the surface and underground workings, and that in the short term at least some form of treatment may be required to produce water quality suitable for discharge. If physical control measures are adopted for control of surface water, treatment may also be required during construction and implementation of these measures. Any surface disturbance in areas of oxidation or disturbed material may result in the release of both total and dissolved metal values, deleterious to water quality.

The possible approaches to control and remediation of the Britannia site are described in the following sections and summarized in Table 6.1. This table provides a general discussion of the approach, advantages and disadvantages and relative costs of each remediation option considered independently. It is recommended however that a combination of control measures is considered for the Britannia site, as described in Section 6.0.

**TABLE 6.1  
PROPOSED REMEDIATION OPTIONS FOR BRITANNIA MINE**

Source	Remediation *	Advantages	Disadvantages	Further Studies
1. Open pits and gloryholes	• divert surface water	• reduce flushing surface area, total flow and loading into the underground workings	• lower available dilution	• hydrology • ground conditions • detailed topography
	• cover exposed walls	• reduce total exposed surface for flushing and limit further acid generation	• not known if this is key source • unstable, steep slopes	• detailed topography • slope/dump stability
2. Underground workings	• flood workings	• limit further acid generation	• flushing of stored products • net positive water balance therefore discharge ultimately required • locating and sealing all openings • control of discharge rate and total volume	• hydrology • locate openings to surface • ground conditions • identify specific sources of acid generation and stored products • available storage volume
	• flooding with alkaline solution	• complexation and precipitation of stored oxidation products • limitation of further acid generation	• quantity of alkali required • potential for subsequent remobilization of precipitate	• extent of underground workings • quantity and nature of stored products • required pH for removal of metals
	• partial flooding	• provide storage of acid drainage prior to treatment plant	• as above • enhanced flushing in early years and higher contaminant loadings	• as above
	• divert surface runoff from points of entry	• reduce total flow and contaminant loading	• lower available dilution	• hydrology • ground conditions • detailed topography
3. Waste dumps	• covering and surface water diversion with ditching	• reduce infiltration and migration of contaminants • lower volume of water to be treated	• locations and stored volumes in waste dumps not known • access to dumps (location and slopes) • stability of ground, dumps • cost - depending on location	• definition of location and volumes in dumps • evaluation of contribution to contaminant loadings in Creek • sources and available quantities of cover materials • dump stability studies
	• remove and dispose in pits below flood elevation	• remove a possible source of contaminated drainage • improve physical stability of area	• increased loading during rehandling • ability to store wastes below flood elevation • materials handling (poor access)	• definition of waste volumes • storage volume available

Source	Remediation *	Advantages	Disadvantages	Further Studies
4. Tailings	<ul style="list-style-type: none"> <li>• installation of covers and diversion of surface water</li> </ul>	<ul style="list-style-type: none"> <li>• reduce flushing and migration of contaminants</li> </ul>	<ul style="list-style-type: none"> <li>• availability of cover materials</li> </ul>	<ul style="list-style-type: none"> <li>• volume of material at all disposal locations</li> <li>• physical stability of impoundment</li> <li>• chemical stability of tailings in each location (are tailings a problem?)</li> <li>• sources and quantities of available cover materials</li> </ul>
	<ul style="list-style-type: none"> <li>• removal to underwater storage either in Howe Sound or in underground workings</li> </ul>	<ul style="list-style-type: none"> <li>• remove possible source of contaminants</li> <li>• improve physical stability in emergency pond area</li> </ul>	<ul style="list-style-type: none"> <li>• release of stored products</li> <li>• cleanup required of current storage area</li> <li>• locating appropriate storage location</li> </ul>	<ul style="list-style-type: none"> <li>• alternate disposal areas</li> <li>• volume of material</li> <li>• potential for water cover</li> <li>• predictions of tailings deposition patterns for submarine disposal</li> </ul>
5. Drainage Water (overview for site specific discussions see Section 6.2)	<ul style="list-style-type: none"> <li>• diversion of fresh water from points of entry</li> </ul>	<ul style="list-style-type: none"> <li>• reduction in total volume to be treated</li> </ul>	<ul style="list-style-type: none"> <li>• construction and maintenance of diversion ditches (slope instability), leakage</li> <li>• considerable precipitation occurs as snow - coping with snow pack</li> <li>• limitations of topography</li> </ul>	<ul style="list-style-type: none"> <li>• hydrology</li> <li>• ground conditions</li> </ul>
	<ul style="list-style-type: none"> <li>• collection and rerouting from surface workings or site facilities</li> </ul>	<ul style="list-style-type: none"> <li>• centralization of contaminated drainage</li> </ul>	<ul style="list-style-type: none"> <li>• construction and maintenance</li> <li>• identification of sources</li> </ul>	<ul style="list-style-type: none"> <li>• water quality monitoring</li> <li>• pumping and piping requirements</li> </ul>
	<ul style="list-style-type: none"> <li>• internal diversions to (re)direct all discharge to 4100 level</li> </ul>	<ul style="list-style-type: none"> <li>• prevention of discharge to Britannia and Mineral Creeks</li> </ul>	<ul style="list-style-type: none"> <li>• identification all surface openings and internal diversions</li> <li>• potential physical instability of underground workings</li> </ul>	<ul style="list-style-type: none"> <li>• hydrological and geotechnical</li> <li>• extent of workings</li> </ul>
	<ul style="list-style-type: none"> <li>• collection and treatment</li> </ul>	<ul style="list-style-type: none"> <li>• control of discharge water quality</li> <li>• removal of contaminants</li> </ul>	<ul style="list-style-type: none"> <li>• long term maintenance and operation requirements and costs</li> <li>• sludge stability</li> <li>• sludge disposal</li> <li>• collection system and upstream surge control</li> </ul>	<ul style="list-style-type: none"> <li>• identification of collection points for all contaminated drainage</li> <li>• upstream surge pond requirements and costs</li> <li>• more detailed influent hydraulic and chemical loading data</li> </ul>

\* each of these remediation options is evaluated independently. It is recommended that a combination of measures may prove most effective, as described in Section 6.0.

At this time it appears that the shaded items should be pursued in the near future.

## **6.1 Water Management - Diversion**

### **6.1.1 Pits**

There are two options for reducing water flow over the pit surfaces:

- i) diversion of runoff around the pits, and,
- ii) placing covers to promote runoff and reduce flushing of pit walls from direct precipitation. There is potential for diverting runoff around the east side of the East Bluff pit and around the west side of the East Bluff and Jane pits. Reducing infiltration to the pits through the use of covers is not recommended because ongoing subsidence would destroy the integrity of the covers. Furthermore, the topography and presence of glory holes would make any cover very difficult to place.

Location of potential diversion ditches and resultant catchments are shown on Figure 6.1 and can be compared to the current catchment areas shown in Figure 4.3. These diversions could reduce the catchment which drains into the underground mine from 1.2 km<sup>2</sup> by a maximum of 0.15 and 0.55 km<sup>2</sup>, respectively. Factors such as the feasibility of ditch location and seepage under the ditches through talus will reduce the efficiency of the ditches.

On the east side diversion a cost savings in ditch construction can be achieved by upgrading the roads so that the upstream edge becomes the diversion ditch, however the diverted area would be only about 50 percent of that area diverted if a ditch was located near the pit crest. It is difficult to determine the best location for a diversion on the west side of the East Bluff pit because the available topographic maps are not sufficiently accurate and the distribution of talus, mine waste, and slide debris influence the practicality of ditch location. It may be necessary to construct lined ditches to prevent losses into talus and subsidence cracks.

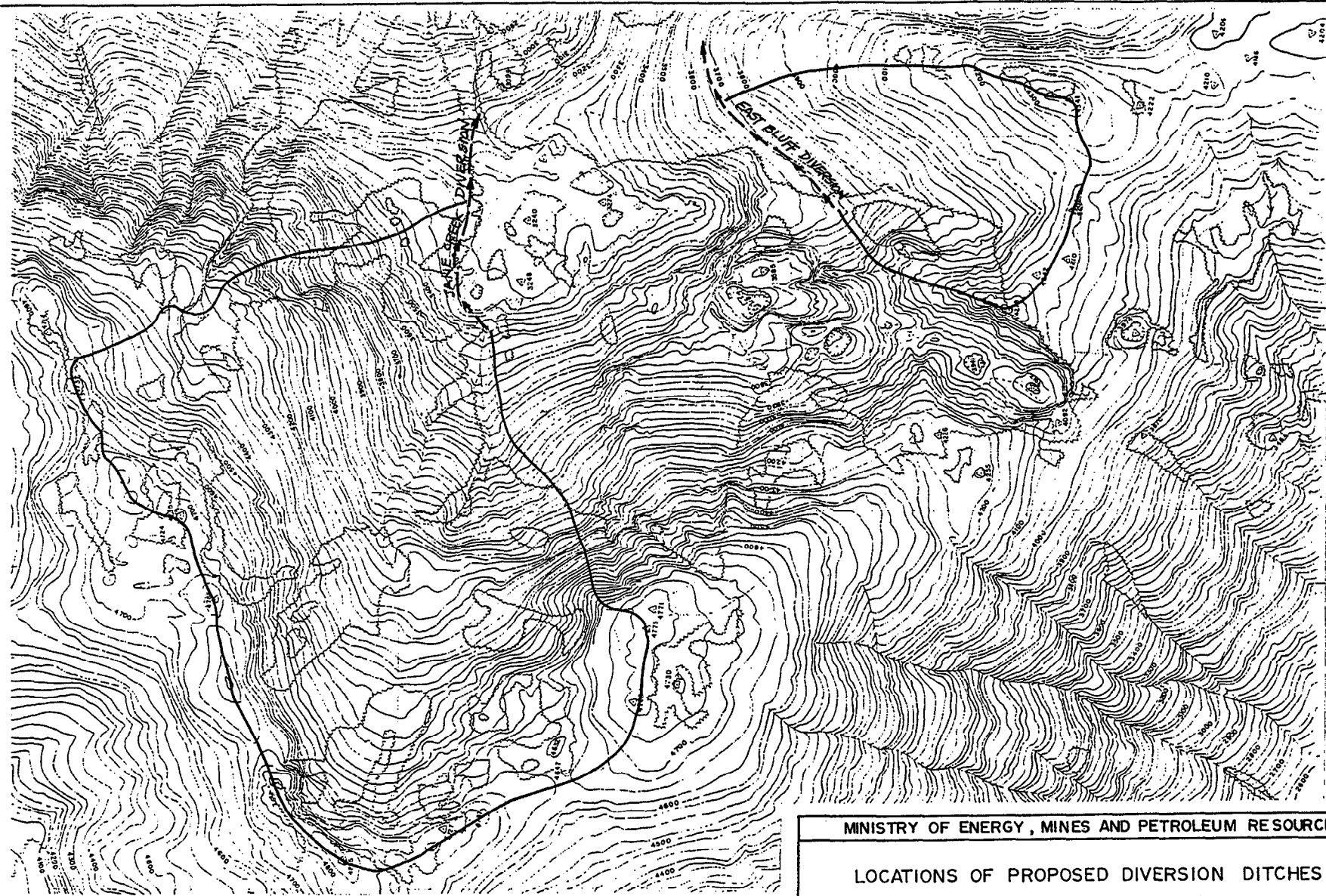
The Empress pit lies in the Furry Creek watershed and is not part of the scope of work for this project. As noted in Section 4.2.1, the Empress pit may be hydraulically connected to the underground mine.

### **6.1.2 Underground Workings**

There are two options for limiting the flow of water through the underground workings. The options include:

- i) divert runoff from the points of entry, and;





**LEGEND**

→ → →  
APPROX. LOCATION  
OF PROPOSED  
DIVERSION DITCH

—————  
CATCHMENT  
BOUNDARY FOR  
AREA COMMANDED  
BY DIVERSION  
DITCHES

MINISTRY OF ENERGY, MINES AND PETROLEUM RESOURCES  LOCATIONS OF PROPOSED DIVERSION DITCHES	DATE MARCH 1991
	PROJ. NO. 66004
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	APPROVED
	FIG. NO. 6.1

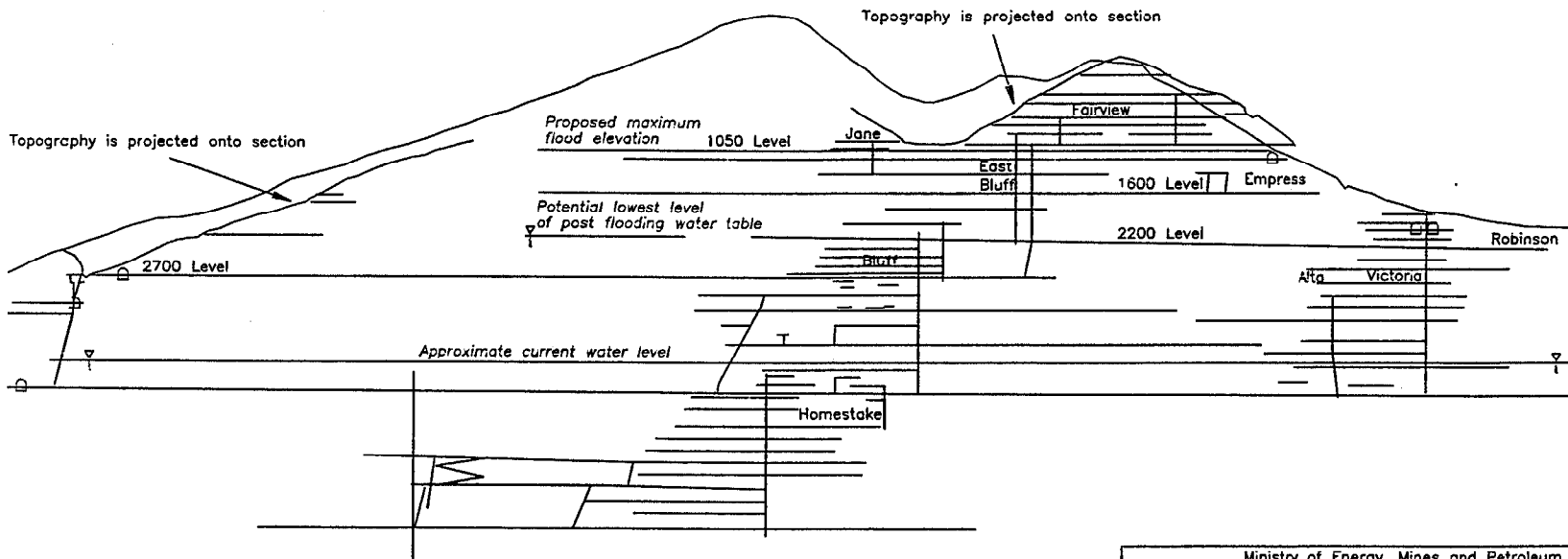
- ii) flood the underground workings to the upper most levels. Diverting runoff from the points of entry, as discussed in Section 6.1.1 will be limited because of the pit area catchment. Diverting runoff from around the pits, as described above, could reduce inflows by approximately 50 percent.

Reducing the flushing of contaminants can be achieved by flooding the underground workings. It should be recognized, however that there could be in the short term, an initial slug of loading from previously unflushed areas. Proposed locations for bulkheads are shown in Figure 6.2. Details of preliminary design of bulkhead requirements for flooding are presented in Appendix A.

A modification of the flooding option that has been implemented at the abandoned BethEnergy Mine in West Virginia, U.S.A., incorporates alkaline additives to the flooding water (House and Willison, 1986). The underground workings were sealed and flooded with alkaline water, and apatite (phosphate) was dusted on the surfaces underground which would not be flooded. Alkalinity was added as lime and caustic soda (NaOH). Prior to this flooding, contaminated drainage was collected and treated at an annual cost (\$1985) of \$480,000. At the time, the present value of this cost on a projected 40 year treatment requirement was \$4.7 million (10 percent discount rate). The cost of flooding was \$1.3 million. It has not been possible to obtain any further information to date, however the paper reports that:

- within four months of beginning flooding (1984) discharge pH increased from 3.9 to 7.0 and remains at 7.0 (1986)
- iron decreased from 133 ppm to  $\pm 1$  ppm.
- alkalinity has increased from 0 to 350 mg/L and acidity decreased from 470 to 20 mg/L.
- at an annual flow of  $512 \times 10^6$  gallons (U.S.) per year, an equivalent flow of four replacement volumes of water have run through the mine workings since flooding, and no additional surface water treatment has been required.

This approach may have merit at Britannia however would require careful consideration of the nature and amount of stored products, and the required type and amount of alkali for metal removal (notably zinc).



**LEGEND**

- Levels
- Recommended Bulkhead Locations
- ▽ Water Table
- Topography

SECTION LOOKING NORTH

Notes: Figure is derived from Copper Beach Estates Section is a simplified projection of the mine workings

Ministry of Energy, Mines and Petroleum Resources		DATE	March, 1991
<b>BRITANNIA MINES</b> <b>PROPOSED BULKHEAD LOCATIONS</b>		FILE NO.	66004
		APPROVED	
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers		NO.	6.2

### 6.1.3 Waste Dumps

There are three options for reducing infiltration into the waste dumps; diversion ditches, covers, and removal for disposal either underwater or within another catchment such as the pits. The three areas of concern are: the main waste dumps adjacent to the East Bluff pit, the waste at the 2200 level portal, and the waste in the mill area.

In the case of the waste dumps adjacent to the pits there are two general approaches for remediation. The first is to leave the waste in place and reduce the infiltration with diversion ditches and a cover. This may not be sufficient to reduce environmental impact to acceptable levels and some form of collection and treatment may be required. Based on a plan area of 38,000 m<sup>2</sup>, a cover 1 m thick, and allowing for recontouring of up to 10,000 m<sup>3</sup> of waste, the cost of constructing a cover would be in the order of \$1.2 million (Assuming costs similar to Mt. Washington at (K. Ferguson, review comments, 1990) \$30/m<sup>3</sup> for imported and compacted fill, and \$2/m<sup>3</sup> for recontouring).

The other option is to remove and dispose the waste in the pits. It is not likely that much of the waste could be dumped so that it would be below the flood elevation in the underground and workings and consequently the waste would continue to generate ARD. The benefit of this approach is that the total volume of water to be treated could be decreased. A concern is that rehandling will promote flushing of ARD products from the waste and may also create new acid generating surfaces, both of which would result in increased loadings to the treatment facility. The cost of this approach, based on a total of 700,000 tonnes and a unit cost of \$2/tonne, the total cost would be in the order of \$1,400,000. Additional data and a cost benefit analysis is necessary to determine the best option for remediation of this waste dump.

The character and quantity of waste stored and used for fill and construction at the 2200 level cannot be determined from surface observation, consequently, it is difficult to identify suitable remediation options. However, given that this waste is not located near to a site where underwater disposal could be implemented, it appears that some form of drainage water diversion and cover will be the most cost effective solution. Protection against future erosion by Britannia Creek is a concern. Further information will be required to quantify and cost remediation for this source.

Waste which has been dumped at the 4100 level near the mill has a surface slope at or near the angle of repose which would make placement of a cover very difficult. This material is not located near to a suitable underwater disposal site, except for Howe Sound which is not recommended because of the short-term potential for releasing stored ARD products in waste. Further data are required to identify potential remediation options.

#### **6.1.4 Tailings**

Options for reducing contaminated drainage from the tailings pond include reduction of infiltration into the tailings through diversions or covers, or removal of the tailings. The following discussion refers to the pond near the mill, and does not include other areas of possible historic tailings deposition, such as along the shoreline of Howe Sound.

Construction of diversion ditches around the pond area, in conjunction with placing a synthetic cover, could significantly reduce the infiltration, possibly by as much as 90 percent. A synthetic cover is probably less expensive to place over the tailings than a soil cover because of the soft nature of the tailing material, additionally a synthetic cover would be more effective at reducing infiltration. A 40 mil low density polyethylene (LDPE) cover over the pond would cost approximately \$7,000, installed. Allowance for regular replacement of the cover would be required. The existing tailings pond could be upgraded to ensure that a permanent water cover is maintained. The nature of the existing embankment containing the tailings is not known so it is difficult to estimate costs for rehabilitation of the structure. This approach could effectively limit the formation of ARD products from the tailings, however, further treatment of the drainage water might be required to remove ARD products flushed out of the tailings.

Options for disposal of the tailings include underwater disposal in Howe Sound, where most of the tailings were placed during operation or placement underground in the mine. Placement of the tailings in Howe Sound is not recommended because this would liberate the accumulated ARD products in the tailings. Removal and placement underground would probably require taking the material into the 2200 level. The cost of this approach could be in the order of \$10/tonne or \$80,000 because either special trucks would be required to enter the mine, or double handling would be required. Underwater disposal in the mine would limit further ARD, however temporary increases in metal loadings to the treatment plant may occur due to flushing of stored products.

#### **6.1.5 Roads**

At this time it is assumed that acid generation from the roads is not a problem and that measures to divert water from the roads are not required.

### **6.2 Drainage Collection Systems**

#### **6.2.1 Pits**

Options for implementing drainage collection for the pits in Jane Basin are limited to allowing the drainage to exit via the underground mine because of the limits of topography and the presence of the

glory holes. Drainage from the Empress pit could be monitored and if acceptable released to the environment or otherwise pumped up to the No. 5 pit where it would flow into the underground mine.

### **6.2.2 Underground Mine**

The geometry of the underground mine is well suited for drainage collection and routing to the 4100 level portal, as is the current practice. Drainage from the 2200 level must be effectively diverted back into the underground workings or, if required, this drainage could be piped down Britannia Creek to the treatment plant.

Seasonal variations in metal loading and drainage volume will need to be reduced in order to effectively run the treatment plant. Some type of flow pond must be considered. The two options for this are to construct a retention pond in the vicinity of the outfall, or to use the storage capacity within the underground workings.

It is estimated that up to  $0.4 \times 10^6$  cubic meters of storage would be required for an average precipitation year. The geometry of the site is not suitable for a retention pond of this size. To use the underground workings as a retention pond would require: i) effective blocking of any portals within the zone of water storage and fluctuation; ii) control of water volumes exiting the mine at the 4150 level. If the underground workings were flooded, these structures would be required at all the mine portals. Flooding the mine would have the advantage of reducing the formation of ARD products, but it may result in higher initial ARD loading to the treatment plant. It is possible that underground storage of water may accelerate subsidence of that portion of the mine which is in the zone of the fluctuating water level.

If required, drainage from the 2200 level portal area could be either piped down Britannia Creek to the treatment plant, or pumped into the mine at the 2200 level. This may require a high pressure pump if the mine is flooded. A pipe along Britannia Creek would drain by gravity and would avoid the need for a power supply.

### **6.2.3 Waste Dumps**

Options for collection of waste dump drainage, assuming that the waste is not relocated, is limited to providing a collection pond downhill of the various waste dumps. Some of the drainage from the waste adjacent to the East Bluff pit may be draining into the underground mine. Drainage from the dumps which is collected could be monitored and if acceptable discharged to the environment or routed to the treatment plant. Drainage from the Jane Basin waste dumps would be most easily routed to the treatment plant, if required, with a pump system to raise the water to where it could drain into the underground mine. It may be difficult to provide power to this location to run the pumps.

Drainage from the 4100 level mill area waste could be collected in ditches and pumped directly to the treatment plant.

#### **6.2.4 Tailings**

Collection of drainage from the tailings pond would be most easily conducted with a ditch excavated near the toe of the embankment. The drainage could be then piped to the treatment plant.

#### **6.2.5 Roads**

It is assumed that the roads in Britannia Creek are not sources of ARD and that collection of drainage is not required.

### **6.3 Treatment**

#### **6.3.1 General**

In developing a treatment strategy for an acid mine drainage situation, a logical sequence of activities is followed. These activities are listed to provide an overview of the approach adopted in this study.

- identify maximum and average flows
- characterize feed as well as possible
- select the probable optimum processing concepts
- test proposed concepts at lab scale
- develop conceptual designs based on lab data
- complete the detailed engineering for best concept
- build the plant

#### **6.3.2 Flows**

Flow data for the period 1983 to 1990 at the 4100 portal was analyzed to generate historical maximum and average design flows. These results, presented in Section 6.6, were in turn used to generate capital and operating costs presented in Sections 6.9.4 and 6.9.5. The design flow assumes that diversion of surface water above the pits will reduce peak flows and that the plugging of the portals and vent raises using concrete bulkheads will equalize hydraulic loading. The maximum design flowrate used to size the equipment was 690 m<sup>3</sup>/h while the mean design flow used for operating cost estimates was 635 m<sup>3</sup>/h.

### 6.3.3 Raw Water Characteristics

The chemical characteristics of the potential feed to a treatment plant were determined by analyzing samples collected from the 4100 and 2200 portals on November 28 and December 6, 1990. The chemical characteristics of the two samples are provided in Tables 6.2 and 6.3. These two samples were used for the initial bench tests presented in this report. The 2200 portal sample had considerably higher acidity and dissolved metals than the 4100 portal sample. Both samples contained elevated levels of dissolved copper, zinc, iron and aluminum.

In addition, a sample from the 4100 portal was collected on March 13, 1991 to conduct the process simulation work. The results from this sample, which indicate a lower acidity than the November 28, 1990 sample, are provided along with the test results in Table 6.7.

### 6.3.4 Process Testwork Description

A series of bench tests has been conducted to screen alternative processes. The purpose of this work was to provide a basis for assessment of the relative technical and economic feasibility of each. Process selection will ultimately involve trade-offs between costs, effluent quality and alternatives for upstream mitigation. Testwork was initially conducted on samples from both the 4100 and 2200 portals due to the marked difference in their chemical characteristics and uncertainty regarding future blending ratios for the drainages. The final test series was conducted on an additional sample from the 4100 level since access to the 2200 level was difficult, and the contribution of the 2200 level solution appears to be minor. In addition, the 2200 solution will likely be redirected to the 4100 level within the mine prior to the onset of treatment, so that its chemical characteristics will be modified in any case.

The following processes were evaluated on a laboratory amenability level:

- 1) Lime Precipitation
- 2) Soda Ash Precipitation
- 3) Sulphide Precipitation
- 4) Lime and Soda Ash Addition
- 5) Ion Exchange
- 6) Seawater Dilution
- 7) Limestone Treatment

The results are reported in the following sections together with descriptions of the processes that were considered, and our evaluation. Test details are provided in Appendix C.



Table 6.2

Raw Water Characteristics - Physical Test, Anions and Nutrients

	2200 Portal	4100 Portal
Date of Collection	Dec 6/90	Nov 28/90

Physical Tests

Total Dissolved Solids		1170	2480
Hardness	CaCO <sub>3</sub>	610	979
pH	Units	3.49	3.51
Total Suspended Solids		21	26
Conductivity	umhos/cm		2730

Dissolved Anions

Acidity	CaCO <sub>3</sub>	764	343
Alkalinity	CaCO <sub>3</sub>	<1	<1
Chloride	Cl	<0.5	<0.5
Fluoride	F	0.85	1.52
Sulphate	SO <sub>4</sub>	1470	1850

Nutrients

Ammonia Nitrogen	N	0.14	0.043
Nitrate Nitrogen	N	<0.005	<0.005
Nitrite Bitrogen	N	<0.001	<0.001
Total Phosphorous	P	0.038	0.017

Results expressed as mg/L except where noted

< = less than

Table 6.3

## Raw Water Characteristics - Metals

Date of Collection		2200 Portal		4100 Portal	
		Dec 6/90		Nov 28/90	
		Total	Dissolved	Total	Dissolved
Aluminum	Al	61.8		35.3	33.3
Antimony	Sb	<0.0001		<0.2	<0.2
Arsenic	As	0.0011		<0.2	<0.2
Barium	Ba			0.012	0.012
Beryllium	Be			<0.005	<0.005
Bismuth	Bi			<0.1	<0.1
Boron	B			0.21	0.2
Cadmium	Cd	0.327		0.232	0.22
Calcium	Ca		118	206	203
Chromium	Cr			<0.015	<0.015
Cobalt	Co			0.101	0.098
Copper	Cu	97.3	97.1	24.3	22.9
Iron	Fe	42.6	39.3	5.97	5.95
Lead	Pb	0.1	0.1	0.175	0.173
Magnesium	Mg		75.1	104	98.4
Manganese	Mn			6.85	6.47
Molybdenum	Mo			<0.03	<0.03
Nickel	Ni			0.075	0.075
Phosphorous	P			1.26	1.16
Potassium	K		0.46		
Mercury	Hg	<0.00005			
Selenium	Se	<0.0005		<0.2	<0.2
Sodium	Na		2.79		
Silicon	Si			16.1	15.2
Silver	Ag			<0.015	<0.015
Strontium	Sr			2.77	2.59
Vanadium	V			<0.03	<0.03
Zinc	Zn	49.1	47	44.4	42.4

All results expressed as mg/L

&lt; = less than

#### 6.3.4.1 Lime Neutralization

The conventional treatment plant for acidic drainage is based on neutralization and precipitation to remove acidity and heavy metals from solution. Lime is added to raise the effluent to a pH where the heavy metal hydroxides are sufficiently insoluble to produce a dischargeable effluent.

The system would consist of a milk of lime mixing system to which hydrated lime (calcium hydroxide) and water would be added to make a 15-20 weight percent slurry. Lime slurry would be dosed from a ring main to the acidic feed in one or more stirred tanks under pH control. At the appropriate pH, the metals would precipitate. The precipitate would need to be removed in a settling device which could range from a simple sludge pond to a fairly sophisticated thickener and/or filter. Iron removal would depend in part on the success of iron oxidation in the lime reactor, which would be achieved by means of an air sparge.

It is likely that use of an organic flocculant chemical would be specified as well, to assist in the settling of the precipitates. The flocculant system would require a mixing tank and a stock tank, together with a metering pump and predilution system to make the appropriate additions.

The NaOH based neutralization curves are provided in Figures 6.3 and 6.4 for the two samples. Equivalent curves based on quicklime are provided in Figures 6.5 and 6.6. These latter plots are calculated based on the equivalent basicity of 1.43 for caustic soda compared to high calcium quicklime (CaO) at a basicity of 1.0. Reagent consumption is expressed as kg/1000 m<sup>3</sup> which is numerically identical to concentration expressed as mg/L. Lime consumption (CaO) to pH 7.0 was approximately 410 and 150 kg/1000 m<sup>3</sup> for the 2200 and 4100 portal drainages respectively. CaO consumption to pH 9.5 increased to 490 and 220 kg/1000 m<sup>3</sup> for the 2200 and 4100 portal drainages respectively.

The results of the lime precipitation tests on the two portal drainages are presented in Table 6.4. The testwork has indicated that a lime dosage of 227 kg CaO/1000 m<sup>3</sup> was required to reduce the soluble metals to acceptable concentrations, based on the sample from the 4100 portal collected in November, 1990, while the equivalent lime dosage for the 2200 portal was 643 kg CaO/1000 m<sup>3</sup>.

#### 6.3.4.2 Iron/Alum Coagulation

In the lime process, removal of the metal hydroxides in the settling device might not be sufficiently effective owing to the gelatinous and colloidal nature of hydroxide precipitates, and the low levels at which they would be present. Settling of the precipitated solids could be enhanced by the addition of a coagulant such as ferric or aluminum sulphate. Iron or aluminum have the additional capability of

Table 6.4

## Lime Precipitation Test Results

Location		2200 Portal				4100 Portal			
Date of Collection		Dec 6/90				Nov 28/90			
Lime Dosage mg/L Ca(OH) <sub>2</sub>		0	700	850	1000	0	200	300	500
CaO Equivalent kg/1000m <sup>3</sup>		0	530	643	757	0	151	227	378
pH		2.92	6.6	8.7	10.09	3.51	7.13	8.57	9.9
		Dissolved Metals				Dissolved Metals			
Aluminum	Al	61.8	<0.2	0.49	1.47	33.3	<0.2	<0.2	0.87
Antimony	Sb	<0.001	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Arsenic	As	0.0011	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Barium	Ba		<0.1	<0.1	<0.1	0.012	<0.01	<0.01	<0.01
Beryllium	Be		<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Bismuth	Bi		<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Boron	B		<0.1	<0.1	<0.1	0.2	0.19	0.18	0.18
Cadmium	Cd	0.327	0.043	<0.01	<0.01	0.22	0.192	0.035	<0.01
Calcium	Ca	118	302	188	194	203	207	206	213
Chromium	Cr		<0.015	<0.015	<0.015	<0.015	<0.015	<0.015	<0.015
Cobalt	Co		<0.015	<0.015	<0.015	0.098	0.076	<0.015	<0.015
Copper	Cu	97.1	0.023	<0.01	<0.01	22.9	0.348	<0.01	<0.01
Iron	Fe	39.3	<0.015	0.041	0.017	5.95	0.065	<0.015	0.039
Lead	Pb	0.1	<0.05	<0.05	<0.05	0.173	<0.05	<0.05	<0.05
Magnesium	Mg	75.1	56.8	52.08	18.4	98.4	83.3	74.8	59.3
Manganese	Mn		2.47	1.17	0.013	6.47	5.19	3.39	0.174
Molybdenum	Mo		<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
Nickel	Ni		<0.02	<0.02	<0.02	0.075	0.05	<0.02	<0.02
Phosphorous	P		<0.03	<0.03	<0.03	1.16	<0.03	<0.03	<0.03
Selenium	Se		<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Silicon	Si		1.71	1.04	0.909	15.2	9.85	2.32	1.42
Silver	Ag		<0.015	<0.015	<0.015	<0.015	<0.015	<0.015	<0.015
Strontium	Sr		0.915	0.792	0.753	2.59	2.5	2.31	2.46
Vanadium	V		<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
Zinc	Zn	47	0.662	0.046	0.007	42.4	26.5	0.288	0.042

All results expressed as mg/L

&lt; = less than

Neutralization of 2200 Portal Drainage  
Sodium Hydroxide  
Dec 6/90 Sample

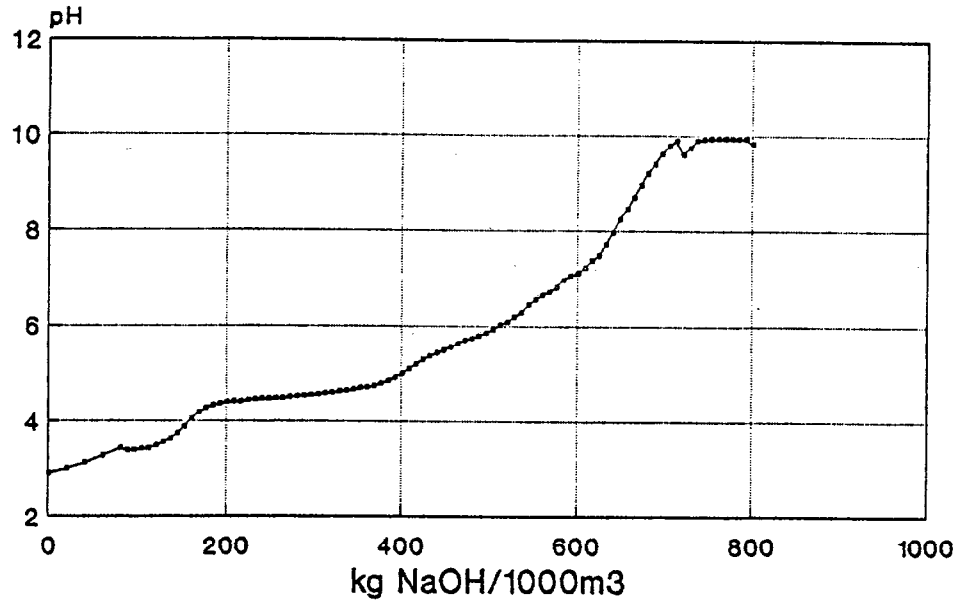


Figure 6.3

Neutralization of 4100 Portal Drainage  
Sodium Hydroxide  
Nov 28/90 Sample

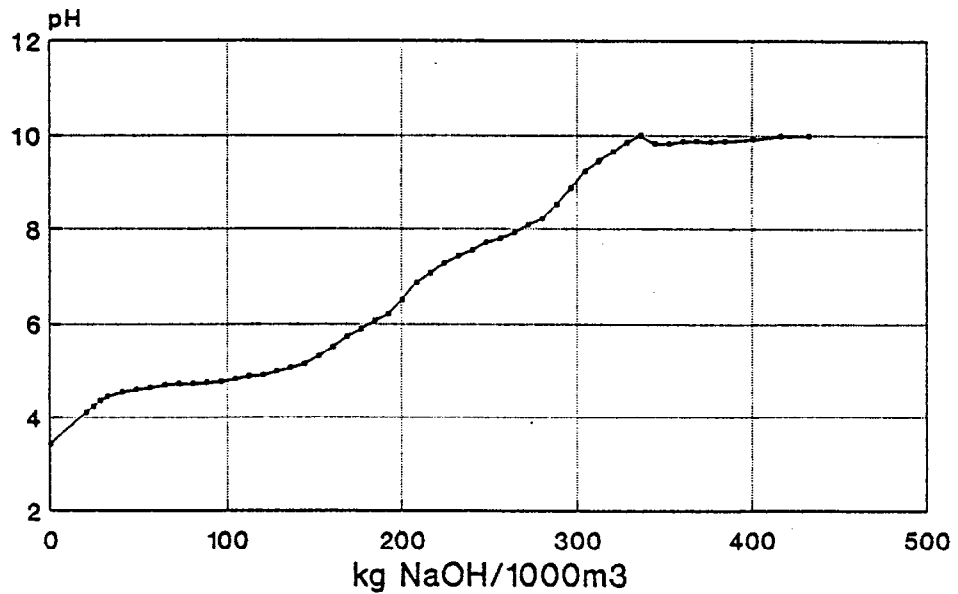


Figure 6.4

Neutralization of 2200 Portal Drainage  
High Calcium Quicklime Equivalent  
Dec 6/90 Sample

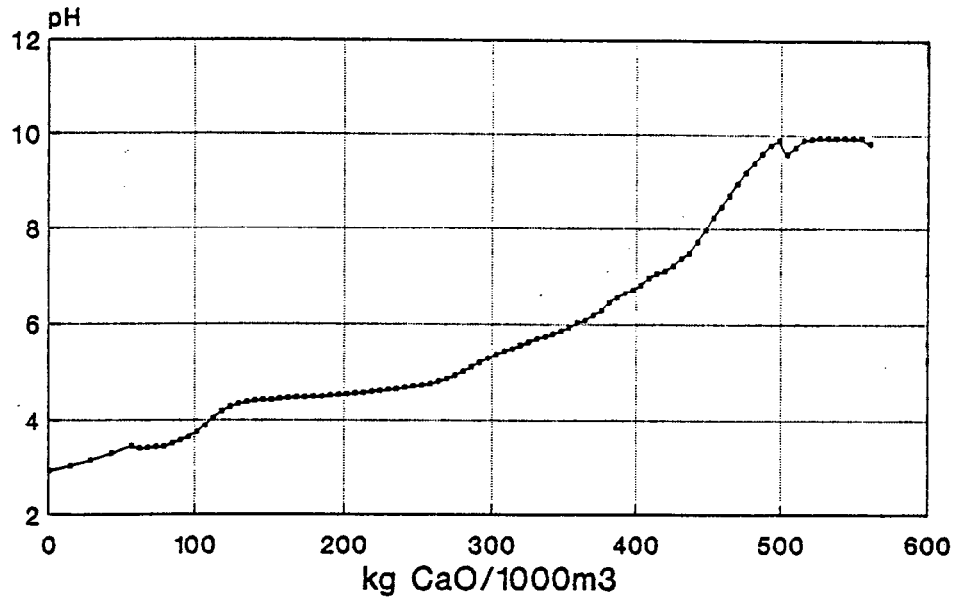


Figure 6.5

Neutralization of 4100 Portal Drainage  
High Calcium Quicklime Equivalent  
Nov 28/90 Sample

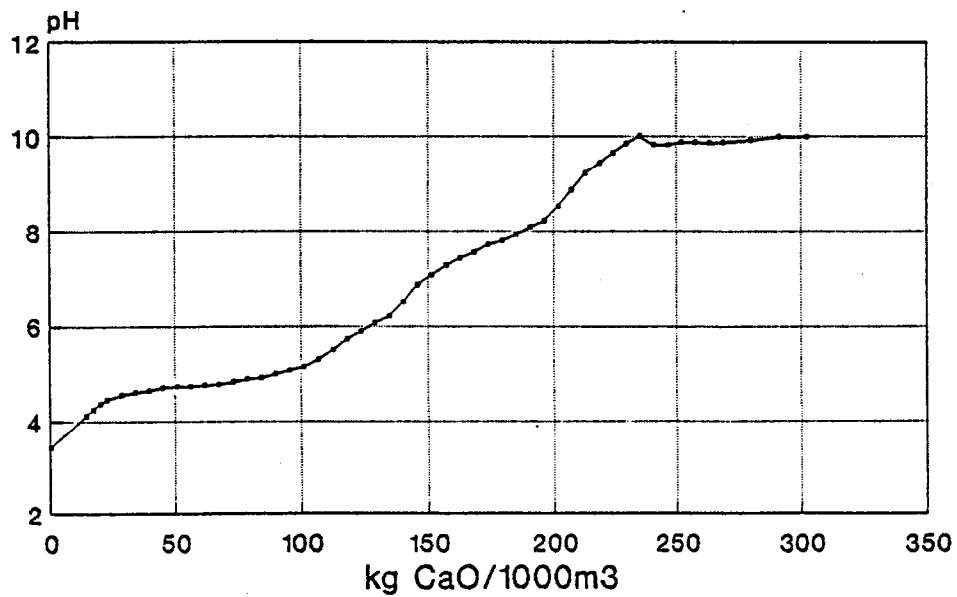


Figure 6.6

coprecipitating a wide range of ions. Thus, their use would both improve settling and reduce the total dissolved solids by adsorbing/coprecipitating additional quantities of undesirable ions.

The process modifications required to use these reagents would consist of a mixing and a stock tank, together with a metering pump to add the reagent solution to the lime reactor. The reagents consume some lime as part of their function, so additional lime capability might be required.

Both iron and alum are present in the feed to the precipitation process (Table 6.3) and constitute a significant portion of the metal loading. Consequently, it was not felt necessary to augment the quantities naturally present, so that this variant of the lime precipitation process is already reflected in the current testwork.

#### 6.3.4.3 Soda Ash

Soda ash can be used for neutralization and precipitation as an alternative to lime. This process relies on the precipitation of metals as both hydroxides and carbonates. Precipitation of zinc carbonate under neutral pH conditions is feasible as an alternative to the pH 9.0 to 9.5 conditions required to remove zinc as a hydroxide. Alternatively, in a predominantly lime-based process, the required lime dosage would be lower than that required to precipitate zinc as a hydroxide if carbonate addition results in the formation of insoluble zinc carbonates or basic zinc carbonates under neutral pH conditions. Potentially this process would allow complete zinc removal without the need to increase pH to precipitate zinc hydroxide, thereby minimizing total alkali consumption.

The results for soda ash addition are provided in Table 6.5. For both feeds, the higher dosage was capable of removing both copper and zinc to acceptable levels, while the lower dosages left residual dissolved zinc in solution. The results indicate that the removal was due to the precipitation of zinc under high pH conditions as opposed to precipitation of zinc carbonate. The calculated CaO equivalent for both samples indicates that partial soda ash substitution has a higher equivalent dosage than solely quicklime, perhaps due to the loss of CO<sub>2</sub> during neutralization. Neutralization to precipitate metals as their hydroxides would not be attractive using soda ash alone due to its higher cost compared to lime. However, soda ash may have an application to precipitate basic carbonates when used in conjunction with lime.

The results of experiments using a sequenced addition of lime and soda ash are provided in Table 6.6. Comparative results with lime only are also included in this table.

Table 6.5

## Soda Ash Neutralization Tests

Soda Ash Dosage		2200 Portal			4100 Portal		
		0	720	1056	0	320	546
Na <sub>2</sub> CO <sub>3</sub>		0	720	1056	0	320	546
CaO Equivalent kg/1000m <sup>3</sup>		0	381	559	0	169	289
pH		2.92	7.76	9.15	3.51	8.07	9.17
		Dissolved Metals			Dissolved Metals		
Aluminum	Al	<0.2	<0.2		33.3	<0.2	<0.2
Antimony	Sb	<0.2	<0.2		<0.2	<0.2	<0.2
Arsenic	As	<0.2	<0.2		<0.2	<0.2	<0.2
Barium	Ba	<0.1	<0.1		0.012	<0.01	<0.01
Beryllium	Be	<0.005	<0.005		<0.005	<0.005	<0.005
Bismuth	Bi	<0.1	<0.1		<0.1	<0.1	<0.1
Boron	B	<0.1	<0.1		0.2	0.18	0.19
Cadmium	Cd	0.072	<0.01		0.22	0.094	<0.01
Calcium	Ca	118	94.1	73	203	192	165
Chromium	Cr	<0.015	<0.015		<0.015	<0.015	<0.015
Cobalt	Co	0.045	<0.015		0.098	0.063	<0.015
Copper	Cu	97.1	1.05	0.433	22.9	0.3	0.051
Iron	Fe	39.3	0.022	<0.015	5.95	0.033	<0.015
Lead	Pb	0.1	<0.05	<0.05	0.173	<0.05	<0.05
Magnesium	Mg	75.1	52.5	62.4	98.4	84.6	76.2
Manganese	Mn		3.12	0.735	6.47	5.34	1.57
Molybdenum	Mo		<0.03	<0.03	<0.03	<0.03	<0.03
Nickel	Ni		0.03	<0.02	0.075	0.045	<0.02
Phosphorous	P		<0.3	<0.3	1.16	<0.3	<0.3
Selenium	Se		<0.2	<0.2	<0.2	<0.2	<0.2
Silicon	Si		11.2	9.61	15.2	8.52	7.55
Silver	Ag		<0.015	<0.015	<0.015	<0.015	<0.015
Strontium	Sr		0.397	0.333	2.59	2.41	1.85
Vanadium	V		<0.03	<0.03	<0.03	<0.03	<0.03
Zinc	Zn	47	3.81	0.094	42.4	3.97	0.052

All results expressed as mg/L

&lt; = less than



Table 6.6

## Lime Precipitation and Soda Ash Addition Tests

Lime Dosage mg/L Ca(OH) <sub>2</sub>	2200 Portal			4100 Portal					
	0	700	700	0	200	200	300	300	500
CaO Equivalent kg/1000m <sup>3</sup>	0	530	530	0	151	151	227	227	378
Soda Ash Dosage mg/L Na <sub>2</sub> CO <sub>3</sub>		0	40		0	20	0	80	0
CaO Equivalent kg/1000m <sup>3</sup>		0	21		0	11	0	42	0
Total CaO Equivalent kg/1000m <sup>3</sup>		530	551		151	162	227	269	378
pH	2.92	6.6	7.1	3.51	7.13	6.56	8.57	7.39	9.9
	Dissolved Metals			Dissolved Metals					
Aluminum Al	61.8	<0.2	0.94	33.3	<0.2	<0.2	<0.2	0.34	0.87
Antimony Sb	<0.001	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Arsenic As	0.0011	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Barium Ba		<0.1	<0.1	0.012	<0.01	<0.01	<0.01	<0.01	<0.01
Beryllium Be		<0.005	<0.005	0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Bismuth Bi		<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Boron B		<0.1	<0.1	0.2	0.19	0.17	0.18	0.16	0.18
Cadmium Cd		0.043	<0.01	0.22	0.192	0.0173	0.035	<0.01	<0.01
Calcium Ca	118	302	340	203	207	200	206	430	213
Chromium Cr		<0.015	<0.015	0.015	<0.015	<0.015	<0.015	<0.015	<0.015
Cobalt Co		<0.015	<0.015	0.098	0.076	0.067	<0.015	<0.015	<0.015
Copper Cu	97.1	0.023	<0.01	22.9	0.348	0.202	<0.01	<0.01	<0.01
Iron Fe	39.3	<0.015	<0.015	5.95	0.065	0.078	<0.015	<0.015	0.039
Lead Pb	0.1	<0.05	<0.05	0.173	<0.05	<0.05	<0.05	<0.05	<0.05
Magnesium Mg	75.1	56.8	52.2	98.4	83.3	75.5	74.8	72.1	59.3
Manganese Mn		2.47	0.416	6.47	5.19	4.74	3.39	1.07	0.174
Molybdenum Mo		<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
Nickel Ni		<0.02	<0.02	0.075	0.05	0.047	<0.02	<0.02	<0.02
Phosphorous P		<0.03	<0.3	1.16	<0.03	<0.03	<0.03	<0.03	<0.03
Selenium Se		<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Silicon Si		1.71	0.45	15.2	9.85	8.53	2.32	1.047	1.42
Silver Ag		<0.015	<0.015	0.015	<0.015	<0.015	<0.015	<0.015	<0.015
Strontium Sr		0.915	0.96	2.59	2.5	2.2	2.31	2.27	2.46
Vanadium V		<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
Zinc Zn	47	0.662	<0.03	42.4	26.5	21.2	0.288	<0.005	0.042

All results expressed as mg/L

&lt; = less than

For both samples, the addition of soda ash after lime dosing was capable of reducing soluble zinc to low levels under neutral pH conditions, whereas the comparative lime-only result reported a measurable quantity of zinc in solution. In the case of the 4100 portal sample, the total equivalent lime dosage calculated using basicity factors was less for the 2 reagent process than the required lime dosage when no other reagent was added.

#### **6.3.4.4 Additional Lime and Soda Ash Process Evaluation Results**

In order to confirm the screening test results for the lime and lime plus soda ash alternatives, a series of follow-up tests were conducted on a sample of 4100 portal drainage collected on March 13, 1991. The purpose of these tests, reported in the Appendix as Experiments 9 and 10, was to simulate the coagulation, flocculation and sedimentation aspects of the lime and lime plus soda ash alternatives.

The results of these tests, reported in Table 6.7, indicate that the lime process was successful at a lime dosage of 250 mg/L CaO, while the lime plus soda ash combination was not as efficient in terms of zinc removal at a lime dosage of 125 mg/L and a soda ash dosage of 25 mg/L. Possibly, a higher dosage of either lime or soda ash would be required to improve the performance of the lime plus soda ash alternative.

The data generated during these tests were used to assemble the operating cost estimates reported in Section 6.6. Lime consumption results from the tests were converted to a lime to acidity ratio in order to estimate the required dosage at a higher acidity concentration. In the case of the lime/soda ash combination, the estimated dosages were increased to 150 mg/L CaO and 40 mg/L Na<sub>2</sub>CO<sub>3</sub> in order to provide a conservative estimate of the reagent consumption prior to calculating a reagent dosage to acidity ratio.

#### **6.3.4.5 Sulphide Precipitation**

In this process, a solution of an inorganic or organic sulphide, or even a slurry of ground iron sulphide, could be added to the effluent after neutralization. The sulphide ions would react with the heavy metals present to precipitate the respective sulphides. If iron sulphide were added, the surface of the sulphide particles would become coated with the heavy metal sulphides (especially copper) driven by the greater insolubility of the respective sulphides compared with that of iron sulphide. The iron would come into solution, and might need to be oxidized and precipitated subsequently.

Table 6.7

## Lime and Soda Ash Treatment Process Simulation Test Results

4100 Portal Sample Collected March 13, 1991

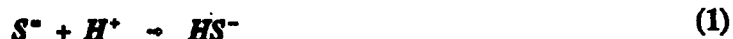
	Raw		Lime		Lime & Soda Ash	
Lime Dosage mg/L Ca(OH) <sub>2</sub>	0		331		165	
CaO Equivalent kg/1000m <sup>3</sup>	0		250		125	
Soda Ash Dosage mg/L Na <sub>2</sub> CO <sub>3</sub>					25	
Total CaO Equivalent kg/1000m <sup>3</sup>			250		178	
pH	3.63		9.2		7.95	
Acidity	285					
Alkalinity	25.5				24	
Sulphate	2520	2520			2540	
Metals	Total	Dissolved	Total	Dissolved	Total	Dissolved
Aluminum Al	32.3	27.9	0.72	0.6	<0.2	<0.2
Antimony Sb	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Arsenic As	0.21	<0.2	<0.2	<0.2	<0.2	<0.2
Barium Ba	<0.010	<0.010	<0.010	<0.010	<0.01	<0.01
Beryllium Be	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Bismuth Bi	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Boron B	0.23	0.2	<0.1	<0.1	0.2	0.17
Cadmium Cd	0.148	0.137	<0.01	<0.01	0.058	0.051
Calcium Ca	467	464	642	631	574	569
Chromium Cr	<0.015	<0.015	<0.015	<0.015	<0.015	<0.015
Cobalt Co	0.097	0.091	<0.015	<0.015	0.043	0.04
Copper Cu	18.1	16.5	0.023	<0.01	0.087	0.02
Iron Fe	5.7	3.09	<0.030	<0.030	<0.030	<0.030
Lead Pb	0.141	0.129	<0.05	<0.05	<0.05	<0.05
Magnesium Mg	95.2	87.5	47	44.4	93.6	86.7
Manganese Mn	6.42	5.95	0.095	0.052	5.25	4.88
Molybdenum Mo	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
Nickel Ni	0.064	0.056	<0.02	<0.02	<0.02	<0.02
Phosphorous P	1.33	1.3	<0.3	<0.3	<0.03	<0.03
Selenium Se	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Silicon Si	16.3	15.2	1.55	1.33	4.36	3.81
Silver Ag	<0.015	<0.015	<0.015	<0.015	<0.015	<0.015
Strontium Sr	2.6	2.33	2.83	2.69	2.75	2.53
Vanadium V	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
Zinc Zn	32	28.7	0.081	0.019	1.26	1.03

All results expressed as mg/L

&lt; = less than

The sulphides produced would be removed by settling or filtration. Use of coagulants might enhance this process, but might require the addition of alkalinity (lime) to maintain a suitable final pH. Recycle of the precipitated solids could be required to enhance solid/liquid separation properties.

Precipitation of metals by sulphide requires the presence of a significant concentration of sulphide ion,  $S^{2-}$ . At increasingly acid pH,  $S^{2-}$  is converted to  $HS^-$  or  $H_2S$ , reducing its availability for metal precipitation:



The concentration of sulphide ion required to precipitate a given metal is dependent on the relative insolubility of the sulphide; thus, copper is precipitated by very low levels of sulphide ion, while zinc and iron require higher levels. This theory was borne out by the testing: at low pH, copper was successfully precipitated by sulphide addition, but much larger additions were required to remove zinc.

As shown by equations 1 and 2, the addition of sulphide ion to acidic effluent consumes acid ( $H^+$ ), and thus, its use in large quantities is tantamount to using it as a neutralizing agent rather than as a highly specific precipitant. When compared with lime or soda ash in this capacity, and assuming addition as sodium sulphide, it is not economically competitive. As well, used in this way, significant quantities of highly toxic hydrogen sulphide gas may be evolved.

The test results are provide in Table 6.8. In both cases the highest sulphide dosage was required to precipitate zinc while iron remained in solution. Copper was precipitated out at the lowest test additions; i.e., 313 mg/L in the case of 4100 level and 1254 mg/L with the 2200 sample. The results indicate that a high sulphide to metal ratio is required in all cases to precipitate metals, making this process unattractive unless a source of sulphide such as a sulphide flotation concentrate were readily available.

#### 6.3.4.6 Sea Water Dilution

The waters of Howe Sound are a convenient large repository of available alkalinity. Although the volume is large, the alkalinity is not high. Garrels and Christ (1965) have calculated the molality of bicarbonate as 0.0024, and carbonate as 0.00027.

Table 6.8

## Sulphide Precipitation Tests

Na <sub>2</sub> S Dosage mg/L S=		2200 Portal					4100 Portal			
		0	313	627	1254	2090	0	313	627	1254
Na <sub>2</sub> S.10H <sub>2</sub> O Equivalent kg/1000m <sup>3</sup>		0	2524	5055	10110	16851	0	2524	5055	10110
pH		2.92	2.95	2.98	3.1	4.14	3.51	3.67		4.76
		Dissolved Metals					Dissolved Metals			
Aluminum	Al	61.8	58.4	53.8	49.7	47.7	33.3	28.5	29.6	12.3
Antimony	Sb	<0.001	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Arsenic	As	0.001	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Barium	Ba		<0.1	<0.1	<0.1	<0.1	0.012	<0.01	<0.01	<0.01
Beryllium	Be		<0.005	<0.005	<0.005	<0.005	0.005	<0.005	<0.005	<0.005
Bismuth	Bi		<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Boron	B		<0.1	<0.1	<0.1	<0.1	0.2	0.17	0.18	0.18
Cadmium	Cd		0.309	0.29	<0.01	<0.01	0.22	<0.01	<0.01	<0.01
Calcium	Ca	118	114	108	100	99.8	203	182	186	190
Chromium	Cr		<0.015	<0.015	<0.015	<0.015	0.015	<0.015	<0.015	<0.015
Cobalt	Co		0.081	0.074	0.067	0.059	0.098	0.075	0.073	0.063
Copper	Cu	97.1	82.5	45.9	<0.01	<0.01	22.9	<0.01	<0.01	<0.01
Iron	Fe	39.3	41.01	<0.015	34.6	31.7	5.95	2.17	2.02	2.54
Lead	Pb	0.1	0.066	<0.05	<0.05	<0.05	0.173	<0.05	<0.05	<0.05
Magnesium	Mg	75.1	68.7	64	57.9	57.7	98.4	74.9	77.5	81.9
Manganese	Mn		4.33	4.01	3.66	3.61	6.47	5.05	5.19	5.47
Molybdenum	Mo		<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
Nickel	Ni		0.074	0.067	0.06	0.062	0.075	0.056	0.056	0.051
Phosphorous	P		<0.03	<0.3	<0.3	<0.3	1.16	<0.3	<0.3	<0.3
Selenium	Se		<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2	<0.2
Silicon	Si		21.1	18.8	18.3	17.4	15.2	13.3	13.5	13.6
Silver	Ag		<0.015	<0.015	<0.015	<0.015	0.015	<0.015	<0.015	<0.015
Strontium	Sr		0.52	0.479	0.43	0.427	2.59	2.02	2.12	2.28
Vanadium	V		<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
Zinc	Zn	47	32.2	38.3	37.9	<0.005	42.4	32.8	6.9	0.012

All results expressed as mg/L

&lt; = less than

The testing (Table 6.9 and Figures 6.7 and 6.8) showed that for the 4100 level sample, a dilution ratio of 15:1 was required to reach background seawater pH (7.58), suggesting an equivalent alkalinity of about 22 mg/L CaCO<sub>3</sub>, or about 0.00024 molal. For the 2200 sample, a ratio of about 55:1 was required, suggesting an equivalent alkalinity of about 14 mg/L CaCO<sub>3</sub>. This comparison illustrates the difficulty of using standard tests such as acidity and alkalinity titrations to predict treatment chemistry in some situations.

Dissolved zinc levels resulting from these tests were above conventional discharge levels. Only 50 % of the reduction in soluble concentration is due to precipitation during neutralization. The kinetics of zinc precipitation via seawater dilution are not favourable. In addition the precipitates formed are colloidal and may not necessarily have been retained on 0.45 micron filter paper.

#### 6.3.4.7 Ion Exchange

This process operates using plastic resin beads that are permeable to the effluent solution. As part of the plastic structure, organic chemical groups are incorporated that are capable of exchanging one ion for another. Thus, for example, in a heavy metal solution, they might exchange hydrogen ion for metal ion (a cation exchange resin). The "loaded" resin beads can be regenerated periodically by washing with an acid solution.

The beads would be charged to a packed or fluidized bed and the effluent passed through, in batch or continuous mode. When the resin was loaded, it would be discharged and regenerated.

For batch operation (most common), at least two parallel beds would be required, one on line and the other in the process of being rejuvenated. The processing would be under the control of a programmable logic controller (computer) which would sequence a set of valves according to a timed program or in response to an on-line analytical signal.

In continuous operation, at least two columns would be required, one for treating effluent, and the other for regeneration. Fully loaded and fully regenerated resin would be transferred at least semi-continuously between the columns. The spent regenerant would be a relatively concentrated solution containing all the contaminants that had been removed. This would need to be treated further (e.g., by liming) or to be shipped to a safe disposal area.

One reason for considering ion exchange is that it could provide a necessary concentration step as a front-end to a solvent extraction process for copper recovery. The results of the ion exchange experiment are provided in Tables 6.10 and 6.11. The dissolved values for zinc and copper are plotted in Figures 6.9 and 6.10 on a bed volume basis.

Table 6.9

Seawater Dilution Test Results

2200 Portal Drainage					4100 Portal Drainage				
Dilution Ratio	pH	Dissolved Metals			Dilution Ratio	pH	Dissolved Metals		
		Copper	Zinc	Iron			Copper	Zinc	Iron
0	2.85	97.10	47.00	39.30	0	3.58	22.9	42.4	5.95
1	3.42				1	4.89			
2	4.35				2	5.37			
3	4.90				3	5.83			
4	5.10				4	6.11			
5	5.30				5	6.28	2.19	5.75	0.073
6	5.52				6	6.43			
7	5.71				7	6.54			
8	5.91				8	6.60			
9	6.02				9	6.70			
10	5.87	4.94	3.72	0.08	10	6.79	0.873	3.14	0.086
15	6.29	3.39	2.62	0.10	11	6.84			
20	6.49				12	6.95			
25	6.75	1.68	1.46	L0.015	15	7.58	0.69	2.17	0.092
30	6.92								
35	7.03								
40	7.17								
45	7.27								
50	7.39	0.78	0.87	0.12					
55	7.44	0.52	0.89	L0.015					

All results expressed as mg/L

L = less than

Table 6.10

Ion Exchange Test Results - 2200 Portal Drainage

Resin - Dowex MSC-1 Strong Acid Cation			
Bed Diameter	10 mm		
Resin Volume	25.12 mL	0.00025 m3	
Bed Depth	320 mm	0.32 m	
Bed Area	0.000078 m2		
Flow Rate	10 mL/min	0.00001 m3/min	
Surface Loading	7.64 m3/m2/h		
	3.31 gpm/ft2		

Sample Time min	Sample Volume	Bed Volumes	pH	Dissolved Metals			
				Zinc	Copper	Calcium	Magnesium
10	100	4.0	5.82				
20	200	8.0	5.71	0.058	0.069	0.05	0.131
30	300	11.9	5.70				
40	400	15.9	5.45	0.036	0.073	0.782	0.098
50	500	19.9	5.08				
60	600	23.9	5.70	0.032	0.057	0.904	0.077
70	700	27.9	5.00				
80	800	31.8	5.48	0.059	0.088	0.941	0.134
90	900	35.8	4.25				
100	1000	39.8	3.76	0.236	0.302	1.57	0.471
110	1100	43.8	3.17	0.391	0.562	0.246	0.767
120	1200	47.8	3.08				
130	1300	51.8	2.76	1.35	1.83	0.655	2.53

All concentrations in mg/L



Table 6.11

Ion Exchange Test Results - 4100 Portal Drainage

Resin -	Dowex MSC-1 Strong Acid		
Bed Diameter	10 mm		
Resin Volume	25.12 mL	0.000025 m3	
Bed Depth	320 mm	0.32 m	
Bed Area	0.000078 m2		
Flow Rate	10 mL/min	0.00001 m3/min	
Surface Loading	7.64 m3/m2/h		
	3.31 gpm/ft2		

Sample Time	Bed Volume	Bed Volumes	pH	Dissolved Metals			
				Zinc	Copper	Calcium	Magnesium
10	100	4.0	5.39				
20	200	8.0	5.65	0.024	0.016	0.05	0.084
30	300	11.9	5.79				
40	400	15.9	5.83	0.012	0.013	0.05	0.098
50	500	19.9	5.88				
60	600	23.9	5.77				
70	700	27.9	5.90	0.014	0.021	0.453	0.069
80	800	31.8	5.78				
90	900	35.8	5.72	0.028	0.109	0.318	0.082
100	1000	39.8	5.60				
110	1100	43.8	5.43	0.288	0.552	0.44	0.817
120	1200	47.8	4.45				
130	1300	51.8	3.95	1.44	1.83	1.62	4.06
140	1400	55.7	3.50	3.45	1.13	3.49	9.63
150	1500	59.7	3.28	7.47	2.57	7.99	20.5

All concentrations in mg/L

### Neutralization via Seawater Dilution 2200 Portal Drainage

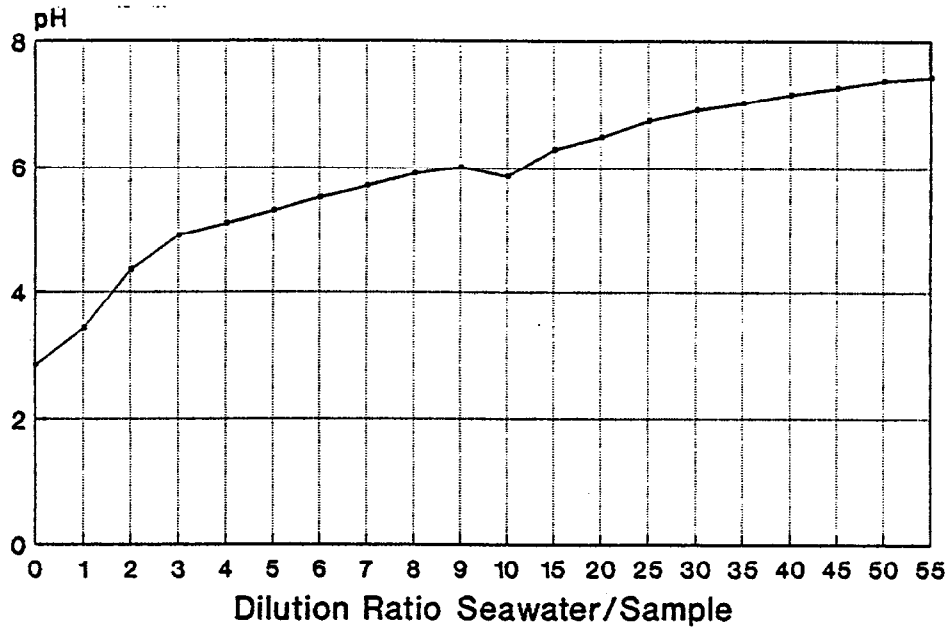


Figure 6.7

### Neutralization via Seawater Dilution 4100 Portal Drainage

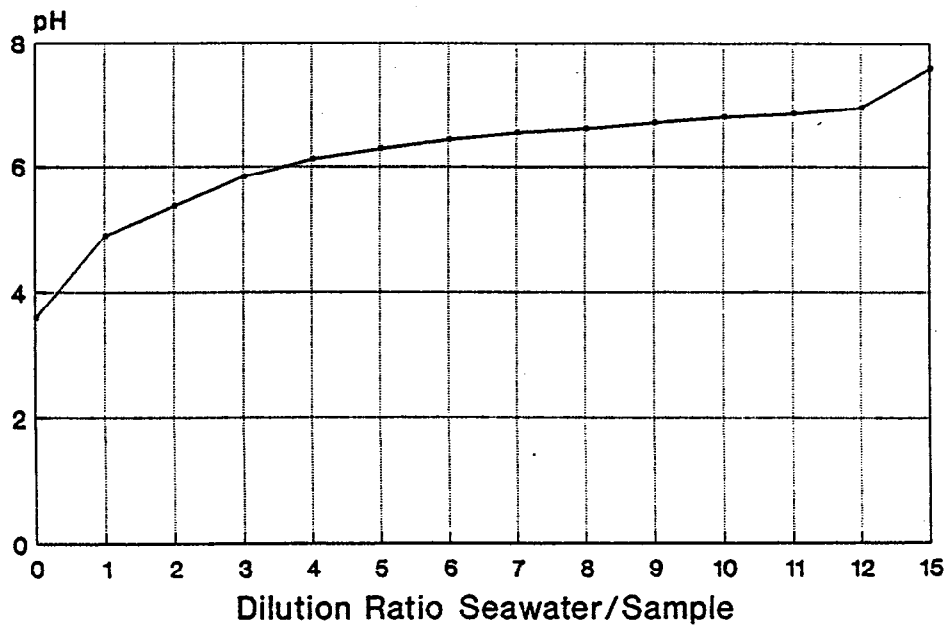


Figure 6.8

### Cation Exchange Test Results 2200 Portal Drainage

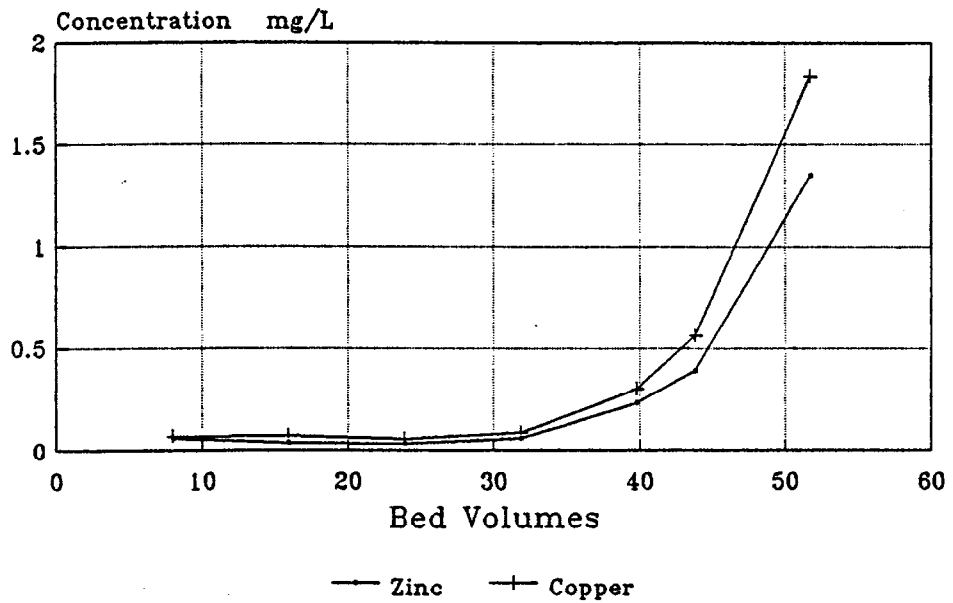


Figure 6.9

### Cation Exchange Test Results 4100 Portal Drainage

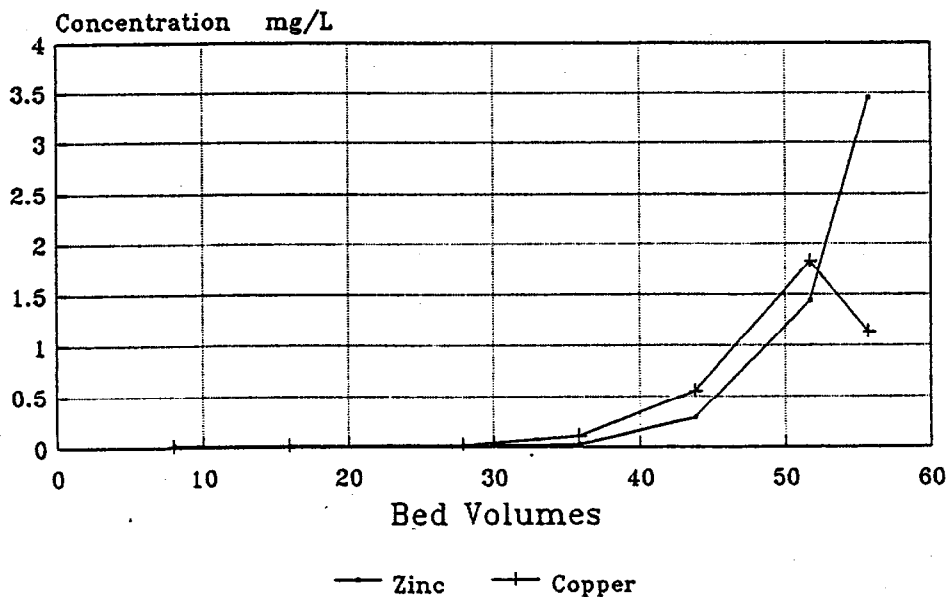


Figure 6.10

In our testing, the resins were successful in picking up the metals; break through occurred at 40 bed volumes in the case of the 2200 sample while breakthrough occurred at 44 bed volumes in the case of the 4100 portal sample. The probable application of ion-exchange to the Britannia situation is discussed further in Section 6.4.3.2.

#### **6.3.4.8 Carbon Adsorption**

Carbon adsorption is a process that would operate in a similar manner to ion exchange, but using activated carbon as the adsorbing medium. Although activated carbon can adsorb metals, the adsorption isotherm has a low X/M ratio (which measures the quantity adsorbed in relation to the concentration in solution). As a result, loadings will be low, and the application of activated carbon given the discharge concentrations at Britannia is not judged practical. No testing was completed on this option.

#### **6.3.4.9 Reverse Osmosis**

This is a rather high cost process that is more suited to smaller operations where recovery of values is important (e.g. plating shop wastes) and for desalination projects. It is not used for treating high volume effluents.

The process involves forcing water through a membrane with high pressure. The membrane is permeable to the water molecules, but not to the salt ions, which remain on the "salt" side of the membrane. Thus, the impurities are rejected as a salty solution rather than as a precipitate which can be discharged to a tailings impoundment. The solution can be recycled if a process is available to use it, but the impurities will build up, usually with detrimental effects on the treatment process (greater pressure will be required, and fouling is likely).

No testing was completed on reverse osmosis.

### **6.4 Contaminant Disposal and Sludge Stability**

The treatment process will remove contaminants from the effluent. There remains the question of how to dispose of these contaminants without harmfully reintroducing them into the environment. This section considers both the possibilities for recovering saleable products, and the options for permanently disposing of the process products.

### 6.4.1 Metal Recovery

The metal content represents the most significant proportion of the contaminants removed. It is conceptually attractive to consider recovering them for sale. This approach is considered in more detail for copper in Section 6.4.3.

MacDonald et al (1989) have summarized some of the considerations when recycling zinc sludges to a conventional electrolytic zinc smelter (as at Trail, for example). Impurities contained in the sludge may interfere with the electrolysis process. In particular, iron in the sludge becomes a disposal problem for the smelter. The water content of sludges may also give problems in handling the sludges, and on the zinc plant water balance.

Given these factors, and the low level of zinc encountered in such sludges, it is unlikely that any payment would be received from the smelter for the metal content. In fact, a fee might be chargeable for accepting the material, in addition to the cost of transporting it to the smelter site.

MacDonald et al (1989) have provided cost estimates for alternatives to conventional lime treatment that incorporate metal recovery. All the alternatives will result in significant cost increases to the mineral industry, but ion exchange and solvent extraction costs are lower in some cases than combined conventional lime precipitation, sludge dewatering, and disposal. Ion exchange and solvent extraction are evaluated specifically in the Britannia context in Section 6.4.3.2.

### 6.4.2 Remaining Contaminants

In addition to copper and zinc, other contaminants (such as iron and cadmium) will be present in the effluent in insufficient quantities or will have insufficient value to justify recovery by known technology. These would be rejected from the treatment processes described above as sludges or concentrated solutions. Metal ions in solution are highly mobile, and so solution disposal would not be considered suitable for final placement. The dissolved metals likely would have to be converted to a solid form where their mobility was restricted to a level tolerable to the deposition environment, and this process would likely result in the formation of a metal-containing sludge.

The stability of the sludge depends on the mode of storage. The sludge should contain some protective alkalinity, which will largely prevent leaching of the precipitated contaminants. However, the alkalinity (calcium carbonate - calcium hydroxide) has a solubility in water, as do the precipitated contaminants. Changes in the sludge's aqueous environment provide the opportunity for continual leaching of the alkalinity and metals to meet their respective equilibria, and eventually, with the alkalinity gone, pH swings could result in heightened dissolution of contaminants.

Thus, the sludge should be deposited in a way that immobilizes the water in its immediate vicinity, discouraging leaching of the sludge constituents. This should not be too difficult to achieve, since the precipitates are very fine, and over lengthy periods should consolidate to a relatively impermeable mass. Local dissolution and reprecipitation of hydroxides and carbonates may help to "cement" the mass together over the long term. Situations to be avoided are those where the sludge may periodically be resuspended and resettled (e.g., shallow ponds), as this provides a change of water and prevents consolidation.

Solidification of the sludge using (for example) a cement binder is an expensive option that may have to be considered in specific circumstances. Sludge solidification is an area of intensive research, and several proprietary solidification systems are available.

Dewatering of the sludge and disposal on the Britannia site may be possible. With sufficient contained alkalinity resulting from unreacted lime, the sludge is unlikely to be a Special Waste as determined by the SWEP leaching test. It could thus be deposited in a properly designed landfill at Britannia, or possibly underground, if non-contact with acidic liquors can be assured. Once dewatered, it could also be transported to a tailing disposal area at one of the minesites in B.C. where such sludges are already being deposited, for example Equity Silver, or preferably an abandoned tailings impoundment closer to Britannia. Acceptable disposition of the Equity sludge after closure has yet to be determined, but whatever solution is adopted might also accommodate sludge from Britannia.

### **6.4.3 Potential for Enhancement and Copper Recovery**

#### **6.4.3.1 Copper Cementation**

The fact that metallic copper has been recovered by cementation at Britannia raises the possibility of updating this process and using the copper produced as a source of revenue to offset treatment costs.

Copper is being produced from bacterial dump leach operations at several locations in the southwestern U.S., as well as at Gibraltar in B.C. The processing sequence used is solvent extraction for copper concentration and purification, followed by electrowinning to wirebar grade cathode copper. Cementation is no longer widely practised due to the low payback received for cement copper and the relatively high cost of scrap steel. The solvent extraction/electrowinning process is simply better and more economic technology for the situations where cementation formerly was practised.

In situations where cementation was practised, the solution feed grade was much higher than is now present at Britannia. When the mine was operating, the solution grade was in the range 30 - 105 ppm copper for the Beach Plant, and 100 - 175 ppm copper for the Mt. Sheer plant (The Anaconda (Canada) Co., 1965). At the low grade experienced now, the "can factor" which measures the iron consumption

relative to the copper production becomes large and the quality of the cement is poor due to contamination with impurities from the large amount of iron that is required. A can factor of 2 to 4 kg cans per kg copper precipitated was a rule of thumb for cementation plants (Spedden *et al.*, 1966); Britannia achieved about 2.2 kg of cans per kg of copper.

The can factor increases with low feed grade because the side reactions that consume iron (e.g. with sulphuric acid, ferric iron, and atmospheric oxygen) become significant relative to the amount of copper that is reacted. Thus, the plant simply contributes ferrous iron to the effluent, while removing copper in an inadequate manner.

At low copper concentrations, the reaction rate of the copper is slow and precipitation efficiency is poor. The recent records for Britannia show little removal of copper from the effluent during its passage through the launders, although the impression is that the launders are somewhat neglected.

The low quality cement must be transported to a smelter, which in the case of Britannia would probably mean Japan. When the mine was operating, it was easy to add the cement to the concentrate which was already being shipped to a smelter. Now, the best option would be to determine if it could be added to concentrates from one of the operating B.C. copper mines as they passed through the terminals in Vancouver. Generally, there would not be much incentive for the concentrate owners to accept it, since it has low copper content, high water content, and poorly controlled impurity content due to the uncontrolled sources of the scrap steel. Net smelter return on this material would be expected to be low.

Thus, copper recovery is dealt with in the next section by considering the alternatives other than cementation for the effluent as it occurs, with adjustment for a reduction of flow to 60 percent of current averages to be achieved by diversion of sources entering the mine in the Jane Basin area. Thereafter, the potential for increasing the amount of copper leached by techniques such as redistribution of solution, crushing the remaining waste and replacing it, *in situ* blasting, etc. is considered.

The basis for this analysis is the flow data summarized in Table 6.12 and the analytical data provided in Table 6.13, and varies somewhat from the data used to develop the cost estimates in Section 6.6 since the hydrological analysis was not complete at the time the work reported here was done. The conclusions from the analysis remain valid.

#### 6.4.3.2 Other Copper Recovery Approaches

Flow and copper assay information for the period 1983 to 1990 was provided by the Ministry. This information was compiled with the data from Moore and van Aggelen to determine plant capacity. The data is summarized in Table 6.12 and 6.13.

Table 6.12 – Summary of Flows, Copper Concentrations and Copper Production from the 4100 Portal, 1983 – 1989

Year	Avg Flow m <sup>3</sup> /h	Avg [Cu] mg/L	Cu Production kg/d
1983	664	19.7	313
1984	561	18.4	247
1985	396	16.1	153
1986	494	19.6	233
1987	597	17.8	256
1988	475	18.5	211
1989	442	15.3	162
<b>Average</b>	<b>518</b>	<b>18</b>	<b>225</b>



Table 6.13 – Summary of Assay Information on 4100 and 2200 Portal Flows

Location	2200	2200	2200	4100	4100	4100	4100	4100	4100	4100	4100
Date	21/06/83	06/12/83	21/04/84	21/06/83	13/07/83	15/08/83	08/09/83	06/12/83	21/04/84	21/05/85	20/06/85
pH	2.9	3.5	2.7	3.4				3.5	3.2	3.8	3.9
Acidity (pH 8.3)	445	109	494	367				306	277	239	258
Sulphate	880	293	895	1620				1700	1620		
Dissolved metals											
Al	31.3	8.05	36.4	39	37.6		35.3	30.6			
B	0.02	0.01		0.17	0.17	0.31	0.34	0.25		0.28	0.20
Cd	0.17	0.06	0.18	0.19	0.14	0.14	0.11	0.14	0.0058	0.12	0.16
Ca	129	53.2	121	441	411	418	478	419		450	387
Co	0.17	<.1	0.22	0.22	0.14	0.3	0.29	0.35		0.1	0.1
Cu	51.4	12.3	43	31.7	30.3	23.6	22.5	18.3	14	11.1	18.1
Fe	36.8	3.16	45.1	11.3	17.1	50.6	56.1	17.1	14	10.8	4.74
Pb	0.12	<.1	0.16	0.22	0.14	0.22	0.09	0.23		0.2	0.2
Mg	38.5	11.3	39.4	88.6	82.4	100	115	83.4		94.3	81.7
Mn	2.6	0.9	3.12	5.47	6.21	8.4	9.2	7.52		7.66	5.98
Ni	0.07	<.05	0.05	0.11	0.08	0.15	0.09	0.07		0.07	0.06
Zn	26.1	8.21	31.6	41.9	30.9	27.1	27.3	28.1	0.21	28.1	30.8
Flow estimate	210										

\* Acidity in mg/L equiv. CaCO3, flow in m3/h, concentrations in mg/L

While there is a downward trend in copper tenor of the solutions since the mine closed, Figure 6.11 shows that the averaging of the copper tenors over the period 1983 - 1989 provides an adequate estimate of copper production for study of the potential for metal recovery. Similarly, Figure 6.12 shows that there is no significant trend in flows, and thus average flows based on this period are adequate for the study.

It is noted (Figure 6.13) that there appears to be an increase in copper tenor with flow rate, and this could be explained by greater flushing and better wetting of sulphide surfaces during wetter years. However, the regression coefficient for the line shown is not statistically different from zero, so the statement cannot be made in certainty. Perhaps a significant correlation could be achieved if the data were analyzed on a monthly basis rather than on an annual basis. Such an activity was outside the scope of this study.

During the period January 1, 1983 to December 31, 1989 the average hourly flow from the 4100 level was 518 m<sup>3</sup>/h carrying an average of 225 kg/day copper for an average concentration of 18 mg/L. We have only one measurement for the 2200 level flow, at 210 m<sup>3</sup>/h in June, 1983, when the 4100 level flow was 927 m<sup>3</sup>/h. Prorating this to the average 4100 level flow, we could say that the average 2200 level flow was 117 m<sup>3</sup>/h. The total average flow requiring treatment would then be 635 m<sup>3</sup>/h.

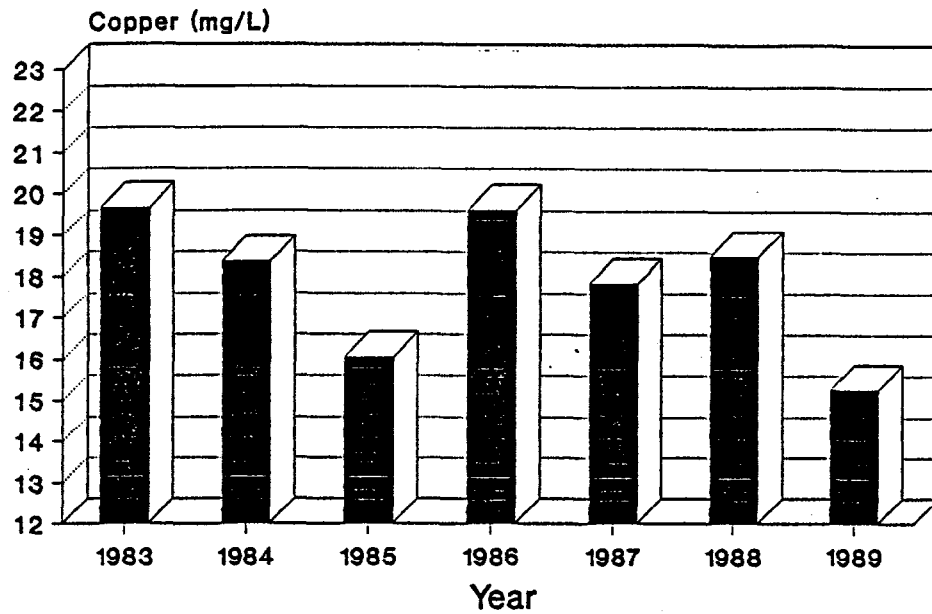
The average plant feed assay would be a flow-weighted average of the two sources. Assume a reasonable first approximation would be to average the given data. When this is done, we get:

	2200	4100	Flow-weighted Average
Acidity:	349.	289.	300.
Sulphate:	689.	1646.	1470.
Copper:	36.	18.	21.
Iron:	28.	23.	24.
Zinc:	22.	31.	29.

The average copper production (100 % recovery) would then be 320 kg/day.

For process design, we must size the plant somewhat above the average flow, since storage will not be enough to compensate for annual variation in flows. The factor to apply will depend on the storage we can provide to average the monthly flow variations, together with the variation in annual flows, which we assume cannot be averaged. For this study, we assume that monthly flows can be averaged by storage behind a bulkhead at the 4100 level, and thus we look at historical data to determine the maximum variability in annual runoff. This indicates that the maximum runoff would be twice the average, so the plant should be sized to treat 1270 m<sup>3</sup>/h. (Analysis subsequent to this study indicates that more flow averaging capability is available, and the treatment plant design is based on a substantially smaller flow.)

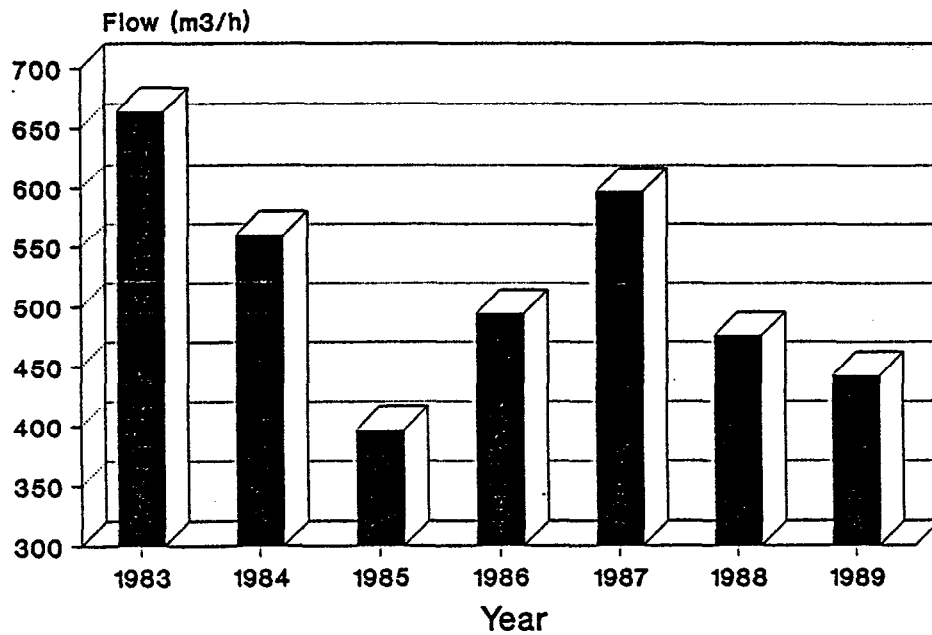
## Trend in Copper Tenor



Based on flow-weighted averages from  
4100 Portal Drainage

Figure 6.11

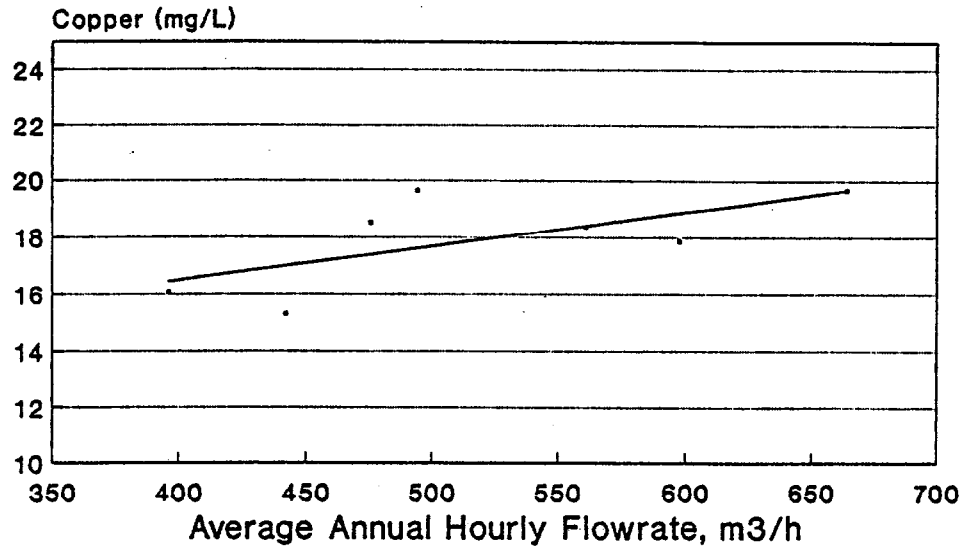
## Trend in Average Hourly Flowrate



Based on 4100 Portal Drainage

Figure 6.12

Correlation Between Copper Tenor  
and Average Annual Hourly Flowrate  
1983 - 1989



Copper concentration based on  
flow-weighted mean - 4100 Portal

Figure 6.13

Two major difficulties arise with this concept: first, even with a reduction of total flow to 60 % of the current rate by appropriate civil works, the plant feed concentration is only 35 ppm copper, worth about \$.08/m<sup>3</sup>. Organic losses to raffinate in a solvent extraction process, even with very careful recovery procedures could hardly be less than 25 ppm (50 ppm is a conventional rule of thumb). The organic phase is valued assuming a 5 % solution of extractant in a proprietary diluent, with extractant at \$16 /kg, and diluent at \$.45/kg:

Extractant	43 kg	\$ 688
Diluent	807 kg	363
	-----	-----
Total	850 kg	\$1051 /m <sup>3</sup>

At 25 ppm (by weight), organic losses total 25 g/m<sup>3</sup> of aqueous raffinate, or 29 x 10<sup>-6</sup> m<sup>3</sup> of the organic mix. The value of this loss is \$0.03 /m<sup>3</sup> of effluent treated. Thus, nearly 40 percent of the revenue is lost in the effluent before any other provision for operating or capital costs is made.

The second difficulty is that the average production rate of 0.32 tonne copper per day is substantially below the level of about 5 tonnes per day considered a minimum economic plant size. From standard cost curves, if a plant were built for Britannia, the capital cost would be of the order of \$3.5 million, with the bulk of the capital in the solvent extraction plant, owing to the large flow (this design assumes only the average flow, not the maximum flow). If we assume a 15 year plant life at 15 % interest, the annual capital charges would be about \$600,000. The annual copper production might be 105 tonnes, so the cost would be about \$6 per kg of copper produced, twice the current spot price.

An ion exchange plant could be added to the front end of the plant to preconcentrate the copper and thereby control organic reagent losses. At an assumed processing rate of 24 m/h (10 USGPM/ft<sup>2</sup>), the columns would require a cross-sectional area of 26 m<sup>2</sup> (average flow only), and the system would cost of the order of \$1.5 million. Their presence would reduce the size, but not eliminate the solvent extraction plant, since conventional ion-exchangers do not offer the selectivity (copper purification capability) that the solvent extraction reagents do. Chelating resins offer better selectivity, but provide a shorter life, and cost approximately ten times as much (MacDonald, et al., 1989).

The regenerant would likely need to be chloride rather than sulphate based, to prevent build up of calcium from causing gypsum fouling of the resin (calcium is the major cation in the AMD). Thus, either sodium chloride or hydrochloric acid could be considered. Recovery of pure copper from chloride solutions is an emerging technology, but cannot now be considered proven commercially. An alternate to solvent extraction and electrowinning would need to be sought.

Another major constituent of the plant feed is iron. A significant portion of this will be in the ferrous form owing to its contact with sulphides in the mine. After adsorption by the cation exchange resin, it may become oxidized and precipitate within the resin as ferric hydroxide. This is a major source of fouling in cation exchange resins (Pelosi and McCarthy, 1982).

All the cations, including all the heavy metal contaminants, would have been removed, but there would be little economic offset in reduced treatment costs. The metal cations (other than copper, assuming it was removed by solvent extraction) would need to be precipitated from the stripping solution, and this would most likely be done by raising the pH to cause metal hydrolysis. The acid remaining in the treated AMD would not have been extracted and would require neutralization.

It therefore appears that recovery of the copper should not be recommended. For the above scheme, neutralization of the effluent would still be required, while the technical and economic feasibility of copper recovery from the stripping solution is questionable. There is no precedent for this type of process on low-strength copper sulphate waste streams. Alternatives such as precipitating a bulk copper-zinc-iron sulphide do not achieve significant purification, and therefore produce a low-value product whose marketability is questionable. The available metal to be produced by any scheme is too small an amount to provide recovery of the amounts of capital likely to be required.

The above work has assumed an enhanced feed concentration due to reduced dilution inflow, while maintaining the metal production from the mine at recent rates. In fact, the evidence to date suggests that the concentration may be reduced at reduced flows due to less effective contacting of solution with oxidized mineral. To achieve the assumed concentrations, there would thus need to be an improvement in solution distribution patterns accompanying the reduced flows. This might be achieved by sprinkling in the glory holes as suggested by Hooper (1970).

Careful thought would need to be applied to the design of such a system; the simple scheme proposed by Hooper may not demonstrate the desired effectiveness. A better alternative may be the flood and drain sequence practised by Denison Mines and recommended by Wright Engineers for Copper Beach Estates in 1983.

These and other alternatives for enhanced copper production are considered next.

#### **6.4.3.3 Enhanced Metal Production Rates**

Enhanced copper production will result from the elimination of bottlenecks constraining higher production rates. In principle, two major activities must take place in order for copper to report in the mine effluent:

the sulphides containing the copper must be oxidized to render the copper water-soluble, and the resulting copper sulphate must be dissolved and transported away from the reaction site.

### Oxidation of copper sulphides

The mechanisms of oxidation of copper sulphides in dumps have been described by Murr (1980). Both direct attack by bacteria and indirect attack by bacterially-produced ferric ion are believed to occur, but the requirement for bacterial involvement to achieve observed rates is not doubted. Thus, optimization of copper sulphide oxidation requires optimization of the conditions for bacterial growth. The bacteria are generally present in all circumstances and do not need to be supplied.

The needs of the bacteria are:

- a source of energy for growth, usually ferrous iron, various sulphides, or elemental sulphur.
- a source of oxygen. The combination of the oxygen with the energy source is the reaction producing the required energy for bacterial processes.
- a source of carbon and nitrogen as building blocks for synthesis of biomass.
- sufficient water to provide a suitable environment for the aqueous-phase biosynthetic and energy producing reactions, as well as to remove the products of bacterial metabolism. A pH of 2-3 in this aqueous medium surrounding the bacteria is optimal, and products of metabolism (e.g., cupric ion) must not rise to toxic concentrations.
- maximum bacterial activity takes place at temperatures around 35 °C.

In a waste dump situation, the energy sources are the iron and sulphide content of the mineralization. The oxygen is provided by circulation of air through the dump (and to a much lesser extent by oxygenated water passing through the dump). Carbon is provided by bacterial fixation of atmospheric carbon dioxide, as well as by carbonates that may be dissolved in the dump water. Nitrogen is supplied as ammonium ion which may be present from breakdown of organic compounds or from blasting agents.

Water is supplied by the dump operator, with groundwater providing an additional source in some situations. The operator may acidify the water if the bacteria are not able to keep the pH down with the products of their metabolism.

Temperature is not usually controlled, and rarely is optimal. Dumps that are actively oxidizing may rise well above the optimal temperature due to the heat of reaction, while those in a more dormant state may be too cold.

Thus, to enhance the oxidation rate at Britannia, it should be determined which factors are limiting, and then steps should be taken to improve the appropriate conditions. Exposed area of oxidizable sulphides is doubtless one important factor, and the reason why the copper production rates have dropped since mining stopped is probably that the available sulphide surface is decreased, both by consumption, and by blinding with reaction products. Wright Engineers (1983) concluded that size reduction of the in-place reserves to expose more surface would be impractical, as material reclamation, crushing, and replacement would be required.

Based on experience with other sulphide dump leach situations, the other probable rate-limiting factors are access to oxygen and water.

### Oxygen

As a result of detailed dump modelling, Cathles and Schlitt (1980) concluded that the optimum dump height should be roughly 15 m, and the optimum width about 30 m. Beyond these limits, convective transfer of oxygen is insufficient to provide for maximum leaching rates, and indeed, in low oxygen areas, copper can be reprecipitated from the leach solutions by contact with sulphides.

Obtaining adequate convective flow in enclosed stopes located between the 2200 and 4100 foot levels at Britannia seems unlikely without the establishment of an extensive system to mechanically introduce air. While this can be tried as an enhancement approach, it could result in the leaching component of the process having a significant operating cost, whereas our initial concept is that the leach solutions are available essentially for free.

As an alternative, a system could be set up where the stopes were periodically flooded and drained. This cycling would draw air into all the interstices having greater than capillary dimensions, ensuring at least some exposure to both oxygen and water. In the Wright Engineers concept, the cycle would take place once a year, with the flooding occurring during run-off, and the draw-down occurring over the remainder of the year. To operate a cycle on any shorter time frame would involve provision of large storage capable of holding the entire flooded solution inventory of the mine, together with pumping facilities for high rate solution movement. Room for such storage probably could not be found at the site.

The volume of solution (and air) that could be exchanged in such a scheme would be the void volume of the broken rock in the stopes. We can calculate that the oxygen content of the introduced solution would have a negligible equivalence in copper.

If the in-place rock has a density of 2.6 tonnes/m<sup>3</sup>, and the swell factor on mining is 1.5, the bulk density of broken rock would be 1.7. The available copper reserves in the mine are stated (Hooper, 1970) to be



245 million pounds (Fairview and West Bluff only), and an average grade has been assumed to be 0.4 % (Stern, 1966). Wright Engineers refer to a C.M.S. report completed in 1980 which estimated reserves available for leaching of about 200 million pounds of copper. It is not clear whether this reserve is currently all broken. Assuming it is, and using the 245 million pound estimate at 0.4 % copper, the rock available for enhancement will be 28 million tonnes occupying a volume of 11 million m<sup>3</sup> if unbroken, 16 million m<sup>3</sup> if broken. The volume of air to be exchanged annually is 5 million m<sup>3</sup>, containing 1 million m<sup>3</sup> or 45,000 kg-moles of oxygen.

The oxygen will be used to oxidize sulphides to sulphate, which appears in the leach solution. From our historical data, the sulphate level is 1470 mg/L when the copper level is 21 mg/L. Since 1 kg-mole of oxygen is equivalent to 0.5 kg-mole of sulphate, the sulphate which could be produced as a result of the air exchange would be 22,000 kg-moles, or 2140 tonnes. Prorating by current solution concentrations, the copper that would accompany this would be about 30 tonnes per year, or 80 kg/day. This does not represent an enhancement over current production.

The above calculation is based on very rough data and is open to criticism. The exchanged air does not represent the only source of oxygen, as can be seen from the on-going acid production in the absence of enforced exchange. Some sulphate that is produced may be deposited in the mine as jarosite and gypsum. Nevertheless, there does not appear to be great scope for oxygen transfer enhancement through cyclic operation.

### Water

Solution management is considered critical in the operation of heaps and dumps. Since no special activities are completed at present to ensure careful solution distribution, it is assumed that enhancement should be possible in this area.

One means of ensuring complete wetting of all the rock is the flooding approach discussed above. Since only one cycle per year seems feasible, the productivity of this approach is related to the ability of the bacteria to carry on oxidation during the dry cycle, at which time concentrations of metals in the pore water surrounding the bacteria could rise to inhibitory levels.

If the 28 million tonnes of rock assumed above drained to 10 percent moisture, the pore water would contain 2.8 million kg of copper at 1 g/L concentration. If entirely flushed during each year, this quantity would represent a copper production of 7.7 tonnes per day, which would likely be an economic production rate.

The bacteria are known to be able to tolerate concentrations much higher than this, so this approach could be feasible as a water distribution method.

The Wright Engineers report questions the ability of a plug at the 4100 level to withstand the pressures that could be reached with a periodic flooding concept. These relate to the strength of the rock used to hold the plug in place. Should periodic flooding appear desirable, a detailed engineering investigation of this problem would be necessary.

If it were desirable to obtain a flow through system for air in the stopes in order to ensure adequate oxygen for the oxidation reactions, a percolation system might be necessary. Certainly, vertical air flow would be restricted during any time the stopes were not completely drained. Hooper (1970) planned to use this approach in his feasibility study.

While the runoff and groundwater could not be captured for redistribution and percolation, a recycle stream was proposed as a means of enhancing solution contact. Recycle of leach solution after copper recovery is routinely employed in dump leaching to make use of the acid it contains, thereby ensuring the necessary low pH for metal dissolution and bacterial activity. The iron content may also be beneficial to the bacteria, who will oxidize ferrous to ferric in preference to other substrates. Finally, it contains bacteria, and thereby serves to redistribute the organisms and provide a good distribution of activity throughout the dump.

A bleed from the recycle loop equivalent to the amount of runoff and groundwater would be taken for copper recovery and subsequent treatment and discharge in order to maintain a water balance on the leaching system.

Three difficulties can be foreseen with this approach. First, the stopes apparently are inclined from the vertical. There is thus a hanging wall, and all ore located beneath this would have poor access to the percolating solutions which tend to move vertically downward.

Second, the vertical depth for percolation is much greater than is encountered in most dump leach situations. Once the solution has been distributed at the broken ore surface in the stope, it is no longer in the operator's control, and the formation of channels which take most of the flow has been demonstrated in pilot testing (Murr, 1980). The greater the percolation depth, the greater is the opportunity for maldistribution of the solution. As an extreme example, it has been found that packed columns in chemical process plants (which are designed to facilitate well-distributed percolation) should incorporate solution redistribution mechanisms at intervals of at least every 6 or 7 m (Treybal, 1980).

Third, as the solutions percolate through the broken ore, migration of the fines content and precipitates results in the formation of impermeable layers in the waste (Bruynesteyn and Cooper, 1974). This results in areas which are limited in access to both oxygen and solution. Murr (1980) observes that "dump fluffing" by blasting to increase permeability results in a decrease, not an increase in permeability. Thus, this should not be a useful strategy at Britannia.

#### Summary of copper production enhancement

The possibility of increasing copper production to economic rates is intriguing, and while questions have been raised especially in the area of oxygen availability, we feel the idea cannot be rejected out of hand. A full scale test is required to determine what can be achieved by approaches such as outlined above. The down side is that attempts at enhancement will eliminate the gradual decline in copper tenor in the effluent that has occurred since the mine closed. The ability to "walk away" from the AMD problem will thus have been postponed (if it can ever be achieved).

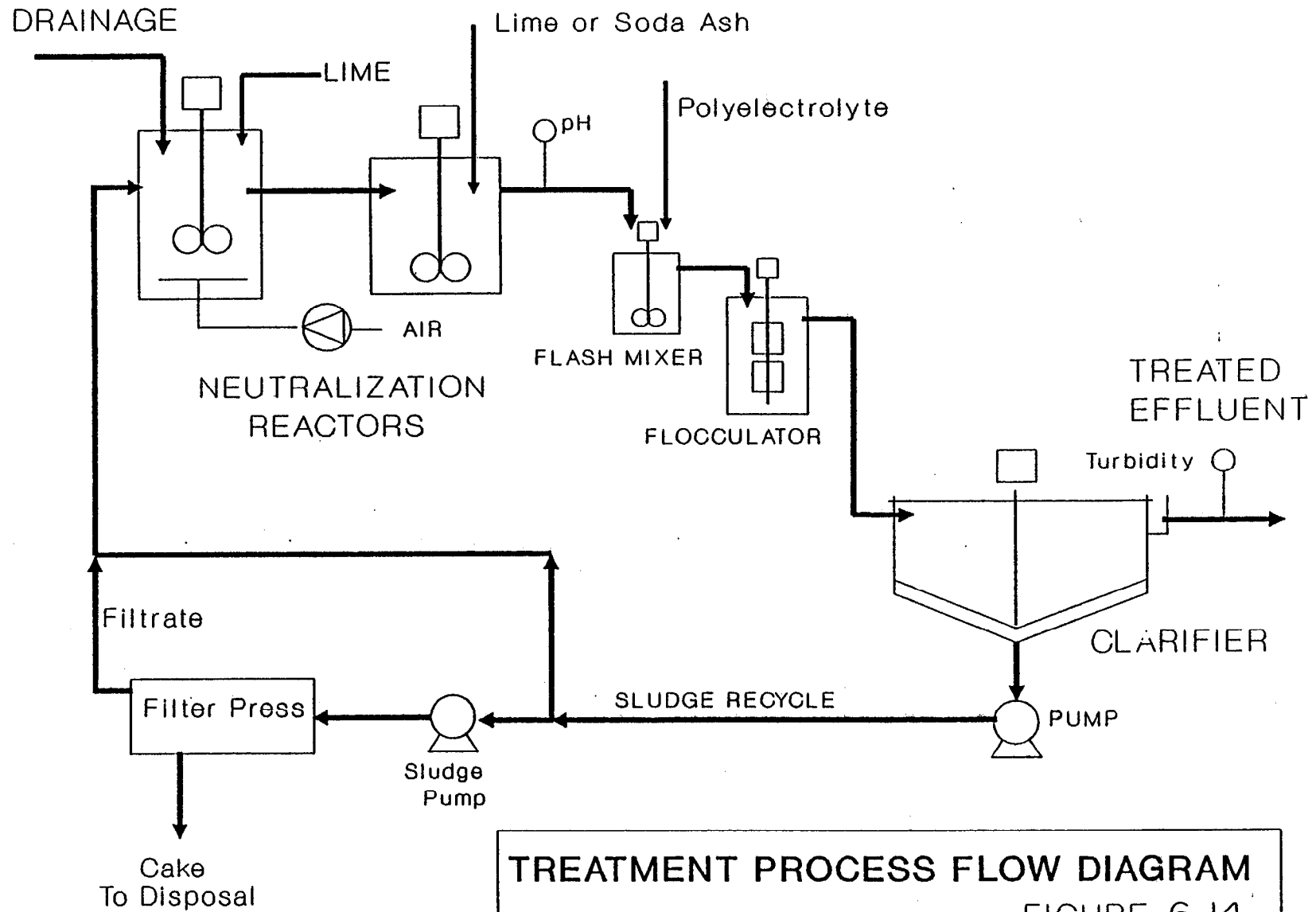
### **6.5 Proposed Treatment Plant Design**

The design sequence consists of the following steps:

- review testwork, experience of others
- decide on a process concept; i.e., high density sludge versus straight through, lime versus limestone, etc.
- determine the process flowsheet to meet the concept
- balance the flows, chemistry on the flowsheet
- list the major equipment
- based on the process flows and design criteria established in testing, calculate the equipment size
- make equipment layout drawings that define the major structural and mechanical requirements
- complete detailed engineering as required to specify and procure the equipment, select contractors and build the plant

The engineering does not necessarily proceed in as defined a sequence as laid out above; in reality, there is usually some iteration, as layout constraints may alter the flowsheet, etc. However, the lime treatment process is fairly straightforward, so that few complications would be expected.

The proposed process flowsheet is given in Figure 6.14. The design assumes a combined lime and soda ash treatment system or a two-stage lime treatment system. The following sections outline the design selected and discuss some of the pertinent considerations when setting the process concept and selecting and sizing the major equipment.



**TREATMENT PROCESS FLOW DIAGRAM**  
**FIGURE 6.14**

## **6.5.1 Reagent Mixing and Storage**

### **6.5.1.1 Lime Slaking**

Burned quicklime is usually received in bulk truckloads as minus 6 mm material. It is pneumatically conveyed to a silo, which is equipped with a baghouse to remove the lime dust from the transporting air.

Lime slaking is usually accomplished in one of three ways: detention slakers, paste slakers, or lime mills. These alternatives, and some others are described in the National Lime Association Bulletin 213 (Anonymous, 1976). Paste slakers are said to provide a higher yield of slaked lime owing to the higher temperature at which they operate. Lime mills offer the advantage that grit removal is not required, since it is ground up and discharged with the product slurry.

The current design incorporates a paste slaker, but this is not a final choice. The relative costs and benefits would be weighed in arriving at a final choice as part of the purchasing process.

### **6.5.1.2 Flocculants**

Flocculants used in lime treatment plants usually will be of the polyacrylamide type, although some natural products are used as well. They are quite soluble in water, but their long chain polymeric nature makes the solid particles tend to stick together if initially wet in a mass, so that so-called "fisheyes" are formed in the stock solution. These may take a considerable time to completely dissolve. Therefore, proprietary mixing equipment is sold by the chemical suppliers and others, which is designed to individually wet each flocculant particle before discharging it into the mixing vessel. These range from simple funnels on educators connected to the feed water flow to complex completely automated pneumatic feeders with wetted wall columns. For this design, a fully automatic preparation unit is included.

As an alternative, flocculants can be purchased as premixed fluid dispersions which combine readily with water.

The long chain nature of the polymers makes the solutions very viscous, and thus stock solution strengths are limited to about 0.2 to 0.5 percent. At this strength, the solution may be stored for a day or so; eventually it degrades due to polymer hydrolysis.

Before addition to the process, the solution is diluted to 0.05 percent, which allows the polymer to fully "unwrap" and hydrate for maximum effectiveness. Ideally, 30 minutes residence time should be provided after dilution to allow this hydration to proceed prior to adding the solution to the process stream.

In order to minimize polymer chain breakage, shearing of flocculant solutions is minimized at all stages. This is a consideration in agitator selection as well as pumping; typically, progressing cavity pumps are used.

Flocculants are not corrosive, and special materials selection for the handling equipment are not required.

### **6.5.1.3 Soda Ash**

Soda ash is available in bulk trucks or in bags. For the loadings considered in the Britannia design, purchase in bags (approximately 25 kg each) would lead to unreasonably high levels of labour, so that bulk purchase would be preferred. Discussions with local chemical suppliers suggest that the most economic means of delivery could be in 1 tonne bags, which could be unloaded into a small hopper as required. Alternatively, unloading of bulk trucks or rail cars to a silo is done pneumatically as for quicklime. Soda ash dissolves readily and usually may be handled without any special materials of construction considerations.

## **6.5.2 Oxidation and Precipitation Reactors and Agitators**

The tanks are sized to provide the required residence time, typically in the range 30 to 60 minutes. The residence time will be more effectively used if at least two tanks are connected in series to defeat short circuiting. Addition of internal feed and/or exit baffles can help eliminate short circuiting as well, but can cause problems for descaling. Best practice will keep the reactor internals as simple as possible.

Detailed design of a reactor in which air is to be dispersed for oxidation is a very complex subject, involving estimation of required dissolved oxygen level, oxidation rates, and mass transfer coefficients. This type of analysis is rarely undertaken in practice.

Instead, a chemical demand for oxygen is calculated based on the reactions desired, and the rate at which they must proceed to be complete in the desired residence time. This rate is used with a rule of thumb oxygen utilisation from dispersed air (typically 10-20 percent) to estimate the rate at which air must be supplied. Tank geometry is usually selected somewhere near a "square" design (tank diameter equals liquid depth) which permits good mixing with a single impeller. Four anti-swirl baffles will be included, equal to approximately 1/10 tank diameter. Agitator vendors are asked to recommend an agitator that will disperse the required amount of air in the reactor, comfortably below the flooding point. Solids suspension is not usually a major consideration in these designs, as the solids are typically very fine and suspend easily. More likely, the selection will be governed by gas dispersion.

Agitator vendors can provide recommendations on the best arrangement for sparging air under the impeller.

Once the feed is neutralized, its corrosive properties diminish. Depending on the pH at which the first stage is operated, it may be possible to construct all the tanks in mild steel. Alternatively, it may be desirable to make at least the first stage of stainless steel. Polymer linings or free-standing FRP are not considered, as they would be damaged by periodic descaling operations.

### 6.5.3 Clarifier Selection and Sizing

Selection of the type of clarifier will be the result of the experience and preference of the designer, the recent experience of others with similar plants, and an economic evaluation based on solicitation of quotes from vendors. Where the effluent quality is specified in terms of total metals rather than dissolved metals, an effective clarifier is essential, and post-filtration may be a useful option to ensure continuous achievement of discharge specifications. Post filtration is not proposed for Britannia.

In the high density sludge process, flocculation is critical to successful performance, and thus, separate reactors may be provided for "flash" mixing of flocculant with the feed followed by gentle agitation while the flocs form and consolidate. Alternatively, flash mixing and flocculant may be provided by addition of the flocculant to a pipeline or launder upstream of the clarifier. Sometimes some experimentation with the addition point and method is required after start up, since the hydraulic conditions in a pipeline or launder can be hard to define. Recommendations for design of flash mixing and flocculation facilities are given by the U.S.E.P.A. (Anonymous, 1975).

The clarifier is sized based on rise rate to achieve overflow clarity, and residence time/sludge depth for underflow density. Test work is completed to verify that standard design factors for this type of application apply, but might not be used directly in sizing the equipment, especially for smaller units. Testwork results for a rise determination using the lime treatment process are provided in Figure 6.15. This predicts an overflow rate of 1.4 m/h (0.57 USGPM/ft<sup>2</sup>).

Rake mechanisms are selected with assistance from the vendor based on the anticipated sludge characteristics. A feature that may be included is a lifting device (of greatest concern when high density sludges are being sought) which may be manually controlled or instrumented to actuate based on torque.

Other details vary with the design selected. Vendors should give assistance in understanding the various features that should be evaluated in arriving at the final specification.

# Settling Test Results Lime Treatment Process

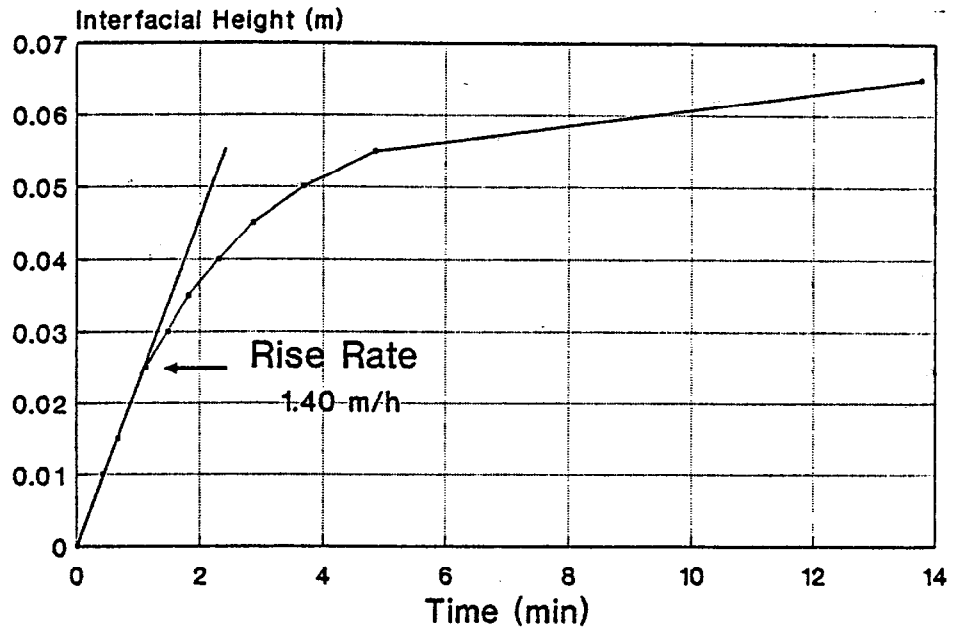


Figure 6.15



Both positive displacement and centrifugal pumps are used in pumping underflows. The sludge can be abrasive (inclusion of some lime grit, for example), and so the pump should be designed for fine abrasive slurry service. Rubber lined centrifugal pumps, and air or mechanically operated diaphragm pumps are common. Diaphragm pumps offer the advantages that their periodic motion can help to "fluidize" the thixotropic sludge and keep it flowing out of the tank; and pumping rate is easily controlled to allow balancing of a suitable recycle rate with obtaining a desired sludge density in the clarifier. Flow rate variation when using centrifugal pumps may be achieved by variable speed drives, or by automatic valves on the pump discharge.

These pieces of equipment are handling neutralised slurries in which the fluid is supposedly discharge quality; materials of construction selection is therefore not restrictive. Mild steel or concrete tanks and steel mechanisms are adequate; various internals such as lamella packs may be fabricated from plastics if desired (e.g., to give a surface with a low friction coefficient).

#### **6.5.4 Instrumentation**

##### **6.5.4.1 Reagent dosing system and control**

Lime is added as a slurry, which makes continuous controlled addition at low rates difficult. To maintain adequate velocities periodic additions at full flow are often used. The controller may set the frequency of pulse or pulse length or both. Automatic valves should be selected suitable for slurry service. Branch lines are short and preferably come vertically off the ring main so that solids settle back into the main when flow is shut off. The valve is then mounted in a horizontal run so that a column of solids cannot settle on the control element, preventing its functioning.

Control of pH and minimization of supersaturation which can lead to scaling will be enhanced by staged addition of lime in a series of stirred tanks. Control also requires sufficient tank residence time to give stable operation. A minimum of 20 minutes per tank is suggested.

Makeup of the soda ash solution to the desired strength requires either instrumentation to weigh out the required amount to make a batch of reagent solution, or a mix tank sized to accept the entire contents of a 1 tonne bag. For Britannia, a system using weighing instrumentation is specified.

Soda ash solution addition rate may be automatically controlled to maintain a desired pH leaving the last stage of the process, or it may simply be set at a constant dosage to match the feed rate (which is not expected to vary much in the short term).

Flocculant addition is not usually automatically controlled. The diluted flocculant solution is metered through a needle valve, and its addition rate is monitored with a rotameter. Flocculant performance will be enhanced if dilution is accomplished with sufficient residence time (30 minutes) to allow complete hydration of the polymer before addition to the process.

#### **6.5.4.2 Turbidity measurement**

Process upsets in AMD and similar types of treatment plants are commonly associated with an increase in the level of suspended solids in the effluent. The condition can be monitored using an on-line turbidity meter coupled to an alarm system. Sampling and analyses of effluent samples during start-up can provide a correlation between suspended solids, turbidity and other parameters. In this way, a reasonably instantaneous measurement of general effluent quality can be provided, and serious upset conditions avoided. Regular maintenance and recalibration of the turbidity meter is essential in order to provide good quality data. The use of an on-line instrument such as a turbidity meter can reduce the amount of sampling and analytical work required for process control and for regulatory compliance.

Automatic control can be used to provide an alarm, and to divert the product stream back to the feed source when turbidity specifications are exceeded.

#### **6.5.5 Analytical support**

At the Britannia treatment plant, analytical capability will probably be limited to transmission spectrophotometry (not atomic absorption spectrophotometry), and simple titrations. Plant operators can be trained to complete this type of analysis, but the levels of detection achievable are often not adequate for monitoring treated solutions. Clean facilities are required to avoid contamination of assays, and independently determined results can be more credible for external reporting. Thus, normal analytical support consists of a mixture of on-site determinations for day to day plant control, together with external assays for reporting purposes. Commercial laboratories have equipment such as inductively coupled plasma spectrophotometers (ICP) that can perform multi-element assays more quickly and cheaply. These labs also have personnel trained in the more difficult wet techniques required for some circumstances.

The frequency of sampling will be specified in the permit under which the facility operates. Beyond this, when a treatment plant is in stable operation, samples once a shift are the most that might be required, and this might be reduced to daily sampling or even less frequently when feed flow and chemical makeup do not vary quickly, as should be the case at Britannia. The flow path through the mine, together with storage capacity to be provided behind the 4100 bulkhead will help to even out the plant loading.

## 6.6 Capital and Operating Cost Estimates

### 6.6.1 Flow and Loading

Critical to the sizing and costing of the treatment process are the flow rate and acidity loading used for design.

#### 6.6.1.1 Geotechnical/Hydrological Analysis

##### Flow

The mean annual flow of a 100 year wet year with half of Jane Basin catchment diverted and 100 % diversion efficiency gives 396 m<sup>3</sup>/h. Use of a diversion efficiency of 70 % and assumption of surges in flow at 10 % above mean to allow for inflow exceeding storage at some point in the year provides a mean annual flow of 511 m<sup>3</sup>/h and a peak flow of 576 m<sup>3</sup>/h.

##### Loading

Assuming diversion of surface water at the open pits, with all drainage directed to the 4100 level and mean annual flow of a 100 year return wet year, the calculated loading from available data compared to "typical" long term flow as shown in December 1990 sampling, the copper loading will be 233 kg/day, and the zinc loading will be 413 kg/day.

#### 6.6.1.2 Historical Data

In Section 6.6, the flow and loading data from the 4100 portal were reviewed for the period 1983 to 1989. It was concluded that allowing for a contribution from the 2200 flow, the mean copper level would be 21 mg/L, with the related acidity at 300 mg/L of CaCO<sub>3</sub> equivalent, in a flow averaging 635 m<sup>3</sup>/h. The analysis showed that the predicted average copper production was 320 kg/day, so the associated acidity loading would be 4.6 tonnes/day CaCO<sub>3</sub> equivalent.

The geotechnical prediction of loading is lower owing to the assumption of the application of diversions to the water entering the mine. However, although a reduction in loading is expected to accompany a reduction in water entering the mine, this is an uncertain prediction. For plant design, a conservative assumption would be no change in the historical loading. The geotechnical predictions should then be examined as a sensitivity case.

### 6.6.2 Design Basis

The peak 100 year return flow estimated will serve as the design basis for sizing equipment where size is dependent on flow rate. Where equipment size is dependent on loading (reagent preparation and dosing equipment), the historical loading of 4.6 tonnes/day CaCO<sub>3</sub> plus a design factor will be used.

A plant should not be designed to operate for any significant time exactly at its ultimate capacity; this leaves no flexibility to correct problems or allow for down time. Considering the uncertainty in the data, we therefore feel it prudent to apply a 20 % design factor. The design conditions for equipment sizing will therefore be:

<u>Flow</u>	576 m <sup>3</sup> /h x 1.20	=	690 m <sup>3</sup> /h
<u>Loading</u>	4.6 tonnes/day x 1.20	=	5.5 tonnes/day CaCO <sub>3</sub>

For the operating cost estimate, reagent consumption will be estimated based on the historical loading of 4.6 tonnes/day CaCO<sub>3</sub> (no contingency included), with the predicted reduced loading due to application of diversion considered as a sensitivity.

### 6.6.3 Equipment List

The flowsheet which is the basis for this estimate is shown in Figure 6.14, and an equipment list is given in Table 6.14. If an all lime system were selected, it is likely that the soda ash facilities would be retained, as the addition to capital cost is not large, and the flexibility to add the second reagent could provide security should the effluent chemistry change.

Acid mine drainage is supplied from behind the bulkhead at the 4100 level at a controlled rate to the first of two treatment tanks in series. Each tank has been sized to provide a residence time of 30 minutes based on feed flow. Lime is added to the first tank to maintain a controlled pH which may be set in the range of 5 to 7. Underflow sludge from the clarifier is recycled to this reactor to increase the solids concentration and improve flocculant effectiveness. Sodium carbonate solution is added to the second tank at a rate which is ratioed to the feed flow rate to provide a dosage of 242 kg/tonne CaCO<sub>3</sub> acidity in the feed.

Slurry overflowing the second tank flows by gravity to a flash mixer of 30 seconds residence time, where diluted flocculant solution is dosed at a rate of 1.5 ppm. From this mixer, it passes to three lamella clarifiers in parallel.

Table 6.14 – Major Equipment List – Britannia  
Water Treatment Plant

Item No.	No. Req'd	Description	HP/ unit	Conn. HP	Total Cost
1	2	Neutralization tanks, 21'x23'	0	0	184,000
2	2	Agitators, ss	30	60	45,000
3	1	Air blower 100 cfm	10	10	7,000
4	1	Clarifier mech. 87' cs	5	5	91,000
5	1	Clarifier tank, 87'dia. cs	0	0	210,000
6	1	Filter press, 100 ft <sup>3</sup>	1	1	123,000
7	4	Diaphr. pumps	0	0	38,000
8	1	Lime silo/slaker/storage pckge	15	15	180,000
9	2	Hor. cent. pump, srl, 2"	5	10	6,000
10	1	S. ash bin, 5 t, c/w scr. conv.	3	3	18,000
11	1	Soda ash mix tank, 9x10', cs	0	0	10,000
12	1	Agitator	3	3	6,000
13	1	Soda ash stock soln tank, 9x10'	0	0	10,000
14	2	Metering pump, 0-5 USGPM	1	2	9,000
15	1	Floc mixing unit	3	3	24,000
16	1	Mix tank, 4x6, pe		0	1,000
17	1	Agitator	1	1	2,000
18	2	Moyno transfer pump	3	6	7,000
19	1	Stock tank 8x8 cs	0	0	7,000
20	2	Moyno process feed pump	3	6	7,000
21	1	Plant air compressor, 500 cfm	150	150	36,000
22	1	Chain hoist	0	0	1,000
23	4	Submersible pumps, 8"	50	200	57,000
<b>Total</b>				<b>475</b>	<b>\$1,079,000</b>

The clarifiers have been sized based on an industry standard loading of 0.5 USGPM/ft<sup>2</sup> rise rate for water treatment-type hydroxide sludges. This is a lower rise rate than demonstrated in testing, but a design factor is necessary to provide an operating range for the equipment. Most of the sludge is withdrawn and recycled to the first reactor tank. As required to maintain a constant sludge inventory, a portion is fed to a filter press using an air-operated diaphragm pump (an additional pump is provided for spare capacity).

The filter press is a caulked, gasketed, recessed plate unit sized to handle the required sludge volume in four batches per 24 hour day. The cake is discharged at a density of 35 percent solids and is trucked to disposal at a site to be determined.

Clarifier overflow is continuously monitored for turbidity. If turbidity rises, the plant feed is stopped, and the overflow is recycled to the first reactor for retreatment. Clear overflow is directed to one of three monitoring ponds, where it is held until analytical results verify that it meets discharge criteria. Submerged pumps then discharge the pond through the existing outfall into Howe Sound. The costs for the monitoring ponds are added in Table 6.15, after the factors for piping, etc. have been applied.

Burned pebble quicklime is delivered in tanker trucks and pneumatically conveyed to a 45 tonne lime silo. It is slaked as required, with the milk of lime being stored in a tank from which a recirculation loop is pumped. Milk of lime is dosed to the process as required to maintain the set pH.

Soda ash is delivered to the plant in 1 tonne bags which are emptied into a hopper as required. It is conveyed to a mix tank to make batches of soda ash solution in water. The soda ash solution is dosed to the process by a metering pump whose rate is automatically adjusted to maintain a constant ratio to the feed rate.

#### **6.6.4 Factored Capital Cost Estimate**

The purchased equipment cost has been factored into a fixed capital cost as shown in Table 6.15. This estimate is of the type described as a Study Estimate by Weaver and Bauman (1973) and is expected to be within 30 percent of the cost to build the plant described.

The capital costs shown do not include GST. We have not investigated the tax position of the plant. The overall capital cost would increase by about 7 percent if the GST is applicable.

The estimated fixed capital cost for the plant is \$3,400,000. To this would need to be added owner's costs, financing charges, head office costs, and other costs as dictated by the owners' accounting methods. This estimate is based on all new equipment, and significant savings might be realized if appropriate used equipment became available at the time purchasing commenced.

**Table 6.15 – Factored Capital Cost Estimate  
for Britannia Treatment Plant**

<b>Design Flow</b>	<b>690</b>	<b>m3/h</b>
<b>Total purchased equipment</b>		<b>1,079,000</b>
<b>Installation at</b>	<b>25%</b>	<b>269,750</b>
		<hr/>
<b>Total installed equipment</b>		<b>1,348,750</b>
<b>Piping</b>	<b>15 %</b>	<b>202,313</b>
<b>Electrical</b>	<b>12</b>	<b>161,850</b>
<b>Instrumentation</b>	<b>7</b>	<b>94,413</b>
<b>Buildings, struct.</b>	<b>20</b>	<b>269,750</b>
		<hr/>
		<b>728,325</b>
		<hr/>
<b>Total Physical Plant</b>		<b>2,077,075</b>
<b>Monitoring ponds</b>		<b>456,000</b>
<b>Capital spares</b>	<b>5%</b>	<b>53,950</b>
<b>Startup and commissioning</b>	<b>2%</b>	<b>21,580</b>
		<hr/>
		<b>531,530</b>
		<hr/>
		<b>2,608,605</b>
<b>Eng. and Construction</b>	<b>18 %</b>	<b>469,549</b>
		<hr/>
		<b>3,078,154</b>
<b>Contingency</b>	<b>10 %</b>	<b>307,815</b>
		<hr/>
<b>Total plant fixed cost</b>		<b>3,385,969</b>
		<hr/> <hr/>
<b>Say</b>		<b>\$3,400,000</b>
		<hr/> <hr/> <hr/>

The factor assumed for structures assumes that a building would be provided to house an office, control room, control and switchgear, reagent storage and preparation equipment, pressure filter, air blower, rudimentary maintenance facilities, and the plant air compressor. The lime silo, process tankage and clarifier would be outside.

## **6.6.5 Operating Cost Estimate**

### **6.6.5.1 Lime Treatment System**

The operating cost based on treatment with lime (no soda ash) is summarized in Table 6.16, and amounts to \$0.16/m<sup>3</sup> treated at the historical mean operating flow of 635 m<sup>3</sup>/h. Reagent usage assumes the treatment of 1669 tonnes acidity (as CaCO<sub>3</sub>) per year, and a lime dosage/acidity ratio derived from testwork of 0.88 kg pebble lime/kg CaCO<sub>3</sub>. The major cost items are for lime, soda ash, labour, and sludge disposal. This covers only the costs at the site, and does not allow for any costs associated with management of the facility beyond a general foreman. Chemical vendors have advised us that GST would apply to these purchases, and this has been included at 7 percent.

Lime consumption has been calculated from the loading assuming that the purchased pebble quicklime would have 92 % active lime content, and in the gypsum-saturated solution, would react to an extent of 82 %. This leaves residual alkalinity to stabilize the sludge.

Operating labour has been costed at \$18.00/h, with allowance of 30% for payroll burdens and 10 % for overtime. The foreman is assumed to be paid an annual salary of \$45,000, plus burdens at 27%. Power is charged at an average cost of 5 cents per kWh. Maintenance supplies are included at 3 % of purchased equipment; operating supplies are allowed at \$51,006 /y. Lime is quoted at \$118 /tonne FOB Britannia. GST is additional to the quoted prices.

### **6.6.5.2 Lime and Soda Ash Treatment System**

Based on the use of conservative lime and soda ash dosages, there is an operating cost advantage of approximately \$20,000/yr at the operating flow of 635 m<sup>3</sup>/h. (Soda ash is quoted at \$0.28 /kg for 1 tonne bags, FOB Britannia.)



Table 6.16 – Operating Cost Estimate

<b>Conditions</b>	
Flow	635 m3/h
Mean Copper	21 mg/L
Equivalent Acidity	300 mg/L CaCO3
Acidity Loading	4.57 tonne/day
	1669 tonne/yr
Sample Acidity	285 mg/L CaCO3
<b>Lime Treatment Option</b>	
Lime Dosage	250 mg/L CaO
Ratio Lime Dosage/Acidity	0.88 kg CaO/kg CaCO3
Percent Reacted Lime	85 %
Annual Lime Consumption	1722 tonne/yr
Lime Cost	126 \$/tonne F.O.B. Brittonia
Annual Lime Cost	\$216,993
<b>Lime and Soda Ash Treatment Option</b>	
Lime Dosage	150 mg/L CaO
Ratio Lime Dosage/Acidity	0.53 kg CaO/kg CaCO3
Percent Reacted Lime	90 %
Annual Lime Consumption	976 tonne
Annual Lime Cost	\$122,963
Soda Ash Dosage	40 mg/L Na2CO3
Ratio Soda Ash Dosage/Acidity	0.14 kg Na2CO3/kg CaCO3
Annual Soda Ash Consumption	234 tonne
Soda Ash Cost	300 \$/tonne FOB Brittonia
Annual Soda Ash Cost	\$70,264
Total Cost Lime and Soda Ash	\$193,227
Percent Savings	11.0
<b>Polyelectrolyte Consumption</b>	
Percol 156 Dosage	1.5 mg/L
Daily Consumption	22.86 kg/day
Ratio Poly/Acidity	5.0 kg/tonne
Annual Consumption	8343.9 kg
Cost	5 \$/kg
Annual Cost	\$41,720

**Summary of Operating Cost Estimates**

	Lime		Lime + Soda Ash	
Labour	\$223,891	24.5%	\$223,891	25.1%
Total Reagent Cost	\$258,713	28.3%	\$234,947	26.4%
Supervision	\$57,150	6.3%	\$57,150	6.4%
Contract Maintenance	\$25,000	2.7%	\$25,000	2.8%
Maintenance Supplies	\$46,260	5.1%	\$46,260	5.2%
Op. Supplies Assay	\$51,006	5.6%	\$51,006	5.7%
Power	\$88,294	9.7%	\$88,294	9.9%
Sludge Disposal	\$164,017	17.9%	\$164,017	18.4%
<b>Total Annual Operating Cost</b>	<b>\$914,331</b>	<b>100.0%</b>	<b>\$890,565</b>	<b>100.0%</b>

Optimization of this system would require additional process development work, especially in terms of evaluating the benefits of recycle, i.e. relief of gypsum supersaturation. The treatment system design that has been proposed would be sufficiently flexible to accommodate additional reagents such as soda ash. A system using lime and soda ash may be preferable to a lime-based system due to the inherent stability of copper and zinc carbonates compared to the equivalent metal hydroxides. For this report, the use of operating cost estimates based just on lime is warranted since this represents proven technology.

#### **6.6.6 Impact of Acidity Load Reduction on Operating Costs**

Diversion of surface water and partial flooding of the workings may result in a reduction in the acidity loadings that the treatment plant would be required to process. In order to evaluate the impact of load reductions on operating costs a series of calculations were conducted to generate Figures 6.16 and 6.17.

Figure 6.16 is a plot of total operating cost in 1991 dollars versus percentage of acidity load reduction, using data in Table 6.16 as the base condition. Two cases are provided; the first case, which includes sludge dewatering and off-site trucking and disposal ("Complete Sludge Cost") and the second case where sludge disposal costs have been eliminated. The second case would apply if the discharge of a thickened sludge to the underground workings is feasible.

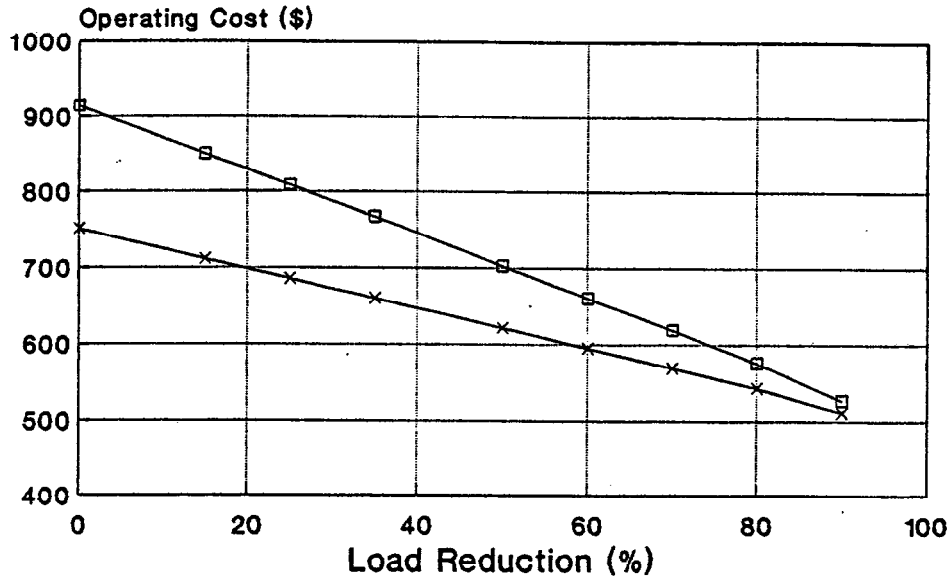
Figure 6.17 is a plot of operating cost reduction versus loading reduction expressed as percentage, again using Table 6.16 data as the base condition. Comparative data for lime and lime plus soda ash are included. As expected, reductions in loading can significantly reduce reagent costs. However, other costs such as labour and power are not as sensitive to loading reductions. For example, based on Figure 6.17, a 50 % reduction in loading reduces the operating cost by approximately 15 % (for the case of underground deposition of sludge).

#### **6.7 Discussion**

Continuous pilot plant operation is needed to better define sequential lime/sodium carbonate requirements, generate sufficient sludge for recycle, settling, pressure filtration, and sludge stability tests. Potential savings from using lime/soda ash vs straight lime should be verified in continuous operation.

A pilot plant could be set up as a demonstration of the technology being applied by the mining industry to solution of the acid mine drainage problem. An opportunity exists to combine this pilot work at the Britannia site with the summer season at the Mining Museum. The exhibit could be expanded to include demonstrations of other approaches as well.

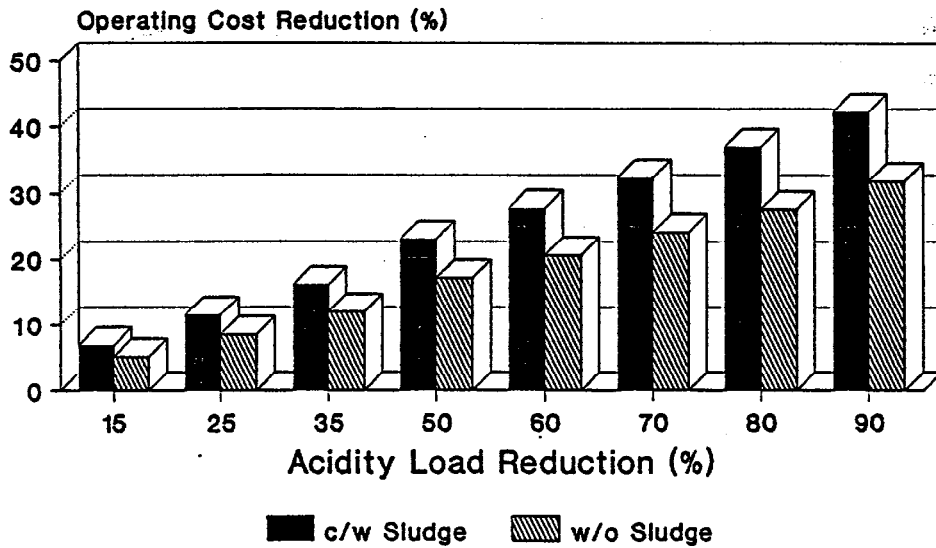
### Evaluation of Operating Cost Annual Operating Cost vs Load Reduction



Based on Lime Treatment Option

Figure 6.16

### Evaluation of Acidity Load Reduction Operating Cost vs Load Reduction



Based on Lime Treatment Option

Figure 6.17

The cost shown is based on the historical loading of 320 kg Cu discharged per day. Because the operating cost contains significant elements that are not related to loading, the change in operating cost with loading reduction is modest, and may not justify some types of expenditures to reduce loading.

Labour is a significant component of operating cost. There is a need to have a responsible person present or nearby at all times to deal with alarms and upsets, make up reagents, and clean the filter press. In fact, cleaning the press is a job better done with two operators, and this may dictate a switch to an automatic press at about three times the capital cost.

A potential labour savings could be obtained if some labour could be associated with the Mining Museum or the real estate development. Perhaps the plant could be a showcase for industry mitigation activities, although lime processes are inherently a bit messy for public exposure.

Further investigation into fate of the sludge is required. Trucking is a significant expense, so that study of further dewatering measures would be worthwhile. An added benefit of an automatic filter press is the probable higher solids content achievable in its product filter cake, with consequent reduced transport costs.

The sludge is not expected to be a Special Waste due to its residual alkalinity. Thus, disposal in a landfill at Britannia is an option, if an appropriate site can be found. The quantity, at about 9000 tonnes wet weight per year is not beyond accommodation in this type of facility. The best solution would be to pump the clarifier underflow to deposit in the workings below the 4100 level. This would eliminate the need for the filter press, eliminating an operating headache, reducing the need for labour and maintenance. To justify this, knowledge about water flows and chemistry in the area of the proposed deposition would need to be developed and interpreted.

The provision of monitoring ponds for the plant discharge represents a significant capital expenditure (nearly one-third the equipment cost). The costs shown for these are based on the volume of material to be excavated and the area to be lined, but do not represent more than an order-of-magnitude estimate. With a turbidity meter installed on the clarifier overflow to detect process upsets, consideration should be given to eliminating these ponds in favour of a direct discharge to Howe Sound through a sea-water dilution chamber as an aid to dispersion.

If flow equalization by storage within the mine turns out not to be feasible, the plant size might be doubled. Using a standard  $2/3$  power law scale factor, the capital cost might be expected to increase by a factor of 1.6. Due to the small size of the plant, and the fact that the lime system quoted to us is already significantly oversized, the increase in cost might not be as great as predicted by this factor.

The GST position of the plant needs to be clarified to make both the capital and operating cost estimates more certain.

## **6.8 Permit Conditions and Monitoring**

### **6.8.1 Discharge Standards**

In the original terms of reference it was requested that consideration be given to the current permit, and to future discharge criteria. Upon review of the draft report, it was agreed that this is the responsibility of the regulators and the company and will be developed with the remediation plan. The following discussion is provided for information, and as an initial approach to discharge criteria.

Since the discharge is not under a Waste Management Permit and is not an operating mine it theoretically falls outside the jurisdiction of the Provincial Objectives established under the Waste Management Act. Assuming that it was an active mine it would be regulated according to the criteria provided in Table 6.17. Under Federal Regulations the Fisheries Act which controls the release of deleterious substances would be the only legislation that applies. Since the effluent is toxic to fish then it would legally be deemed to be in contravention of the Fisheries act. However since the previous owner is no longer responsible for operation of the facility it would be difficult for Federal Fisheries to advance a case under the Fisheries Act that would in fact result in any improvement in the current status. The Metal Mines Liquid Effluent Regulations do not apply in this case since Britannia is no longer an active mine. If Britannia were an active mine then it would be controlled by the Metal Mine Liquid Effluent Regulations established under the Fisheries Act (Table 6.17). Once remedial measures such as installation of a treatment plant are complete then it will possible to bring the discharge into compliance with both Federal and Provincial regulations and objectives and issue a permit.

Discharge Standards can be set by two methods, either at the point of discharge or on a receiving water basis. MMLER criteria or Level "A" Provincial Objectives would apply if the discharge is regulated on a "end of pipe basis". Criteria based on the receiving environment could utilize criteria such as the CREEM guidelines. In this case these criteria would apply outside a specific zone of influence of the discharge. This zone of influence could be defined spatially as a specified offset both horizontally and vertically from the point of discharge. The use of receiving water criteria may result in a less stringent discharge criteria than "end of pipe" specifications, especially if the allowed zone of influence is large due to the fact that the receiving environment is already heavily impacted. Based on studies conducted by EP in 1985 (Goyette & Ferguson) the current discharge is having a significant impact on the Howe Sound in that elevated copper levels are evident in oyster and mussel tissue over a wide spatial distribution. However it should be pointed out that the discharge of tailings and cyclone overflow at the mouth of Britannia Creek over a 50 year period may be responsible in part for the tissue metal levels noted.

Table 6.17 Effluent Water Quality Regulations and Objectives

Parameter	Effluent		Receiving Environment
	MMLER*	Provincial Objectives **	CCREM Objectives ***
pH	5	6.5-8.5	6.5-9.0
Total Suspended Solids	25-50	25 - 75	<10% increase
Toxicity	100-80%	100%	
Cyanide		0.1-0.5	0.005
Ammonia N		1.0-10	1.4-2.2
Metals			
Aluminum		0.5-1.0	0.005-0.1
Arsenic	0.5-1.0	0.05-0.25	
Cadmium		0.01-0.1	0.0002-0.0018
Copper	0.3-0.6	0.05-0.3	0.002-0.004
Iron		0.3-1.0	0.03
Lead	0.2-0.4	0.05-0.2	0.001-0.007
Manganese		0.1-1.0	
Nickel	0.5-1.0	0.05-0.5	0.025-0.015
Zinc	0.5-1.0	0.2-1.0	0.03

*All concentrations in mg/L unless otherwise specified*

\* Metal Mines Liquid Effluent Regulations - Fisheries Act 1970

\*\* Pollution Control Objectives for the Mining, Smelting and Related Industries of B.C. (1979), Pollution Control Board, Ministry of Environment, Victoria

\*\*\* Canadian Water Quality Guidelines, 1978, Task Forces Water Quality Guidelines of the Canadian Council of Resource and Environmental Ministers (CRREM), 1987

The establishment of future discharge standards will be a function of three factors:

- i) receiving water objectives;
- ii) compliance with existing regulations established under the Waste Management Act;
- iii) availability of the technology to meet the desired effluent quality.

In regards to the available technology it can be assumed that if a treatment system is installed such as one that involves the use of lime, the incremental costs associated with meeting a specific "end of pipe" criteria are small relative to the total capital cost of the facility and baseline operating costs. Some other treatment options such metal recovery w/o neutralization may however require some flexibility in terms of discharge standards.

In summary it is recommended that "end of pipe" criteria be established for this facility. These criteria should consider both the receiving environment and the treatment technology used.

### **6.8.2 Surveillance Monitoring Requirements**

Acidic drainage is currently originating from both the 2200 and 4100 portals. Diversion of the 2200 drainage will be essential to minimize the impact on Britannia Creek. Potentially a treatment facility could be constructed in the vicinity of the 4100 portal or the existing copper recovery plant in order to treat all acidic drainage from the property. Once these treatment facilities are in place it will be necessary to monitor the success of the diversion and drainage collection system. This will potentially involve the establishment of a surveillance network that would be included as part of the permit. The network would include stations both upstream and downstream of the property. The network would be monitored on a regular basis to determine whether the collection and diversion facilities are functioning as designed. Stations would be monitored via sample collection and the use of portable instruments. Trends in the data would in turn be used to evaluate the efficiency of collection and diversion. A secondary purpose of the surveillance program would be documentation of improvements in water quality in the drainage basin relative to both historic information and data collected as part of this program.

The surveillance network would be monitored for physical parameters such as pH, conductivity and suspended solids as well as acidity and sulphate plus both total and dissolved aluminum, cadmium, copper, iron, lead, nickel and zinc.

Surveillance would be conducted monthly at most stations that are accessible however it would not likely be necessary to conduct all the analyses on a frequent basis since in many cases physical measurements

can be used for assessment once a relationship between items such as dissolved metals and conductivity are established.

The location of the surveillance network stations would mirror that conducted as part of this program. Design of a surveillance program will depend to some extent on the availability of funds and personnel to conduct the work.

### **6.8.3 Description of Available Monitoring Equipment**

The surveillance network described above would rely on the use of both samples and on-site monitoring equipment. Analyses of metals will depend on the collection and analyses of samples using commercial labs while some of the physical measurements can be conducted using either hand held instruments or portable equipment. Physical measurements that can be made using portable equipment include,

- pH
- turbidity
- conductivity
- dissolved oxygen

These instruments are readily available commercially at a reasonable cost. The use of instruments does however require some training regarding operation and maintenance.

pH would be used in specific areas to determine if acidic upstream drainage is reaching the receiving environment and whether there is any deviation from base line conditions

Turbidity measurements can be used to measure the impact of bank erosion in the upper portion of the drainage basin on downstream water quality. Field turbidity data can be correlated with lab suspended solids data.

Conductivity can be used to detect the presence of contaminated groundwater resulting from acidic drainage. since the presence of dissolved salts resulting from the generation of acid will increase conductivity readings above background.

Dissolved oxygen can be used to assess the impact of acidic drainage on the receiving environment since these drainages can contain oxygen demanding materials, however in most cases changes in conductivity would be more sensitive to the presence of acidic drainage in a particular stream. Dissolved oxygen would be of more significance to documenting the status of a stream in regards to water quality for aquatic habitat.



Instrumentation would naturally be included in the installation of a treatment facility. Chemical dosage of reagents such as lime would be controlled by a combination of flow measurement and pH while final effluent quality would be monitored using on-line turbidity. The level of sophistication of the instrumentations installed in the plant will be controlled by the process used and the availability of trained operators. Experience has indicated that instrumentation can significantly improve performance and reduce the frequency and severity of upsets. However it is essential that operators be trained to operate and maintain instrumentation to maximize plant performance.

## **7.0 CONCLUSIONS AND RECOMMENDATIONS**

### **7.1 Conclusions**

The following discussion of conclusions and recommendations is based on the investigations completed to date. The site assessment within the scope of this investigation has been completed, and the areas in which remediation measures for acid drainage are required both in the short and long term identified. Potential physical and chemical control and remediation measures have been identified. The next stage of the investigation requires more detailed information in specific areas than has been available to date. These recommendations for further studies specific to the control measures have been included in Section 6, Table 6.1. In some areas, further field investigations are required to define technical details and construction costs.

As a result of the investigation to date, the following conclusions and recommendations are indicated:

- acid generation and metal leaching is continuing from the Britannia site resulting in elevated metal concentrations, notably copper and zinc, and other metal loading to Britannia Creek and Howe Sound;
- water quality data are limited from the site, metal loads currently discharged are less than those discharged during operation, they have however not significantly decreased in the past five years;
- variability in precipitation and runoff affects the sources and relative contribution of fresh water and contaminated drainage, and the drainage quality and loadings from the mine workings. The fluctuations can be both seasonal and vary within shorter periods. Long antecedent dry periods followed by high precipitation events (such as the summer/fall of 1990) can result in short term high metal loading in drainage waters;
- the primary sources of metal loading appear to be drainage from the underground workings: from the 2200 and 4100 portals and to a lesser degree the 2700 portal. Pit walls and waste rock above the 2200 level appear to be oxidized and potentially contribute oxidized products to the drainage which enters the underground workings;
- there is a significant store of oxidation products within the underground workings and within waste rock, tailings and fill material used throughout the site;
- the Britannia site could therefore potentially continue to generate contaminated drainage for many years. Physical and/or chemical control measures would be required for improvement in drainage

water quality to Britannia Creek and Howe Sound. It must also be recognized however that in the short term, as control measures are implemented, drainage water quality may worsen due to physical disturbance of wastes and release of stored oxidation products;

- acid generation is well established at the Britannia Site and, given the extent of development underground, and surface openings (gloryholes) which essentially act as "funnels" for precipitation, control of acid generation and contaminant migration are not feasible;
- diversion of surface water inflows is estimated to potentially reduce the quantity of flow through the workings by approximately 50 percent;
- flooding of all of the mine workings is not feasible, however partial flooding may significantly reduce the volume and loading to the treatment plant;
- diversion of inflows with ditches and covers to the mine waste at the pits and the 2200 level portal may be successful at reducing inflows for the control of acid generation and metal leaching. Some collection and treatment, removal of waste or covering may be required, depending on the extent of the waste, and hydrogeological regime;
- the underground workings could be used for flow attenuation to the treatment plant. Partial flooding of the workings would provide surge control prior to the treatment plant, to allow control of the influent concentrations and flow rates to the treatment plant;
- diversion of flow through the mine waste at the 4100 and mill levels does not appear feasible for control of ARD, although the extent of the waste and drainage water quality is not known. Collection and treatment may be required;
- remediation or reclamation of the "tailings" pond near the 4150 level will probably require a cover and/or flooding if it is determined that this material is (or could potentially) contribute contaminants to drainage water. If flooding is selected, then the embankment stability must be examined;
- an opportunity exists to test copper leaching enhancement and recovery if desired. The outcome of such an activity cannot be predicted with certainty, but it should delay eventual "burnout" of the acid forming reactions if the enhancement is only moderately successful;
- copper recovery from the present effluent is expected to be uneconomic;

- significant reduction in copper, zinc and acidity loading are technically feasible by application of a lime or lime and soda ash treatment process. The capital cost for such a treatment process is estimated at \$3,400,000 assuming two stage lime or lime plus soda ash treatment, with clarification and sludge dewatering by filter press. The annual operating cost for the lime process is estimated at \$914,000 assuming the historical loading. Capital expenditures to reduce the loading will bring about modest reductions in annual operating costs.

## **7.2 Recommended Approach to Future Work**

The recommended approach to future work at the site outlined in the following section incorporates in a staged approach both potential remediation options and specific investigations. More information is required about the physical and chemical nature of the site which must be acquired in the next year or two. Experience with site investigations and remediation of abandoned sites has shown the importance of thoroughly understanding the specific sources of acid generation and contaminant loading, and relative contribution to drainage water, prior to implementing remediation measures. As remediation measures are put in place, in the staged program developed below, specific monitoring must be conducted to determine the effect of the control measure, and refine the treatment design and overall remediation requirements.

The Britannia site could also provide a rather unique opportunity for research and investigation into ARD processes and control. The site is readily accessible by road with a long history of ARD potentially from all components of any mine site; open pits, underground, tailings, waste rock and construction materials. Consideration should be given to developing the site as a research facility, and also as an opportunity to disseminate information to the public. The water quality information and topographical information has been incorporated into a CAD package and also a GIS system, providing an ideal medium for preparation of visual displays of information for the public. More recent topographical information is being acquired at this time, and incorporated into the GIS package for further investigation into the general applications of GIS to environmental site assessment by SRK.

The potential control options for the Britannia site and required investigations specific to those options have been described and tabulated in Section 6. Additional investigations to more fully understand the site and select remediation measures are defined in the following discussion, with control measures that should be considered in the near future. Within the next year it is recommended that the following be implemented:

- water quality monitoring should be conducted on at least a monthly basis for a 14 month period along Britannia Creek, Jane Creek, and Mineral Creek for key parameters including flow, pH, acidity/alkalinity, conductivity, sulphate, and total and dissolved Cu, Fe and Zn;

- detailed seep surveys in the spring (freshet) and fall rain periods. The areas around which these surveys should be concentrated are the waste dumps, pit walls, tailings and settling ponds, and in areas where waste has been used for fill;
- surface investigations to identify and sample drainage to the Furry Creek basin from underground. Access to this area is restricted however and it is not clear which adits may still extend to surface. If the 1800 level adit is still free draining, this would provide useful information on the relative contribution of oxidized products over the depth of the underground workings;
- surface solids sampling including acid-base accounting and shake flask test from open pit/ glory hole walls, waste rock, tailings, and from underground (if access is possible) to determine the potential for further oxidation and the extent of stored products;
- geotechnical investigation in Fairview and East Bluff pit area to evaluate ground conditions and availability of material for diversion of the upper reaches of Jane Creek from entering the glory holes (open pits) and limiting the downstream contribution to Britannia Creek near the 2200 level;
- if feasible, design and construct surface water diversion(s); and,
- conduct detailed monitoring to evaluate effect on drainage water quality and flows;
- refine treatment system design and overall remediation plan.

It is also important to reroute drainage from the 2200 level portal away from Britannia Creek, as this drainage is a major source of loading to the Creek. Investigations by Klohn Leonoff (1985), and the provincial mines inspector have indicated limited access into the 2200 level adit, and obstructions within a few hundred feet of the portal. The physical stability of the workings and obstructions to flow must be investigated to ensure that plugging of the portal will not simply cause the drainage to issue from another opening. This type of investigation should be conducted by qualified mining/geotechnical engineers in cooperation with the Mines Inspection Branch officers. The available records of the development of the underground workings are limited (the Copper Beach Estates files and Museum of Mining files were both searched), and it may be necessary to interview former mine employees and review their personnel files to develop a better understanding of the extent of the underground workings and surface openings.

Another key issue that should be addressed within the next year is waste at the 2200 level. The nature and quantity of wastes at this level, must be quantified through original topography, survey of current elevations and sampling (by trenching, test pits or drilling). A limited hydrogeological investigation to determine flushing and contaminant loading to Britannia Creek should be considered.

Diversion of surface water and elimination of the 2200 portal drainage should improve the drainage quality and quantity to Britannia Creek and Howe Sound. Within the next year however it would not be possible to identify, design and install remediation and control measures throughout the site; the site is simply too complex, with insufficient physical, chemical and hydrologic data available. In the short term therefore some form of collection and/or chemical treatment may be required if an immediate improvement in water quality is feasible. It should be recognized however that if treatment is selected as the long term control for the site, this interim treatment plant design may not be appropriate.

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**APPENDIX A**

**DETAILED RESULTS OF SITE INVESTIGATION**

WATER QUALITY DATA

	Britannia Creek B1	Jane Creek J1	2200 Portal	Jane Creek J2	Jane Creek J3	Britannia Creek B2	Britannia Creek B3	Britannia Creek B4	Mineral Creek M1	Britannia Creek B6	4150 Portal	Furry Creek
	1990	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec
<b>Physical Tests</b>												
Total Dissolved Solids	10	50	1170	430	430	20	60	40	30	30	1250	10
Hardness CaCO3	7.5	37.1	610	198	197	13.7	30.9	25.7	22.7	20	979	8.13
pH	7.02	7.49	3.49	3.99	3.97	6.97	5.41	5.72	7.93	6.42	4.24	8.36
Total Suspended Solids	<1.0	4	21	20	17	5	6	5	<1	5	13	<1
<b>Dissolved Anions</b>												
Acidity CaCO3	2.7	3	764	184	181	5.3	20	10.4	2	5	332	<1
Alkalinity CaCO3	6.2	8.2	<1	<1	<1	3.8	<1	<1	8.7	1.2	<1	5.6
Chloride Cl	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Fluoride F	0.03	0.08	0.85	0.42	0.42	0.03	0.07	0.06	0.05	0.04	1.52	0.03
Sulphate SO4	<1.0	30.9	1470	390	391	10.2	44.1	30.1	12.8	19.4	1930	<1
<b>Nutrients</b>												
Ammonia Nitrogen N	0.007	0.009	0.14	0.037	0.034	0.018	0.015	0.013	0.007	0.011	0.061	0.008
Nitrate Nitrogen N	0.062	0.16	<0.005	<0.005	<0.005	0.059	0.031	0.059	0.012	0.085	<0.005	0.1
Nitrite Nitrogen N	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Total Phosphorus P	0.003	0.002	0.038	0.009	0.014	0.002	0.004	0.003	0.004	0.002	0.009	0.002
<b>Total Metals</b>												
Aluminum T-Al	0.074	0.878	61.8	15.5	15.4	0.468	1.76	1.19	0.094	0.44	32.2	0.092
Antimony T-Sb	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001
Arsenic T-As	<0.0001	0.0002	0.0011	0.0005	0.0004	0.0002	0.0002	0.0002	0.0003	0.0001	0.0005	0.0002
Cadmium T-Cd	0.0002	0.0055	0.327	0.083	0.079	0.0017	0.0098	0.0064	0.0003	0.0031	0.199	<0.0002
Copper T-Cu	0.015	0.455	97.3	23.5	23.1	0.453	2.49	1.75	0.017	0.838	20.9	0.011
Iron T-Fe	<0.03	0.042	42.6	10.5	9.13	0.414	1.02	0.702	0.114	0.212	9.32	0.033
Lead T-Pb	<0.001	<0.001	0.1	0.029	0.029	<0.001	0.002	<0.001	<0.001	<0.001	0.24	<0.001
Mercury T-Hg	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005
Selenium T-Se	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
Zinc T-Zn	0.008	0.85	49.1	13	12.7	0.264	1.44	1.02	0.045	0.546	39.1	0.006
<b>Dissolved Metals</b>												
Calcium D-Ca	2.94	12.7	118	46.1	46.2	4.24	8.31	7.2	8.21	5.99	244	2.75
Copper D-Cu	0.015	0.302	97.1	23.4	23.1	0.397	2.48	1.74	0.013	0.813	20	0.011
Iron D-Fe	<0.03	<0.03	39.3	9.72	8.46	0.355	0.172	0.128	0.093	0.051	7.84	<0.03
Lead D-Pb	<0.001	<0.001	0.1	0.029	0.029	<0.001	0.002	<0.001	<0.001	<0.001	0.24	<0.001
Magnesium D-Mg	0.26	1.28	75.1	19.6	19.5	0.73	2.4	1.84	0.523	1.2	87.9	0.3
Potassium D-K	0.12	0.24	0.46	0.22	0.23	0.12	0.13	0.15	0.11	0.14	1.82	0.25
Sodium D-Na	0.63	1.28	2.79	1.55	1.59	0.66	0.75	0.78	0.94	0.84	11.4	0.84
Zinc D-Zn	0.008	0.815	47	12.7	12.7	0.264	1.39	0.992	0.043	0.544	36.5	0.006

Results are expressed as milligrams per litre except were indicated otherwise.  
 < - Less than detection limit

CALCULATED LOADINGS (kg/d)

	Britannia Creek B1	Jane Creek J1	2200 Portal	Jane Creek J2	Jane Creek J3	Britannia Creek B2	Britannia Creek B3	Britannia Creek B4	Mineral Creek M1	Britannia Creek B6	4150 Portal	Furry Creek
	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec	06-Dec
Estimated Flow L/s	560	56	13	69	69	661 calc	730	1040	130	1770	160	NA
<b>Physical Tests</b>												
Total Dissolved Solids	484	242	1314	2563	2563	1142	3784	3594	337	4588	17280	NA
Hardness CaCO3	363	180	685	1180	1174	782	1949	2309	255	3059	13534	NA
pH	7.02	7.49	3.49	3.99	3.97	6.97	5.41	5.72	7.93	6.42	4.24	8.36
Total Suspended Solids	0	19	24	119	101	286	378	449	0	765	180	NA
<b>Dissolved Anions</b>												
Acidity CaCO3	131	14.5	858	1097	1079	303	1261	935	22.5	765	4590	NA
Alkalinity CaCO3	300	39.7	0.00	0.00	0.00	217	0.00	0.00	98	184	0.00	NA
Chloride Cl	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	NA
Fluoride F	1.5	0.4	1.0	2.5	2.5	1.7	4.4	5.4	0.6	6.1	21.0	NA
Sulphate SO4	0.0	150	1651	2325	2331	583	2781	2705	144	2967	26680	NA
<b>Nutrients</b>												
Ammonia Nitrogen N	0.34	0.04	0.16	0.22	0.20	1.03	0.95	1.17	0.08	1.68	0.84	NA
Nitrate Nitrogen N	3.00	0.77	0.00	0.00	0.00	3.37	1.96	5.30	0.13	13.00	0.00	NA
Nitrite Nitrogen N	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	NA
Total Phosphorus P	0.15	0.01	0.04	0.05	0.08	0.11	0.25	0.27	0.04	0.31	0.12	NA
<b>Total Metals</b>												
Aluminum T-Al	3.58	4.25	69.4	92.4	91.8	26.7	111.0	106.9	1.06	67.3	445.1	NA
Antimony T-Sb	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	NA
Arsenic T-As	0.00	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.02	0.01	NA
Cadmium T-Cd	0.01	0.03	0.37	0.49	0.47	0.10	0.62	0.58	0.00	0.47	2.75	NA
Copper T-Cu	0.73	2.20	109	140	138	25.9	157	157	0.2	128	289	NA
Iron T-Fe	0.00	0.20	47.8	62.6	54.4	23.6	64.3	63.1	1.3	32.4	129	NA
Lead T-Pb	0.00	0.00	0.11	0.17	0.17	0.00	0.13	0.00	0.00	0.00	3.32	NA
Mercury T-Hg	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	NA
Selenium T-Se	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	NA
Zinc T-Zn	0.39	4.1	55.1	77.5	75.7	15.1	90.8	91.7	0.51	83.5	541	NA
<b>Dissolved Metals</b>												
Calcium D-Ca	142	61.4	133	275	275	242	524	647	92	916	3373	NA
Copper D-Cu	0.73	1.5	109	140	138	23	156	156	0.15	124	276	NA
Iron D-Fe	0.00	0.00	44.1	57.9	50.4	20.3	10.8	11.5	1.0	7.8	108	NA
Lead D-Pb	0.00	0.00	0.11	0.17	0.17	0.00	0.13	0.00	0.00	0.00	3.32	NA
Magnesium D-Mg	12.6	6.2	84	117	116	42	151	165	5.9	184	1215	NA
Potassium D-K	5.8	1.2	0.52	1.3	1.4	6.9	8.2	13.5	1.2	21.4	25.2	NA
Sodium D-Na	30.5	6.2	3.1	9.2	9.5	37.7	47.3	70.1	10.6	128	158	NA
Zinc D-Zn	0.39	3.9	52.8	75.7	75.7	15.1	87.7	89.1	0.48	83.2	505	NA

Results are expressed as kilograms per day except where indicated otherwise.  
 < - Less than detection limit

WATER QUALITY DATA

	1990	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep
	Britannia Creek B1	Jane Creek J1	2200 Portal	Britannia Creek B2 CALC	Jane Creek J2	Britannia Creek B3	Mineral Creek	Britannia Creek B5	4150 Portal	Treatment Plant In	Treatment Plant Out	Britannia Creek B6	
<b>Physical Tests</b>													
Total Dissolved Solids	20	110	990	66	110	70	130	50	50	3000	3400	60	
Hardness CaCO3	9.5	52.6	412	26.3	49.6	28.5	55.9	22.1	1400	1400	1380	23.2	
pH	7.02	7.37	4.29	NA	7.56	6.98	7.23	6.92	4.46	4.55	4.38	6.98	
Total Suspended Solids	<1	5	14	10	6	10	<1	4	11	2	17	2	
<b>Dissolved Anions</b>													
Acidity CaCO3	1	4	142	NA	3	15	2	3	266	254	242	2	
Alkalinity CaCO3	8.2	16.3	<1.0	NA	14.6	5.9	14.2	7.4	<1.0	<1.0	<1.0	7.1	
Chloride Cl	<0.5	<0.5	<0.5	0.9	<0.5	0.8	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	
Fluoride F	0.05	0.05	0.47	0.07	0.07	0.07	0.06	0.06	1.29	1.31	1.30	0.07	
Sulphate SO4	3	36	559	27	37	28	44	16	1740	1730	2030	18	
<b>Nutrients</b>													
Ammonia Nitrogen N	<0.005	<0.005	0.045	0.045	<0.005	0.041	<0.005	<0.005	0.047	0.045	0.045	<0.005	
Nitrate Nitrogen N	0.120	0.140	<0.005	0.072	0.150	0.079	0.093	0.130	<0.005	<0.005	<0.005	0.140	
Nitrite Nitrogen N	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	
Total Phosphorus P	0.004	0.007	0.005	0.006	0.005	0.006	0.190	0.013	0.014	0.002	0.003	0.011	
<b>Total Metals</b>													
Aluminum T-Al	0.04	0.53	15.00	0.93	0.57	0.90	0.02	0.14	28.00	28.40	28.60	0.20	
Antimony T-Sb	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	
Arsenic T-As	0.0002	0.0005	0.0002	<0.0001	0.0005	<0.0001	<0.0001	<0.0001	0.0005	0.0007	0.0002	<0.0001	
Cadmium T-Cd	<0.0002	0.0037	0.0570	0.0028	0.0036	0.0029	<0.0002	0.0010	0.1240	0.1240	0.1170	0.0010	
Copper T-Cu	<0.001	0.14	13.70	0.80	0.31	0.75	0.01	0.11	15.60	15.40	14.00	0.11	
Iron T-Fe	<0.030	<0.030	6.45	1.90	0.13	1.73	<0.030	0.13	3.64	4.13	13.40	0.13	
Lead T-Pb	<0.001	<0.001	0.054	0.003	0.003	0.003	<0.001	0.003	0.230	0.230	0.250	<0.001	
Mercury T-Hg	0.00009	0.00012	0.00012	0.0001	0.00012	0.00012	0.00012	0.00014	0.00020	0.00012	0.00012	0.00020	
Selenium T-Se	<0.0005	0.0006	<0.0005	<0.0005	0.0008	<0.0005	<0.0005	<0.0005	0.0006	<0.0005	<0.0005	<0.0005	
Zinc T-Zn	<0.005	0.559	11.7	0.714	0.858	0.728	0.048	0.242	30.2	32.4	30.3	0.232	
<b>Dissolved Metals</b>													
Calcium D-Ca	3.32	18.4	125	7.99	17.3	8.88	20.4	7.18	421	419	415	7.55	
Copper D-Cu	<0.001	0.100	13.5	0.620	0.100	0.570	0.008	0.092	15.4	15.3	13.8	0.073	
Iron D-Fe	<0.030	<0.030	3.82	1.42	<0.030	1.28	<0.030	0.036	1.38	1.36	12.0	<0.030	
Lead D-Pb	<0.001	<0.001	0.048	<0.001	<0.001	<0.001	<0.001	<0.001	0.230	0.310	0.280	<0.001	
Magnesium D-Mg	0.28	1.56	23.6	1.51	1.52	1.51	1.16	0.99	83	85	81	1.04	
Potassium D-K	0.14	0.24	0.66	0.39	0.18	0.37	0.52	0.31	1.58	1.67	1.50	0.24	
Sodium D-Na	1.23	1.48	11.10	1.29	1.48	1.31	2.29	1.41	11.70	11.80	11.60	1.50	
Zinc D-Zn	<0.005	0.54	11.70	0.69	0.79	0.70	0.05	0.24	30.20	30.80	30.30	0.23	

Results are expressed as milligrams per litre except where indicated otherwise.  
 < - Less than detection limit

**CALCULATED LOADINGS (kg/d)**

	Britannia Creek B1	Jane Creek J1	2200 Portal	Britannia Creek B2 CALC 21-Sep	Jane Creek J2	Britannia Creek B3	Mineral Creek	Britannia Creek B5	4150 Portal	Treatment Plant In	Treatment Plant Out	Britannia Creek B6
	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep	21-Sep
Estimated Flow L/s	58	4.2	0	39.8	4.2	44	8	92	42	42	42	97
<b>Physical Tests</b>												
Total Dissolved Solids	100	40	0	226	40	266	90	397	181	10886	12338	503
Hardness CaCO3	47	19	0	90	18	108	39	176	5080	5080	5008	194
pH	7.02	7.37	4.29	NA	7.56	6.98	7.23	6.92	4.46	4.55	4.38	6.98
Total Suspended Solids	0	2	0	36	2	38	0	32	39	7	61	17
<b>Dissolved Anions</b>												
Acidity CaCO3	5.0	1.5	0.0	NA	1.1	57.0	1.4	23.8	965.3	921.7	878.2	16.8
Alkalinity CaCO3	41.1	5.9	0.0	NA	5.3	22.4	9.8	58.8	0.0	0.0	0.0	59.5
Chloride Cl	0.00	0.00	0.00	3.04	0.00	3.04	0.00	0.00	0.00	0.00	0.00	0.00
Fluoride F	0.25	0.02	0.00	0.24	0.03	0.27	0.04	0.48	4.68	4.75	4.72	0.59
Sulphate SO4	16.0	13.2	0.0	94.0	13.5	107.6	30.3	127.2	6314.1	6277.8	7366.5	150.0
<b>Nutrients</b>												
Ammonia Nitrogen N	0.00	0.00	0.00	0.16	0.00	0.16	0.00	0.00	0.17	0.16	0.16	0.00
Nitrate Nitrogen N	0.60	0.05	0.00	0.25	0.05	0.30	0.06	1.03	0.00	0.00	0.00	1.17
Nitrite Nitrogen N	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Total Phosphorus P	0.02	0.00	0.00	0.02	0.00	0.02	0.13	0.10	0.05	0.01	0.01	0.09
<b>Total Metals</b>												
Aluminum T-Al	0.20	0.19	0.00	3.2	0.21	3.4	0.01	1.1	102	103	104	1.7
Antimony T-Sb	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Arsenic T-As	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Cadmium T-Cd	0.00	0.00	0.00	0.01	0.00	0.01	0.00	0.01	0.45	0.45	0.42	0.01
Copper T-Cu	0.00	0.05	0.00	2.7	0.11	2.9	0.01	0.87	56.6	55.9	50.8	0.92
Iron T-Fe	0.00	0.00	0.00	6.5	0.05	6.6	0.00	1.0	13.2	15.0	48.6	1.1
Lead T-Pb	0.00	0.00	0.00	0.01	0.00	0.01	0.00	0.02	0.83	0.83	0.91	0.00
Mercury T-Hg	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Selenium T-Se	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Zinc T-Zn	0.00	0.20	0.00	2.5	0.31	2.8	0.03	1.9	110	118	110	1.9
<b>Dissolved Metals</b>												
Calcium D-Ca	16.6	6.7	0.00	27.5	6.3	33.8	14.1	57.1	1528	1520	1506	63.3
Copper D-Cu	0.00	0.04	0.00	2.1	0.04	2.2	0.01	0.73	55.9	55.5	50.1	0.61
Iron D-Fe	0.00	0.00	0.00	4.9	0.00	4.9	0.00	0.29	5.0	4.9	43.5	0.00
Lead D-Pb	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.83	1.12	1.02	0.00
Magnesium D-Mg	1.4	0.57	0.00	5.2	0.55	5.7	0.80	7.9	301	307	292	8.7
Potassium D-K	0.70	0.09	0.00	1.3	0.07	1.4	0.36	2.5	5.7	6.1	5.4	2.0
Sodium D-Na	6.2	0.54	0.00	4.4	0.54	5.0	1.6	11.2	42.5	42.8	42.1	12.6
Zinc D-Zn	0.00	0.20	0.00	2.4	0.29	2.7	0.03	1.9	110	112	110	1.9

Results are expressed as kilograms per day except where indicated otherwise.  
 < - Less than detection limit

**APPENDIX B**

**PRELIMINARY BULKHEAD DESIGN ESTIMATE**



## APPENDIX B

## PRELIMINARY BULKHEAD DESIGN ESTIMATE

**BULKHEAD DESIGN FOR PARTIAL FLOODING OF UNDERGROUND WORKINGS**

This approach would require construction of a concrete bulkhead in all portals, vent raises and shafts which daylight, below the maximum flood level. Based on a review of a composite section of the underground mine and the location of the pits, Figures 4.1 and 4.3, it appears that it is possible to flood up to the 1050 level, which is approximately the lower edge of the East Bluff pit. It will be necessary to review detailed mine plans to determine the number of openings to be sealed, however the available data indicates that there are at least 6 openings to be sealed, as shown on Figure 4.3. The number of vent raises is not known, however, allowance for sealing up to 5 of these openings should be made. It is assumed here that sealing of the upper adits in Mineral Creek and in the Fairwest deposit is not required.

There are no specific criteria for the design of permanent bulkheads in mines, however experience in South African mines indicates that the bulkhead should be designed so that the hydraulic gradient through the bulkhead is in the range of 790 to 1020 kPa/m, with the higher gradient being allowed when the surrounding rock is of a good quality. In addition, the minimum length of the bulkhead should not be less than the width of the drift opening. These criteria result in plug type bulkheads which do not require reinforcing steel. In order to achieve an effective bulkhead it is essential to scale all loose rock from the floor, walls and roof of the drift, and to pour the concrete so that it is tight against the roof. Once this is complete the rock all around the bulkhead should be pressure grouted. In the case of bulkheads for the Britannia Mine, sulphate resisting concrete is recommended. The required length of bulkhead for some levels in the mine are shown below, based on flooding up to the 1050 level and installation in good rock. These lengths assume that the drift size is not greater than 3 x 3 m.

Bulkhead Level	Length, m
4100	9
2700	5
2200	3.5

Based on estimates for construction of similar bulkheads at a proposed mine in B.C., the average bulkhead would cost in the order of \$50,000. A total of 11 bulkheads would cost \$550,000. This cost does not include any costs associated with gaining access to the areas where the bulkheads should be installed. As some of the accesses to the mine were caved during mine closure some additional costs would be incurred. The cost indicated above may be low for construction of seals in vertical openings, depending on the condition of the timbers.

It is estimated that it would take approximately 1.5 years to flood the mine up to the 1050 level, based on the following evaluation. A total of  $47.8 \times 10^6$  tonnes have been mined from the Britannia Mine, of which, say  $1.5 \times 10^6$  tonnes was taken from the pits and  $5 \times 10^6$  tonnes was mined from the Fairview mine above the 1050 level. Assuming that this includes development waste and the mean specific gravity of the material is 3.0, then this gives a total excavated volume below 1050 level of  $13.77 \times 10^6 \text{ m}^3$ . Assuming that 30% of the mine has been backfilled and that the lower 20% of the mine is currently flooded there is  $6.9 \times 10^6 \text{ m}^3$  to be flooded. Based on discharge records from the 4100 level, the mean annual flow through the mine is 150 lps or  $4.73 \times 10^6 \text{ m}^3/\text{year}$ . It is therefore estimated that this flow rate would require 1.5 years to flood up to the 1050 level, however this estimate is based on numerous assumptions and could easily vary by a factor of two. The time to reach the 1050 level would be approximately twice as long if diversion ditches were constructed in Jane Basin prior to the start of flooding.

**APPENDIX C**  
**RESULTS OF TESTING**

## APPENDIX C

### RESULTS OF TESTING

#### Experiment # 1 - pH Neutralization Curves

##### Objective

Evaluate the requirements for base to neutralize acidity in the two samples.

##### Method

1 N NaOH was added stepwise using increments of 0.1 and 0.2 mL to 1 L stirred samples. pH was measured continuously and recorded after equilibration following each reagent addition step.

##### Results

The NaOH based neutralization curves are provided in Figures 6.3 and 6.2 for the two samples. Equivalent curves based on quicklime are provided in Figures 6.5 and 6.6. These latter plots are calculated based on the equivalent basicity of 1.43 for caustic soda compared to high calcium quicklime (CaO) at a basicity of 1.0. Reagent consumption is expressed as kg/1000 m<sup>3</sup> which is numerically identical to concentration expressed as mg/L. Lime consumption (CaO) to pH 7.0 was approximately 410 and 150 kg/1000 m<sup>3</sup> for the 2200 and 4100 portal drainages respectively. CaO consumption to pH 9.5 increased to 490 and 220 kg/1000 m<sup>3</sup> for the 2200 and 4100 portal drainages respectively.

## Experiment # 2 - Lime Precipitation Test

### Objective

Evaluate batch addition of lime to neutralize acid and precipitate metals. This process relies on the precipitation of metals as insoluble hydroxides.

### Methods

A 5 % slurry of hydrated lime,  $\text{Ca(OH)}_2$ , was added batchwise to a stirred 1 L sample. Each sample was tested at three dosages. The samples were allowed approximately 15 minutes of reaction time after reagent addition. The samples were then filtered through 0.45 micron filter paper and retained for metal analyses. A spot test using 10%  $\text{Na}_2\text{S}$  solution was used on product filtrate to indicate whether metals remained in solution. A black precipitate formed around the droplet if zinc or copper were present. This procedure was used throughout the bench test program to assist in selecting dosages. It was noted however that iron in solution did not readily form a black precipitate until the sample was allowed to stand for 1 hr.

### Results

The results for the lime precipitation experiments are provided in Table 6.4 for both the samples. The 2200 portal sample was tested at 1000, 850 and 700 mg/L  $\text{Ca(OH)}_2$ , while the 4150 sample was tested at 500, 300 and 200 mg/L  $\text{Ca(OH)}_2$  concentrations.

For the 2200 portal sample a dosage of 700 mg/L  $\text{Ca(OH)}_2$ , equivalent to 530 kg/1000  $\text{m}^3$  CaO, removed soluble copper. However, a measurable quantity of zinc remained in solution. A dosage of 850 mg/L  $\text{Ca(OH)}_2$  [643 kg/1000  $\text{m}^3$  as CaO] was required to reduce zinc to low levels.

A similar relationship occurred with the 4100 portal sample in that a dosage of 300 mg/L  $\text{Ca(OH)}_2$  removed all soluble copper but left a small residue of zinc. A dosage of 500 mg/L  $\text{Ca(OH)}_2$  [378 kg/1000  $\text{m}^3$  as CaO] was required for complete zinc removal.

In both cases, the testwork demonstrated that it was necessary to increase pH above 8.5 in order to adequately remove zinc, potentially due to the solubility of zinc hydroxide under neutral pH conditions.

All other metals, including aluminum and iron, were removed by neutralization to pH 7.

Additional bench tests using continuous or staged lime addition would be required to more accurately determine the required dosage using the lime treatment process.

### Experiment # 3 - Soda Ash Neutralization

#### Objective

To evaluate the use of soda ash for neutralization and precipitation as a alternative to lime. This process relies on the precipitation of metals as both hydroxides and carbonates. Precipitation of zinc carbonate under neutral pH conditions is feasible as an alternative to the pH 9.0 to 9.5 conditions required to remove zinc as a hydroxide. An added purpose of the test is to evaluate the role of carbonate during hydroxide precipitation. Carbonate precipitation is generally applicable to the removal of cadmium, lead and nickel if these metals are present. However these metals were at relatively low concentrations in both the raw water samples.

#### Method

The procedure used in Experiment # 2 was followed. A soda ash solution containing 4%  $\text{Na}_2\text{CO}_3$  was used for the testwork. Soda ash dosages tested were 720 and 1056 mg/L for the 2200 sample and 320 and 546 mg/L for the 4100 portal drainage sample.

#### Results

The results are provided in Table 6.5. In both cases, the higher dosage was capable of removing both copper and zinc to acceptable levels, while the lower dosages left residual dissolved zinc in solution. The results indicate that the removal was due to the precipitation of zinc under high pH conditions as opposed to precipitation of zinc carbonate. The calculated CaO equivalent for both samples indicates that partial soda ash substitution has a higher equivalent dosage than solely quicklime, perhaps due to the loss of  $\text{CO}_2$  during neutralization.

## Experiment # 4 - Sulphide Precipitation

### Objective

Evaluate the use of direct sodium sulphide addition to precipitate metals in the acid pH range without neutralization.

### Method

The method used in Experiment # 2 was followed, except sodium sulphide replaced the use of lime. A 10% solution of  $\text{Na}_2\text{S}$  was made up by addition of 100 g of  $\text{Na}_2\text{S} \cdot 9\text{H}_2\text{O}$  to 1 L of Milli-Q water. This solution had a sulphide concentration of 20.9 mg/mL based on analyses of a diluted stock solution for sodium. The two portal drainage samples were tested at sulphide concentrations of 313, 627, 1254 mg/L  $\text{S}^{2-}$  for the 4100 sample and 313, 627, 1254 and 2090 mg/L  $\text{S}^{2-}$  concentration for the 2200 sample.

### Results

The test results are provide in Table 6.8. In both cases the highest sulphide dosage was required to precipitate zinc while iron remained in solution. Copper was however precipitated out at the lowest test concentrations i.e. 313 mg/L in the case of 4100 level and 1254 mg/L with the 2200 sample. The results indicate that a high sulphide to metal ratio is required in all cases to precipitate metals, making this process unattractive unless a source of sulphide such as a sulphide flotation concentrate were readily available.

## Experiment # 5 - Lime and Soda Ash Precipitation

### Objective

To evaluate the use of a combination of lime and soda ash to neutralize the sample and precipitate metals. The basis of this alternate is that the required lime dosage would be lower than that required to precipitate zinc as a hydroxide if carbonate addition results in the formation of insoluble zinc carbonates or basic zinc carbonates under neutral pH conditions. Potentially this process would allow complete zinc removal without the need to increase pH to precipitate zinc hydroxide, thereby minimizing lime consumption.

### Method

The procedure used in Experiment #2 was followed. Lime addition preceded soda ash addition in all cases.

### Results

The results of these experiments are provided in Table 6.6. Comparative results from Experiment # 2 with lime only are also included in the Table.

For both samples the addition of soda ash after lime addition was capable of reducing soluble zinc to low levels under neutral pH conditions, whereas the comparative lime-only result reported a measurable quantity of zinc in solution. The total equivalent lime dosage in the case of the 4100 portal sample was less the required lime dosage reported in Experiment #2. Additional testwork will be required to accurately define the reagent requirements for both lime and soda ash if this alternative is developed further.



## Experiment # 6 - Ion Exchange Tests

### Objective

Evaluate the use of ion exchange to selectively remove metal in the acid range using a cation exchange resin.

### Method

The experiment was conducted using a 100 mL burette with a bed depth of 330 mm and bed volume of 25 mL. The resin was suspended in Milli-Q water prior to placing in the column. Sample was fed through the column by gravity at a rate of 10 mL/min. Samples were collected at 10 min intervals. Periodically, pH and response to the sulphide spot test (described in Experiment #2) were determined for each sample. Solution feed continued until the spot test yielded positive readings indicating that soluble zinc or copper was present.

### Results

The results of the ion exchange experiment are provided in Tables 6.10 and 6.11. The dissolved values for zinc and copper are plotted in Figures 6.9 and 6.10 on a bed volume basis. Breakthrough occurred at 40 bed volumes in the case of the 2200 sample while breakthrough occurred at 44 bed volumes in the case of the 4100 portal sample.

## Experiment # 7 - Seawater Dilution

### Objective

Determine the relationship between seawater dilution ratio, pH, and dissolved metals. Evaluate the precipitation of metals in the acidic drainage via reaction with alkalinity in seawater. This alternative could be utilized in conjunction with other processes that remove some dissolved metal prior to disposal via an outfall.

### Method

The experiment was conducted by adding seawater in increments to a fixed volume of sample, followed by measurement of pH once the sample had equilibrated. With the relationship between pH and dilution ratio established, a series of dilutions were made, filtered, and the filtrates retained for metal analyses.

### Results

The results are provided in Table 6.9, while the relationships between pH and dilution ratio for the two samples are provided in Figures 6.7 and 6.8. The results indicate that some metal remains in solution after neutralization by seawater dilution. In the case of zinc, only 50 % of the reduction in soluble concentration is due to precipitation during neutralization. The kinetics of zinc precipitation via seawater dilution are not favourable. In addition, the precipitates formed are colloidal and may not necessarily have been retained on 0.45 micron filter paper.

## Experiment # 8 - Limestone Neutralization

### Objective

Evaluate the potential use of limestone to accomplish partial neutralization and to precipitate metals. This alternative could be used in conjunction with a lime treatment system.

### Method

The experiment was conducted by adding pulverized dolomitic limestone to 1 L samples of each portal drainage. The initial dosage was 20 g/L. In the case of the 2200 portal drainage sample two additional 10 g batches of pulverized were added after 20 minutes and 2 hours of mixing. The same sequence was followed with the 4100 portal sample except the two additional batches of pulverized limestone were added after 10 minutes and 1 hour of mixing. The total mixing time for the sample was 3 hours, after which each slurry was filtered and the filtrate retained for metal analyses.

### Results

The results, provided in Table C1, indicated that the use of limestone did raise pH to a limited extent but did not precipitate zinc and copper. Limestone addition did however assist in the removal of soluble iron probably due the oxidation of ferrous iron. The use of excess limestone did not appear to have any added benefit in terms of neutralization. It is assumed that gypsum formation on the limestone particles blocked neutralization via limestone. The pH increase generated by limestone addition might justify the use of a separate system for handling and feeding pulverized limestone in conjunction with a conventional lime treatment system.

## Experiment # 9 - Process Simulation - Lime Treatment System

### Objective

The purpose of this experiment was to simulate the use of a lime treatment system for treating the 4100 Portal Drainage. Following the results obtained with the previous series of tests using lime, this experiment was conducted to refine dosage estimates, predict supernatant quality and demonstrate the use of sludge recycle.

### Method

A sample of 4100 Portal Drainage was obtained on March 13, 1991 in order to carry out the tests. The neutralization curve (Figure C1) for the sample (using NaOH as described in Experiment #1) was determined in order to compare the results with data generated using the November 28 sample. A series of two - 2 L samples of portal effluent was treated using hydrated lime. 0.663g of dry hydrated lime was added to the first 2 L sample, to increase pH to 9.5. This was equivalent to a dosage of 251 mg/L as quicklime (CaO). After 15 minutes of mixing, 2 mg/L of Percol 156 was added using 4 mL of 0.05% polyelectrolyte solution. 4.4 mL of 15 % lime slurry was added to the second 2 L aliquot. At this point the treated samples were combined. 6 mL of 0.05% polyelectrolyte was added to flocculate the solids. The slurry was allowed 20 minutes of settling time, after which the supernatant was discarded and the sludge retained. The 600mL of sludge was then combined with a 1 L sample of 4100 Portal Drainage. 2.2 mL of 15% lime slurry was added to the combined sample followed by flocculation using 2 mg/L of Percol 156 as before. The flocculated slurry was then used in a settling test to predict clarifier overflow rates. A supernatant sample was collected and retained for analyses. The sludge was also retained for total metals analyses.

### Results

The results of settling test are plotted in Figure 6.15. The results predict an overflow rate of 1.4 m/h (0.57 USgpm/ft<sup>2</sup>). The analytical results are provided in Table 6.7. The results indicate that the lime treatment system should be capable of meeting low copper and zinc criteria at a dosage of 250 mg/L CaO.

## Experiment # 10 - Process Simulation - Lime and Soda Ash Treatment System

### Objective

The purpose of this experiment was to simulate the use of a lime and soda ash treatment system for treating the 4100 Portal Drainage. This experiment was conducted to refine dosage estimates, predict supernatant quality and demonstrate the use of sludge recycle.

### Method

A series of two - 2 L samples of portal drainage was treated using both hydrated lime and soda ash. Sufficient lime was added to the first sample to increase pH to 6.94. Secondly, 1 mL of 5%  $\text{Na}_2\text{CO}_3$  solution was added to increase pH to 7.9. 8 mL of 0.05% Percol solution were then added and the resultant slurry was allowed to settle of 25 minutes. The resultant sludge was retained for subsequent testing. The second 2 L sample aliquot was treated in the same manner as the first using lime followed by soda ash. The sludge from the first aliquot was then combined with the second aliquot prior to the addition of 8 mL of 0.05% Percol 156. The sample was allowed 25 minutes of settling time, after which the supernatant and sludge were retained for analyses.

### Results

The analytical results for the supernatant sample are provided in Table C1 along with the results using lime. The results indicate that a lime-soda ash system should be capable of treating portal drainage to meet a 1.0 mg/L zinc criteria at the dosages reported. However a higher soda ash dosage would likely be required to produce results equivalent to the lime-only system. A detailed evaluation of the process would require some pilot testing in order to fully assess the benefits of recycle and determine the optimum lime and soda ash combinations.

**Table C.1**

**Limestone Addition Testwork – Britannia**

**Pulverized Dolomitic Limestone**

**Reacted for 3 Hours**

**Continuous Agitation Provided by Mixer**

Location	Limestone Dosage g/L	pH		Alkalinity mg CaCO <sub>3</sub> /L	Sulphate	Dissolved Metals		
		Initial	Final			Zinc	Copper	Iron
2200 Portal	40	2.85	4.02	L1	973	41.2	79	0.96
4100 Portal	40	3.31	4.83	10.4	1230	37.6	17	0.582

**Note: Lack of reactivity assumed due to gypsum coating of limestone particles**

Neutralization of 4100 Portal Drainage  
High Calcium Quicklime Equivalent  
March 13/91 Sample

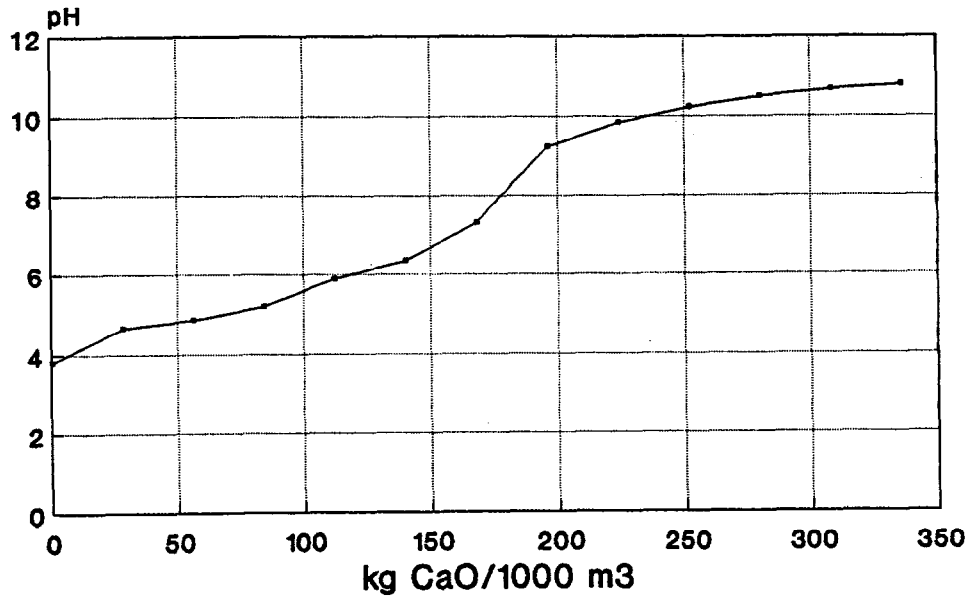


Figure C.1