FARO DECOMMISSIONING

OVERVIEW OF THE ENVIRONMENTAL PLANS

REPORT #WH9108

CURRAGH RESOURCES INC.
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1 EXECUTIVE SUMMARY

Since 1986, Curragh Resources Inc. has operated one of the world's largest open-pit lead-zinc mines at Faro, in Yukon. Now (December 1991) mining in the original Faro Pit is close to ending. An interrelated plan involving closure of the Faro Pit, conversion of the pit to a tailings pond, utilization of the Faro mill and tailing facilities for new mines at Grum and Vangorda, recycling of tailings, and the final decommissioning of the tailings facilities, has been devised. Regulatory approval has already been received for the operational aspects of this plan: the new mines, continued use of the Faro Mill and use of the Faro Pit for tailings storage. Now under consideration are the closure aspects of the Faro Site and, in particular, the Rose Creek Valley tailings facility and the Faro Pit Tailings facility.

In April 1991 Curragh Resources Inc. submitted to the Yukon Territory Water Board a plan for decommissioning of the Rose Creek Tailings facility (also referred to as the Down Valley Tailings Impoundment). The report was the result of several years of research and analytical effort, with assistance from experienced engineers and scientists, and in consultation with government officials, regulators, and the public.

In July 1991 an additional report was submitted to the Yukon Territory Water Board. This report (Kilborn 1991) presented Curragh's plans for the conversion of the Faro Pit to a tailings facility and the plans for water recycle related to that conversion. The July 1991 report dealt with some aspects of decommissioning. In particular, plans for closure of the Faro Pit tailings facility, demonstration that the Faro Pit has sufficient volume to complete the reprocessing/recycling operation outlined in the April 1991 report, and examination of some aspects of tailings management related to the reprocessing/recycling operation.

A decommissioning plan must serve multiple objectives. The Faro plan has been developed to:

- provide for environmental protection both in the immediate future and longer term;
- comply with all Government regulations;
- employ state-of-the-art technology;
- ensure the full utilization of a valuable resource. The plan must exploit every opportunity to recycle materials so that useful minerals are not wasted;
- be consistent with the available financial resources.

Curragh's decommissioning plan for Rose Creek Valley Tailings area is designed to ensure a final storage place for the tailings in a low-risk environmental location: submerged under a permanent water cover so that oxidation of the sulphides in the tailings is severely impeded by lack of available oxygen. The plan also creates direct employment for 90 persons for six to eight years after mining operations cease, excluding any multiplier effect.

Curragh's report to the Yukon Territory Water Board examined several alternatives and it was concluded that the most environmentally secure--and most financially viable--alternative involved recycling most of the tailings at Faro for additional extraction of metal, with the transfer of residual tailings into the mined-out pit which would be flooded. Costs would be recovered by revenues generated from the sale of the mineral concentrates derived from the recycling/reprocessing operations.

At the conclusion of reclamation, tailings still remaining in Rose Creek Valley would be isolated and protected with a water cover, and to the extent practicable, the entire site would be restored to nature.

Plans and supporting technical data were filed with the Yukon Territory Water Board in April and July, 1991. Several areas requiring additional work were identified and commitments were made to gather further information with definitive studies proposed. Some of the additional test work has already been completed while other studies are on-going. This report is a supplement to the April 1991 and July 1991 Yukon Territory Water Board filings. It summarizes technical aspects of the overall closure plan, describes the results of additional test work, and provides supporting detail.

Included in this report is consideration of:

- Plans for Reclamation of Pits, Ponds, Dams and Waste Dumps.

- Tailings Transfer, which involves the recovery and transfer of tailings back to the mill for reprocessing, using techniques of "hydromonitoring" and slurry pumping. Significant quantities of tailings will be pumped from the current tailings ponds back to the mill for reprocessing in the mill, following which the tailings will be placed in the Faro Pit. In support of this plan we provide working examples, a schedule for startup and shutdown, and identification of the required capital equipment and operating costs.

- Metallurgical Reprocessing, which involves the re-treatment of tailings in the Faro concentrator to produce marketable bulk concentrate. This process will require modifications to the existing Faro mill (or "concentrator"). Sampling and test work in support of the plan are described. The required process technology is defined in general terms.

- Marketing Bulk Concentrate The decommissioning plan is funded from metals markets on a pay-as-you-go basis. Accordingly, the state of metal markets will govern the magnitude of cash flows available to pay for the plan. Included in the report are assumptions derived from reviews of the

lead and zinc metal markets, historic prices and expected future prices. Projected revenues from bulk concentrate sales are compared to projected costs derived from detailed calculation of project operating, transportation, and smelter treatment charges. This comparison indicates that at reasonable future prices the project will break even.

- Environmental Management. It is planned to maintain the environmental integrity of the Faro site by controlling effluent from the site during tailings reprocessing, in periods between seasonal reprocessing, and throughout the inevitable metal market cycles which may force suspension of operations from time to time. As well it is planned to collect and treat water, provide surveillance and maintenance of the site and its facilities, and monitor water quality on an on-going basis.

- Biological Treatment of Waste Dump Drainage This section deals with Curragh's planned biological treatment of waste dump drainage, employing a new and exciting technology with great potential applicability. Biological technology is altering industry thinking with respect to the troublesome problem of mine drainage.

- Water Quality expected of the water management incorporated within Curragh's plan has been studied with encouraging results. Those findings are summarized in this report.

- Timing and Sequencing Curragh's decommissioning plan incorporates a carefully sequenced series of projects, until final decommissioning is achieved. A complicating factor is the need to inter-weave the Faro decommissioning plan with the ongoing mining at the adjacent Grum and Vangorda Pits, since common facilities are involved. The resulting inter-connected schedule is presented in this report. It is important to note that "final" decommissioning will not occur until well past the end of this decade.

The essence of the proposed plan will be to:

- ☐ remove all the tailings above the 1044.3m (3426 ft.)* elevation in the Rose Creek Valley;
- ☐ recover the base and precious metals in a marketable bulk concentrate;
- ☐ condition secondary tailings arising from reprocessing in the mill with lime (to precipitate any water soluble metals);
- ☐ store the secondary tailings in the mined-out Faro Pit where they will be submerged beneath a water cover;
- ☐ maintain a water cover over tailings below elevation 1044.3m (3426 ft.) in the Rose Creek Valley so they will remain non-reactive;
- ☐ treat dump seepage biologically.

* see addendum on elevations - Appendix F

So that the reader may obtain an overview of the plan for decommissioning the Faro site two artist's sketches are shown herein, one showing the site facilities before decommissioning (Plate 1) and the other after decommissioning (Plate 2).

In addition, artist's sketches of the Rose Creek Valley tailings area have been prepared. They depict the current impoundment facilities (Plate 3), the same area during tailings reclamation (Plate 4) and how it will look after decommissioning (Plate 5).

2 THE REGULATORY SETTING

The Yukon Territory Water Board Public Hearing

The Yukon Territory Water Board has called for a hearing on January 15, 1992, for public discussion of the implications of various technical studies and engineering reports submitted by Curragh Resources Inc. in connection with it's plan for Rose Creek Tailings Area decommissioning and the closure aspects of its plan for water recycle and tailings deposition. More particularly, the hearing will consider the implications of two reports submitted by Curragh in April 1991 and June 1991, respectively, as a condition of Curragh's water license. These studies focus on environmental management of tailings ponds and waste dumps, and other water quality issues.

The Purpose of This Report

The decommissioning of a large facility, such as exists at Faro, in today's regulatory and technical milieu, is no small undertaking. In preparing for the January 1992 hearing, Curragh officials began to appreciate that the accumulating mass of technical and scientific studies could be overwhelming and possibly hinder the proper assessment of the decommissioning plans.

Curragh officials concluded that it might be helpful to produce an all-encompassing "umbrella" summary report in non-technical language, so that various stakeholders could more readily grasp the broad outlines of the plan. The goal of this report is to provide this overview of the Faro environmental plan in all of its various interrelated aspects.

This report also presents new information and technical data which could improve the environmental plan. As well, this report further attempts to clarify any inconsistencies between the various other reports that have been produced on decommissioning as those plans have evolved during Curragh's operation of the Faro Mine.

Status of Faro Water License

The current Rose Creek water license (No. IN89-001) applicable to the Faro Mine and Mill was granted to Curragh Resources by the Yukon Territory Water Board on January 23, 1990 and expires on January 30, 1997. This was a renewal of the water license originally granted to Cyprus Anvil Mining Corporation seventeen years ago, and assigned by the Yukon Territory Water Board to Curragh Resources Inc. in 1985.

Scope of the Water License

The existing water license, and this report, pertain to all of Curragh's operations affecting water resources in the Rose Creek Valley, including the Faro Pit, Mill and the tailings facilities which will serve the Grum and Vangorda Mines and possibly later the Dy Mine. The Faro water license is not otherwise concerned with the Grum and Vangorda pits and dumps, nor with the expected future Dy Mine, all of which are located in other nearby watersheds.

3 PITS AND PONDS, DAMS AND DUMPS

The Rose Creek Valley watershed around Faro encompasses several medium-to-small sized streams, and an intricate complex of "pits and ponds, dams and dumps". Over the years, a carefully balanced system of dams and diversion ditches, canals, settling basins, tailings impoundments, special drainage systems, and pumping arrangements, have evolved at Faro. These have performed several functions: providing water for the mill, keeping the mining area dry, and protecting the environment.

The dominant water course of the area is Rose Creek, which is more than a "creek" as it is the source of drinking and process water for the Faro operations. The broad (1.5 km wide) Rose Creek Valley is also the home of Faro's tailings facilities. Upstream from the Faro mine site, Rose Creek splits into a "North Fork" and a "South Fork". Faro Creek is a small creek which, before mining commenced, flowed directly over the Faro ore body, running down the hill into Rose Creek. In the early stages of mine development Faro Creek was diverted around the Faro Pit and into North Fork to create dry ground for mining.

The various structures which have been developed at the Faro site over the years can be broadly categorized into the following functional areas:

a) Pits and Underground Workings: from which the ore is removed;
b) Rock Dumps: where rock removed from the mines to get at the ore has been placed;
c) Roads: on which ore is carried to the mill and waste rock is carried to the dumps;
d) Mill: in which the ore is ground to a powder and most (about 80%) of the valuable minerals are separated to produce a mineral concentrate containing either lead, zinc or both together (bulk concentrate);
e) Water Supply: which provides a source of water, year round, to the mill for use in the mineral separation process. The water is used to make a slurry of the pulverized ore and is the medium in which the mineral separation occurs. The water is also used for drinking, washing and fire protection;
f) Rose Creek Tailings Area: where the leftover minerals from the milling process (along with 20% of the lead and zinc in the ore) are deposited and the leftover water separated;
g) Faro Pit Tailings Area: when the Rose Creek Tailings Area is full and the Faro Pit is mined out the pit will be converted to a tailings impoundment for the newer mines.
h) Other Facilities: which include the shops, office complex, guard house, warehouse, explosives magazines, lube shack, petroleum storage and all the other structures needed by a mine and the people who work there.

The sections which follow elaborate on the various components of the Faro site in the context of the

above functional categories. We describe first what the structure is, or what it does, then how it will be dealt with at final closure and decommissioning of the complex. The location of all of these structures can be found on Figure 1. Many of the structures are also identified on Plates 1 through 5.

A. PITS AND UNDERGROUND WORKINGS

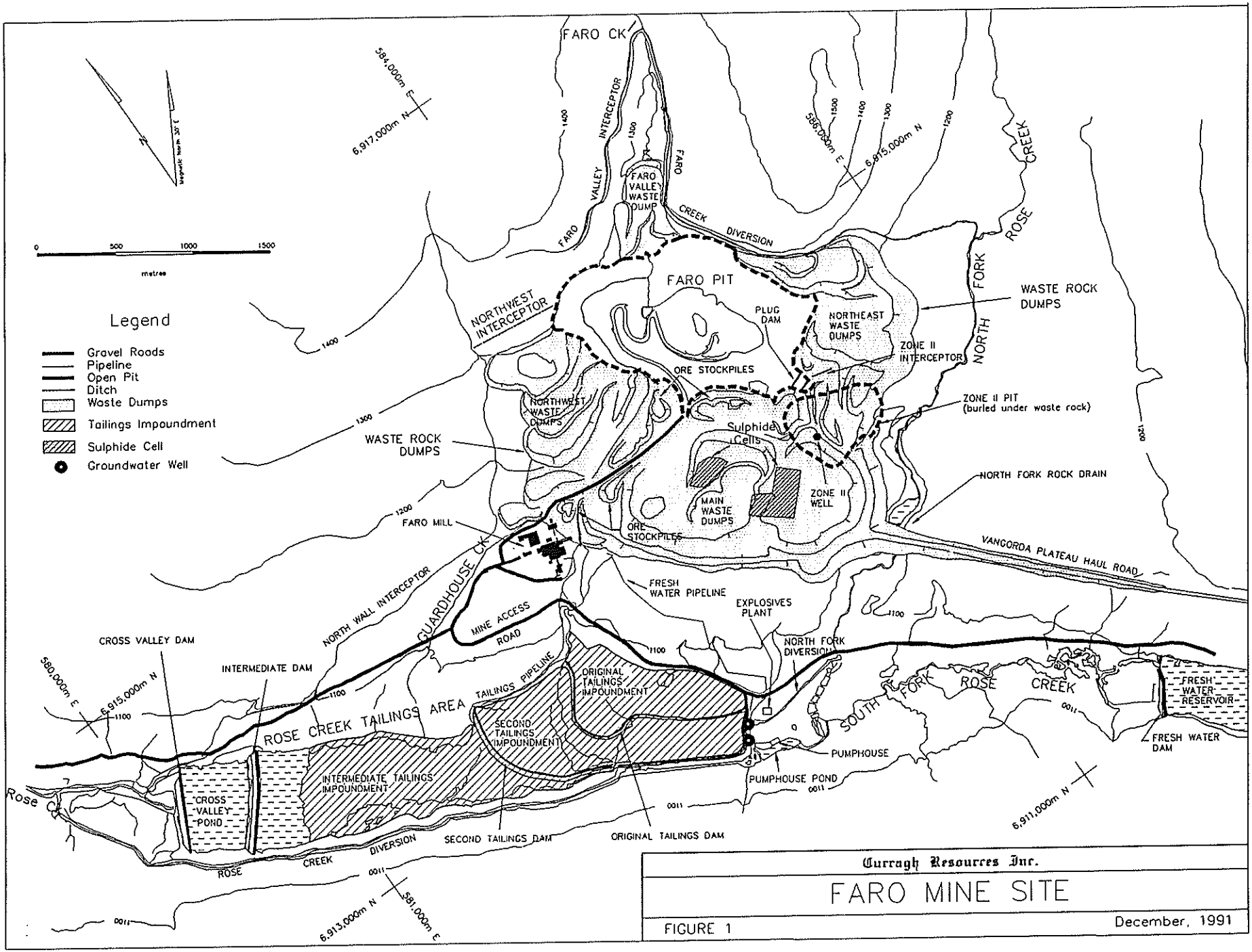
The Faro Pit. The Faro Pit is approximately three-quarters of a kilometre wide, one and one-quarter kilometres long, and about one-third of a kilometre deep. For approximately 23 years, Cyprus Anvil Mining Corporation and later Curragh Resources Inc. excavated material from the pit. Currently there is approximately six (6) months of ore supply remaining and the pit will soon be freed for other uses. The obvious option was to utilize the pit as a place for permanent storage of tailings. This change was approved by the Yukon Territory Water Board and is now integral to the Faro environmental plan. The pit as a tailings facility is discussed in more detail later. After final decommissioning, the pit will remain flooded and a lake 4.25m deep will cover the tailings. We will refer to this lake as "**Faro Lake**".

The Zone II Pit. Zone II was a separate, smaller pit mined by Cyprus Anvil between 1978 and 1981. It lies to the southeast of the Faro Pit. Once depleted, it was filled with waste rock from the Faro Pit, first by Cyprus Anvil and then by Curragh. In Curragh's environmental plan the Zone II Pit plays a key role in the collection of seepage waters from various sources, particularly the Waste Dumps, so that such water can be treated before release to the environment.

Faro Underground Mine. Since early 1990 an underground mine has been operated through a opening in the southwest wall near the bottom of the Faro Pit. The mine openings follow the ore deposit to the southwest and exploit areas where the deposit was too thin to be mined by open pit methods. The fate of the underground workings is discussed later where they are shown to be an environmental solution rather than a problem.

Metal-Rich Water from the Pits. Naturally occurring sulphide rock remains in both the Faro Pit walls and the Zone II Pit walls. With the passage of time, and when exposed to the combination of air, water and bacteria, rocks containing sulphides will oxidize, forming soluble sulphates and turning adjacent water acidic. The acidic water dissolves various metals from the host rocks. If not controlled, these metals will eventually enter the environment. This undesirable chain of events must be prevented--or at least inhibited--from occurring.

In the Faro decommissioning plan, most exposed sulphides in the pit walls are inundated by tailings, or, by a cover of water so they cannot readily oxidize. Nevertheless, small quantities of sulphide material at the northwest end and along the northeast wall of the Faro pit will be exposed. Due to the insignificant area exposed, the amount of metal picked up by surface run-off water will be minimal.



Curragh Resources Inc.

FARO MINE SITE

FIGURE 1

December, 1991

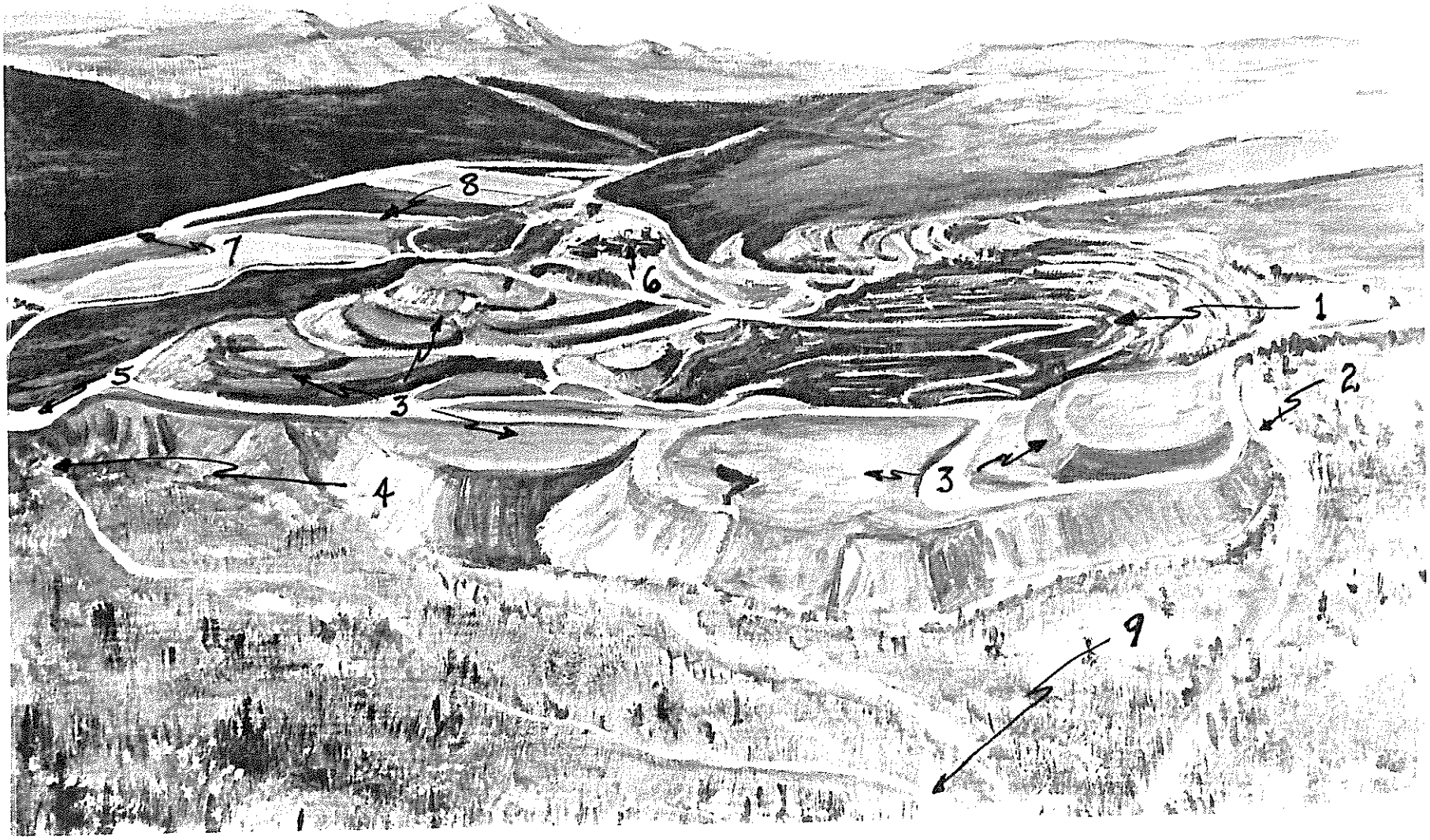


PLATE 1 - FARO MINE SITE - EXISTING - The Faro Pit and Waste Rock Dumps when mining is completed and before tailings deposition in the Pit begins. The Rose Creek Tailings Area is very near completion. Faro Creek is diverted around the pit into the North Fork of Rose Creek which flows through a rock drain under the Vangorda Plateau Haul Road. The Rose Creek Diversion carries the flow above and around the Rose Creek Tailings Area.

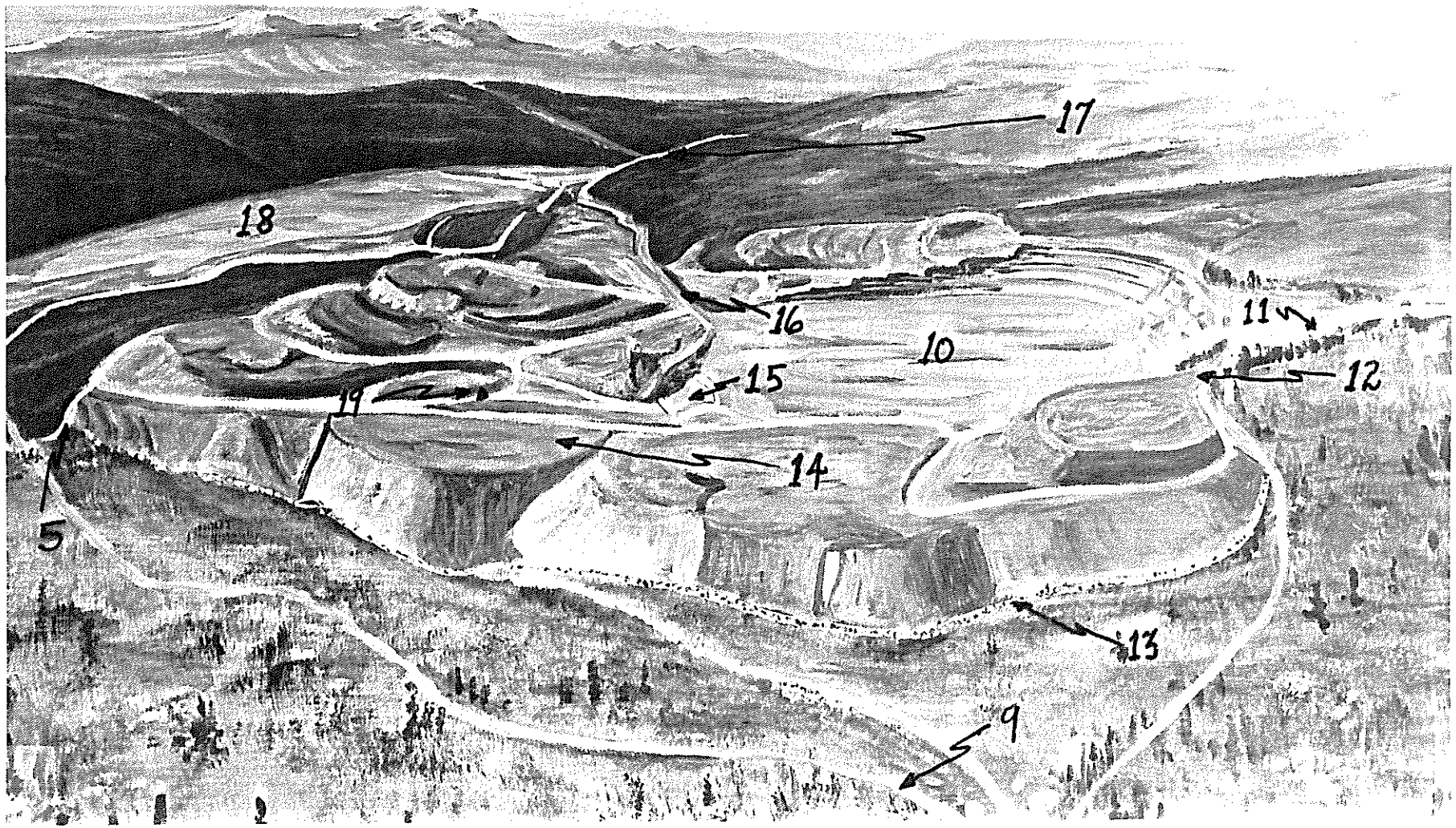


PLATE 2 - FARO MINE SITE - POST DECOMMISSIONING - Faro Lake is shown at its final water elevation with tailings deposited below water surface (year 2026). Faro Creek is permanently diverted into the lake over the pit wall and its outflow is directed to Rose Creek. The Plug Dam prohibits surface flow and restricts seepage to the Zone II Pit. The North Fork of Rose Creek flows through a partial breach in the Vangorda Plateau Haul Road, joins the South Fork and enters the Valley Lake. The Rose Creek Diversion is removed. The Mill, other buildings and facilities are dismantled and the area revegetated with grass/legume cover crop. The Waste Rock Dumps and area around the Rose Creek Tailings Area are also revegetated.

Seepage from Faro Pit into the Zone II Pit. A fault zone exists at the southeast limit of the Faro Pit which could provide a natural pathway for water to seep from Faro Lake into Zone II where it not only would become contaminated but also would add to the total volume of water needing treatment. In order to limit this seepage, a "Plug Dam" (see below) has been designed which will cover the fault zone with highly impermeable fine tailings and effectively seal the pit wall.

Use of Zone II Pit as Sump. Paradoxically, while considerable money and effort will be expended to minimize seepage out of Faro Lake into Zone II Pit, the Faro environmental plan also uses Zone II Pit as a sump to collect seepage from various waste dumps, prior to being treated.

Faro Pit Wall Stability. The steep, high-standing (over 300 metres) northeast wall of the Faro Pit has shown signs of instability when wet. This rock face was stabilized during mining operations through surface water control and internal dewatering (via drill holes). Under the decommissioning plan, the Faro Pit will be flooded with water. Initially water will cover the toe of the pit wall. Three years later the wall will be inundated to its maximum level but only 60m of tailings will be deposited in the pit. Eventually the pit will be nearly full of tailings with only 4.25m of water covering them.

The stability of the pit wall, when saturated with water initially, prior to the pit being full of tailings, could be in question. Current geotechnical advice suggests that if the pit wall collapses during the early backfilling stages, the consequences are not serious since a wave could not go over the top of the pit walls. Once the pit wall is substantially flooded with water its stability will actually increase because of the combined result of two opposite effects a) water in the wall will reduce the rocks resistance to movement, however this will be offet by b) the buoyancy of the rock in the water which will reduce the weight of the wall and reduce its tendency to move downward. After the pit is filled with tailings, stability will be further increased and the impact of any pit wall subsidence would be negligible due to the minor volumes which could collapse unhindered. Nevertheless, monitoring of the pit wall will be continued.

The complete geotechnical opinion, as provided by Mr. A. Stewart of Piteau & Associates Ltd, is included in Appendix D.

B. DUMPS

Waste Rock Dumps. The existing Faro Mine has generated approximately 100 million cubic metres of waste rock. Today, all of this material resides in "waste dumps" around the perimeter of the Faro Pit and within the former Zone II Pit.

Fortunately for the environment, the vast majority of this material is benign; that is, it presents no hazard to the environment. However, mixed within the dumps, more or less at random, are low-grade sulphide rocks which (like the sulphide rocks left behind in the pit walls) will oxidize. When in contact with

rainwater and snow melt percolating through the dumps, the oxidized rock will generate metal-rich seepage which will emerge from the toe of the dump.

The oxidation process is proceeding slowly at present and the technical estimate is that, due to the inherent nature of the rock and low temperature conditions which prevail in Yukon, oxidation will continue to be slow. Nevertheless, waste rock seepage, if unattended and unmanaged, could become deleterious to the environment. Thus, an over-riding environmental management issue is to ensure collection and appropriate treatment of this drainage.

Dump tops and slopes will be revegetated with a grass/legume cover crop to accelerate their natural reversion to wild terrain.

Sulphide Cells. During Curragh's tenure at the Faro Site, sulphide rocks have been concentrated into specific areas of the dumps. There are two such areas identified on Figure 1. Both the sulphide cells will be covered by a layer of compacted phyllite as described in the 1988 Closure Plan. Some sulphide dumps have been established within the Faro Pit. These will be inundated by tailings and water at final closure.

Faro Valley Interceptor. This is a minor tributary diversion ditch located northwest of Faro Creek. It prevents surface water from running through the Waste Dump areas during mining operations. During decommissioning, this Interceptor will be upgraded and maintained, so as to cut off water before it runs into the waste dumps where it could become contaminated.

Northwest Interceptor. This is another ditch to be constructed above the Northwest dumps to intercept surface runoff before it contacts the dumps. It will be built as part of the post closure water management plan.

Ore Stockpiles. In Curragh's plan, the various ore stockpiles, including low-grade ores, oxidized ores, and Skagway gravels returned to the mine for reprocessing, are to be fed into the mill during the later stages of regular operations. They will be fully consumed.

C. ROADS

Vangorda Plateau Haul Road. This 14-kilometre causeway was constructed from non-acid-generating materials extracted from the Faro Pit. At closure, the roadway will be breached to accommodate the North Fork of Rose Creek. Several culverts on the haul road will be breached if it becomes evident that future blockage of the culverts is a clear threat. Elsewhere the haul road will be scarified and seeded.

North Fork Rock Drain. This rock drain permits water in the North Fork to flow under the haul road and causeway running from Vangorda Plateau to the Faro Mill. During decommissioning, the haul road embankment will be breached where it crosses North Fork and a spillway will be established to accommodate freshet water flows in North Fork. This spillway will be close to the original stream elevation and will allow fish passage to resume.

D. MILL

The mill is one of the major assets of the Faro Site. It produces mineral concentrates from ores from the mines. The concentrates have more unit value than the ore and can be shipped to smelters around the world to produce refined metal. The Faro Mill is one of the largest lead-zinc concentrators in the world. It was originally built in the late sixties but has been expanded several times, most recently in 1981. Since Curragh re-opened the mine in 1986 the mill has been further modernized and improved through installation of new equipment and repair of old.

The Faro Mill will be refitted to reprocess tailings once all ores are consumed as described in this report. When all tailings have been reprocessed the Mill will be disassembled and sold for salvage. The above ground concrete structures will be demolished and/or buried and concrete debris as well as scrap metal will be buried. Below ground level excavations will be filled in. Areas with significant contamination will be covered with soil and the area revegetated.

E. WATER SUPPLY

Fresh Water Dam. During operations, fresh water for the mill, particularly needed during arid winter months, has been stored in a reservoir behind the Fresh Water Dam on the South Fork of Rose Creek. During decommissioning the overflow ditch bottom will be lowered slightly, so as to preserve valuable fish habitat in the reservoir while allowing fish passage to the lower South Fork. The dam will be breached sufficiently, however, to provide a very high factor of dam safety and stability.

Freshwater Reservoir. The reservoir will be partly retained since it has created valuable over-wintering habitat for Arctic grayling fish. The South Fork Reservoir, reduced in size, will continue as permanent fish habitat. The reservoir water level will be 4m lower than it is today, however its area will be little changed. The reservoir will have substantial area of water depth greater than 8m.

Pumphouse Pond. This pond, about three kilometres downstream from the Fresh Water Reservoir, feeds water into the process water supply system for the mill. In decommissioning, the pump house will be removed, the impoundment structure will be breached, and Rose Creek will be returned to its natural elevation.

North Fork Diversion. This is a short diversion channel which carries the water flow of the North Fork of Rose Creek around the Pumphouse Pond during periods of spring freshet. The purpose is to reduce suspended sediment load in the process water supply for the concentrator. The diversion channel is not used in winter when water flows are low and the North Fork is directed into Pumphouse Pond to augment the water supply. During decommissioning, the diversion channel will be upgraded to accommodate flood events while the original course will be maintained in the flood plain as an emergency overflow.

Groundwater Wells. Several wells were developed in the area of the junction of the north and south forks of Rose Creek. These wells augment surface water supply in winter. They are connected by pipelines to the Pumphouse Pond. All surface facilities connected with the wells will be removed at final decommissioning.

Fresh Water Pipeline. This pipeline conveys fresh water from the pumphouse pond to the mill and will be dismantled at the conclusion of operations.

F. ROSE CREEK TAILINGS AREA

The Rose Creek tailings area is shown in Plate 3 in essentially its current configuration.

Tailings Impoundments and Dams. Tailings produced by mill operations at Faro have been stored behind three retaining dams stretching across Rose Creek Valley, as follows:

- The **Original Tailings Impoundment** behind the **Original Tailings Dam** was used for tailings between 1969 and 1975. It is located just north of the junction of the North and South Forks of Rose Creek. The low retaining dam was built during Cyprus Anvil's early years of operation and raised several times.

- The **Second Tailings Impoundment** behind the **Second Tailings Dam** was used for tailings between 1975 and 1981 and occasionally since Curragh reopened the mine. It was built and used by Cyprus Anvil and is located below the original impoundment dam.

- The **Intermediate Tailings Impoundment** behind the **Intermediate Dam** was constructed by Cyprus Anvil (1981) and used for tailings storage by Curragh Resources (1986-present). It is the newest and most sophisticated of the tailings facilities and has been raised on two occasions since 1986.

Cross Valley Dam. Downstream in the Rose Creek Valley, below the three tailings dams, there is a final dam: the Cross Valley Dam. This dam was built to create a final "polishing pond" which provides the settling for suspended sediments which might escape the tailings ponds. It thereby helped to ensure suitable water quality before final release to the environment. The Cross Valley Pond will be retained

PLATE 3 - ROSE CREEK TAILINGS AREA - EXISTING - The Original and Second Tailings Impoundments were almost entirely filled by Cyprus Anvil Mining Company and are no longer in use. The Intermediate Tailings Impoundment is the site of current tailings deposition and contain exclusively Curragh tailings. The Cross Valley Pond is for water clarification only and contains no significant tailings. The Rose Creek Diversion carries the flows of Rose Creek around the tailings area. The Pumphouse Pond is the source of water for mill operations and is fed by the South Fork of Rose Creek whose flow is regulated by the upstream Freshwater Reservoir.

throughout the tailings reprocessing operation to assist in maintaining water quality. During decommissioning, the Cross Valley Dam will be breached as it is no longer required. Accumulated fines behind the dam will be removed and placed behind the Intermediate Dam under permanent water cover.

Rose Creek Diversion. In order to develop Cyprus Anvil's second tailings impoundment area and the currently used Intermediate Tailings Impoundment area in the Rose Creek Valley, it was necessary to divert Rose Creek from its natural course by constructing a canal running along the south wall of the Valley. This canal will be retained during reprocessing; however, after decommissioning Rose Creek will be re-routed into a permanent pond behind the Intermediate Dam. Rose Creek Diversion will then be abandoned.

North Wall Interceptor. The North Wall Water Interceptor is a ditch established to re-direct stream flow and surface water run-off which might have otherwise entered the Intermediate Dam tailings impoundment. With decommissioning this ditch will no longer be required. Some flows will be returned to their natural courses and portions of the ditch will be abandoned. Other parts will be upgraded and will be used for Faro Lake discharge.

Tailings Pipelines. The pipelines which conveyed tailings to the tailings ponds will be retained during the reprocessing operation as an emergency discharge line and to deliver lime slurry as needed to the Intermediate Tailings Impoundment. At final closure the pipelines will be dismantled.

The Valley Lake. During reprocessing, most of the tailings in the original and second impoundment and a small amount of the tailings behind the Intermediate Dam, will be removed. This will result in all tailings in the Rose Creek Valley above the elevation of 1044.3m (3426 ft.) being reclaimed. Flooding over the residual tailings will therefore be feasible to an elevation of 1046.3m (3433 ft.). This water cap will permanently isolate the tailings from contact with oxygen in the atmosphere. A lake, **Valley Lake,** will be created in Rose Creek Valley which will be a minimum of two metres deep.

This plan avoids the extensive capital expenditure which would be required to flood all of the Rose Creek Valley Tailings area to the current elevation. This was a financially unrealistic option which was rejected in the past.

Spillway on Intermediate Dam. A side channel spillway will be constructed at the northwest edge of the Valley Lake at the north abutment of the Intermediate Dam. The spillway is sized to safely pass the magnitude of flood that would be expected to occur once every 500 years in Rose Creek.

Rose Creek Inlet. During decommissioning Rose Creek will be returned to its original course, flowing through the (to be breached) Pumphouse Pond and entering Valley Lake. An energy dispersing rip-rap structure will be provided to prevent any disturbance of the submerged tailings as Rose Creek enters the Valley Lake.

Clean Up of Tailing Pond Shorelines. Underlying the tailings ponds of Rose Creek Valley are the natural valley bottoms. The bottoms behind the original and secondary dams were overlaid with tailings by Cyprus Anvil . Since this environmental plan specifies that these tailings are in large measure to be removed, the exposed tailing pond shoreline will be cleaned during decommissioning and returned to nature. The cleaning technique to be employed will involve hydromonitoring with mechanical excavation in selected areas. The area will then be re-vegetated.

G. FARO PIT TAILINGS AREA

Faro Creek and Faro Creek Diversion. Upon completion of mining in the Faro Pit, the pit will be flooded and backfilled with tailings. Commencing in 1992 and for approximately three years thereafter, Faro Creek will be diverted back to its natural course, flowing into the Faro Pit to assist in its flooding. Once the pit is flooded, and while the pit is still being used for active tailings deposition, the Creek will be temporarily re-routed back to its diversion channel. Later, when the pit has been filled with tailings, Faro Creek will be returned to its natural course and run into Faro Lake (see below) formed by the flooding of the pit, and exit over a constructed spillway to rejoin Rose Creek in the Valley.

Creation of Faro Lake. The minimum elevation of the bottom of the Faro Pit is approximately 3200 feet (976m) above sea level. Tailings are to be protected with a permanent water cover to prevent oxidation. Flooding will create a lake three-quarters of a kilometre across by one and one-quarter kilometre wide; which we will refer to as "Faro Lake".

At the conclusion of Faro Pit backfilling, the tailings surface will be approximately 3836 feet (1169m) above sea level. The tailings deposited under the Lake will be approximately 636 feet (196m) thick, or about the height of one of the taller office towers in Toronto. The surface of Faro Lake will be 3850 feet (1173m) above sea level. Water depth in the lake will be 4.25m (14 ft.).

The surface water in Faro Lake will be used as a mill water supply source once the Pit is flooded to capacity. Water level in Faro Lake will be controlled by a siphon structure until a permanent spillway is constructed upon decommissioning.

Water quality in Faro Lake will be excellent during the filling stage due to the high alkalinity of the mill tailings discharged into it, thereby inhibiting any uptake of metals during pond initiation. Once the lake is filled, water quality will remain high due to the limited sources of contamination which could enter the lake. Discharge from the lake will meet the relevant water quality standards.

The Plug Dam. In order to eliminate water seepage from the flooded Faro Pit into the Zone II Pit, a Plug Dam must be installed between them. This Plug Dam will consist of two dams plus a water barrier: a downstream impermeable dam, an upstream rock-fill dam, and fine materials (cycloned out of the tailings) placed in-between to ensure an adequate water barrier thereby preventing leakage. The

development of the Plug Dam is illustrated in Figure 2.

Faro Lake Spillway. The spillway from Faro Lake will be constructed to the southwest of the Faro Pit. It will lead into a channel about 300m long excavated through fill material, followed by an additional 1 km of channel at the current ground level, all lined with impermeable clay. The remaining 2.9 km of channel will be an open ditch in natural ground. The spillway structure itself will be rip-rap armoured.

The south side of the flooded Faro Pit and the above two water management structures are illustrated in Figure 3.

Faro Creek Cascade. Following decommissioning, Faro Creek will be allowed to cascade over the pit wall into Faro Lake below. An energy-dispersing structure will be placed at the bottom of the cascade to avoid agitation of the tailings at the Lake bottom.

H. OTHER FACILITIES

Buildings and Structures. Several buildings around the minesite must eventually be decommissioned and removed. All buildings not needed for an ongoing environmental maintenance and monitoring, will be dismantled and removed. Cash raised through the dismantling and sale of equipment will be available for environmental funding. The area will be made safe for human and wildlife passage by backfilling foundations and demolishing and/or burying concrete structures. Certain areas will be scarified and covered with soil, including those, such as parking lots, which might contain superficial hydrocarbon or metal contamination.

COMPARISON WITH THE 1988 CLOSURE PLAN

Certain aspects of the 1988 Closure Plan filed with the Yukon Territory Water Board differ from the plans submitted more recently. The more current plans replace those provisions of the 1988 plan. The most significant changes are:

- The Faro Pit will be flooded with tailings and water, rather than simply water as in the 1988 Plan. The Pit will be filled to a higher elevation: 1173m (3850 ft.) ASL versus 1161.5m (3811 ft.) ASL previously, thus water level will be 12m (39 ft.) higher. All but the top 4.25m (14 ft.) of the filled pit will contain tailings which will seal the pit walls, reduce the effect of the higher water level and reduce seepage, improving groundwater control.

- Discharge from the Faro Pit will be to the southwest rather than the southeast as in the 1988 plan avoiding the possibility of leakage from the discharge spillway into Zone II.

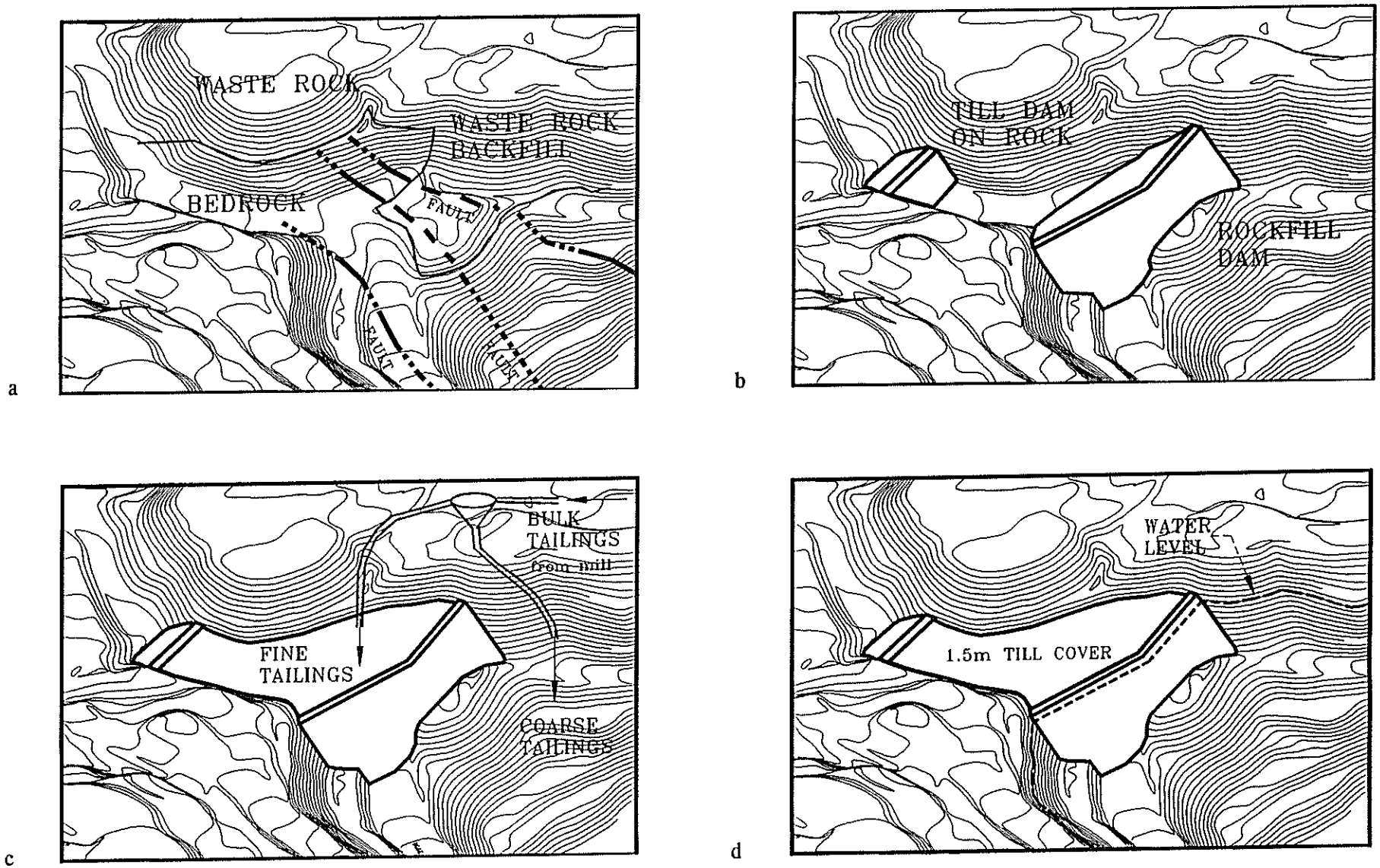


FIGURE 2 - Sequence showing the development of the Plug Dam at the southeast end of the Faro pit **a)** area as it is today showing area already backfilled and the two major faults **b)** construction of the two dams **c)** placement of cycloned fine tailings between the two dams **d)** placement of a 1.5m thick till cover over the consolidated fine tailings, water level in the flooded pit is indicated.

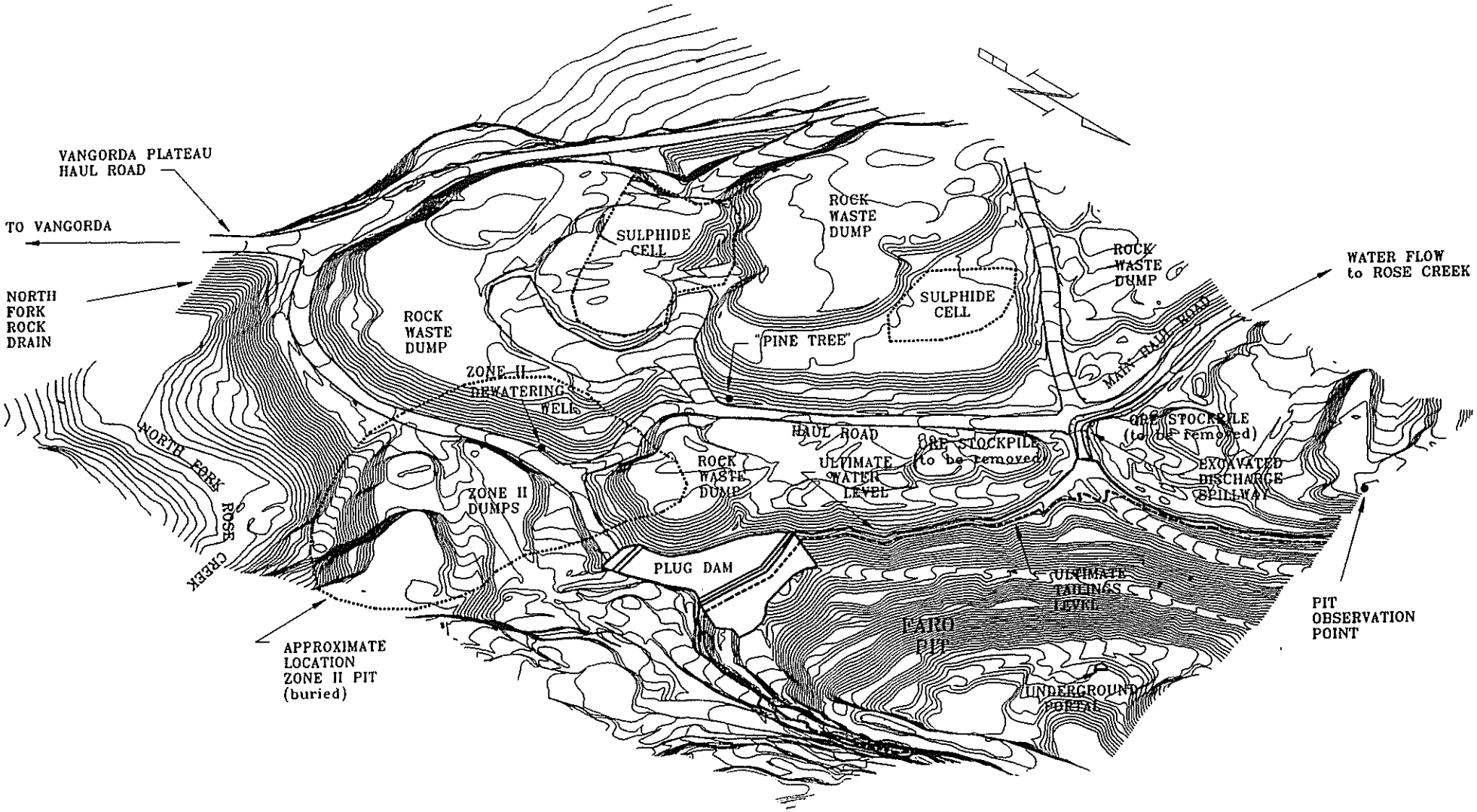


FIGURE 3 - Oblique view (looking southwest) of the south side of the Faro Pit and a portion of the Waste Rock Dumps to the southwest showing the water control system that will be in place when the pit is flooded. The level of pit flooding will be raised by the Plug Dam so that water will flow out of the pit to the west, rather than the southeast, to the North Fork of Rose Creek as proposed in the 1988 plan. Discharge from the Lake in the pit will be through an excavated spillway on the south pit wall and then via a channel following the current main haul road and portions of the North Wall Interceptor to Rose Creek (now shown).

- The current plan incorporates a Plug Dam to the southeast of the Faro Pit, in order to prevent water seepage into Zone II. This represents a significant further improvement in groundwater control and a structure much improved over the dam in the 1988 Plan.

- Water quality concerns, attendant to the filling of Faro Pit, are reduced in the current plan due to the alkalinity introduced by process water flowing into the Pit with the tailings. In the 1988 plan, the pit would have been flooded with only natural stream run-off. This change alleviates one of the major uncertainties of the 1988 plan.

- North Fork Rock drain will be breached to a substantially lower elevation than previously, so that impounded water cannot back up into the Zone II area and interfere with dump seepage collection ditches. The bulk of the rock drain will be removed and the channel of the North Fork will be suitable for fish passage.

- Faro Creek diversion will be managed as described in this report, with resultant changes from the 1988 report.

- Collection and treatment of dump seepage was proposed as a contingency measure in the 1988 Report. Seepage collection and treatment has been greatly elaborated in the current plan (through specification of ditches, sumps, pumps, and biological sulphate reduction).

- Zone II interceptor ditches are changed, as outlined in this report, in order to take advantage of a dewatering well which has been installed in Zone II since the 1988 report was written, and to enhance water control.

- Water level in Zone II is now controlled by a dewatering well rather than a drain providing improved control over the water level in the pit.

- Fresh water reservoir elevation at closure will be 2m lower in the current plan than in the 1988 plan. This will provide increased dam stability, preserve the arctic grayling overwintering habitat within the reservoir and allow fish passage between the reservoir and the lower South Fork.

4 TAILINGS RECYCLING/REPROCESSING: PHYSICAL

Between 1969 and 1991, a large volume of tailings was produced as a byproduct of the milling operations at Faro. In total, approximately 56 million tonnes of tailings were deposited behind the three tailings dams in Rose Valley. Of this total, approximately 32 million tonnes were produced by Cyprus Anvil and approximately 24 million by Curragh Resources Inc.

Tailings in the Rose Creek Valley have economic value and studies conducted by Curragh Resources, its various technical consultants and advisors have demonstrated that prospects for the economic recovery of zinc and lead sulphides from the tailings are good. The minerals to be recovered were rejected during the original milling process at Faro, for a variety of technical reasons:

- Earlier milling technology under Cyprus Anvil employed a coarser grind which failed to successfully liberate as much metal from the ore as is currently achieved;
- Cyprus Anvil operations gave highest priority to high-grade ores with correspondingly richer tailings;
- Initial operations were subject to frequent mill upsets resulting in wastage of mineral product into tailings;
- Curragh has "forced" mill throughput, maximizing production tonnages at some sacrifice of minerals to the tailings pond.

The lower the efficiency of past processing, the greater the opportunity for future reprocessing. It is estimated that approximately 800,000 tonnes of metal is contained in the Faro tailings. A portion of this metal can be reclaimed and marketed.

Curragh's environmental plan involves reclaiming approximately two-thirds of the accumulated Rose Creek Valley tailings for reprocessing in the Faro Mill and sending the residual tailings into the Pit. The primary environmental goal is to remove sufficient tailings (37.7 million tonnes) to bring the elevation of the final tailings surface (behind all three Rose Creek Valley tailings dams) down to at least 1044.3m (3426 ft.). Therefore not all of the tailings are to be reprocessed. Skimming off the highest portion of the tailings will permit the remaining Intermediate Dam Impoundment tailings to achieve the presently accepted optimum form of abandonment: permanent under-water storage. The permanent water cover will be established behind the existing Intermediate Dam. It should be noted that the 1044.3m (3425 ft.) cutoff elevation was determined to be within the storage capacity volume of Faro Pit, and consistent with the tailings facility needs of the Grum, Vangorda and Dy mines.

The motivation of the tailing reprocessing plan is environmental; however, it is also financial for it must be affordable. Previous closure plans for Faro promised flooding behind a new Cross Valley Dam of

enormous scale, to be built after the mine had closed. The cost was impossibly high. Using the existing Intermediate Dam, as opposed to some new or greatly enlarged and costly structure, is an extremely important factor in the plan economics.

At decommissioning, virtually all of Cyprus Anvil tailings and some of the Curragh tailings from Faro orebodies will have been reprocessed and permanently protected under about 4.25m (14 ft.) of water in Faro Lake; while most of Curragh's tailings from the Faro orebodies will have been permanently stored under a minimum 2m of water behind the Intermediate Dam. Bulk concentrates produced in the mill will have paid for virtually all of the operating costs involved.

In order to recycle the tailings, it is first necessary to recover them physically for transfer back to the mill. Kilborn Engineering Inc. was retained by Curragh Resources Inc. to evaluate various physical transfer options, including:

- Dredging the tailings.
- Truck and shovel excavation.
- Hydromonitoring.

The most cost effective option was hydromonitoring. This involves the use of high-pressure water guns to liquefy the tailings and transfer them as a slurry to pumping points downstream. The process is not unlike hydraulic placer mining common to Yukon and Alaska, however in this case there will be a closed system with no water escaping without having been treated, if necessary. The slurry is screened to remove oversize, non-tailings material and then pumped to the mill for re-processing.

Kilborn Engineering staff has extensive experience with hydromonitoring through their involvement in the ERG project in Timmins, Ontario and the ERGO project in South Africa. At ERGO large tailings impoundments have been cleaned up by hydromonitoring. The slurried tailings have been pumped as far as 20-30km to a mill for reprocessing. At ERG, the technique was highly successful in moving large quantities of tailings and the work there gave valuable cold weather experience.

The equipment involved is fairly simple: three hydraulic monitors (two for cutting, and one cutting part time and helping to wash tailings down the ditch to the pumping sump) which can be remotely controlled; a booster pump to feed high-pressure water to the monitors; a trench along which the remobilized tailings slurry moves away from the cutting face; a concrete basin (sump) for collecting the slurry; two low lift pumps capable of moving material up to eight inches in diameter; a oversize rejection screen; a battery of high-head booster pumps to transfer the tailings to the mill (lifting it an average of 75 metres), and the transfer pipeline to the mill.

At the conclusion of reprocessing, the final tailings (or secondary tailings) are sent to the Faro Pit for permanent disposal.

This will be an environmentally desirable closed circuit. The source of water for the high-pressure monitors will be Faro Lake, and the recovery operation returns this water, with residual tailings, to the Lake. Water quality will be controlled through the addition of lime in the mill to remove base metals from solution before the water emerges from Faro Lake.

The hydromonitoring operation will employ approximately 25 persons on a 2-shift, 24-hour per day, 7-days per week operation. Some will also be employed in ongoing rehabilitation and grooming of the Down Valley tailings shoreline.

Tailings reprocessing will be a seasonal operation as hydromonitoring is not feasible when temperatures drop below freezing. Thus, both hydromonitoring and milling will be confined to approximately six months per year. This necessitates a spring mobilization and an autumn demobilization of hydromonitoring and mill processing with the inactive months providing an opportunity for equipment maintenance.

Hydromonitoring may also be interrupted by the business cycle. Since reclaiming is being funded through the sale of bulk concentrates, world metal markets may not allow the generation of sufficient cash flows from time to time, causing temporary shutdown. In this regard, the economics of reclamation will resemble the economics of most mines in a competitive, cyclical market. However, unlike most mines, this operation will be designed to shutdown and startup on a regular basis thus will have the option of much more flexible response to business cycles.

The tailings reprocessing operation anticipates the movement of 800,000 tonnes of tailings per month, or 4.8 million tonnes per year. Kilborn Engineering estimates the capital cost of the hydromonitoring equipment at approximately $6.6 million. The cost of the tailings transfer operation will be 39 cents (Cdn) per tonne of material moved or about $1.8 million (Cdn) per annum. This figure excludes the expense of metallurgical reprocessing in the mill as well as conditioning of the water with lime.

With slightly under 5.0 million tonnes of tailings being reprocessed each year, the requisite 38 million tonnes of Down Valley tailings can be reprocessed and transferred to permanent storage under Faro Lake in about 8 operating years.

The engineering study by Kilborn is included as Appendix A of this report where more detailed information can be found on hydromonitoring as contemplated for the Faro site. Plate 4 shows an artist's conception of the Rose Creek Valley when hydromonitoring is in progress. Plate 5 shows the Rose Creek Valley when hydromonitoring is finished and the area has been completely reclaimed.

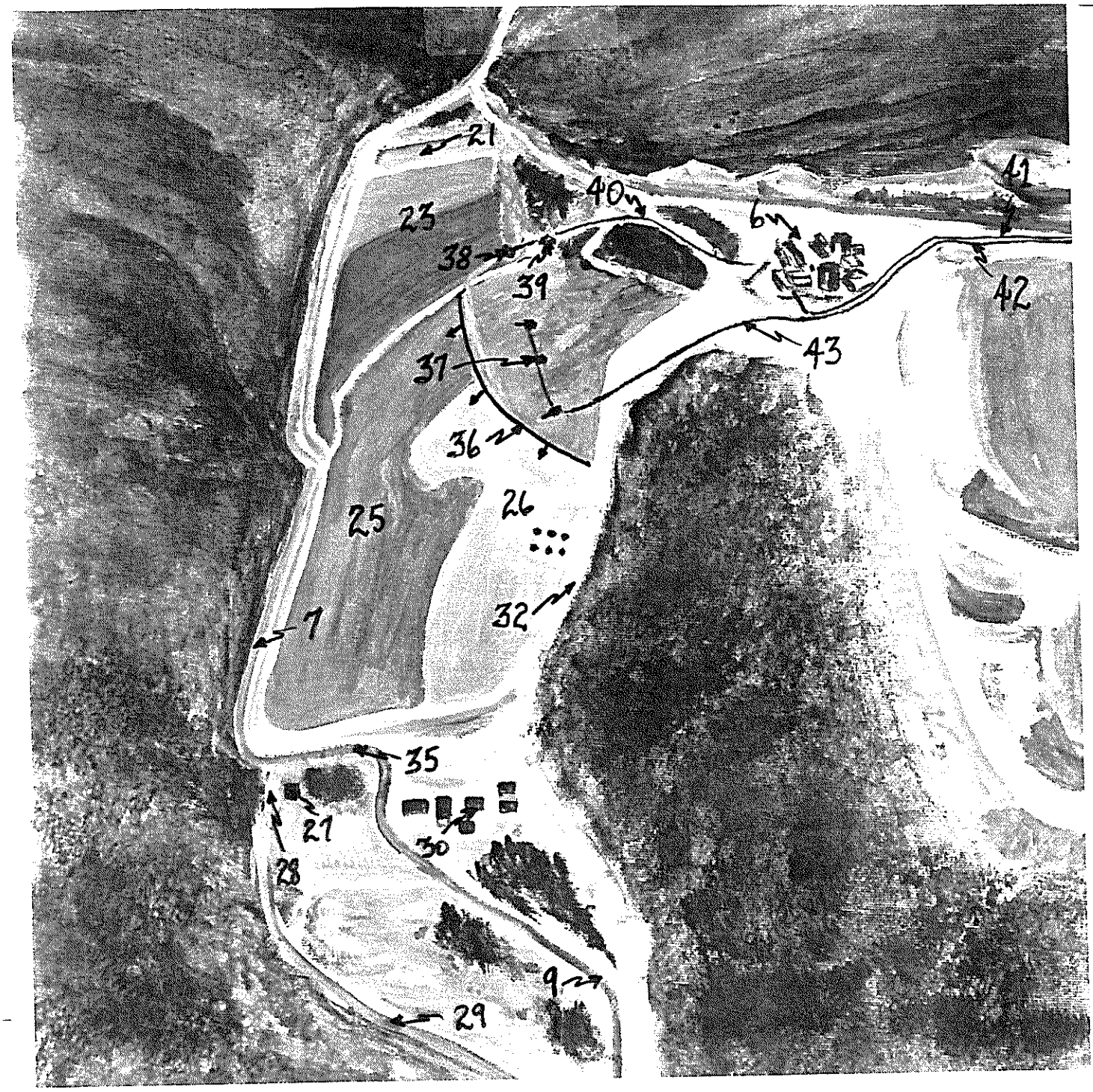


PLATE 4 - ROSE CREEK TAILINGS AREA - HYDROMONITORING IN PROGRESS - The area is shown shortly after the hydromonitoring process has begun (year 2010). The water supply pipeline from the Faro Pit is feeding the three hydromonitors which are advancing a cut face in the tailings from right to left. The slurried tailings wash to a sump, are pumped through a classifying screen and then are pumped to the mill for reprocessing. The tailings line from the Mill to Faro Pit completes the closed system.

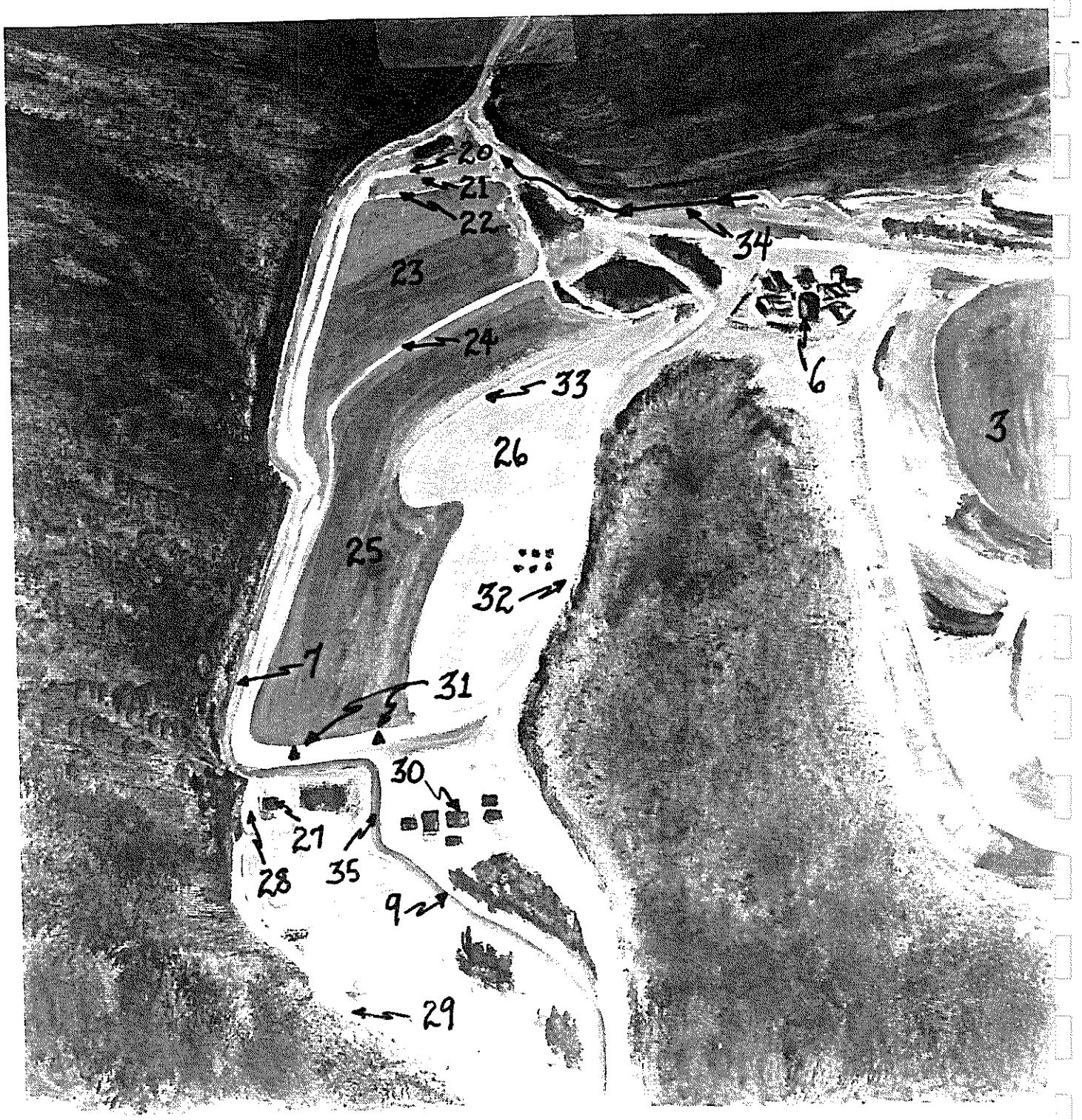


PLATE 5 - ROSE CREEK TAILINGS AREA - POST DECOMMISSIONING - An artists' conception of the Valley Lake formed behind the Intermediate Dam (year 2018). The Cross Valley Dam is breached. The Mill and all other buildings and facilities are removed. Disturbed area around Valley Lake is revegetated and wetland vegetation is established along the shoreline.

5 TAILNGS RECYCLING/REPROCESSING: METALLURGICAL

Value of the Rose Creek Valley Tailings

Rose Creek Valley tailings to be reprocessed have the following composition, in comparison with typical Faro ore:

	Tailings	Faro Ore
Lead	0.79%	3.50%
Zinc	1.23%	5.00%
Copper	0.15%	0.15%
Silver	15 g/t	30 g/t
Gold	0.12 g/t	0.12 g/t

Viewing the Rose Creek Valley tailings dump as just another orebody, we see that in comparison with original Faro ore, it has a much-reduced lead and zinc content, a somewhat reduced silver content, but virtually an unchanged gold content. The Rose Creek Valley "low-grade ore" has several advantages: it has already been crushed and ground, no other material needs to be moved to mine it, it is physically accessible, can be cheaply moved, and it can be metallurgically processed in facilities whose capital cost have already been paid back.

At the prices assumed for this study one tonne of tailings contains metal with a gross value of about $28 (US) as shown below:

Lead @ 30¢ US per lb.	= $ 5.22 US
Zinc @ 60¢ US per lb.	= $16.27 US
Copper @ $1.06 US per lb.	= $ 3.51 US
Silver @ $4.50 US per oz.	= $ 2.17 US
Gold @ $365 US per oz.	= $ 1.41 US
Gross Value	= $28.58 US per tonne of tailings

In comparison, at today's prices for lead (24¢ US/lb) and zinc (55¢ US/lb), the gross value of a tonne of tailings is $26.18 (US). In the first quarter of 1989 prices were considerably higher (lead averaged 28.6¢ US/lb and zinc 86.9¢ US/lb) and the gross value was $35.63 US per tonne.

The calculated worth of the metal in all 56 million tonnes of Down Valley tailings at $28.58 US per tonne is $1.5 billion dollars. However, this sum must be balanced against the cost of recovering only a fraction of those metals, shipping that fraction to market, smelting it, as well as paying for environmental shutdown at the end of the day. The question to be answered is what, realistically, can be recovered?

In order to answer that question, Curragh Resources Inc. retained a number of metallurgical consultants and a test laboratory, as well as its own metallurgists and laboratory facilities.

The Metallurgical Test Program

In September 1991, the original and second tailings impoundment areas were sampled by digging 17 holes on a grid plan with a backhoe. Samples were cut, bagged and shipped for laboratory testwork. The initial laboratory program was designed to determine the physical and chemical description of each sample, metal distribution in each screen fraction, pH variation with depth of the hole and analysis of the supernatant water leached from each grid hole composite.

Screen analyses of each sample gave varied results. Some samples indicated mineral enrichment of the finer fractions. However this could not be used to advantage in reprocessing because the indication was not universal. It was concluded that the entire tailings would have to be reprocessed rather than one specific grain size range.

Microscopic examination of the composite sample indicated that some of the zinc sulphide (sphalerite) and lead sulphide (galena) present in very fine tailings could be recovered in a commercial milling process. The balance of mineral values were bound up in grains that are a mixture of pyrite (iron sulphide), sphalerite and galena crystals which would be difficult to recover under present technology.

A composite of samples was prepared at the Faro laboratory and sent to Lakefield Laboratories in Ontario for preliminary flotation testwork. The pH of the composite tailings was 4.0 to 5.2 and indicated the need for lime for neutralization.

The Lakefield program subjected the composite sample to selective flotation procedures (endeavouring to recover a separate zinc concentrate and separate lead concentrate) and bulk flotation procedures (endeavouring to recover a mixed lead-zinc concentrate). Tests were run on the total tailings and on separated sand and fines fractions of the tailings.

Because of unfavourable economics (reagent costs exceeded metal values) the selective flotation approach was abandoned and bulk concentrate production became the focus of testwork.

Batch flotation tests with the sand size fraction and finer fraction of the tailings treated separately produced flotation results similar to the total tailings results. Reagent consumptions were high so the sand-and-fines approach was also abandoned and further work focused on test of the total tailings.

In the next series of flotation tests at Lakefield, various flowsheets and reagent schemes were tested. Total tailings were used in the evaluation of several different collectors (a reagent) to produce a bulk concentrate. The reagents tested are experimental but can be purchased commercially. Results to date

are encouraging and testwork is ongoing. This work offers potential to further improve the reprocessing plan however at this time a simpler reagent scheme is proposed.

A second laboratory campaign was designed to produce a bulk concentrate from the total tailings with the simplified reagent scheme. Preliminary flotation test results indicated that it is possible in the laboratory to produce a bulk concentrate assaying 2.2% lead and 12.9% zinc at a calculated recovery of 24.5% and 63% respectively. These are encouraging results which call for further testwork to optimize concentrate grade and metal recovery. This testwork is ongoing. Considering the metallurgical response obtained, Curragh concluded that a marketable bulk concentrate is achievable.

Expected Plant Metallurgical Balance

The expected metallurgical balance was derived from the preliminary test data outlined above, a review of sample mineralogy, knowledge of the Faro plant performance and discussions with laboratory metallurgists at Lakefield and Faro, as well as outside metallurgical consultants.

The conclusion is that a bulk concentrate with the following specifications can be produced at the Faro mill from Rose Creek Valley tailings feedstock:

Lead content	14.2%
Zinc content	37.1%
Silver content	165 grams per tonne
Metal Recovery	25% (combined Pb/Zn)
Weight Recovery	1% of feed weight will be recovered; i.e. it is necessary to process 100 tonnes of tailings to recover one tonne of concentrate

Reprocessing Flows in the Mill

While it appears technically possible to produce a bulk concentrate from Down Valley tailings, it is necessary to accomplish this within the processing capabilities of a modified Faro mill. Metallurgical engineering at Curragh has concluded that a simple, modified circuit can do the job.

Hydromonitored tailings from the Down Valley will be pumped to a slurry surge tank (converted fine ore bin) in the mill. Lime will be added to the slurry, as well as the following reagents: an activator, a collector and a frother. The slurry will be pumped to six banks of flotation cells where preliminary bulk concentrate will be recovered. The preliminary bulk concentrate will be subjected to finer grinding; then it will be treated again with the same suite of reagents; and subjected to three stages of further flotation (cleaning). The final bulk concentrate will be thickened, filtered, and thermally dried so that it is ready for truck and ship transport to market. The residual tailings will be pumped to the Faro Pit.

Mill Operations

In order to assess the economics of this operation, it was assumed that the Faro Mill circuitry would be modified (simplified) to produce bulk concentrate, and that it would process Rose Creek Valley tailings 2-shifts per day, 24 hours per day, 6 months per year. Other key operating assumptions are:

Plant availability:	90%
Tailings processed:	1,234 tonnes (dry) per hour
	4.8 million tonnes (dry) per year
Concentrate output:	48,000 tonnes (dry) per year

It may be noted that, during reprocessing, the 6 month concentrate tonnage output will be slightly under one-tenth the output achieved by the Faro Mill in a typical 12 month year during regular operations.

A crew consisting of operators and contractors will mobilize the equipment each year before the spring start up, and winterize the equipment after operations cease in the fall.

The operating crew will comprise of:

Management/supervision	9
Operators, Tradesmen, Service	20
Total	29

Two employees will be retained on the payroll at the end of each operating season to caretake the minesite (buildings and equipment) and perform any monitoring and other environmental duties necessary. Caretakers will be required to make daily inspections of the minesite.

The electric power contract will have to be renegotiated to reflect seasonal use (summer usage only) which should be advantageous to the utility company.

For environmental monitoring, sampling will be done in the winter by the caretakers and in the summer by a designated staff person. During periods of more intense sampling, assistance will be organized by the Curragh Resources Inc. regional office in Whitehorse. All data will be recorded and interpreted at the Curragh regional office.

Total employment sustained by the tailings recycling/reprocessing operation will be 90 persons, excluding any multiplier impact on the Yukon. This will consist of 25 persons in hydromonitoring, 29 persons in milling and 35 persons in trucking and shipping.

Capital Cost Estimate

Capital costs for Mill modifications are estimated to be $500,000. The existing mill circuit will have to be modified to accommodate the reprocessing of Rose Creek Valley tailings, the production of a bulk concentrate, and the pumping of the tailings to the Faro Pit for final deposition. The capital cost estimate does not include provision for a second tailings pipeline and pumping station nor does it provide for a second reclaim water siphon pipeline from the open pit to the mill since these capital costs have been included in the hydromonitoring capital costs estimated by Kilborn Engineering.

Operating Cost

The operating cost for reprocessing has been based on reagent consumptions derived from the metallurgical tests described above, the staffing requirements outlined above and on extensive operational experience within the mill. The cost projection for hydromonitoring is based on the Kilborn Engineering Inc. estimate described above. Costs are presented in Canadian dollars per metric tonne of tailings reprocessed. The costs tabulated below include all costs required to move the tailings from Rose Creek valley, reprocess them, provide water conditioning to maintain water quality and pump the secondary tailings to the Faro Pit for final disposal.

Cost Centre	Cdn Dollars per Tonne of tailings
Manpower	$0.20
Reagents	$1.28
Supplies	$0.10
Power	$0.31
Subtotal Mill operating cost	$1.89
Hydromonitoring operating cost	$0.39
Total Operating Costs	$2.28

Minesite "Net Back" Value of the Bulk Concentrate

The estimated bulk concentrate dollar value returned to the minesite is based on projected smelter treatment charges and actual historical transportation and marketing costs as incurred by Curragh Resources in Yukon. The metal prices assumed were:

zinc	$0.60 US per pound
lead	$0.30 US per pound
silver	$4.50 US per ounce
Rate of exchange	$1.14 Canadian per US dollar

In valuing the netback at the Faro site for Rose Creek Valley bulk concentrate, we have assumed the standard ISF smelter contract with a zinc payment calculated on the basis of the lesser of 85% of the contained zinc metal or a deduction of 7% from the grade of the concentrate (due to the low grade of the bulk concentrate the 7% grade deduction gives the lower payable metal). Similarly the lead payment is the lesser of 95% of the contained lead or a deduction of 3% off the lead grade (again the grade deduction gives the lowest grade payable lead). Silver payment is the lesser of 95% or a deduction of 75 grams per tonne from the silver grade (the grade deduction again gives the lower payable silver). No payment for contained copper or gold was included.

The net-back value at mine site in Faro, under these assumptions, is $199 (US) per tonne of bulk concentrate shipped. Because 100 tonnes of tailings must be reprocessed to produce one tonne of concentrate this is equivalent to $1.99 (US) OR $2.27 Cdn at the assumed exchange rate, per tonne, of Rose Creek Valley concentrate processed.

The mine site net back represents the gross value of the metal less a) the payable metal deductions; b) the smelter treatment charge; c) marketing costs; d) ocean freight; e) storage and loading of the concentrate at the Skagway port facility; and f) trucking to Skagway. All site costs must be paid for from the net back value. If the site costs equal the net back the project breaks even. As noted above, site operating costs are $2.28 Cdn per tonne of tailings reprocessed. Thus, at the prices assumed the project will breakeven. At higher prices the project will make money and at lower it will lose money. The key to operating this project is to average the prices assumed.

Annual average zinc prices for the last twenty years are tabulated below. The average price for 20 years expressed in 1991 dollars is 60¢ US per pound, thus the assumed prices are reasonable. The average price for lead for the same period is 43¢ US per pound in 1991 dollars. The lead price selected is lower and more representative of the price average for the last 10 years in 1991 dollars.

Lead and zinc markets are expected to remain healthy for the foreseeable future. Zinc will see increased use in galvanizing and its pricing can be expected to follow activity cycles in heavy industry, automobiles and construction. Lead will see increased use as electric cars become more widespread but this demand may decline as viable alternative technologies for storage batteries are developed in the future. Little new supply of lead has been developed thus prices may remain stable or rise.

Technical detail and backup to support and elaborate upon the statements in this section can be found in Appendix C of this report.

	Zinc LME* SHG Cash US cents/lb	Zinc price In 1991 US cents/lb	1990 GDP Deflater
1970	13.9	45.3	37.88
1971	15.5	47.8	40.01
1972	17.7	52.1	41.93
1973	19.7	54.4	44.65
1974	35.3	89.6	48.62
1975	36.9	85.2	53.44
1976	36.1	79.4	56.08
1977	32.4	66.0	60.61
1978	27.5	52.2	65.01
1979	35.8	62.4	70.82
1980	36.1	57.7	77.27
1981	41.3	60.2	84.71
1982	38.0	52.0	90.14
1983	37.8	49.9	93.60
1984	45.4	57.7	97.10
1985	38.3	47.3	100.00
1986	36.3	43.7	102.59
1987	37.5	43.7	105.84
1988	56.6	63.8	109.36
1989	78.0	84.6	113.84
1990	68.9	71.8	118.48
1991**	52.9	52.9	123.45
Average	38.1	60.0	

* London Metal Exchange Super High Grade - European Producer Price used for years between 1970 and 1987

** 1991 Price is the year's average to July 3, 1991

6 ENVIRONMENTAL MANAGEMENT

Curragh Resources has a commitment to environmental management and monitoring throughout the life of mining at Faro and thereafter. Our primary environmental objective is to ensure high water quality.

Environmental Policy

Curragh has adopted the Mining Association of Canada Environmental policy and has committed the Company to this strong statement on environmental protection (see Appendix E).

Management Organization for the Environment

The organizational structure for environmental management of the Faro complex currently encompasses:

- A full-time Manager of Environmental Affairs, based in Whitehorse
- A full-time Environmental Engineer at Faro
- A full-time Environmental Technician at Faro
- Long standing relationships with the following consultants with special technical and environmental expertise:
  - --Steffen, Robertson and Kirsten
  - --Kilborn Inc.
  - --Richard F. Down, P. Eng.
  - --A. S. (Daniel) Webster
  - --Lakefield Research
  - --Rescan Environmental Services
  - --Cominco Engineering Services Ltd.
  - --Golder and Associates
  - --Piteau and Associates Ltd.
- Regulatory input from the Yukon Territory Water Board
- Technical input from the following Federal Government Agencies:
  - --Department of Indian Affairs and Northern Development
  - --Environment Canada, Environmental Protection Service
  - --Department of Fisheries and Oceans
- Technical input from the following Yukon Territory Government Agencies:
  - --Department of Renewable Resources
  - --Department of Economic Development
- A senior management and Board of Directors committed to funding the required environmental activities (in recent years at least 10% of Faro's capital expenditures have been environmental).

This network of private and public expertise assists Curragh in discharging its responsibilities with respect to environmental management issues for the Faro site.

In the context of the decommissioning plan for the Faro site these include the following areas.

Rerouting Tailings Deposition to the Faro Pit from Rose Creek Valley

When tailings are deposited into the Faro Pit instead of Rose Creek Valley, water and alkalinity inputs to the Rose Creek Valley tailings will decrease, however, the tailings pond water quality will be maintained. In the event that soluble metals values increase, Curragh will employ lime as a neutralizing factor and corrective, probably discharged as lime slurry from the mill via the current tailings line. Other methods of lime application as outlined in the April 1991 report are being investigated. Regular monitoring of water quality will allow for the detection of any harmful zinc enrichment and initiate the appropriate remedial action .

Handling Excess Water from the Faro Pit

Inflows to the Faro Pit, once flooded, will be greater than outflows. To prevent overtopping of Faro Lake the water-recycling siphon to be installed has been designed with excess capacity. Excess water will be siphoned off to the mill, and treated if necessary before discharge to the Intermediate Tailings Impoundment and Cross Valley Pond where it will be held before discharge to the environment.

Larger than normal inflows to the pit will be handled by operating the siphon so that the Faro Lake water level is drawn down and excess flow can be stored in the pit until discharge.

Closed Circuit During Reprocessing

During reprocessing of the Rose Creek Valley tailings, a closed circuit will exist among the Faro Pit, Mill and Hydromonitoring. No water will be released into Rose Creek directly unless it meets discharge standards. Lime will be added in the mill to prevent excessive acidity and to ensure that dissolved base metals are precipitated into the pit tailings. Excess flows will be managed as noted above.

Environmental Design Standards After Decommissioning

During the decommissioning at Faro, Curragh will construct, modify and/or eliminate various elements of the complex water-handling system described earlier in this report (Section 3). For example, new dams, new spillways, and new interceptor ditches must be constructed. The design standard will vary, but at a minimum new structures in Rose Creek Valley will withstand a one in 500 year flood. The seismic event (earthquake) standard for the Intermediate Dam in Rose Creek Valley will be one event in 475 years. The intake and outlet for Faro Lake will be designed to accommodate a one in 500 year event.

Environmental Management Organization after Decommissioning

Upon final closure, now estimated to occur in the year 2026, the Trusteed Environmental Fund which has been established for Faro will create sufficient revenues to support an on-going environmental management organization and program. A site supervisor and two full-time technicians will be employed to:

- operate the biological treatment system
- monitor water quality
- implement the downstream aquatic life monitoring program
- maintain site infrastructure (pumps, pipelines and instruments, etc.)
- inspect physical structures for safety and stability
- provide site security
- communicate and report

7 WATER TREATMENT

An important feature of the Faro environmental plan is provision for the treatment of water seeping from the waste rock dumps and from exposed sulphide areas at in the pits. The first line of environmental defence is to minimize water and oxygen contact with sulphide materials. Since much of the Faro waste dumps were constructed before acid mine drainage was realized to be a significant concern, the first defense is not available and we must retreat to a second and more expensive line of defense: to collect such seepage and to render it harmless.

Dump Seepage Collection

Dump seepage must be collected before it can be treated. There will be five collection points:

- a sump on the northwest corner of Faro Pit
- the Zone II Pit
- interceptor ditches at the toe of northeast dumps
- interceptor ditches at the toe of south dumps
- interceptor ditches at the toe of the southeast corner of the dumps immediately below the Vangorda Plateau Haul Road

From these collector points, contaminated water will be pumped to the underground biological treatment system. The system is capable of treating a sizeable volume of water at comparatively high base metal loadings.

Biological Water Treatment

In view of the importance of the water treatment issue, Curragh Resources Inc. requested SRK to evaluate the potential of establishing an in-situ sulphate reduction process as a water treatment system. Curragh's motivation was reports of a similar natural phenomenon occurring at the Løkken Mine in Norway.

The study indicated that, upon closure, the underground mine can be used as a "biological precipitator" in a sulphate-reduction system.

It should be noted that while the Faro Mine has been mainly open-pit, underground mining has also been carried out at Faro since 1990. Underground mining was the most efficient way to extract the lower extremities of ore. Today, a network of tunnels extend from the bottom of the pit. When underground mining terminates at Faro in the spring of 1992, a complex of abandoned tunnels will remain.

In mine water, sulphate originates from the naturally occurring chemical and bacterial oxidation of sulphide minerals exposed by mining to air and water. The scientific community has recognized the potential for application of a naturally occurring biological process to reverse the oxidation of sulphide minerals. Such biological process can remove the contaminants from mine and waste dump water and convert undesirable metal-rich water into environmentally acceptable water. Bacteria therefore return dissolved metals to the minerals from which they started: to zinc sulphide and lead sulphide, only now it is sludge deposited safely on the floor of the underground workings.

As applied to Faro, potentially harmful drainage will be collected as it emerges from the waste dumps, and will be pumped into the underground workings which will become "biological precipitators". The underground mine workings at Faro will be fitted with appropriate drill holes from the surface, pumps and instrumentation to manage water in-flows and out-flows. Multiple inlet and outlet drill holes will make process control more efficient and optimize use of the available volume in the mine workings. A schematic view of the treatment system is provided in Figure 4.

The SRK study discusses the overall biological sulphate reduction process, its success in other locations, and its potential at Faro. Once set into motion, the process is a model of simplicity: untreated, acidic, metal-rich, and potentially hazardous waste dump drainage is pumped in; benign water acceptable for discharge to the environment is pumped out.

A preliminary water management system was developed by SRK primarily for the evaluation of the biological water treatment system, but also to demonstrate the feasibility of overall water handling systems at Faro. The system provides for the collection of contaminated run-off in five retention structures. From these retention sumps, contaminated water will be pumped into the "biological precipitators". Naturally clean run-off will be diverted to prevent contamination.

The calculated rate of sulphate reduction will generate sufficient hydrogen sulphide to remove effectively all of the base metals in solution. Total treatment capacity will be considerably in excess of the expected volume of drainage and metals load.

The bacteria which sustain the reaction must be fed. Evaluation of potential food sources to sustain the sulphate-reducing bacterial culture indicates that feeding with sugar will be most cost-effective. Approximately one truck-load of raw industrial sugar will be pumped down from the surface every 4 to 6 weeks to keep the bacteria on the job.

An analysis of the stability of the existing underground mine shows that the existing workings should provide an adequate network of natural travel paths for efficient operation although local instabilities could alter the flow path over time. Sludge build-up (from precipitation of metal sulphides) will not significantly affect the process for many years, after which redistribution of the sludges by flushing may be required to maintain high reaction rates.

Details of the Biological Treatment system are provide in Appendix B of this Report.

BIOLOGICAL TREATMENT SYSTEM

FARO UNDERGROUND

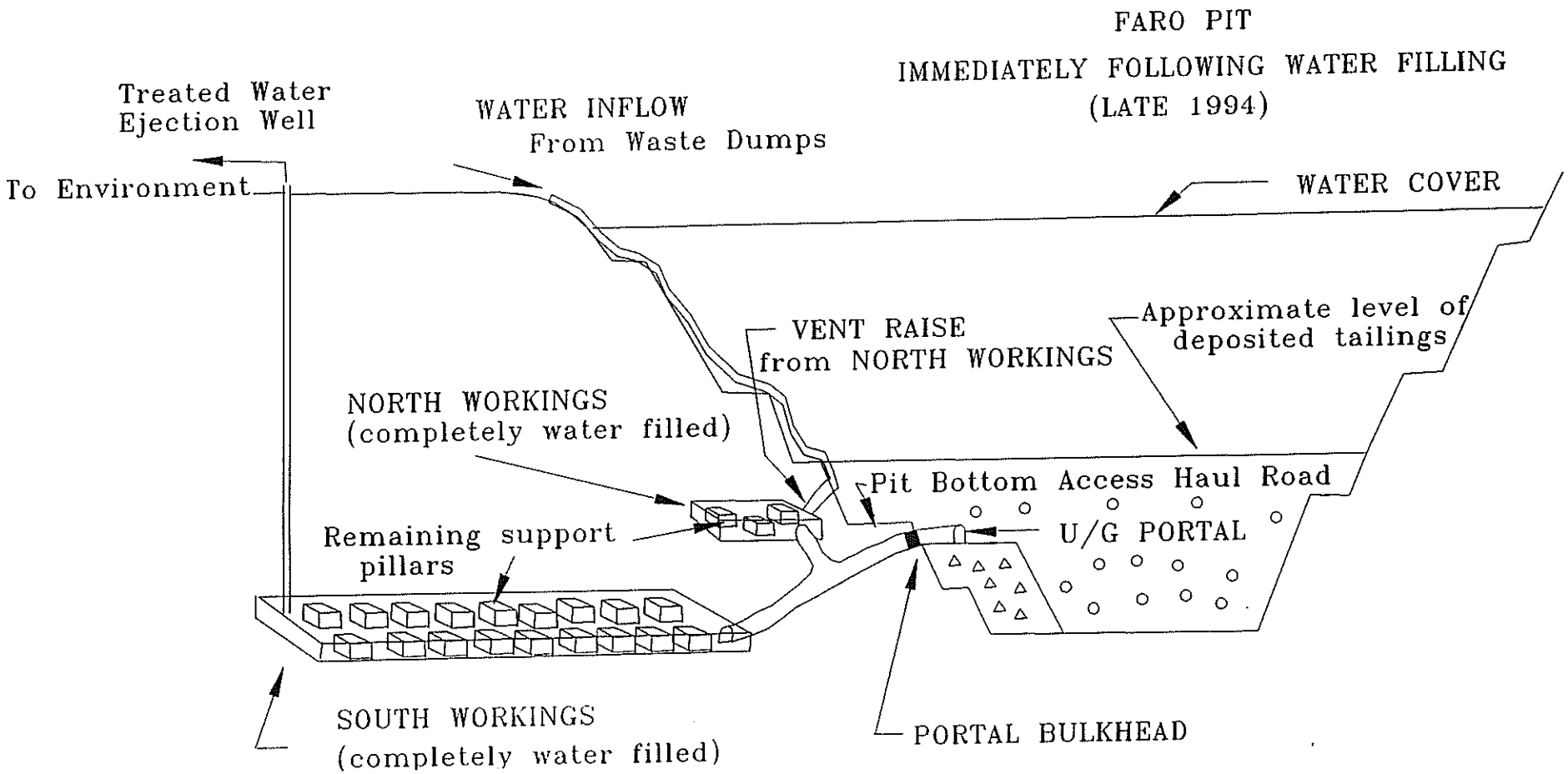


FIGURE 4 - BIOLOGICAL TREATMENT SYSTEM SCHEMATIC - Contaminated, dump seepage is pumped underground through a pipeline connected to the vent raise of the North Workings. This area increases the water temperature such that bacterial sulphate reduction may occur at an accelerated rate within the South Workings. Dissolved metals in the water combine with the free sulphide produced by the bacteria and settle out as a small volume of sludge. Water, suitable for release to the environment, is then pumped from the ejection well at the downstream end of the South Workings.

8 WATER QUALITY

Preservation of water quality is a primary objective in Curragh's environmental plan. Schedule D of license IN89-001 is a series of studies and investigations into various closure issues at the Faro Site. The Schedule D researches culminate in the development of an all inclusive water quality model for the site. This model will be submitted to the Yukon Territory Water Board with the final Schedule D report in March, 1992. We have used a preliminary version of this model to make the following estimates of downstream water quality in Rose Creek.

Final water quality will be a blend of the quality of water discharged in three different areas:

1. Water Quality from the Rose Creek Tailings Area

Five alternatives were examined for the decommissioning of the Rose Creek Tailings Area:

- Tailings exposed to the atmosphere
- Cover the tailings with natural soil material
- Water cover without reprocessing of tailings (the original Cyprus Anvil "large dam" approach)
- Composite soil/fine tailings/coarse rock cover on Original and Second Tailings Impoundments, and water cover over the Intermediate Tailings Impoundment
- Reprocessing of tailings and permanent water cover

Of the five alternatives evaluated, the one selected (re-processing of tailings and permanent water cover) yielded the most favourable final water quality results. This was because the selected alternative:

- minimizes zinc and other base metals concentration in surface waters by placing a water cover over the tailings to prevent further oxidation;
- minimizes the volume of seepage into the groundwater in comparison with other water-cover alternatives; and
- totally removes the oldest and coarsest tailings piled above Rose Creek rather than simply covering them.

Water quality modelling by SRK applied to Rose Creek downstream from the tailings area suggests that peak monthly zinc concentrations will be very low (0.007 mg/l) and flow-weighed average zinc concentrations will be correspondingly low (0.001 mg/l). Drainage from the tailings area at Faro has in the past been essentially neutral (pH about 7.5) and is expected to continue to be so in the future.

2. Water Quality from Faro Pit (Faro Lake)

During the placement of tailings in the Faro Pit, a water cover exists through the diversion of Faro Creek. This water cover will prevent oxidation of the tailings and subsequent metal pickup in the water. The alkalinity of the fresh tailings will suppress any soluble base metals released from the pit walls and floor. This plan is far more effective than the previous plan (1988 Abandonment Plan) which allowed the pit to flood with natural surface water.

The plan incorporates filling (hence sealing) the pit with tailings, and the building of a Plug Dam of low permeability which will restrict the leakage of pit water into Zone II where it would become contaminated.

Water quality modelling indicates that downstream flows out of Faro Lake after decommissioning will be of a high quality.

3. Water Quality from the Waste Dumps

Almost all seepage from the waste dumps will be collected and treated in the biological water treatment system. The previous (1988) plan called for some form of collection and treatment, but details were not provided. The core assumption, at that time, was treatment by adding lime to the effluent. While lime is effective, it is expensive and creates a sludge disposal problem.

The technology underlying Curragh's current (1991) plan--biological treatment--is based on a process demonstrated to occur naturally in the Løkken underground mine in Norway. Conditions at the Faro minesite are conducive to biological treatment since pH of the contaminated water is near-neutral and copper content is very low. The North Workings of the Faro Underground Mine will be used primarily to warm the water to make the biological process most efficient in the larger South Workings.

The input to the biological precipitator will be quite contaminated while the output is anticipated to be benign.

Total Impact on Water Quality

The total impact of the Faro site after implementation of the decommissioning plan on water quality is encouraging. Application of a preliminary water quality model to the entire Faro complex indicates that expected water quality downstream in Rose Creek will be very good.

Water quality will meet the CCREM (Canadian Council of Resource and Environmental Ministers) aquatic environment toxicity guidelines for zinc (maximum zinc 0.030 mg/l) below the tailings in Rose Creek. With a discharge quality from the underground biological precipitator of 0.020 mg/l, peak

monthly zinc values below the tailings area are estimated at 0.029 mg/l, and the flow-weighted average for the year is estimated at 0.022mg/l. As a result, there should be no long term toxic impact on downstream aquatic life.

9 TIMING

The scheduling of the Decommissioning at the Faro site must take several inter-related factors into account:

- the need to find storage for the tailings produced from Grum, Vangorda and the potentially new mine, Dy;
- Curragh's inability to reprocess Faro tailings in the mill while processing other ores;
- the need to secure the environment on an interim basis, until the full decommissioning program has been completed;
- the vagaries of world metal markets, which will shape the cash flows achievable from bulk concentrate sales; and
- new technology, which could dramatically improve the economics of tailings reprocessing and accelerate the schedule.

Time sequence of events are planned as follows:

1992	Cessation of mining the Faro Pit
	Cessation of mining in Faro Underground
	Modify underground facilities for future water treatment
	Accelerate mining at Vangorda Pit
	Stripping program commences at Grum Pit
	Faro Pit prepared for use as tailings facility
	Diversion of Faro Creek into Faro Pit
	Commence deposition of Grum/Vangorda tailings in Faro Pit and halt deposition of tailings in Rose Creek Valley on regular basis
	Continued lime treatment of waste dump seepage
	Commence lime stabilization of Rose Creek Tailings
1993	Commence construction of the Zone II Plug Dam
1994	Faro Lake at final water elevation
	Commence recycling of process water from Faro Lake

From this point in time onward all dates are assumed on the basis of the current mine plan.

1997	Commence development of Dy Mine
1998	Commence placement of Dy tailings under Faro Lake
2008	Completion of mining at Grum, Vangorda and Dy
	Completion of mining in the Anvil District
	Terminate placement of Grum/Vangorda/Dy tailings in Faro Lake
	Commence conversion of Faro Mill to reprocessing circuitry
2009	Commence hydromonitoring of Rose Creek Tailings
	Commence reprocessing of Rose Creek Tailings
	Placement of Rose Creek Tailings into Faro Lake
	Marketing of bulk concentrates
2017	Reprocessing of tailings completed
	Tailings in Faro Lake at final submerged elevation
2018	Commence final decommissioning program on structures
	Build spillways for Faro Lake inlet and outlet
	Build Intermediate Dam spillway
	Breach Cross Valley Dam
	Reduce height of Freshwater Reservoir Dam
	Breach the North Fork Rock Drain
	Construct waste dump interceptor ditches and sumps
	Drill holes into underground precipitator
	Complete facilities for underground water treatment
	Commence biological treatment of seepage from waste dumps
2020	Divert Rose Creek into Intermediate Dam
	Flooding of Rose Creek Tailings
	Re-vegetation of waste dumps and disturbed areas
	Create wetlands environment in Rose Creek Valley
2021	Begin intensive post-closure monitoring
2023	Begin long-term maintenance monitoring program

While the above program is consistent with best available technology and known economics as of 1991, it should be recognized that metallurgical and reclamation technology continues to evolve rapidly, as do world markets. Accordingly, the timing and methodology of closure may change in the future, necessitating modifications to the plan.

APPENDICES

APPENDIX A. *(bound separately)*

Curragh Resources, Faro Mine Tailings Relocation Project, Preliminary Design and Cost Estimates, Report Number 3509-62, Kilborn Inc., Toronto, Ontario, December 1991

APPENDIX B. *(bound separately)*

Sulphate Reduction as a Water Treatment Alternative at the Faro Mine, Report Number 60643, Steffen, Robertson & Kirsten (B.C.) Inc., Vancouver, B.C., December 1991

APPENDIX C. *(bound separately)*

Curragh Resources Inc., Faro Division, Reprocessing Tailings, Report Number WH91-07, R.F. Downs, P. Eng. and G.W. McDonald, Toronto, Ontario, December 1991

APPENDIX D. *(bound separately)*

Letter Report on Geotechnical considerations for Faro Pit Decommissioning, File #85-80413, Alan F. Stewart, P. Eng., Piteau & Associates Engineering Ltd., December 13, 1991

APPENDIX E.

Signed copy of the Mining Association of Canada Environmental Policy adopted by Curragh Resources Inc.

APPENDIX F.

A cautionary note to the reader on Faro Elevations

THE MINING ASSOCIATION OF CANADA
ENVIRONMENTAL POLICY

Member companies of The Mining Association of Canada are committed to the concept of sustainable development which requires balancing good stewardship in the protection of human health and the natural environment with the need for economic growth. Diligent application of technically proven and economically feasible environmental protection measures will be exercised throughout exploration, mining, processing and decommissioning activities to meet the requirements of legislation and to ensure the adoption of best management practices. To implement this policy, whether in Canada or abroad, the member companies of The Mining Association of Canada will:

- assess, plan, construct and operate their facilities in compliance with all applicable legislation providing for the protection of the environment, employees and the public;
- in the absence of legislation, apply cost-effective best management practices to advance environmental protection and to minimize environmental risks;
- maintain an active, continuing, self-monitoring program to ensure compliance with government and company requirements;
- foster research directed at expanding scientific knowledge of the impact of industry's activities on the environment, of environment/economy linkages, and of improved treatment technologies;
- work pro-actively with government and the public in the development of equitable, cost-effective and realistic laws for the protection of the environment; and
- enhance communications and understanding with governments, employees and the public.

Curragh Resources Inc.

hereby agrees with The Mining Association of Canada to adhere to the above policy.

FOR THE COMPANY

Chairman

Vice Chairman

May 29th, 1990
DATE

FOR THE MINING ASSOCIATION OF CANADA

CHAIRMAN

PRESIDENT

APPENDIX F

A CAUTIONARY NOTE TO THE READER ON FARO ELEVATIONS

Elevations used in Pit operations at Faro have historically been expressed in feet, and are often but not always given with respect to "Mine Datum". Elevations used in Tailings Pond operations at Faro have historically been expressed in metres, and are often but not always given with respect to "Down Valley Datum". The benchmarks for elevations used in the Pit and the Tailings Area are not the same (the differential being approximately 100 feet). The mean sea level datum is not quite consistent with either of the two benchmarks. These surveying peccadilloes derive from the original wilderness staking of these claims, and they long predate Curragh Resources Inc. involvement at Faro. For clarity in this report, all elevations are expressed in metres and feet, translated to the same common and identical benchmark. That benchmark is the most recent mean sea level datum for the area and is approximately the same as the elevation datum for the most recent Government of Canada 1:50,000 maps for the Region. Readers of reports on the Faro area, including technical reports tabled before the Yukon Territory Water Board by Curragh Resources Inc., its consultants, and its predecessors, are advised to confirm the elevation data used before comparing elevations.

Elevation conversion rules:

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

REFERENCES

("CRI Abandonment April 1988") Faro Mine Abandonment Plan, Whitehorse, Yukon, April 1988; prepared by R. McLenehan, Environmental Engineer and J. Bower, Mine Engineer.

("CRI Other Facilities 1989) Curragh Resources Inc., E. Soprovich, Environmental Engineer, Other Facilities Abandonment Plan, Whitehorse, Yukon, June 1989

("Kilborn Recycle 1991") Kilborn Inc., Curragh Resources Inc., Faro Mine, Water Recycle and Tailings Deposition Plan, Kilborn Engineering, Report Number 350928, Toronto, Ontario, June 1991. Filed with Yukon Territorial Water Board in July, 1991.

("SRK Down Valley 1991") Steffen Robertson & Kirsten, Down Valley Tailings Impoundment Decommissioning Plan, Faro, Yukon. SRK Report Number 60635, Vancouver, B. C., April 1991.

Volume I: Report
Volume II: Appendices A and B
Volume III: Appendices C to K

Filed with Yukon Territorial Water Board in April 1991.

("SRK Water Quality 1991") Steffen Robertson and Kirsten, Water Quality Assessment, Tailings Disposal in the Faro Pit, SRK Report No 81703, July 1991, included as Appendix A to Kilborn Recycle 1991 report.

REFERENCES

("CRI Abandonment April 1988") Faro Mine Abandonment Plan, Whitehorse, Yukon, April 1988; prepared by R. McLenehan, Environmental Engineer and J. Bower, Mine Engineer.

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FARO DECOMMISSIONING

OVERVIEW OF THE ENVIRONMENTAL PLANS

APPENDICES A to D

Appendices to Report # WH9108

CURRAGH RESOURCES INC.
Whitehorse, Yukon
December 1991

FARO DECOMMISSIONING

OVERVIEW OF THE ENVIRONMENTAL PLANS

LIST OF APPENDICES

A. Curragh Resources, Faro Mine Tailings Relocation Project, Preliminary Design and Cost Estimates, Report Number 3509-62, Kilborn Inc., Toronto, Ontario, December 1991

B. Sulphate Reduction as a Water Treatment Alternative at the Faro Mine, Report Number 60643, Steffen, Robertson & Kirsten (B.C.) Inc., Vancouver, B.C., December 1991

C. Curragh Resources Inc., Faro Division, Reprocessing Tailings, Report Number WH91-07, R.F. Downs, P. Eng. and G.W. McDonald, Toronto, Ontario, December 1991

D. Letter Report on Geotechnical considerations for Faro Pit Decommissioning, File #85-80413, Alan F. Stewart, P. Eng., Piteau & Associates Engineering Ltd., December 13, 1991

Appendices E and F are bound with the Overview Report

FARO DECOMMISSIONING

OVERVIEW OF THE ENVIRONMENTAL PLANS

APPENDIX A

Curragh Resources, Faro Mine Tailings Relocation Project, Preliminary Design and Cost Estimates, Report Number 3509-62, Kilborn Inc., Toronto, Ontario, December 1991

wp/3509-32

CURRAGH RESOURCES INC.

FARO MINE

TAILINGS RELOCATION PROJECT
PRELIMINARY DESIGN AND
COST ESTIMATES

DECEMBER 1991

Prepared By:

KILBORN INC.

2200 Lake Shore Blvd. West

Toronto, Ontario

M8V 1A4

December 4, 1991

Mr.Gerry Acott P.Biol.
Manager, Environmental Affairs
117 Industrial Road
Whitehorse
Yukon
Y1A 2T8

Dear Gerry:

Reference: Faro Tailings Relocation

As requested, we have completed a preliminary design and cost estimates for an 800 000 t/month tailings relocation system for the Rose Creek tailings. The proposed operation uses water delivered from the Faro pit using an existing siphon supplemented by an additional siphon into a new pipeline. It delivers trash-free tailings (100 % passing 2 mm) into the mill building for reprocessing. The treated tailings are then pumped to the Faro pit using the pumps and pumpbox which are currently being installed. Allowances are included in the estimate to use one of the existing Zinc conditioners as a pumpbox.

I hope we have successfully incorporated all comments into this final version of the report.

We have based the labour operating costs on a 6 month operating season only as we understand that you have included allowances for annual mobilization, start up and demobilization in your report covering the reprocessing. The cost data are for the relocation operation only and do not include for reprocessing or chemical treatment of the tailings. The power costs for pumping to the Faro pit from the mill are shown separately.

The capital cost of the proposed relocation scheme is $6.6 million and operating costs are estimated to be $0.39/t. An average operating season of 6 months should be achievable.

I trust the attached report satisfies your immediate needs. Please call the undersigned, John Goode or Ted Jurgens if any questions arise.

Yours very truly,

KILBORN INC.

R.H. Walton, P.Eng
Senior Metallurgist

RHW/rw

cc: C. Benner, G. Jilson, G. McDonald,
J. Mitchell, J.R. Goode, E. Jurgens,
A. Webster.

wp/3509-32

CURRAGH RESOURCES INC.

FARO MINE

TAILINGS RELOCATION PROJECT

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2.0	**SUMMARY**
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CURRAGH RESOURCES INC.
FARO MINE
TAILINGS RELOCATION PROJECT

1.0 INTRODUCTION AND SCOPE

In November of 1990, Kilborn Inc. (Kilborn) were contracted by Curragh Resources Inc. (Curragh) to evaluate various options for future tailings deposition and water recycle. The selected method, described in final report No. 3509-28 entitled "Water Recycle and Tailings Deposition Plan", submitted to the Yukon Territory Water Board in July of 1991, was based on environmental and economic considerations. The above study dealt both with the operational phase of tailings management and with the closure related aspects of tailings reprocessing and relocation.

Curragh has since requested that Kilborn conduct a more comprehensive evaluation of the hydraulic monitoring component of tailings relocation using input from a worldwide expert who offers working experience in this field. The report identifies a working methodology for the relocation of the Faro tailings, defines a conceptual mining plan, sizes and locates equipment and includes capital and operating costs for the plan.

In the proposed relocation scheme as outlined in SRK report 60635, it is only necessary to remove the tailings above the 1 044 metre elevation (UTM datum). The remaining tailings will then be allowed to flood behind the Intermediate Dam, which was raised to its final elevation of 1 049.4 metre (1 081.7 metre Down Valley Datum) in October 1991, and a permanent water cover will be established.

This study does not include examination of the metallurgical aspects of reprocessing tailings as Curragh are independently evaluating potential methods to maximise the profitability of reprocessing. No costs are included for the reprocessing of tailings or chemical treatment of the tailings should they not be reprocessed.

The results of this study cannot be considered definitive. The estimates are however sufficiently accurate to be used for preliminary evaluation of the project.

Metric units of measurement are used throughout this report except where specifically noted to the contrary. All costs are stated in fourth quarter 1991 Canadian dollars.

CURRAGH RESOURCES INC.

FARO MINE

TAILINGS RELOCATION PROJECT

2.0 SUMMARY

Kilborn has selected and examined a method for relocation of the tailings which consists of a downstream hydraulic monitoring operation. The proposed operation is described in the report as Alternative 1 in section 3 and uses a single in-dam permanent pump station which pumps slurry to a trash screening plant and booster pumping station located outside of the dam area. Some bulldozing may be necessary where hydroxide crusts have formed or where the tailings depth is insufficient to allow cost efficient monitoring.

In the proposed relocation scheme, only the tailings above the 1 044 metre elevation (UTM datum) are removed. The remaining tailings are then submerged in the man-made lake behind the Intermediate Dam, which was raised to its final elevation of 1 049.4 metre (1 081.7 Down Valley Datum) in October 1991. The relocation rate of 800 000 tonnes per month was selected by Curragh as being the maximum level consistent with a reprocessing operation in the existing mill.

The capital and operating costs for the operation have been estimated as $6,582,000 and $0.39/t respectively for an operation moving 4 800 000 tonnes per year during a 6 month annual operating season. Depending on the bulk density of the tailings, and the exact area which has to be cleared, the operating life of the project will be between 5 and 8 operating seasons.

CURRAGH RESOURCES INC.
FARO MINE
TAILINGS RELOCATION PROJECT

3.0 PROJECT DESCRIPTION

3.1 PRESENT TAILINGS STORAGE AREA

The present tailings storage facilities for the Faro Mine are located in the Rose Creek Valley, below the mine site. The tailings (from east to west) comprise the original 1969 tailings area (approximately 42 hectares), the 1974 tailings area (approximately 62 hectares), the Down Valley Intermediate Dam impoundment (approximately 89 hectares), and finally the Cross Valley Dam, which provides an approximate 22 hectare polishing pond area. These last two dams were both constructed in 1980.

The original dyke system was constructed by the former operators, Cyprus Anvil, in 1969 using borrow materials from excavated glacial tills and sands from the Rose Creek Valley. Tailings were discharged from a single point at the north west corner of the pond with spigotting during the summer months along the inner wall of the dyke, to raise the dyke with tailings sands. The water level in the original tailings area was first maintained by a decant tower. This was modified in 1971 when a surface spillway decant was installed at the eastern end of the pond. An attempt was made to raise the original tailings dyke at the western end using cycloned sand fill. This method was labour intensive and it was decided to employ a dragline to stack the tailings sand from the beach area to raise the dyke. The eastern end of the dyke near the slimes end of the basin was raised using mine waste rock. In the final two summers of operating this area, the dyke was raised further with excavated gravels.

A geotechnical study of the tailings structure carried out by Golder Associates Ltd. (Golder) in the summer of 1973 concluded that the original dykes as constructed by a variety of techniques were not capable of resisting anticipated seismic forces. Consequently, an expanded storage area, the second tailings impoundment, was designed to encompass the original tailings area with an engineered dyke system. The new starter dyke system was constructed in 1974 and completed in 1975. Discharge to this was from a single point discharge adjacent to the starter dyke. Dyke raising was accomplished by mechanical placement of tailings sand borrowed from within the pond area or gravel from the surrounding area. Construction of this expanded tailings area required that Rose Creek be diverted along the base of the south slope of the valley.

It was recognized during the mill expansion in 1980 - 1981 that appreciably more tailings would need to be stored in the valley and consequently a major expansion was undertaken to develop the Down Valley Impoundment Area. New facilities included the construction of the Intermediate Dam for the containment of the tailings solids, and the Cross Valley Dam located 500 metre downstream for decanted supernatant water. The polishing pond between the two dams provided sufficient retention time to permit chemical degradation and settlement of fine particles prior to release into Rose Creek.

Since both dams extended across Rose Creek, it was necessary to undertake a major Rose Creek realignment by a further extension of the 1974 - 1975 diversion. Excavated material from the diversion and additional imported fill was used to create a levee dyke parallel to the diversion channel.

Both dams and diversion channel were designed by Golder as engineered structures with criteria mandated by the Yukon Water Board with respect to flood flow handling and earthworks stability. An operating license was issued subject to the submission of an acceptable abandonment plan. Several consultants reports were prepared and issued on alternative abandonment plans during the period from 1982 to 1985, when the Faro operation was temporarily closed down.

Following resumption of milling operations in 1986, Curragh started depositing tailings into the Down Valley area, from a single point at the east end of the impoundment adjacent to the 1974 dam. The Intermediate Dam originally constructed in 1982 has been raised in stages three times. Each raise has been nominally 5 metre. All raisings have been carried out on the downstream side and involved placement of a number of zones of local natural borrow materials. Special filter zones have been prepared by crushing and screening operations.

A detailed account of the design and construction of the dams and the Rose Creek diversion channel is given in the various reports prepared by Golder in the period from 1980 to 1990.

3.2 TAILINGS RELOCATION METHODS

Although this study is not intended to evaluate alternate methods of tailings relocation, Kilborn has considered dredging and mechanical mining. Dredging is not considered practical for this project as it is usually applied to lake bottom or submerged deposits to allow the dredge to be floated. Mechanical mining could be more expensive than hydraulic monitoring, mainly due to the long haul distance. As the tailings have to enter a reprocessing plant, a repulping section would also be required to accept tailings delivered by truck.

Hydraulic monitoring is a low cost method, from both a capital and operating viewpoint, for removing large volumes of tailings. It is proposed to use this method for the majority of the tailings. Mechanical removal will be used to relocate tailings which are difficult to monitor for reasons such as low depth or hydroxide crusting.

It was decided that any pumps located within the dam boundary would have hydraulic drives to avoid flooding of electric motors. Also, in order to reduce the operating pressures and hence the maintenance of these pumps which operate under an arduous duty, it was decided that a booster pump station would be constructed at the north side of the dam. This is also a convenient location for a combined trash screening plant and monitor water booster station. A specialist consultant, Mr. A.S. Webster, the ex General Manager of East Rand Gold and Uranium Co. Ltd. (ERGO) was contacted regarding methods of hydraulic monitoring. His report is appended, it was with his input that the preferred alternative was developed. Based on his recommendations, normal operating slurry trench gradients to transport the material to the pump station will be 1 in a 100, and in the final stages of dam clean-up this will be reduced to 1 in 150.

3.3 ALTERNATE HYDRAULIC MONITORING METHODS

Selected Alternative (Alternative 1). The selected hydraulic monitoring method consists of downstream monitoring with a permanent pump station located at the end of a trench in a deep excavation.

This option has a number of similarities with the McIntyre reclamation site at the ERG project. Although this project was not a financial success, the hydraulic monitoring of tailings was effective and tailings tonnages in excess of 1.2 million tonnes per month were successfully and economically moved by this method. The technique is also widely used in South Africa for large above ground tailings dams. The advantages of this method are:

1) it avoids the extra work involved with repeatedly moving a satellite pump station.
2) clean up of the area can continue behind the monitoring operation without interference

The permanent pump station will be located in the north east corner of the Intermediate tailings storage area, adjacent to the former plug dam. It will have a floor elevation of 1 036 metres. Monitoring will start on the 1974 tailings dam, near the pump station, to establish a face running from south west to north east along the northerly edge of the tailings dam. The exact direction of the face will be determined by the necessity to achieve a suitable slope for the slurry to gravitate to the collection trenches on the site of the former plug dam. The face would be at its greatest depth at the western end and then taper out on the east. When the initial pass from south west to north east has traversed the tailings dam, the monitors will be moved back near the permanent pump station and a second traverse will be made. The monitoring of the tailings will continue in this manner. The furthest point of the monitoring operation will move progressively around in a clockwise manner with the pump station at the centre across the entire area of the tailings dam which is to be reclaimed. This is illustrated on Drawing no. 100-03-003, included in section 6.

The permanent pump station consists of two horizontal dredge pumps (one standby), capable of handling large pieces of trash (Goulds/Morris CKB or equivalent).

Several alternative hydraulic monitoring methods were considered and rejected. These are briefly described below.

Alternative 2. Downstream monitoring with temporary satellite pump stations located in shallow excavations.

With this approach, monitoring would begin at the extreme eastern end of the dam with a 5 metre cut through the tailings from a monitor gun located on the surface of the tailings. The satellite pump station would consist of a vertical pump suspended from a frame into an excavation made into the tailings located at the western end of a 100 metre trench. Once a 5 metre cut had been successfully made, the excavation would be deepened and a new trench dug. Subsequent 5 metre cuts would continue until an approximately 20 metre face had been formed and monitoring would proceed in a westerly direction. The satellite pump station would then be moved to a location convenient for the ground slope.

This method together with the selected alternative has the advantage of working down the slope so that the slurried tailings do not interfere with the monitor operations. However, the use of movable satellite pumps and pump stations involves extra work but allows flexibility to changing ground conditions and slopes.

Alternative 3. Upstream monitoring with a permanent pump station located outside of the second tailings dam on the dam wall in the south western corner of the dam.

Upstream monitoring has the disadvantage that monitored slurries have to be diverted around the ongoing monitoring operations. However once the initial excavation had been made, a high vertical face is available for monitoring. As the tailings relocation project progresses, an increasingly long ditch will be required to transport the tailings to the pump station. This will require some maintenance and the provision of adequate slope angles will be critical.

Alternative 4. Upstream monitoring with a pump station located outside of the second tailings dam connected to the second dam impoundment through a culvert.

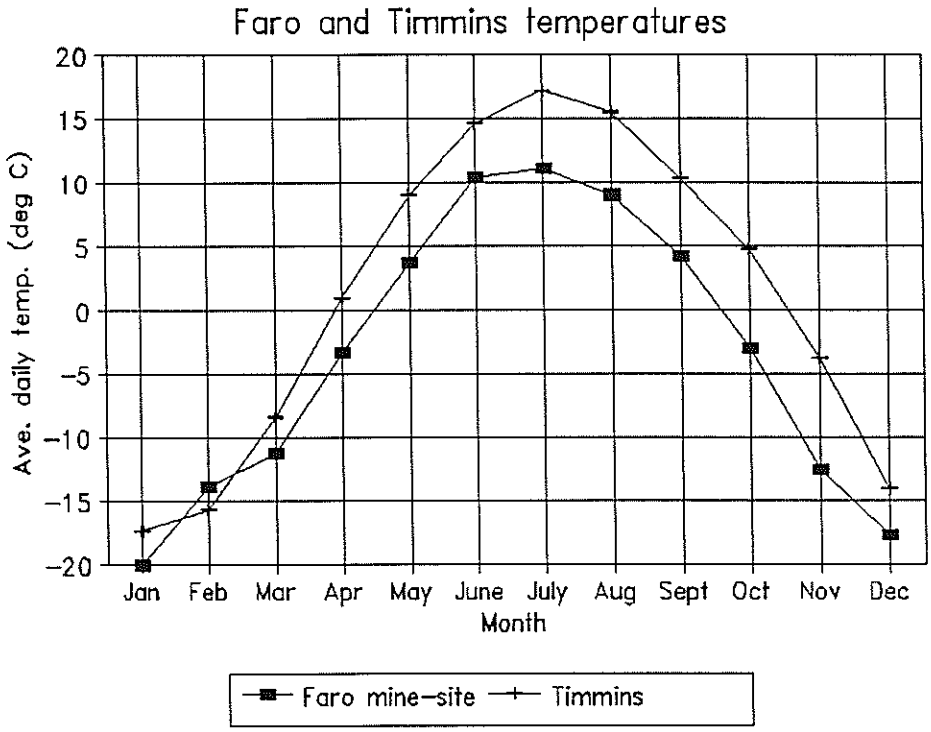
This method is similar to Alternative 3 but employs a culvert to direct the slurried tails to the pumps. Even large diameter culverts give problems with choking and plugging with trash and they are generally to be avoided.

3.4 OPERATING SCHEDULE AND LABOUR REQUIREMENTS

The proposed relocation method uses water as the medium for moving the tailings and it is not workable at water temperatures close to the freezing point. The annual operating period is dictated by the temperature of the water recycled from the open-pit which in turn will depend primarily on ambient air temperatures, but also on other factors such as the frequency of cloud cover.

The ERG tailings reprocessing plant near Timmins, Ontario operated over a one and a half year period. Experience at ERG suggests that an annual operating period of between 7 and 8 months was achievable.

Temperature ranges at the Faro mine site and Timmins are compared in Table 3.1. included overleaf and are illustrated on the graph below.



By measuring the distance between the lines as they cross the zero degree Celsius line, a comparison can be made between water temperatures in Faro and Timmins. It can be seen that ambient temperatures in Timmins rise above zero approximately 2 weeks earlier and fall below zero 4 weeks later in Timmins than at the Faro mine site. It would seem reasonable to assume that a monitoring season at the Faro mine would be 6 weeks shorter than in Timmins for a total of 6 months.

Operations would begin in mid April with preparations for the forthcoming year. The actual monitoring operations would begin in May and run for 6 months until the end of October at which time the operation would be shutdown and the equipment winterized. Contouring of the reclaimed area and general clean up would be an ongoing operational activity. The staff and labour will take leave from

mid November to mid April. The operating labour costs are based on a 6 month per year, 24 hour per day, 7 day per week operation. The mobilization, start up and demobilization costs are included in the Curragh reprocessing costs. The shift system is described in section 4.2.

The proposed operation will employ a labour force of 25. All operating positions could be filled by unskilled personnel following a brief training period.

Table 3.1 Air temperatures in Timmins and Faro.

Month	Average Daily Temperatures °C	
	Faro Mine Site	Timmins
January	-20.0	-17.3
February	-13.9	-15.6
March	-11.2	-8.4
April	-3.3	1.0
May	3.7	9.0
June	10.4	14.6
July	11.1	17.2
August	9.0	15.5
September	4.2	10.3
October	-3.0	4.8
November	-12.6	-3.8
December	-17.6	-14.0

Construction of the project would probably be done in a 4 month period between April and July following completion of detailed engineering and procurement during the previous January to March period. This would allow a short monitoring season of 3 months in the first year of operation.

The calculations included in section 6 show that a project life of between 5 and 6 years could be expected at a reclamation rate of 4 800 000 tonnes per year depending on the bulk density used. This will increase to 8 years if the tailings in the Intermediate Dam are included.

CURRAGH RESOURCES INC.

FARO MINE

TAILINGS RELOCATION PROJECT

4.0 OPERATING COSTS

The operating costs for a 4 800 000 t/a tailings relocation project are presented in the following sections and tables. No costs are included for the reprocessing of tailings or chemical treatment of the tailings should they not be reprocessed or the supernatant water in the pit.

4.1 SUMMARY

Operating costs for the proposed operation are $0.39/t. The costs are summarized below in Table 1.

Table 1 - Summary

ITEM	ANNUAL COST ($)	TOTAL COST ($/t)
Labour costs	600,615	0.13
Subcontractor costs	290,000	0.06
Supplies costs	430,000	0.09
Power costs	570,905	0.12
Total process costs	1,891,520	0.39

4.2 LABOUR COSTS

The labour costs shown below in Table 2 reflect the 6 month operating season only. The mobilization, start up and demobilization costs are included in the Curragh reprocessing costs.

It is assumed that two, twelve hour shifts will be worked per day, seven days per week during the 6 month operating season. Four workers are required for each shift operating position. Maintenance support will be provided by the mill crew who will be involved in reprocessing the tailings.

Table 2 Labour costs

POSITION	NO.	DIRECT RATE $/h	SALARY $/a/PERS	PAYROLL $/a/PERS	TOTAL $/a
Site foreman	1	-	22,500	30,375	30,375
Monitor operator	8	17	18,700	25,245	201,960
Booster operator	8	17	18,700	25,245	201,960
Labourers	8	14	15,400	20,790	166,320
Mech./electricians	incl'd in Curragh processing costs				
Total	25	-	-	-	600,615

4.3 SUPPLIES COSTS

The costs included below in Table 3 are to cover consumable items such as pump, pipe and screen spares, lubricants and off-site equipment repair.

Table 3 Supplies costs

ITEM	ANNUAL COST $
Pump/pipe/screen spares	300,000
Electrical spares	35,000
Lubricants	15,000
Major repairs and overhaul	50,000
Vehicle repairs	30,000
Total supplies cost	430,000

4.4 POWER COSTS

The power costs included in Table 4 are for relocation of the tailings from their present position to the mill. The additional annual costs for pumping from the mill to the open-pit using both of the available pumps which have 500 hp motors are estimated at $220 000 for the 6 month operating season, [(4

motors) x (500 hp each) x (0.746 hp to kW) x ($0.05/kWh) x (85% availability) x (80 % power draw) x (24 h/d) x (180 d)] or $0.05/t.

Table 4 Motor list and power costs

ITEM	MOTOR hp	NO. INSTALLED	INSTALLED POWER hp	LOAD FACTOR	AVERAGE hp	AVERAGE kW
Reclaim pumps	600	2	1,200	0.4	480.0	358.1
Slurry booster pumps	700	4	2,800	0.6	1,680.0	1,253.3
Trash screen	50	2	100	0.8	80.0	59.7
Sump pump	15	2	30	0.1	3.0	2.2
Monitor water booster pumps	800	2	1,600	0.8	1,280.0	954.9
Misc.lighting	50	1	50	0.4	20.0	14.9
Total		13	5,780	-	3,543	2,643
Annual operating cost for 180 d/a at $0.05/kWh						570,905

4.5 SUBCONTRACTOR COSTS

It is assumed that routine activities such as trench-digging, trash removal, vehicle maintenance and other activities will be done by contractor. Allowances have also been included for removal of edge material within the dam, rough contouring and road and building maintenance. These are shown below in Table 5.

Table 5 Subcontractor costs

ITEM	ANNUAL COST $
Trench construction, and edge material removal	200,000
Trash removal	40,000
Roads/building maint.	25,000
Surface vehicle operation	25,000
Total subcontract cost	290,000

CURRAGH RESOURCES INC.
FARO MINE
TAILINGS RELOCATION PROJECT

5.0 CAPITAL COSTS

5.1 COST BASIS

The capital costs for the tailings relocation project are presented on the summary page (page 1) of the attached estimate. No attempt has been made to utilize existing equipment except the existing siphon from the pit to the mill, a zinc conditioner, and the existing tailings pumps and pipeline from the mill to the pit. The scope of the estimate is described on the 5 pages attached to the summary page, the flowsheet in section 6 and the equipment list presented below.

The estimate is expressed in fourth quarter 1991 Canadian dollars. The construction crew manhour rate used is $50/hour.

Specific exclusions from the estimate are financing costs, escalation, taxes including G.S.T., further studies, permitting and working capital.

5.2 EQUIPMENT LIST

The proposed tailings relocation system uses water delivered from the Faro pit and delivers trash free tailings (100 % passing 2 mm) into the existing mill building using the equipment listed below. After being reprocessed, the tailings are then pumped to the Faro pit. Allowances are included to relocate one set of tailings pumps to the zinc conditioner which will be used as a tailings pumpbox. No costs are included for the tailings reprocessing operation itself.

- Second siphon - open pit to mill
- Second pump delivery line - mill to open pit
- Bypass pipeline for mill water tank
- Pipeline - mill to monitor water booster pumps
- Two monitor water booster pumps c/w motors (both operating)
- Pressure reducers and filters to provide gland service from the monitor water
- Six, 6" hydraulic monitor guns c/w controllers (includes one spare)
- Permanent pump station c/w one sump pump c/w motor, two reclaim pumps c/w motors (one standby) and hydraulic drives
- Pipeline - Permanent pump station to booster pump station

- Booster pump station, two trash screens c/w decks, splitter box and chutes, pumpbox and small building for electrical supply and operator shelter only
- Four Slurry booster pumps c/w motors (one variable speed) including one standby
- Pipeline - booster pump station to mill
- Relocation of tails pumps to zinc conditioner
- All misc. piping and valves, civils, structural steel, electrical and instrumentation

ACCOUNT CODE Ar. Gp. No.	DESCRIPTION	TYPE	QTY	UNIT	LABOUR mh/Unit	LABOUR Total mh	LABOUR mh Rate	LABOUR Cost	MATERIAL Unit Cost	MATERIAL Cost	SUB-CONTRACT Unit Cost	SUB-CONTRACT Cost	TOTAL COST
	* 001 FARO TAILINGS RELOCATION PROJECT *												
001	FARO TAILINGS RELOCATION PROJECT					13,765	-	688,250	-	3,533,546	-	652,720	4,875,000
	CONSTRUCTION INDIRECTS	(5 % of Direct Costs)				-	-	-	-	-	-	-	244,000
	ENGINEER.. PROCUREMENT. CONST. MAN.	(10 % of Direct Costs)				-	-	-	-	-	-	-	488,000
	CONTINGENCY	(20 % of Above Costs)				-	-	-	-	-	-	-	975,000
	TOTAL CAPITAL COSTS					-		13,765	-	688,250	-	3,533,546	-	652,720	6,582,000

NOTES:

1 ESTIMATE IS EXPRESSED IN FOURTH QUARTER 1991 CANADIAN DOLLARS

2 ESTIMATE MANHOURS REFLECT YUKON CONTRACTORS REQUIREMENTS, MANHOUR RATE $50.00 REFLECTS LOCAL RATES AND PRODUCTIVITY.

3 EXCLUSIONS FROM COST ESTIMATE TABULATED HEREIN ARE:

1. FINANCING COSTS
2. ESCALATION
3. TAXES (INCL.G.S.T.)
4. FURTHER STUDIES
5. PERMITTING
6. WORKING CAPITAL

ACCOUNT CODE Ar. Gp. No.	DESCRIPTION	TYPE	QTY	UNIT	LABOUR mh/Unit	LABOUR Total mh	LABOUR mh Rate	LABOUR Cost	MATERIAL Unit Cost	MATERIAL Cost	SUB-CONTRACT Unit Cost	SUB-CONTRACT Cost	TOTAL COST
001													
001	* AREA 001 * FARO TAILINGS RELOCATION PROJECT *												
001													
001	Water tank		1	each	Existing at mill								
001	Gravity piping (2nd siphon) - Faro pit to mill				see below								
001	Piping 500mm DR 21		1,550	m	0.60	930	50.00	46,500	94	145,700	0.00	0	192,200
001	Gravity piping - mill to monitor booster pumps				see below								
001	Piping 700mm DR 17		650	m	0.70	455	50.00	22,750	335	217,750	0.00	0	240,500
001	Piping 700mm DR 13.5		900	m	0.70	630	50.00	31,500	410	369,000	0.00	0	400,500
001	Monitor water booster pump (Goulds 10 x 12)		2	each	100	200	50.00	10,000	40,000	80,000	0	0	90,000
001	Motor for Above (rpm and HP) 1,800 800		2	each	400	800	50.00	40,000	45,000	90,000	0	0	130,000
001	Piping - monitor booster pumps to monitor		see below										
001	Piping 450mm (CS welded)		600	m	2.40	1,440	50.00	72,000	240	144,000	0.00	0	216,000
001	Piping 150mm (vic.groove)		100	m	0.70	70	50.00	3,500	60	6,000	0.00	0	9,500
001	Excavation for concrete pumpstation		320	m^3	0	0	50.00	0	0	0	10.00	3,200	3,200
001	Backfill for concrete pumpstation		320	m^3	0	0	50.00	0	0	0	10.00	3,200	3,200
001	Concrete for pumpstation		35	m^3	0	0	50.00	0	0	0	700.00	24,640	24,640

ACCOUNT CODE Ar. Gp. No.	DESCRIPTION		TYPE	QTY	UNIT	LABOUR mh/Unit	LABOUR Total mh	LABOUR mh Rate	LABOUR Cost	MATERIAL Unit Cost	MATERIAL Cost	SUB-CONTRACT Unit Cost	SUB-CONTRACT Cost	TOTAL COST
001														
001	Reclaim pumps (14" x 16" x 42")			2	each	80	160	50.00	8,000	50,000	100,000	0	0	108,000
001	Goulds model CKX or equiv.													
001														
001	Variable speed drive for above			1	each	200	200	50.00	10,000	90,000	90,000	0	0	100,000
001														
001	Motor for Above (rpm and HP)	1,800	600	2	each	300	600	50.00	30,000	38,000	76,000	0	0	106,000
001														
001	Allowance for hydraulics			1	lot	200	200	50.00	10,000	75,000	75,000	0	0	85,000
001														
001	Sump pump			1	each	30	30	50.00	1,500	9,000	9,000	0	0	10,500
001														
001	Motor for Above (rpm and HP)	1,800	15	1	each	10	10	50.00	500	968	968	0	0	1,468
001														
001	Hoist for pump maintenance			1	each	100	100	50.00	5,000	10,000	10,000	0	0	15,000
001														
001	Piping - Reclaim pump to booster pumps			see below										
001														
001	Piping 700mm DR 26			200	m	0.70	140	50.00	7,000	150	30,000	0.00	0	37,000
001														
001	150mm hydraulic monitor gun			6	each	100	600	50.00	30,000	30,000	180,000	0	0	210,000
001														
001	Drive control unit for above (1 spare)			7	each	100	700	50.00	35,000	14,000	98,000	0	0	133,000
001														
001	Motor for Above (rpm and HP)	1,800	3	7	each	10	70	50.00	3,500	incl. above	0	0	0	3,500
001														
001	Slurry collection trench and sump			2,000	m3	0	0	50.00	0	0	0	10	20,000	20,000
001														
001	Slurry booster pumps - 3 stage			4	each	100	400	50.00	20,000	55,000	220,000	0	0	240,000
001	(20" x 18" x 40") SRL XT													
001														

ACCOUNT CODE Ar. Gp. No.	DESCRIPTION	TYPE	QTY	UNIT	LABOUR mh/Unit	LABOUR Total mh	LABOUR mn Rate	LABOUR Cost	MATERIAL Unit Cost	MATERIAL Cost	SUB-CONTRACT Unit Cost	SUB-CONTRACT Cost	TOTAL COST
001	Variable speed drive for above		1	each	200	200	50.00	10,000	90,000	90,000	0	0	100,000
001													
001	Motor for Above (rpm and HP)	1,800 700	4	each	350	1,400	50.00	70,000	35,000	140,000	0	0	210,000
001													
001	Gland water supply pressure reducers		1	lot	0	0	50.00	0	0	0	15,000	15,000	15,000
001													
001	Piping - Booster pumps to mill		see below										
001													
001	Piping 800mm DR 11		900	m	0.80	720	50.00	36,000	438	394,200	0.00	0	430,200
001													
001	Piping 800mm DR 17		650	m	0.80	520	50.00	26,000	286	185,900	0.00	0	211,900
001													
001	Misc.piping allowance		1	lot	0	0	50.00	0	0	0	50,000	50,000	50,000
001													
001	Misc. valve allowance		1	lot	0	0	50.00	0	0	0	70,000	70,000	70,000
001													
001	Pipe bed grading		1	lot	0	0	50.00	0	0	0	15,000	15,000	15,000
001													
001	Access road crossing		1	lot	0	0	50.00	0	0	0	12,000	12,000	12,000
001													
001	Excavation for booster station		720	m^3	0	0	50.00	0	0	0	10.00	7,200	7,200
001													
001	Backfill for booster station		720	m^3	0	0	50.00	0	0	0	10.00	7,200	7,200
001													
001	Concrete foundations and floor for booster station		49	m3	0	0	50.00	0	0	0	700.00	34,160	34,160
001													
001	Building for booster station electrics		64	m3	0	0	50.00	0	0	0	80.00	5,120	5,120
001													
001	Trash screen 2400 x 6000		2	each	450	900	50.00	45,000	160,000	320,000	0	0	365,000
001													

ACCOUNT CODE Ar. Gp. No.	DESCRIPTION		TYPE	QTY	UNIT	LABOUR mh/Unit	LABOUR Total mh	LABOUR mh Rate	LABOUR Cost	MATERIAL Unit Cost	MATERIAL Cost	SUB-CONTRACT Unit Cost	SUB-CONTRACT Cost	TOTAL COST
001	Motor for Above (rpm and HP)	1,800	50	2	each	25	50	50.00	2,500	4,530	9,060	0	0	11,560
001	Pumpbox 6000 x 8000			1	each	150	150	50.00	7,500	40,000	40,000	0	0	47,500
001	Sump pump			1	each	30	30	50.00	1,500	9,000	9,000	0	0	10,500
001	Motor for Above (rpm and HP)	1,800	10	1	each	10	10	50.00	500	968	968	0	0	1,468
001	Struct. steel for screen etc. incl.chutes			25	t	0	0	50.00	0	0	0	3,000	75,000	75,000
001	Building services			1	lot	0	0	50.00	0	0	0	10,000	10,000	10,000
001	4 WD pick-up			1	each	existing								
001	2nd Pipeline 600mm HDPE (mill to pit)			1,800	m	0.80	1,440	50.00	72,000	130	234,000	0.00	0	306,000
001	Allowance to relocate existing tails pumps to mill at Zn cond.			1	each	0	0	50.00	0	0	0	100,000	100,000	100,000
001	Allowance for new pipeline (float cells to Zn cond.)			1	each	0	0	50.00	0	0	0	40,000	40,000	40,000
001	Allowance for elect. connections at mill			1	lot	0	0	50.00	0	0	0	30,000	30,000	30,000
001	4 kV overhead line			900	m	0	0	50.00	0	0	0	40.00	36,000	36,000
001	4 kV starters			6	each	40	240	50.00	12,000	15,000	90,000	0	0	102,000
001	Interruptor switch			1	each	30	30	50.00	1,500	10,000	10,000	0	0	11,500
001	750 kVA transformer			1	each	30	30	50.00	1,500	20,000	20,000	0	0	21,500

ACCOUNT CODE Ar. Gp. No.	DESCRIPTION	TYPE	QTY	UNIT	LABOUR mh/Unit	LABOUR Total mh	LABOUR mh Rate	LABOUR Cost	MATERIAL Unit Cost	MATERIAL Cost	SUB-CONTRACT Unit Cost	SUB-CONTRACT Cost	TOTAL COST
001													
001	600 / switchgear		1	each	40	40	50.00	2,000	15,000	15,000	0	0	17,000
001													
001	Bus duct		1	each	20	20	50.00	1,000	5,000	5,000	0	0	6,000
001													
001	MCC		1	each	10	10	50.00	500	10,000	10,000	0	0	10,500
001													
001	MCC/Motor wiring		1	each	240	240	50.00	12,000	19,000	19,000	0	0	31,000
001													
001	Allowance for lighting (fixed and portable)		1	lot	0	0	50.00	0	0	0	30,000	30,000	30,000
001													
001	Allowance for weatherproofing elect.equipment		1	lot	0	0	50.00	0	0	0	20,000	20,000	20,000
001													
001	Allowance for Freight and insurance		1	lot	0	0	50.00	0	0	0	25,000	25,000	25,000
001													
001	Allowance for Instrumentation		1	lot	0	0	50.00	0	0	0	20,000	20,000	20,000
001													
001	FARO TAILINGS RELOCATION PROJECT		-		-	13,765	-	688,250		3,533,546	-	652,720	4,874,516

CURRAGH RESOURCES INC.

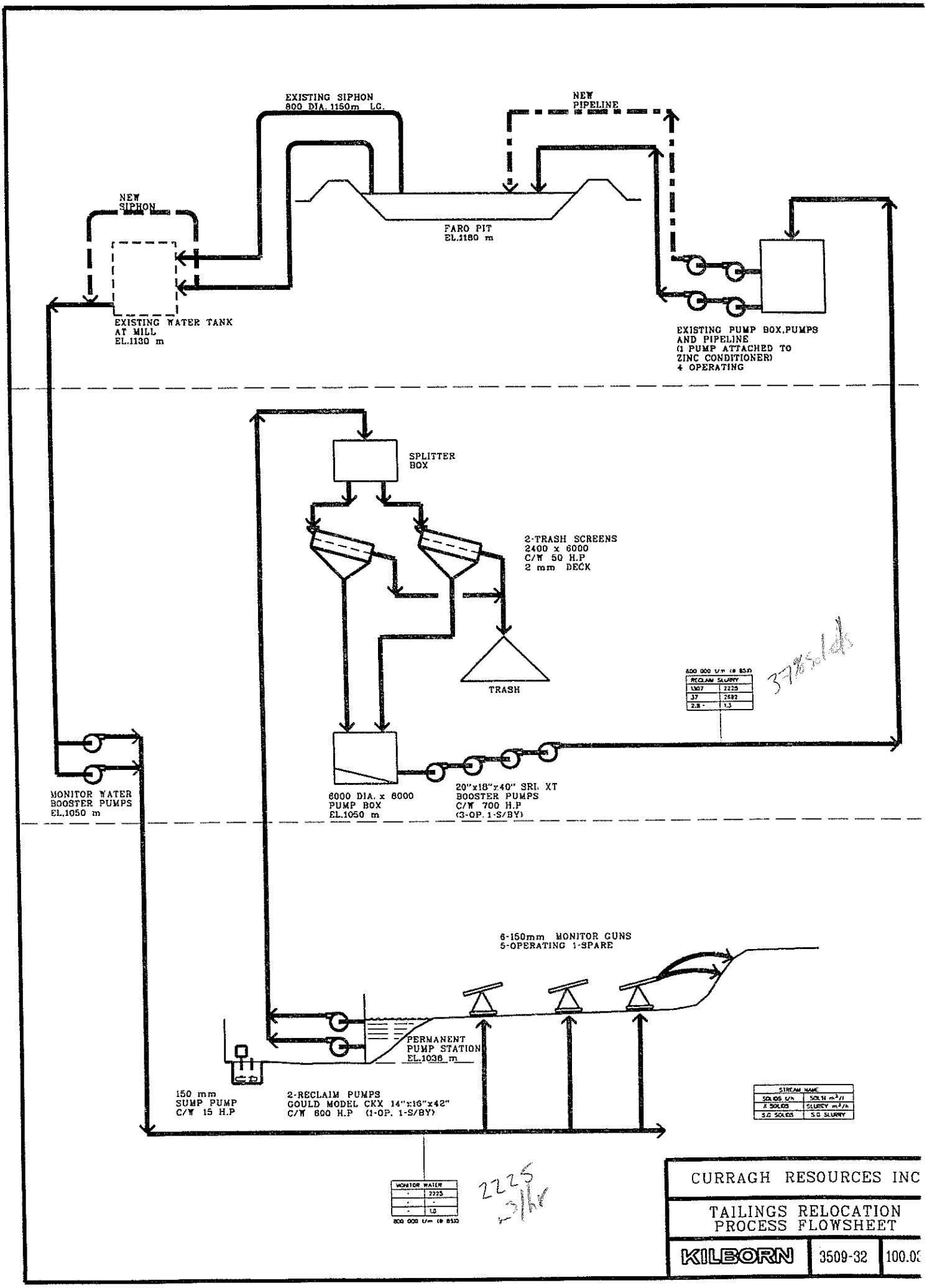
FARO MINE

TAILINGS RELOCATION PROJECT

6.0 DRAWINGS

The following drawings are included to describe the tailings relocation project.

Flowsheet	Dwg. No. 100-03-001
General Arrangement, Booster Station	Dwg. No. 100-03-002
Site Plan	Dwg. No. 100-03-003
Tailings Area Plan and Profile	Dwg. No. 100-03-004



RECLAIM SLURRY	
1307	2225
37	2692
2.8 -	1.3

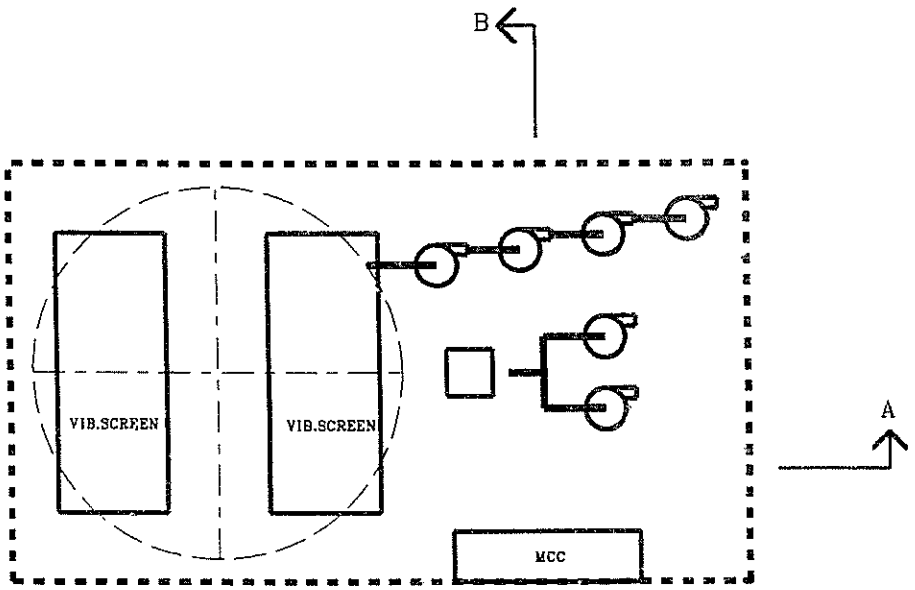
MONITOR WATER	
-	2225
-	-
-	1.0

STREAM NAME	
SOLIDS t/h	SOLN m³/h
% SOLIDS	SLURRY m³/h
S.G. SOLIDS	S.G. SLURRY

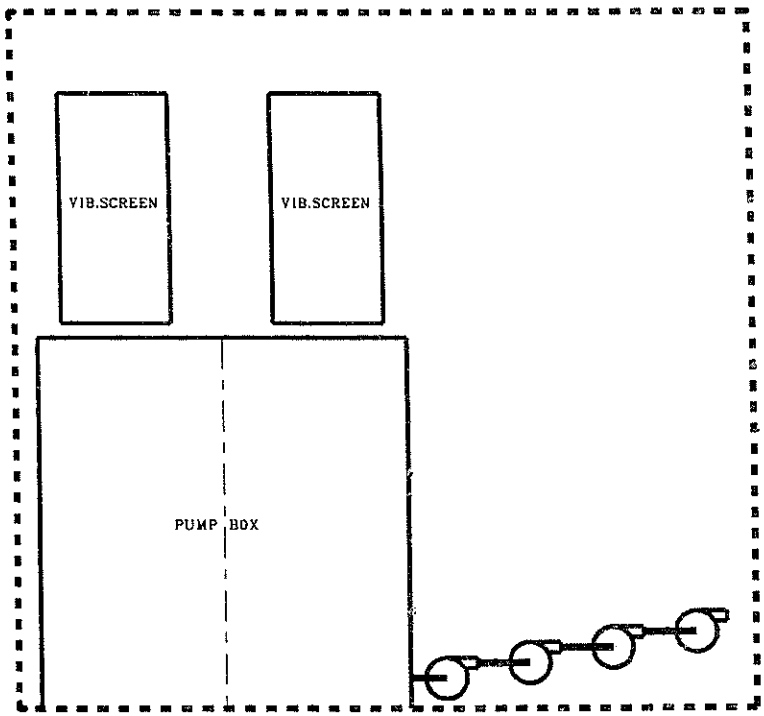
CURRAGH RESOURCES INC

TAILINGS RELOCATION
PROCESS FLOWSHEET

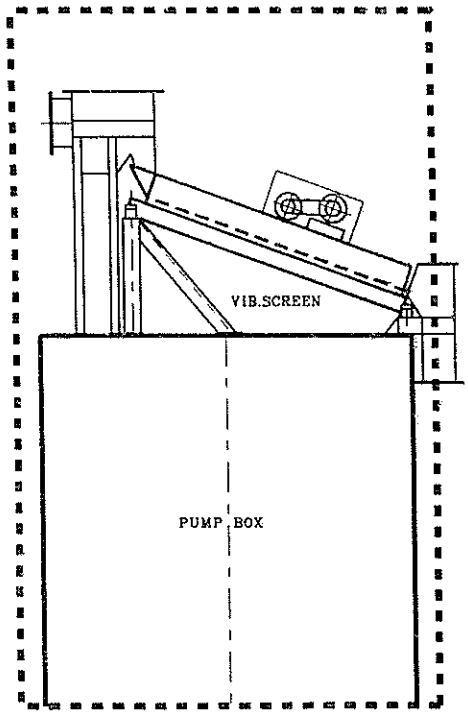
KILBORN | 3509-32 | 100.0



PLAN



SECTION A-A



SECTION B-B

CURRAGH RESOURCES INC.		
TAILINGS RELOCATION BOOSTER STATION-GEN.ARRG		
KILBORN	3509-32	100.03.

Oct 24/91

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CURRAGH RESOURCES INC
FARO MINE
TAILINGS RELOCATION PROJECT

Comments on the project report prepared by KILBORN INC October 1991

Proposed relocation method

The option selected, of hydraulic downstream monitoring is the most economic and flexible method for the reclamation of tailings. The only limitation being a short season of operation due to low temperatures but this equally applies to dredging and to a lesser extent to earthmoving. Downstream monitoring where the ground slope falls away from the monitor guns is clean, safe and an efficient method. Upstream monitoring is dirty, often unsafe and inefficient as all services become inundated with slurry and access extremely difficult.

Siting of primary pumping installations

Alternative I proposed a satellite station consisting of a vertical pump suspended from a frame into an excavation made into the tailings located at the end of a collecting trench. Sited 'in dam' this arrangement has similar disadvantages to upstream monitoring, namely messy and difficult access for maintenance and the removal of floating trash or trash too large to be pumped. Furthermore the total installation, pump and services require to be resited frequently. The ideal installation (option I) is a permanent pump station located outside the dam receiving the stream of monitored slurry over a vibrating screen. This requires a collecting trench to be constructed outside the dam and the invert at the screen to allow for a 1:100 but not less than 1:150 slope to the furthest point of reclamation.

A deeper (and more expensive) excavation is required to accommodate the screen and conventional slurry pumps with a flooded suction. A conveyor is required to carry the trash to a stockpile for further removal.

There is some preference to construct a cheaper primary pumps station (option II) to house a hydraulic drive vertical pump to deliver to a vibrating screen mounted over a large sump box feeding conventional sets of slurry pumps. This installation can be located above ground level and provides far better all round access and trash removal.

Option I may require a deep excavation to accommodate the vibrating screen and pump basement. An alternative sump site shown on the sketch could possibly eliminate some of this if a site could be found low enough to accommodate the equipment with minimum excavation.

However, both options should be evaluated for the advantage of removing all trash before any pumping has distinct advantages with fewer operating problems and minimum outage.

In either case the siting of the permanent collecting pump station should be carefully selected and a site visit - with a surveyor may be necessary to establish this.

Most slimes will follow along a slope of 1:100 and possibly 1:150 minimum if the product is not too coarse. An invert of 1 040m would allow for most of the tailings above the 1 044m level to be reclaimed.

Monitor Requirements

The proposal to use 4-150mm guns 3 operating and one spare is more than adequate to achieve the mining rate of 690tph. 690tph requires 14 375 litres per min to produce slurry at a density of 1,40. One 150mm gun consumes 7 128 litres per min with a 58mm nozzle and a pressure of 2,04 MPa (300psi). Therefore 2 operating guns would give the required tonnage.

The pressure of 2,04 MPa is recommended not only for the large trajectory but has the force to break up the hydroxide crust where necessary. Densities in excess of 1,5 can be achieved even with relatively low faces. A gun with a larger nozzle can be used for washing down at ground level where a lower pressure is required to avoid 'mining' below the original ground level.

Mining Method

It is suggested that the collection sump be located in the area of the former plug dam, where tailings have been deposited up to an elevation of 1 044m approximately. Monitoring should start at the NW corner of the dam to establish a face running from SW to NE as shown on the sketch. The exact direction of the face would be determined by the necessity to achieve a suitable slope for the slurry to gravitate to the collection trenches on the site of the former plug dam. The face would be at its greatest depth on the western end and then taper out to ground level on the east. The temptation to push for tonnage at the western end should be resisted and strict discipline maintained to ensure that the face is maintained in a reasonably straight line compatible with the slurry flow slopes required.

Apart from ensuring a clean and tidy operation it is the safest way to ensure that any wall collapse can only be into the hillside.

Two 150mm guns would have no difficulty in meeting the tonnage, the third gun could be used for 'washing the floor' as previously indicated.

The water supply to the monitors from the mill is likely to contain its fair share of suspended matter. Multi-stage pumps

are not suitable for this duty. The required pressure should be achieved by using single stage pumps in series.

As regards slurry pumping the following data derived at ERGO may be of some use when checking your own calculations.

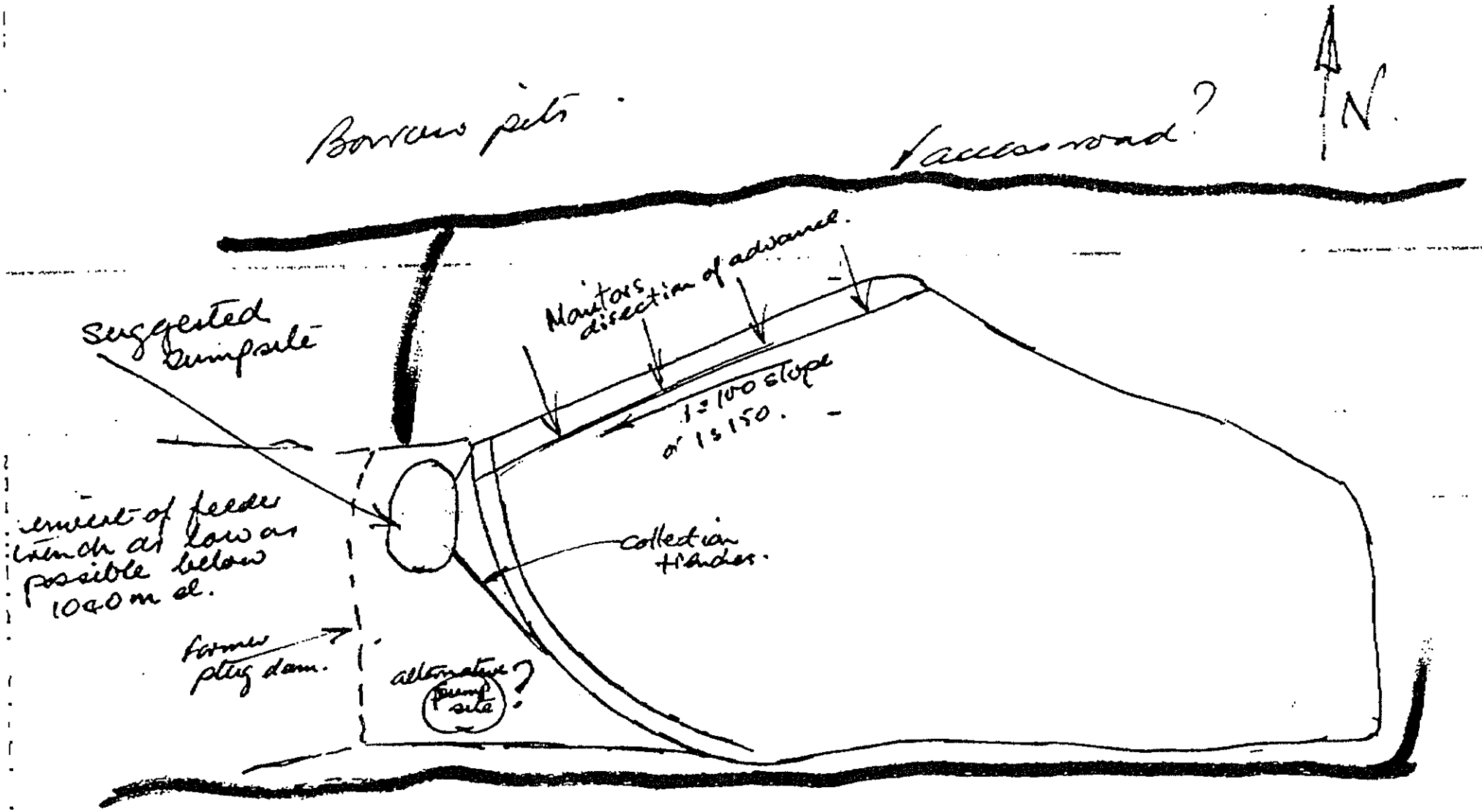
Flow at 1,9m/sec 450mm dia pipe.

Pressure gradients

Water	5,75m H_2O per kilometre					
Slurry	45%	solids	8,4m	H_2O	per	kilometre
"	35%	"	7,2m	"	"	"
"	55%	"	9,3m	"	"	"

Conclusion

The siting of the receiving sump for either option is of paramount importance and necessitates a site visit with a surveyor to thoroughly search the area for the optimum position. Once this is established there should be no difficulty in reclaiming the tonnage required.



SUGGESTED MINING METHOD.

FARO DECOMMISSIONING

OVERVIEW OF THE ENVIRONMENTAL PLANS

APPENDIX B

Sulphate Reduction as a Water Treatment Alternative at the Faro Mine, Report Number 60643, Steffen, Robertson & Kirsten (B.C.) Inc., Vancouver, B.C., December 1991

60643

SULPHATE REDUCTION AS A WATER TREATMENT ALTERNATIVE AT THE FARO MINE

Prepared for:

Curragh Resources Inc.
117 Industrial Road
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Prepared by:

STEFFEN, ROBERTSON AND KIRSTEN (B.C.) INC.
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DECEMBER 1991

SULPHATE REDUCTION AS A WATER TREATMENT ALTERNATIVE AT THE FARO MINE

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EXECUTIVE SUMMARY

The Faro lead-zinc mine, owned and operated by Curragh Resources Inc., is located approximately 200 km north-north-west of Whitehorse, Yukon Territory. During open pit mining, several rock piles were developed on the mine site along the perimeter of the open pit. The process of acid generation is causing poor water quality drainage from these rock piles.

Curragh Resources requested Steffen Robertson and Kirsten to evaluate the potential of establishing *in-situ* sulphate reduction in reactors developed on site as a water treatment system. The underground section of the Main Pit, and rock filled Zone II Pit, provide potential reactor sites for biological treatment. Sulphate reducing bacteria (SRB) convert sulphate produced during the oxidation of sulphide minerals to hydrogen sulphide. Alkalinity is produced during the process, essentially buffering the drainage to a pH of 7.0 while removing solubilised metals as sulphides. Contaminated drainage would be collected and introduced to the reactors, with treated effluents acceptable for discharge to the environment. Natural clean runoff diversion would be maximized to minimize treatment requirements. The study presents a discussion of the overall biological sulphate reduction process, and describes the potential for implementing this technology at the Faro site. A preliminary water management plan was developed primarily for the evaluation of the practical application of the sulphate reduction water treatment system, but also to demonstrate that water handling systems are feasible. The plan provides for the collection of contaminated drainage in two storage structures along with three sumps. Drainage water quality was calculated at an average zinc concentration of 26 mg/L at an average flow rate of 33 L/s.

The study indicates that, upon closure, the underground mine represents a potential biological reactor, ideally suited to the implementation of a sulphate reduction system. A base-case evaluation was completed on the assumption that the most direct route between inlet and outlet in the underground mine is in operation. A more tortuous route would yield higher treatment efficiency. The base case assumes a total volume of 18,800 m^3 available to the process. Through careful implementation, this volume may be expanded to approximately 100,400 m^3. Thus the base case evaluation only represents 19% of the full underground volumetric capacity. A preliminary temperature evaluation for the base case evaluation indicates that the water will be heated to approximately 11.8°C during its passage through the underground mine. At this temperature, a sulphate reduction rate of 0.06 g SO_4^{2-}/L/day is expected. This rate of sulphate reduction will generate sufficient hydrogen sulphide to effectively remove zinc in solution down to a concentration of 0.02mg/L., as well as several other heavy metals. Total treatment capacity would accommodate the load delivered by the average drainage rate. During a significant wet year event, controlled spillage to the environment would occur for the peak flow month only. Adoption of the treatment system should provide adequate protection of receiving water quality.

Evaluation of the potential substrates that may be used for sustaining the sulphate reducing bacterial culture indicated that the use of sugar will present the most cost effective option. Increased retention time may be achieved through the installation of multiple inlet and outlet drill holes.

An analysis of the underground mine stability shows that the mine should provide a network of travel paths well into the future. In the base case evaluation, it was determined that sludge accumulation should not significantly affect the process for approximately 25 to 30 years. If the system retention time is maximised, sludge accumulation will not significantly effect the process for at least 360 years.

SULPHATE REDUCTION AS A WATER TREATMENT ALTERNATIVE AT THE FARO MINE

1.0 INTRODUCTION

1.1 Background

The Faro Mine, is owned and operated by Curragh Resources Inc., and is located approximately 20 km north-west of the town of Faro, and 200 km north-north east of Whitehorse, the capital of Yukon Territory. Lead and zinc concentrates have been produced from the Faro mine, initially from an open pit operation, which was later extended into an underground operation. The metallurgical plant located on the south side of the pit produced 900 tonnes of zinc and 500 tonnes of lead concentrates per day in 1990. During the mining operations, several mined rock piles were developed at various locations around the southern and eastern perimeters of the main pit, and within an earlier minor pit, the Zone II pit. The locations of the metallurgical processing plant, the Main pit, the rock filled Zone II pit and the various mine rock piles are illustrated in Figure 1.

Poor quality runoff from the mine rock piles is occurring with zinc being the major metal contaminant of concern. On decommissioning of the site when pH adjustment from normal mill operations is no longer available, it will be necessary to institute remedial action on the runoff water issuing from the mine rock piles.

In their letter dated September 23, 1991, Curragh Resources Inc. requested Steffen Robertson and Kirsten (B.C.) Inc. (SRK) to evaluate the potential of implementing *in situ* sulphate reduction as a mine rock pile runoff water treatment alternative at the Faro mine site. Biological sulphate reduction has been recognised as one of the primary mechanisms in the remediation of water in wetlands (Reynolds *et al.*, 1991), and has been observed to occur naturally in underground mines (Arnesen *et al.*, 1991; SRK Report 66006, 1991). A work scope for the evaluation of the potential for implementing sulphate reduction at the Faro mine site was defined in the letter dated 26 September, 1991, (Proposal Number E5486) and is reported herein.

This report will present some background information on the biological sulphate reduction process, as illustrated in the sulphur cycle, and in more detail, discuss the metabolism and various mechanisms involved in the application of the process as a whole. The current hydrological and water quality of the rock pile drainage at the Faro mine will be summarised. The conditions and the proposed application at the Løkken mine will be addressed. Subsequently, the possibility of implementing a sulphate reduction system at Faro within the rock filled Zone II Pit and the underground workings off the Main Pit will be addressed. The overall system performance and implication on the drainage quality improvement from the site will be estimated.

1.2 Description of the Proposed Process

As will be discussed in Section 3.2, the underground mine presents the most viable location for the implementation of an *in situ* sulphate reduction system. The proposed water treatment system configuration will therefore consist of:

- contaminated rock pile drainage collection network and a fresh water diversion system (see section 5.0 for detailed discussion). Water retention structures will be constructed at the locations indicated on Figure 10. A sidehill dam will be constructed north of the mine access road in the old channel of Faro creek. In addition, two sumps will be constructed for the collection of runoff currently entering the North Fork of Rose Creek. These sumps will also contribute to overall drainage retention. Drainage from the rock pile located in the Faro Valley upstream of the Main Pit will be diverted by gravity directly to the Zone II Pit via Sump A30. The Zone II Pit will be utilised as a combined collection and storage system. From the two storage locations, contaminated rate will be pumped to the treatment system at a steady flow rate equal to the weighted annual drainage rate of 33 L/s;

- the *in situ* sulphate reduction treatment system (see section 3.0 for detailed analysis) will be located in the underground mine assessed from the Main Pit. The entrance portal joining the mine and the Main Pit will be sealed off. For the base case evaluation, the water to be treated will be introduced at the existing ventilation raise to the northern section of the underground mine. A drill hole will be installed at the furthest point of the southern section of the underground mine, and will serve as the discharge point (see Figure 5). The water will be discharged directly into the Faro Creek diversion channel to be constructed for overflow from the Main Pit once flooded.

The base case evaluation presents a conservative representation for the implementation of a sulphate reduction process in terms of retention time. The volume utilised for the base case evaluation is approximately 18,800m^3. Upon termination of mining, the estimated void in the underground mine will be approximately 251,000 m^3. Due to blind ramps and backfilled sections, it is estimated that approximately 100,400m^3 may be available as retention space for the process. Therefore, the base case evaluation represents only 18.7% of the mine volume available for the process. Should the process prove successful under these extremely conservative operating conditions, performance under enhanced operating conditions should far exceed those indicated by this evaluation.

2.0 SULPHATE REDUCTION

2.1 Introduction

In mine waters sulphate originates from the chemical and bacterial oxidation of sulphide minerals exposed during mining activities to air and water. Sulphate, and the associated heavy metals solubilised during the oxidation process, may be removed by well established demineralization procedures, such as reverse osmosis, electrodialysis and conventional chemical treatment, however, these processes are costly. More recently, the potential for the application of naturally occurring processes to achieve the removal of contaminants from mine and rock pile water has been recognised.

Natural wetlands have been observed to raise pH and reduce metal concentrations of acidic rock drainage. Recent research projects have studied several mechanisms in wetlands that result in the removal of metals. These include the adsorption of metals onto organic materials, uptake of metals by vegetation and the precipitation of metals. The biological reduction of sulphate to hydrogen sulphide has been identified to occur naturally in these wetlands. The precipitation of metals as the sulphide mineral following reaction with the hydrogen sulphide, is considered as the primary mechanisms by which metal removal is achieved in wetlands. Artificial wetlands are currently being considered as possible alternative passive treatment systems for the treatment of acidic drainage.

Recently, sulphate reduction has been found to occur in a flooded underground mine in Norway. At the Løkken mine site in Norway, two distinctly different sulphate reducing species have been identified in the mine water, and over time the water quality within the mine has been observed to improve, as indicated by a reduction in the metal and sulphate concentrations. This occurrence presents the alternative of engineering a sulphate reducing reactor within the underground mine, which, amongst others, presents the advantage of a reduced seasonal temperature impact on the performance of the system, to which wetlands are subject.

2.2 The Sulphur Cycle

Micro-organisms are frequently involved in the transformation and translocation of minerals. These activities may be associated directly or indirectly with the metabolism of the micro-organism. Indirect activities include the dissolution of the minerals under acidic conditions that result from microbial metabolism, the precipitation of minerals produced by anaerobic conditions, the adsorption to microbes, and the formation and destruction of organometallic complexes. Direct metabolic activity requires minerals as either soluble trace elements within the cell metabolic activity, or to serve as specific oxidizing substrates, electron donors/acceptors in the oxidation reduction activities.

In general there are three types of oxidation reduction reactions involving minerals:

i) Reduced minerals are oxidised by autotrophic (cell carbon from carbon dioxide) or mixotrophic (cell carbon from carbon dioxide or organic matter) organisms. The energy derived from the oxidation reaction is utilised in cell synthesis.

ii) Reduced minerals act as electron donors for bacterial photosynthetic activity - light energy drives the reaction.

iii) Oxidised minerals act as electron acceptors for heterotrophic (cell carbon from organic matter) and mixotrophic bacteria. The chemical energy is utilized to produce new cell material from an organic substrate.

In all three of the oxidation reduction reactions sulphur is the major mineral element involved. The natural sulphur cycle is illustrated in Figure 2, and involves various natural oxidation and microbiological transformation processes. The transformation of sulphur, which is cycled throughout the environment, involves diverse types of micro-organisms.

Sulphate is reduced to hydrogen sulphide indirectly through uptake by plants and is incorporated in plant protein, and, through the food chain, is subsequently incorporated into animal protein. Decay of the plant and animal matter through bacterial action results in the production of hydrogen sulphide and completes the cycle. During the decay process many bacteria, fungi, and actinomycetes release hydrogen sulphide to the environment. For example, a common bacterium that produces hydrogen sulphide in the presence of protein is the heterotroph *Proteus vulgaris*, where hydrogen sulphide is generated from organic sources such as those in sewage waste treatment plants where sulphide levels as high as 24 mg/L have been reported.

Hydrogen sulphide is oxidised to elemental sulphur by a variety of bacteria including a number of filamentous forms. The metabolism of these bacteria uses oxygen as the terminal electron acceptor, depositing elemental sulphur inside their cells. They function in transition zones between aerobic and anaerobic environments, where hydrogen sulphide and molecular oxygen can co-exist. Hydrogen sulphide may also be oxidised photosynthetically to elemental sulphur and sulphate by bacteria belonging to the families Chromtiacceae and Chlorobiaceae.

The oxidation of elemental sulphur and reduced inorganic sulphur compounds for energy production is limited to the bacterial genus *Thiobacillus*, the genus *Thiomicrospira* and the genus *Sulfolobus*. These bacteria produce sulphuric acid as a metabolic by-product and are responsible for accelerating the generation of acidic drainage where sulphide minerals are exposed to air and water.

The direct reduction of sulphate to hydrogen sulphide is brought about by specialized strictly anaerobic bacteria and is accomplished by two genera: *Desulfovibrio* (five species) and *Desulfotomaculum* (three species).

Many species of micro-organisms are involved in the cycling of sulphur in the environment. In many instances the impact of their activity is readily apparent, for example the production of acid from mine wastes and the evolution of noxious gases and odours. The interaction and transformation of sulphur species in the sulphur cycle is complex and multi faceted.

2.3 Sulphate Reducing Bacteria

Sulphate reducing bacteria are all gram negative, strictly anaerobic, heterotrophic (that is they obtain their carbon from organic compounds) organisms. Sulphates, thiosulphates, sulphites, or other reducible sulphur compounds serve as the terminal electron acceptors in the respiratory metabolism of these bacteria (Metabolic type iii in the sulphur cycle), with the resultant reduction of sulphate to hydrogen sulphide.

The organic substrates utilised by these bacteria are generally short chain acids such as lactic and pyruvic acid, which, in nature, are provided through the fermentation activities of other anaerobic bacteria on more complex organic substrates. Therefore, although the sulphate reducing bacteria are only able to utilise less complex organic compounds, a complex organic carbon source may be used in a mixed system, which is most likely to occur in nature. During anaerobic respiration sulphate reducing bacteria utilise lactate to produce acetate according to the following reaction (Cork and Cusanovich, 1979):

$$2CH_3CHOHCOO^- + SO_4^{2-} \rightarrow 2CH_3COO^- + 2HCO_3^- + H_2S$$

Under anaerobic conditions *Desulfovibrio desulfuricans* can act as a catalyst in the depolarization of the corrosion of iron. The following equations illustrate the process:

Anodic Solution of Iron:

$$4Fe^o + 8H^+ \rightarrow 4Fe^{2+} + 8H$$

Depolarization:

$$CaSO_4 + 8H \xrightarrow{D.\ desulfuricans} H_2S + 2H_2O + Ca(OH)_2$$

Corrosion Products:

$$Fe^{2+} + H_2S \rightarrow FeS + 2H^+$$
$$3Fe^{2+} + 6(OH)^- \rightarrow 3Fe(OH)_2$$

In this instance, the sulphate replaces oxygen and its' reduction is effected through the metabolic activity of *D. desulfuricans*. This process assumes that *D. desulfuricans* can use the hydrogen produced during the anodic solution of iron as the oxidisable substrate. The mechanism for the corrosion of iron is important for consideration of the stability of the underground mine, since the roof is principally re-inforced with iron bolts.

2.4 Kinetics of Sulphate Reduction

Influence of Substrate

The overall rate of sulphate reduction is dependent on various parameters. Amongst others, the rate is dictated by the concentration of the active biomass. The bacterial concentration in turn will be dependent on the substrate availability and will vary according to the efficiency with which the bacteria may utilise it.

Various researchers have evaluated the rate of sulphate reduction for different substrates utilised, and under various experimental conditions, and are summarised in Table 2.1.

TABLE 2.1

Biological sulphate reduction rates

Reference	$gSO_4^{2-}/L/day$	$gSO_4^{2-}/gVSS/day$	Temperature (°C)	Carbon Source
Middleton (1977)	-	0.03		acetic acid
Middleton *et al.* (1980)	2.25	0.11	35	primary sewage sludge
Pipes (1960)	1.20	0.09	35	waste activated sludge
Obarsky *et al.* (1978)	0.24	-	35	rubber waste
Rabolini (1971)	2.79	-	35	sewage sludge
Burgess *et al.* (1961)	4.50	-	35	primary sewage sludge
Oleszkiewicz *et al.* (1986)	1.50	-	35	cheese whey
Maree (1989)	6.40	0.11	30.5	molasses
McIntire *et al.* (1990)	0.029	-	-	wetland application
Reynolds *et al.* (1991)	0.230	-	18	sodium lactate
	0.413	-	18	hay extract
	0.087	-	18	mushroom compost
Reynolds *et al.* (1991)	0.115	-	18	3/4 cow manure and 1/4 planting soil

(VSS - Volatile suspended solids)

The sulphate reduction rates reported by Reynolds *et al.* (1991) indicate the sulphate reducing bacteria not only are able to utilise hay extract as an organic carbon source, but the relative reduction rate was greater than that obtained while utilizing molasses as a supplemental substrate to the mushroom compost. However, the use of hay only as a substrate source may require nitrogen supplementing to optimise sulphate reduction rates (Personal Communication Dr. R. Klusman). In the study completed by Reynolds *et al.* (1991), where the cow manure - planting soil substrate was evaluated, it was noted that cow manure has an indigenous active population of sulphate reducing bacteria.

Substrate Utilisation

The substrate utilisation efficiency will depend directly on the availability of the substrate to the sulphate reducing bacteria. For example, should the sulphate reducing bacteria have to rely on an intermediate step, such as fermentation, to convert the substrate into a consumable form, the utilisation will be lower than if the substrate was directly utilised by the sulphate reducing bacteria.

Selected substrate utilisations as a function of the total mass of sulphate reduced are provided in Table 2.2.

TABLE 2.2

Substrate Utilisation

Reference	Substrate	Utilisation (per gSO_4^{2-} reduced)
Cork (1978)	acetate	0.625 g
Maree (1983)	sugar	0.694 g
Maree (1989)	molasses	1.20 ml
	pulp mill effluent	9.30 ml
	raw sewage sludge	95.60 ml
Reynolds *et al.* (1991)	mushroom compost	37.0 g*

* Calculated on an availability of 8 months

Reynolds *et al.* (1991) however does not present sufficient data to enable a calculation of the utilisation for hay. The hay extract was prepared by simply contacting distilled water with hay for a period of one week. The supernatant was separated from the residue and used in the experimental test work. Addition of the hay extract resulted in a sulphate reduction rate increase in excess of four times the baseline rate obtained for mushroom compost over the same test period. It is anticipated that the utilisation for hay will be significantly better than that for the mushroom compost. Since only a hay extract was utilised for the

experimental testwork, it is anticipated that the actual requirement will be approximately at a ratio of 2 to 3 parts per mass equivalent part sulphate reduced.

Influence of Temperature

The reaction rate dependency on the temperature is illustrated in Figure 3. The optimum growth rate reported by Maree (1989) is 30.5 °C. Most bacterial growth is inhibited at temperatures greater than 38 °C. The reaction rates reported by Maree are 6.48 gSO_4^{2-}/L/day at 30.5 °C and 3.20 gSO_4^{2-}/L/day at 20 °C. This corresponds to a Q_{10} (Q_{10} represents the ratio of the rates of reaction at temperatures T and T-10 °C) value of 2.

Using the sulphate reduction rate of 0.412 at 18 °C for growth on a hay substrate (Reynolds *et al.* 1991), and a Q_{10} value of 2, the expected sulphate reduction rate at 9°C will be 0.206 gSO_4^{2-} /L/day.

Dr. R. Klusman, Colorado School of Mines, (personal communication) suggested a sulphate reduction rate of 0.0288 at 15 °C, 0.0096 at 10 °C and 0.0048 gSO_4^{2-}/L/day at 5 °C, which compare favourably with the rate suggested by McIntire, (1990) but are significantly lower than those reported by Reynolds *et al.* (1991), as summarised in Table 2.1, and reported elsewhere.

Metal Tolerance

Growth of sulphate reducing bacteria may be inhibited by conditions of acidic rock drainage. Copper is known to be toxic in very low concentrations to sulphate reducing bacteria. For example, the growth of *Desulfobacter* was significantly affected by the addition of copper at a concentration of 6.4 mg/L (Widdel, 1987). Similarly, sulphate reducing bacteria isolated from the Løkken mine waters were not able to tolerate high copper concentrations in solution. Arnesen *et al.* (1991) reported 50% growth inhibition at a copper concentration of 2.36 mgCu/L. In contrast, Reynolds *et al.* (1991) reported copper tolerances in excess of 100 mg/L with vigorous sulphate reduction activity.

However, sulphate reducing bacteria have a significantly greater tolerance of zinc in solution. Active bacteria were isolated from the Løkken mine in Norway at zinc concentrations exceeding 2000 mgZn/L.

Although pure cultures of acidophyllic sulphate reducing bacteria have not been isolated, sulphate reduction has been observed at low pH values for mixed cultures (Guyre *et al.* 1990). Arnesen *et al.*, (1991), isolated active sulphate reducing bacteria from water at a pH of 4.0. However, a pure strain of this culture did not tolerate pH values of less than 5.5.

In conclusion, it is noted that in nature where sulphate reducing bacteria occur as mixed cultures, extreme pH and elevated metal concentrations may be tolerated, while significant sulphate reduction rates are maintained.

2.5 Mechanism of Metals Removal

The activities of sulphate reducing bacteria is important not only due to the production of hydrogen sulphide through the reduction of sulphate, but also due to the subsequent metal interaction with hydrogen sulphide and the increase in alkalinity.

As an alternative to hydroxide precipitation, it has been shown that sulphide precipitation can be used effectively for the removal of heavy metals from contaminated effluents. The advantages of sulphide precipitation can be summarised as follows:

- solubilities of the sulphide minerals are extremely low (See Figure 4 for a comparison of the metal sulphide and hydroxide solubilities);
- metal sulphides are not amphoteric as is the case with some metal hydroxides;
- most heavy metals can be removed at a single pH;
- metal sulphides have better dewatering characteristics than metal hydroxides, and therefore form more compact (dense) sludges.

Hydrogen sulphide readily reacts with aqueous copper at low pH to form an insoluble sulphide. Zinc can be precipitated as a sulphide in a similar way. However, the reaction kinetics for the formation of this metal sulphide is significantly slower than that observed for copper and cadmium. Close to neutral pH values experienced for the current drainage will aid the formation of zinc sulphide.

2.6 Applications

2.6.1 General

Due to the natural occurrence of the sulphate reducing bacteria, sulphate reduction can be utilised *in situ* for the treatment of acid rock drainage provided the correct conditions can be maintained for sustained bacterial activity. Alternatively, recent developments have been directed at commercialization of an integrated process. Thus far, fixed bed reactors have been utilised. Stirred type reactors with a suspended solid medium have also been investigated with the objective of achieving higher reduction rates through improved operating conditions and reactor utilisation.

The integrated processing requires the recycling of inert carrier gases for the removal of H_2S from the reactor vessel, and dispersion of the nutrient gases within the solution medium.

The hydrogen sulphide is utilised in a pretreatment step where metals are removed as sulphides, prior to entering the anaerobic biological sulphate reducing reactor. This step removes contaminants in excess of the bacterial tolerance prior to entry in the biological reactor, thus ensuring operating conditions suitable for maximum sulphate reduction. Due to the differences in reaction kinetics and conditions under which the metals in solution react with the hydrogen sulphide, it is usually possible to produce high grade mineral concentrates. For example Cu is extracted at a low pH (less than 3.5), and with an adjustment in pH to approximately 4.0, Zn readily reacts. Due to the highly insoluble nature of the sulphide minerals, solution qualities within environmental requirements can readily be achieved for these elements.

Excess hydrogen sulphide can readily be converted to elemental sulphur (i.e. reaction with Fe^{3+}; bacteriologically etc.) or can be used to produce value added sulphur compounds (i.e. react with soda ash to produce NaHS).

2.6.2 The Big Five Wetland

Sulphate reduction in the anaerobic reaction zones of a wetlands has been recognized as one of the primary mechanisms contributing to the removal of heavy metal contaminants in such a system.

As such, artificial wetlands are currently being considered as possible passive treatment systems for the treatment of acidic rock drainage. The Big Five wetland in Idaho Springs, Colorado was constructed to determine the effectiveness of such a treatment system. Results of microbiological studies of the substrate at this site indicate the presence (in large numbers) of sulphate reducing bacteria. The effluent from the wetland shows a significant decrease in the concentrations of copper, iron and zinc, with consistent removal of copper, and up to 100 % removal of zinc and iron, depending on the retention time. Furthermore, significant increases in acid volatile sulphides (i.e. metal monosulphide precipitates) within the residues of the wetland have been recorded. The wetland was constructed with mushroom compost as a substrate. Naturally occurring bacterial sulphate reduction within the wetland has been identified as an important process in the treatment of acidic rock drainage at this location.

2.6.3 The Løkken Mine in Norway

Background

The water quality in the Wallenberg shaft at the Løkken mine in Norway has been monitored by the Norwegian Institute for Water Research (NIVA) after termination of mining activities. From the analyses of water samples taken by NIVA in the Wallenberg shaft it is known that there is an increase in pH and

decrease in sulphate and metal concentrations. The sulphate reduction could occur in the presence and activity of sulphate reducing bacteria, which will produce free sulphide resulting in the formation and precipitation of insoluble metal sulphide minerals. Such bacteria have been isolated in the Wallenberg shaft waters by NIVA.

Two distinctly different strains of sulphate reducing bacteria have been isolated by the Norwegian Institute for Water Research (NIVA) from the Wallenberg shaft water at depth. These strains have been designated as LØ 200 and LØ 490, of which the numeric designator coincides with the depth in the shaft in meters. The strain detected at the 490 level was also found to occur at the 300 level. The strains have been cultivated at the NIVA laboratories using batch reactor procedures and providing lactate as an organic substrate.

High dissolved carbon dioxide concentrations have been measured in the mine water at depth. As a consequence of sample depressurization during extraction, and since the water sample pH values were determined at the surface, it is anticipated that the actual pH at depth is likely to be affected by the high carbon dioxide concentrations. It was established that the host rock is alkaline in nature. Calcite is present, and significant quantities were observed in the mine rock at various locations on the site. Reaction of the calcite with the acidic water would produce high carbon dioxide concentrations at the elevated pressures at depth. It was also established that wood utilization in the underground workings during the 1950's and the 1960's ranged between 100 and 300 m^3/y. The wood was preserved using conventional arsenic compound impregnation techniques.

Sulphate Reducing Bacterial Activity

Since sulphate reducing bacteria have limited tolerance with respect to heavy metals concentrations and pH, it is likely that these bacteria were established under the relatively hostile conditions existing within the mine as a result of the formation of micro-environments well suited to these anaerobes. These localised micro-environments were probably created where sufficient alkalinity existed at the water interface with the mine walls to increase the pH, and to reduce the metal concentration through precipitation. Once established, and provided an adequate supply of organic substrate is available, the sulphate reducing bacteria produce sufficient alkalinity to maintain the pH at an acceptable level. The transfer of nutrients to, and products from the micro environments is reliant on a diffusion mechanism.

The sulphate reducing bacteria may rely on any one, or a combination of various respiration processes, as previously discussed. Based on the observations at the site, the following is noted:

- The surface runoff entering the mine has a high natural organic content. This organic content may present a consumable substrate for the bacteria. However, the contribution of this to the water at depth (490 level) during the initial flooding, and in conjunction with the transfer of mine water

from the Astrup workings (assuming mainly ground water) this source is considered secondary for the bacterial population at depth. However, this source may be more readily available to the bacterial population established at the 200 level, and may explain the difference in the cultures observed at this depth and at the 490 level.

- As previously mentioned, *Desulfovibrio desulfuricans* can act as a catalyst in the depolarization of the corrosion of iron. This mechanism, where the hydrogen produced during the anodic dissolution of iron may be utilised as the oxidisable substrate, may have aided the establishment of the bacterial population at depth.

- Considerable quantities of wood were used during the mining process. The hydrolysis of wood, where the carbohydrate content is converted to simple sugars for fermentation, is commercially achieved at elevated temperatures with the aid of dilute sulphuric acid. All wood types can be hydrolysed. The rate of hydrolysis is primarily dictated by the temperature and the strength of the acid. However, it is expected that with the relatively long contact time of the acidic water with the wood, that hydrolysis of the wood may be occurring. Also, considering the vast quantity of wood available in the mine, even if only fractional hydrolysis has occurred, it would result in the release of a vast quantity of organic substrate. It is assumed that this mechanism may provide a readily available substrate for the bacteria at depth.

It is estimated that a sulphate reduction rate of approximately 1.4 to 1.6 g SO_4^{2-}/L/day (utilizing molasses as a substrate) is possible at the existing temperature in the mine (15°C), provided that substrate inhibition does not exist. Based on the current data, a rough estimate indicates an observed sulphate reduction rate of approximately **0.04 gSO_4^{2-}/L/day**. It should also be noted that the bacteria are present under extreme conditions, with zinc concentrations exceeding 2000 mg/L and sulphate concentrations in excess of 50 000 mg/L. Since the bacteria are probably active within micro-environments, the rate of reduction is likely diffusion dependent. In addition the activity and population density of the sulphate reducing bacteria is unknown.

Proposed Decommissioning Plan at the Løkken Mine

The results of a water sampling program conducted on the water depth profile in the Wallenberg shaft indicated a natural improvement, with time, in the water quality. Based on these observations, a decommissioning plan was developed for the Løkken mine which utilizes this natural process. It is intended that seepage from the rock piles will be collected in a system of ditches leading to a pump station from where it will be pumped to the underground mine via an entrance at the Fearnley shaft, located to the opposite of the mine workings in relation to the Wallenberg shaft . Simultaneously, water will be pumped and discharged from the Wallenberg shaft. It is envisaged that, through enhancement of the

natural processes in the mine, improvement in the water quality will be sustained thereby reducing copper and other heavy metal concentrations in the acidic rock drainage as it flows through the mine.

The Norwegian State Pollution Control is actively investigating the development of sulphate reduction technology, in particular for implementation of this technology at the Løkken mine site to serve as a passive treatment system to reduce the acidic rock drainage from this mine site. A two year research and engineering study, funded by the pollution control authority, will commence in 1992.

3.0 SULPHATE REDUCTION APPLICATION AT THE FARO MINE

3.1 Current Faro Mine Site Drainage Water Quality

For the evaluation of the applicability of *in situ* sulphate reduction as a viable passive water treatment option at the Faro mine site, the current water quality will be used as a preliminary indicator.

A water model has been generated on a Lotus spreadsheet to model the zinc concentrations from various sources on the mine site, and to enable a prediction of the overall site runoff and receiving water quality. The total volume of site drainage has been estimated at an annual average flow of 33 L/s. For the baseline case evaluation of the sulphate reduction system performance, it will be assumed that a constant feed rate will be maintained equivalent to the annual average site drainage flow. The development and estimation of site drainage and water quality is discussed more comprehensively in Section 5. Receiving site water quality was modelled for zinc only. The weighted zinc concentration was found to be 26 mg/L.

Proportionally to the zinc concentration, water qualities from the Schedule D progress report, September 30, 1991, (Report number WH9105) provided by Curragh Resources Inc. were used for the estimation of the baseline water quality for the current evaluation. A list of water quality parameters were derived for the purpose of the current sulphate reduction system evaluation based on site data, and is summarised in Table 3.1.

It should be noted that pH is in the range of 5.0 to 6.0 which is suited to sulphate reducing bacteria. Also significant is the relatively low copper concentration anticipated for site runoff. Based on levels given in Table 3.1, toxicity to the sulphate reducing bacteria is not anticipated.

Site runoff water temperature was modelled based on weighted average flows. The annual mean and mean monthly temperatures are presented in Table 3.2.

TABLE 3.1

Current Weighted Site Runoff Water Quality Estimate

Parameter	Concentration (mg/L)
pH	6.0 - 5.0
SO_4^{2-}	3100
Cl	n/a
HCO_3^-	n/a
Total Hardness	240 (as $CaCO_3$)
Al	0.6
As	<0.05
Ba	0.005
Cd	0.02
Ca	570
Co	0.75
Cu	0.10
Fe	5.0
Pb	0.06
K	15.0
Li	0.25
Mg	350
Mo	0.01
Na	60.0
Ni	2.50
P	0.05
Zn	26.0

TABLE 3.2

Monthly Site Runoff Water Temperatures

Month	Temperature (°C)
Jan	4.4
Feb	4.7
Mar	4.7
Apr	4.7
May	4.2
Jun	5.3
Jul	7.3
Aug	5.1
Sep	4.0
Oct	3.1
Nov	4.5
Dec	4.2
Mean	4.7

3.2 Potential Reactors

The extreme weather conditions of the northern climate experienced at the Faro mine will require a controlled and protected environment for the implementation of a biological sulphate reduction process. The severe temperatures and the presence of permafrost dramatically reduces the effectiveness of a conventional wetlands water treatment system. In addition to the temperature requirements, strict anaerobic conditions will also be required for the sustenance of the sulphate reducing bacteria. Based on these requirements, potential reactors for the implementation of a sulphate reduction system have been considered.

On termination of mining activities at the Faro mine, it is intended to fill the Main Pit with tailings generated from ore mined at the Grum and Vangorda deposits and from reprocessing of the Down Valley tailings. Therefore, the Main Pit is eliminated as a potential reactor location.

Two other potential locations at the Faro mine site exist for the development of an *in situ* sulphate reduction system. These have been identified as:

- an underground mine, extending and accessed from the Main Pit, and;
- the Zone II pit.

3.2.1 Underground Mine

In the evaluation of the underground mine as a potential sulphate reduction reactor, a worst case scenario approach will be followed to demonstrate the applicability of the technology. Subsequently, the possible modifications that may be introduced to enhance performance, and the relative performance gain that may be achieved, will be discussed. Should the process prove to be successful at the conservative worst case conditions, process performance will be assured at improved operating conditions.

For the purpose of the present evaluation, it will be assumed that the water to be treated will be introduced via the ventilation shaft which accesses the northern section of the underground mine (as illustrated in Figure 5). Water will follow the course of least resistance to the exit, which will be via a cased drill hole to be provided at the most extreme point of the southern section of the mine. A single body of flow corresponding with the average ramp dimensions, has been evaluated. Assuming an average ramp width and height of approximately 4.87 m (16 feet), the available volume for the shortest flowpath through the mine is approximately 18 800 m^3. Therefore, assuming a direct flow through the workings, at a flow rate of 33 L/s, a minimum retention time of 160 hours is estimated. The total void of the adjacent room and pillar development is approximately 251,000m^3 of which it is estimated that 100,400m^3 may be available to the process through careful design and implementation of the system. If fully utilised, a retention time in excess of 900 hours may be possible at this flow rate.

Temperature

The temperature of stagnant water in the underground mine was measured at 15°C. This suggests that the rock face of the underground mine is at or close to this temperature, and should influence the temperature of the water flowing through the mine.

At a flow rate of 33 L/s, the heat requirement to increase the water temperature by 1°C is approximately 137 kW. To increase the temperature to 15°C, a total heat load of 1 370 kW is required, assuming an inflow water temperature of 5°C.

In considering the heat balance for the system, it is important to consider the Main Pit, which will be filled with water and tailings, and, in effect will represent a heat sink. This may have a negative influence

on the overall heat balance for the mine. An initial, worst case scenario estimate of the water temperature profile for the water passing through the underground mine was made based on the following assumptions:

- as a result of the relatively low flow rate, virtually no turbulence is expected for the water flow path. Based on an average ramp dimension of 4.87 m x 4.87 m (16x16 feet), the flow velocity will be approximately 0.0014 m/s. It is therefore assumed that the only heat transfer mechanism will be conduction (turbulent conditions will not exist at the anticipated flow);

- since the main pit wall is located above the mine, and will represent a heat sink resulting in a net heat loss from the roof of the mine, it is assumed that heat will be transferred from the floor of the ramp only. The contributions of the walls will be insignificant due to the room and pillar mining method;

- it is further assumed that, since the water will result in a decrease of the actual interface rock temperature, a constant rock temperature of 15°C will only be observed on average at a distance in excess of 6 times the mean ramp height below the mine floor (i.e. at a depth of 30 m below the channel floor).

- it is assumed that the inlet water temperature will be approximately 5°C.

Thermal conductivities for various materials are presented in Table 3.3.

TABLE 3.3

Thermal Conductivities of Various Materials (Perry and Chilton, 1973)

Material	k (W/(m°C))
Granite	1.73 - 3.46
Quartz	1.26
Dolomite	1.73
Magnesite	3.80
Sandstone	1.83
Water	0.57

An average rock thermal conductivity of 1.73 W/(m°C) was assumed for the purpose of this evaluation. The R-values for both the rock and water phases, based on a one second flow interval, were calculated where R is defined as:

$$R = x/(kA)$$

and x = length of conduction path;
k = thermal conductivity, and;
A = area of heat transfer surface.

Subsequently, the rate of the overall heat flux was calculated and assumed available for heating the water body. Accordingly, the fractional increase in the water temperature was calculated, and translated into a distance function. This process was repeated for each 0.25 °C increment in temperature, and the temperature increase as a function of the distance passed through the mine was determined.

The results of this evaluation are presented in Table 3.4 and illustrated in Figure 6.

TABLE 3.4

Conduction Heat Transfer - Water temperature and required pathway distance.

Water Temperature (°C)	Distance (m)
5.0	0
6.0	87
7.0	166
8.0	256
9.0	360
10.0	484
11.0	636
12.0	833

The shortest pathway length in the underground mine, measured from the extreme north to the extreme south, is approximately 790 m. The exit temperature of the water is therefore expected to be approximately 11.7°C.

The distance from the entry point at the ventilation shaft in the north section of the mine to the junction with the main entrance to the mine is approximately 180 m, and represents a retention time of approximately 30 to 35 hours. The temperature in this section is expected to rise to approximately 7°C.

This section of the mine is therefore not expected to maintain a significant rate of sulphate reduction as a result of the relatively low temperature.

The minimum water temperature for the south section will be 7 °C with a weighted average of approximately 9.95°C, and represents a retention time of approximately 130 hours.

At a longer retention time anticipated for operating under enhanced conditions, the initial increase in the temperature will be achieved more rapidly, along with an increase in the effluent temperature, resulting in an increased average temperature approaching 15°C. This will provide enhanced sulphate reduction and sulphide mineral formation kinetics.

Substrate Supply

Two modes of substrate supply are envisaged for the implementation of the process in the underground mine. Initially, to develop an active bacterial culture within the confines of the underground mine, it is anticipated that a readily available substrate will be placed in the anticipated reactor flow path prior to flooding the mine. If a solid substrate, such as hay is used, the substrate will provide additional surface area as attachment sites and reaction surfaces for the sulphate reducing bacteria.

Subsequently, substrates may be introduced into the system in the form of a slurry or a solution, depending on the type of substrate utilised, in combination with the drainage water to be treated. Should a solid substrate be utilised, it should be ensured that the inlet drill holes will be of sufficient diameter to allow passage of the substrate. It is anticipated that the substrate will be carried by the water flow to the anaerobic reaction zone.

3.2.2 Zone II Pit

Assuming a 30% porosity of the mined rock that has been placed in the Zone II pit, the hydraulic storage capacity is estimated at approximately 500 000 m^3 (See Figure 7 at elevation 1128). As such, the Zone II pit provides a significant storage capacity, and, should all of the site drainage be redirected into this location, it will provide a retention time of approximately 175 days. This retention in isolation would be ideally suited for a sulphate reduction application. However, there are other physical parameters that have to be considered for the implementation of sulphate reduction as an *in situ* process.

Temperature

The most recent data for the water temperature being pumped from the Zone II pit, provided by Curragh Resources Inc., are provided in Table 3.5.

TABLE 3.5

Zone II Pit Water Temperature

Date (1991)	Temperature (°C)
8 March	2.5
18 March	2.0
27 March	3.0
3 June	5.0
3 July	4.0
24 July	4.3

Preliminary modelling of the site runoff indicates that the overall water temperature for the combined flow from the site exceeds 5.0 °C only for approximately three months of the year, with a maximum of approximately 7°C in July. This suggests that the mean annual water temperature in the pit will remain at or less than 5°C. At this temperature, the activity of the sulphate reducing bacteria is expected to be minimal, and even at the extended retention time provided by the storage capacity, the extent of sulphate reduction will be very small.

Substrate Supply

Since the *in situ* sulphate reduction process, if implemented in the Zone II pit, would rely on the voidage within the waste rock for the establishment and maintenance of the bacterial population, it will be necessary to provide the organic carbon substrate to be utilised by the bacteria at the actual sites where the bacteria will be occurring. To achieve this, the carbon source will have to be supplied in a liquid form, such as water soluble molasses or a soluble sugar product.

Flow Path Management

For the successful operation of a sulphate reduction system in the Zone II pit it will be necessary to introduce the water to be treated in such a way as to avoid short circuiting. A primary requirement is that the water should pass through the anaerobic zone where the sulphate reducing bacteria are active. This would ensure sulphate reduction and metals removal through contact with hydrogen sulphide. This may require installation of an irrigation system to disperse water evenly over the catchment of the pit. Extraction of the water will require the installation of a cased drillhole to the deepest section of the pit to ensure the downward passage of the water through the anaerobic zone.

Exclusion of Oxygen

The water-table surface within the Zone II pit may be relatively exposed to oxygen entry as a result of the relatively high porosity of mine rock placed in the pit. It may be necessary to limit oxygen entry to preserve the anaerobic zone at depth.

In conclusion, because of the low water temperatures, difficulty of supplying substrates and nutrients, and due to potential aerobic conditions, the Zone II pit at present is not considered suitable for the implementation of a sulphate reduction system. Therefore, the current study will only address the applicability and performance of such a system in the underground mine.

3.3 Underground Mine Treatment Capacity

3.3.1 Sulphate Reduction Performance

Temperature

The evaluation of the water temperature as function of the passage through the mine as discussed in Section 2.1.1 has indicated that the North section of the mine will not support significantly active sulphate reducing bacteria populations, but incorporation of this section in the reactor system will contribute to the overall process as follows:

- The water temperature will be increased
- The water conditions will be stabilised, allowing anaerobic conditions to establish without impacting on the bacterial population, which is expected to be most active in the southern section of the mine.
- The north section may also serve as a precipitation site for any supersaturated minerals and salts or a settling area for any suspended solids which may be present in the inflow.

The temperature evaluation for the southern section of the mine indicated a weighted average temperature of 9.95°C. This temperature was established for a constant influent temperature of 5°C, assuming an inflow rate of 33 L/s (the estimated total runoff), and therefore presents a conservative estimate. It is likely that this temperature will be seasonally dependent, with the low end of the cycle equivalent to the analysis presented. It is expected that during the summer months, i.e. the high end of the cycle, the temperature will exceed that predicted by the analysis as a result of a higher influent temperature.

Conservatively, the evaluation of the sulphate reduction system performance will be based on the mean temperature and retention time estimated for the southern section of the underground mine, i.e 9.95°C, and a retention indicated by the maximum capacity of the system as indicated by the sulphate reduction rate. Although it may be possible to increase the actual through flow during the summer months, for the present evaluation a constant feed rate will be assumed for year round treatment.

Sulphate Reduction Rate

The sulphate reduction rate, suggested by Dr. Ronald Klusman of the Colorado School of Mines, at a temperature of 15 °C is 0.0288 gSO_4^{2-} /L/day, and at 10 °C is 0.0096 gSO_4^{2-} /L/day, which is representative of a wetlands situation. However, the rate determined experimentally by Reynolds *et al.* (1991) on hay extract was 0.413 gSO_4^{2-}/L/day at 18°C. At a Q_{10} value of 2 observed for the sulphate reducing bacteria (Maree 1989), the reduction rate expected at 9°C will be approximately 0.206 gSO_4^{2-} /L/day, which is approximately an order of magnitude greater than that suggested by Klusman at 15 °C for a wetlands application. However, it should be noted that the system proposed herein has several advantages over the conventional wetlands system. These advantages are as follows:

- The system is wholly contained in the underground mine, and the maintenance of anaerobic conditions will be significantly more efficient than that observed for a wetlands situation;

- The seasonal effect of temperature will be significantly less for the proposed system than for a wetlands system, and thus the lag phases in bacterial activity associated with fluctuations in temperature will essentially be eliminated.

At the Løkken mine an estimated naturally occurring sulphate reduction rate of 0.04 gSO_4^{2-}/L/day was observed for a temperature of approximately 15°C under extremely non-ideal conditions. This rate is more than four times greater than that suggested by Klusman. It can therefore be concluded that the rate suggested by Klusman, although applicable to wetlands, is not representative of the proposed system. Since the current system will be controlled, an improvement in the rate observed at the Løkken mine will be expected.

In the light of the advantages of the proposed system over and above a wetlands system and the comparable data from around the world, a sulphate reduction rate of 0.06 gSO_4^{2-}/L/day should be reasonable for the proposed water treatment system.

3.3.2 Volumetric Capacity

The volumetric treatment capacity of the sulphate reduction system will be dependent on the required metal removal efficiency. Zinc will be considered as the primary contaminant, and for the purposes of

this study, removal of this metal to the solubility level of the equivalent sulphide mineral will be considered as the primary requirement.

As discussed previously, metal sulphides are in general highly insoluble. The reactivity of the heavy metals with hydrogen sulphide varies with temperature, pH and the concentration and availability of the sulphide anion. While copper reacts readily with sulphide at a relatively low pH, a higher pH is required for the precipitation of zinc as a sulphide. The order of reactivity of various metals with hydrogen sulphide is as follows:

$$Cu > Cd > Pb > Zn > Ni$$

Thus copper will react before cadmium and cadmium before lead etc., therefore, not all the hydrogen sulphide generated will be available for reaction with zinc. Furthermore, the kinetics for the zinc reaction are relatively slow compared with copper.

It should be noted that the underground workings are inclined approximately 20° for the southern section, and 15° for the northern section, with the northern section elevated above the southern section (Pers. Comm. Nick Rose, Fox Geology). Since the water will be introduced in the northern section and extracted from the southern section, and considering the slow water velocity, any free hydrogen sulphide gas will rise countercurrent to the water flow path. This will improve sulphide contact with the metal bearing aqueous phase, effectively causing precipitation of the metals before entering the sulphate reduction zone. This will effectively reduce the retention time requirements for zinc precipitation in the biological zone of the system. (This will also result in a more even deposition of the metal bearing sulphide mineral sludge throughout the underground mine.) Therefore, for the current evaluation, the kinetic requirements for zinc sulphide mineralization are not considered as limiting in terms of the retention time requirements.

Based on the estimated sulphate reduction rate of 0.06 $gSO_4^{2-}/L/day$, and assuming only 12% of the sulphide generated will be available for reaction with zinc, sufficient hydrogen sulphide will be generated within a retention time of approximately 88 hours in the biological reaction zone. The relatively low hydrogen sulphide availability estimate of only 12% provides a conservative evaluation and considers:

- reaction with other metals in solution prior to zinc;
- losses as a result of removal in the effluent as hydrogen sulphide in solution, and;
- possible losses from the system through open drill holes.

This means that at a mean concentration of 26 mg Zn/L, the single flow channel system may have a potential to treat approximately 48 L/s. However, at a higher flow rate, a decrease in the temperature profile is expected for the current evaluation, which will result in a decrease of the reduction rate. Thus the actual capacity for the base case evaluation will likely be less than the 48 L/s, but more than 33 L/s.

Under enhanced operating conditions, the actual capacity of the system should significantly exceed that indicated for the base case evaluation as a result of increased reactor volume, thus retention time, resulting improved temperature profile

3.3.3 Substrate Requirement

The overall sulphate load to the system at an average concentration of 3100 mg/L and at a water flow rate of 33 L/s will be 3,226 metric tons per annum.

At the anticipated retention time of 130 hours in the sulphate reduction zone and at a projected sulphate reduction rate of 0.06 $gSO_4^{2-}/L/day$, the total annual mass sulphate reduced to hydrogen sulphide will be approximately 338.2 metric tons as sulphate, or a reduction in the sulphate concentration of 325 mg/L. The equivalent annual substrate requirement is summarised in Table 3.6.

TABLE 3.6

Anticipated Annual Substrate Requirement

Substrate	Requirement (t/y)	Cost ($/t)	Transport ($/t)	Total Cost ($ F.O.B Mine Site)
Sugar	235	655	325	230,300
Molasses	406	450	325	314,650
Hay	1 015	250	50	304,500
Pulp Mill Effluent	3 145	N/C	150	471,750
Raw Sewage	32 331	N/C	150	4,849,650
Mushroom Compost	12 513	N/A	N/A	N/A

In the case of hay, the estimated 1,015 tons per annum would probably be provided on a once per six month basis to the reactor system. At a cost of $300 per ton delivered to Faro, the operating cost for this substrate is estimated to be $304,500. In the case of the raw sewage, a 'treatment levy' may be charged to counterbalance the high transport costs however, it is not anticipated that this option will be economically feasible. In conclusion, the system is likely to be operated most cost effectively if sugar is used as a substrate.

In practice the amount of substrate supplied to the system will be dictated by reactor performance. Thus, if a high sulphide utilisation efficiency for the zinc removal is observed, the actual demand for sulphide production may be less, and will therefore reduce the substrate requirement.

3.3.4 Storage Capacity

Mineralised Sludge Generation

Sulphide minerals will be generated in proportion to the metals removed. In addition, as a result of the pH adjustment resulting from the alkalinity generated by the sulphate reduction process, some metal carbonates and hydroxides are expected to precipitate. Annual metal loads were estimated from the weighted average flow rate of 33 L/s and the water quality analysis provided in Table 3.7. Table 3.7 presents total loads of some of the metals that will contribute to the sludge accumulation. Also shown are the expected mineralization and the total annual mass and volume sludge generation expected from this source.

TABLE 3.7

Estimated Sludge Accumulation as a Result of Mineralisation

Element	Mineral	Load (t/y)	Density (t/m^3)	Volume (m^3/y)
Al	$Al(OH)_3$	1.80	2.3	0.78
Ca	$CaCO_3$	493.59	2.7	182.81
Fe	$FeCO_3$	10.79	3.7	2.92
Pb	PbS	0.72	7.4	0.10
Mn	$MnCO_3$	39.19	4.2	9.33
Ni	NiS	4.02	5.3	0.76
Zn	ZnS	40.30	3.9	10.33
			TOTAL	207.03

It should be noted that manganese may not be completely removed during the sulphate reduction process. Gypsum may be precipitated from the supersaturated solution and is likely to occur at the point of entry into the underground mine. However, at the current site water quality this occurrence is considered to be insignificant.

Organic Matter Accumulation

i) Hay. It is anticipated that, should hay be used as the primary organic substrate for the sulphate reduction system, a total annual hay requirement of 1 015 tons is expected. The mass loss, and thus the hay volume reduction should range from 60 to 80 %, depending on the degree of bacteriological consumption. At an average mass loss of 65 % the total sludge accumulation from this source is anticipated to be approximately 350 m^3 per annum. In comparison, the relative biomass accumulation is anticipated to be negligible.

Therefore, the combined sludge accumulation from these sources is estimated to be approximately 557 m^3 per annum. This estimate is dependent on the total organic substrate addition rate, and may be significantly lower than expected should the actual sulphide utilization for the mineralisation process be higher than that estimated. Under such conditions it may be possible to control the sulphate reduction rate by minimising the substrate addition rate, and thus the total sludge accumulation rate should be reduced.

However, for the current conservative single channel evaluation, the reduction in the available volume for the baseline evaluation will result in a decrease of the retention time by approximately 4.7 hours per annum. At this rate, no significant impact on the system performance is expected within the first 10 to 15 years of operation. Based on the total volume available to the process, no significant impact on the operating performance as a result of sludge accumulation is expected within the first 145 to 150 years.

ii) Sugar. Maree (1989) reports a total volatile suspended solids (VSS) or equivalent biomass accumulation in the sludge generated of 0.05gram per gram sulphate reduced while utilising molasses as a substrate. The organic sludge generation for sugar, which has a higher solubility, is expected to be less than that for molasses. Thus, a sludge generation rate of 0.05 should present a conservative estimate. The total organic sludge accumulation will therefore be approximately 16.91 m^3/y. The total sludge accumulation will only be approximately 225 m^3/y. Thus, for the baseline single channel flow evaluation, the sludge accumulation while utilising sugar as the primary carbon source will result in an annual retention time reduction of 1.9 hours. At this rate, no noticeable impact on the system is expected within the first 25 to 30 years. Based on the total anticipated volume available to the process, sludge accumulation should not impact on the process within the first 360 to 365 years, at the current metal and anticipated organic substrate load.

In conclusion, sugar presents the most cost effective and efficient substrate. The use of this substrate should result in minimal impact on the operating performance. Sugar may be transported as a solid, and since it is water soluble, no specialised equipment will be required to prepare it for introduction into the underground sulphate reduction reactor system. This will also allow the controlled addition of the substrate to effectively control the sulphate reduction rate. Similarly, significantly higher sulphate reduction rates may be possible, allowing for increased treatment capacity of the reactor system.

3.4 Mathematical Modelling

Dr. Klusman of the Colorado School of Mining has compiled a computer program for the evaluation of sulphate reduction and the associated interaction of metal species with the sulphide and alkalinity generated during the sulphate reduction process. He was retained to evaluate the application potential for the implementation of a sulphate reduction process at the Faro mine. Since the model developed by Dr. Klusman deals strictly with the chemical interactions of the metals in solution and the hydrogen sulphide generated, it is necessary to estimate the rate of sulphate reduction expected for the system.

Dr. Klusman evaluated the performance of sulphate reduction for the following conditions:

Retention time (hours)	130 and 200
Sulphate Reduction rate (gSO_4^{2-}/L/day)	0.0048, 0.0096 and 0.0288
Temperature equivalent for above rates (°C)	5, 10 and 15

Zinc removal and some water quality results are summarised in Table 3.8.

TABLE 3.8

Summary of the Computer Simulation Results

Component	Initial (mg/L)	RATE (gSO_4^{2-}/L/day)	
		0.0288	0.0096
Temperature (°C)		15	10
Retention time = 130 hours			
Ca	570	282	281
Ba	0.005	0.003	0.003
Cu	0.1	0.00	0.00
Fe	5.0	0.10	5.00
Mn	18.0	18.0	18.0
Cd	0.02	0.00	0.00
Pb	0.06	0.00	0.00
Zn	26.0	0.00	15.5
Ni	2.5	0.00	2.50
Al	0.6	0.013	0.014
SO_4^{2-}	3000	2265	2287
HCO_3^-	0.001	3.90	4.00
HS^-	10^{-18}	0.338	10^{-10}
pH	5.00	7.00	7.00
Retention Time = 200 hours			
Ca	570	282	281
Ba	0.005	0.003	0.003
Cu	0.1	0.00	0.00
Fe	5.0	0.016	5.00
Mn	18.0	18.0	18.0
Cd	0.02	0.00	0.00
Pb	0.06	0.00	0.00
Zn	26.0	0.00	9.83
Ni	2.5	0.00	2.50
Al	0.6	0.014	0.14
SO_4^{2-}	3000	2238	2286
HCO_3^-	0.001	3.90	3.99
HS^-	10^{-18}	9.90	10^{-10}
pH	5.00	7.00	7.00

At a retention time of 130 hours, the results show that a sulphate reduction rate in excess 0.0096 g SO_4^{2-} /L/day will be required to achieve complete removal of the zinc. At this reduction rate a total annual zinc removal of approximately 10.9 metric tons will be possible. At a sulphate reduction rate of 0.0288 gSO_4^{2-} /L/day complete removal of the zinc will be achieved. The high free sulphide (HS^-) concentration of 0.338 mg/L at this rate suggests that sulphide would be produced in excess of that required for metal removal. Therefore, complete zinc removal should be possible at sulphate reduction rates less than the 0.0288 gSO_4^{2-} /L/day suggested, or alternatively, at an operating temperature less than 15°C, as suggested by Klusman for a wetlands situation. However, as a result of the advantages presented by the system currently evaluated, a reduction rate of 0.06 gSO_4^{2-}/L/day is considered more appropriate. Complete zinc removal is therefore anticipated under the proposed operating conditions.

Similarly, if the retention time is extended to 200 hours for the wetlands situation, the results indicate that the lower reduction rate of 0.0096 gSO_4^{2-}/L/day will not be adequate to achieve complete zinc removal. A total of 16.8 tons will be removed annually, resulting in a discharge concentration of 9.83 mg/L for zinc.

Dr. Klusman conducted an additional computer run to confirm the complete removal of zinc at the anticipated sulphate reduction rate. The base case operating parameters of 130 hour retention and a sulphate reduction rate of 0.06 gSO_4^{2-}/L/day were evaluated. The results are summarised in Table 3.9.

The results obtained for the base case indicates that, at the anticipated sulphate reduction rate, all of the zinc and other heavy metals will essentially be removed. Excess free hydrogen sulphide is observed, which suggests that the sulphate reduction rate may be controlled at a lower level, by limiting the substrate addition rate, without affecting the process performance. Thus, substrate addition may be less than that anticipated in Table 3.6. During the test run, a carbon dioxide partial pressure of 0.1 atm. was maintained, to simulate the retention of this gas in the enclosed conditions of the underground mine. At these conditions most of the manganese was removed. Under the increased retention time for the fully implemented system, complete removal of manganese is expected.

TABLE 3.9

Computer Modelling Results at a Sulphate Reduction Rate of 0.06 gSO_4^{2-}/L/day and a Retention Time of 130 Hours.

Component	Initial (mg/L)	Final (mg/L)
Ca	570	282
Ba	0.005	0.0003
Cu	0.10	0.00
Fe	5.00	0.025
Mn	18.0	2.1
Cd	0.02	0.00
Pb	0.06	0.00
Zn	26.0	0.00
Ni	2.50	0.00
Al	0.60	0.00
SO^{2-}_4	3000	2211
HCO^{2-}_3	0.001	1236
HS-	10^{-18}	19.7
pH	5.00	7.00
pCO_2		$10^{-1.00}$

3.5 Process Enhancement Alternatives

Several process enhancement alternatives exist that may be pursued to improve the overall anticipated sulphate reduction rates and equivalent metal removal. These are summarised below.

3.5.1 Retention Time

Increasing the system retention time will greatly enhance the overall system performance and will provide a degree of assurance in terms of quality control of the effluent product. Deposition and accumulation of sludges will impact on the overall retention time, and thus on the process performance. It will therefore be prudent to enhance the process retention time to minimise the impact of retention time reduction. An increase in the retention time may be achieved through various means.

Multiple Borehole Access and Discharge Points

It is proposed that multiple inflow wells of up to three units should be installed in addition to the existing ventilation well. These drill holes will access the northern section of the mine at three separate locations to improve the dispersion of the influent water to be treated.

It is proposed that drill holes be installed at the following points of entry to the northern section of the mine (See Figure 7):

1 DHN1 J-700
2 DHN2 NB-725
3 DHN3 NF RAMP

The anticipated increase in retention times for an evaluation at a feed rate of 33 L/s, for utilising various combinations of the access wells, are summarised in Table 3.10.

TABLE 3.10

Access Drill Hole Retention Time Increment

Number (#)	Combination	Retention Gain (hours)	Overall Retention (hours)
2	VS + 1	25	185
2	VS + 2	51	211
2	VS + 3	44	204
4	VS+1+2+3	122	282

The advantage presented by multiple entry points are therefore clearly evident in the retention time gain that is accomplished.

However, it is recognised that the successful operation of the system will be dependent on the continued availability of the main ramp joining the northern and southern sections of the mine. It is therefore considered prudent that a second set of inlet portals be installed in the extreme north of the southern mine section, in the event that this passage is blocked as a result of a rock fall. The potential locations for these drill holes are:

1 DHM1 Intersect of SF 100 and SF-200
2 DHM2 Intersect of SE and SD RAMPS
3 DHM3 Intersect of SB RAMP and SB-200

If operated in parallel, feeding equivalent volumes to each location at a rate of 11 L/s, for a total of 33 L/s, this configuration should provide for a retention time of in excess of 160 hours. Due to the greater mining activities in the southern section, it is not possible to accurately define the potential flowpaths, and thus an accurate estimate of the retention time is not possible at this stage. Tracer tests should be conducted after the mine has been filled to determine the actual retention time.

To aid in diverting the water flowpath from a single to multiple routes through the southern section of the mine, it is recommended that multiple extraction points be installed. This will effectively increase the retention time and result in the full utilization of the mine voidage available for the process. The following locations are recommended for the installation of the discharge drill holes (See Figure 7):

1 DHS1 In proposed mining area
2 DHS2 Intersection of SN RAMP and SN 100
3 DHS3 SL 300

Internal Baffles

Since the water flowpath will be dictated by the route of least resistance, i.e. the most direct route between the inflow and discharge, it will be possible to improve the overall mine volume utilisation by redirecting the flow path. This may be achieved with the installation of internal baffles. However, since the internal baffle must be permanent structures to perform effectively, the risk of plugging a potential flowpath, should an existing flowpath be blocked as a result of a rock fall elsewhere, is great.

Conceptually, baffles may be installed with a pipe section extending through the baffle. The pipe section may be suitably sized to allow transfer of the total fluid load through the baffle, should the alternative passage be blocked, but provide sufficient head loss to redirect the majority of the flow to the alternative route under normal operating conditions. In order to maximise the potential benefit that may be gained. the locations of these internal baffles should be evaluated upon completion of mining, when the final layout of the underground mine has been established.

3.6 Conceptual Process Description and Operation

3.6.1 Process Description

A conceptual process flow diagram is illustrated in Figure 9. Based on the obvious advantages of installing multiple inlet and outlet wells, it is assumed that drill holes will be installed at all the locations (9) described in Section 3.5. Water transfer pumps will be installed at each of the three retention structure locations. The capacity of each pump will be 33 L/s, and are sized at approximately 75 Hp (57kW) with

a head of 325 feet (99 m). Pipes (203 mm diameter (8" Sc40) will be installed to transfer water from each of the locations to the inlet wells of the underground mine.

Sugar, considered as the primary organic substrate source, will be introduced into the sulphate reduction system as a liquified stream. The feed system will consist of a feed hopper, equipped with a screw feeder which will transfer the substrate into a mixing tank, where it will be liquified and pumped to the sulphate reduction system feed line. An inline mixer will be used to disperse the substrate in the feed, prior to injection into the sulphate reduction system.

It is anticipated that a trace inorganic nutrient feed system will be required. This system will consist of an agitated solution holding tank, and a reagent dosing pump. Similar to the organic substrate, the inorganic substrate will be dispersed within the feed to the sulphate reduction system in the inline mixer.

The discharge from the outlet of the system will be pumped to the Faro Creek discharge from the Main Pit. It is anticipated that a control system will be installed, complete with level detection devices for each of the water storage units. Furthermore, a control valve system and control strategy will be developed to best utilise the capacity of the underground mine reactor volume.

3.6.2 System Operation

Similar to the substrate feed system, trace nutrient addition will be continuous to ensure availability at all times. This will prevent bacterial growth limitation to the extent where process performance does not comply with discharge criteria, as a result of uncontrolled substrate inhibition.

Initially, it is recommended that the system be monitored (complete inlet and outlet flow rates and water quality) on a biweekly basis to determine total loads and process performance until satisfactory steady state operating conditions have been established. Subsequently, the sampling frequency may be reduced to quarterly or semi-annually.

4.0 UNDERGROUND MINE STABILITY

The rock mechanics portion of this study is restricted to strictly physical aspects of the project. These aspects are:

- underground stability
- slope stability above the underground workings
- water inlet/outlet drillhole location
- bulkhead design for underground closure

Each of these topics will be discussed in the following sections.

4.1 Underground Stability

A room and pillar mine, if properly designed, is always skirting the stable/unstable boundary. This for the simple reason that the optimum extraction, and thus optimum profit, is returned as the maximum amount of material is extracted. Therefore, any changes in the stress field due to physical changes in the environment will cause local instabilities.

In terms of safety, this mine has had the stable/unstable boundary addressed by using a safety factor of 1.5 for the pillar design. This factor of safety is not used for the intervening roof spans between the pillars. It is the stability of the rooms underneath these spans, not the pillars, which is critical to the success of the project. Therefore, the question to be addressed is whether sufficient volume and permeability will remain in the mine, over time, after roof failure.

4.1.1 Short-term stability

The short term stability is here referred to as that period of time immediately before, during and after flooding of the underground workings.

When final mining has been completed on a retreat basis from the rooms in the mines, many areas, especially the roof, will be very near limiting equilibrium. It will, however, be stable. As the mine is flooded, the sudden rise in pore pressure in the rock as the water rises will promote local instability. Local falls will be more prominent in areas of weak rock mass strength, i.e. the white mica schist, than in areas of high rock mass strength, i.e. massive sulphides. Areas which are very marginally stable upon mine closure will collapse during the filling period.

Once the mine has been filled with water, stability will improve over what it was upon closure. This is due to the buoyant force of the water acting to internally support the openings. Thus, if an opening

survives the filling phase of the project without having failure being initiated, it is relatively certain that the opening will remain as long as the local support upon which stability depends is maintained.

Most failures in the short-term should be relatively small and not promote large scale, caving type, failures. Areas which are most prone to this type of behaviour are areas where faulted and sheared materials have been encountered. As such areas have acted as effective limits for mining in the orebody, only small areas of this material are exposed, generally at the end of a drift.

Failures, even if they are located in strong, non-argillaceous rocks, will not be extremely permeable. Given the restricted nature of tunnels through which the water must pass, any ground fall will reduce the amount of water which may flow through an area at a given time. Given the platy nature of the host rocks and the relatively plastic nature of the white mica schists (with included clays), a relatively impermeable rock fall may result. However, as the falls will probably be local, non-progressive events, there should exist a gap over the top of the fall which would allow water passage.

4.1.2 Long-term stability

As the mine grows older, the artificial support which was installed in the rock mass during mining will decay. This support consists essentially of friction bolts (split-sets), some mechanically anchored bolts, steel strap, and wire mesh.

The wire mesh will dissolve very rapidly, especially in an acidic environment. This will allow the small scale, ravelling type failures, which it was installed to prevent. Friction bolts, being thin-walled, high carbon steel tubes, will also disappear rapidly, likely in less than 5 years. Their disappearance will allow the rock mass above the openings to behave as independent blocks instead of a network of blocks linked with bolts. This could cause relatively major rock falls, even, conceivably, caving action. The steel straps, which are installed to hold block ground in place, will also fall, as they depend on the friction bolts for anchorage to the rock. Therefore, the potential for major falls exist.

However, the water acts as an offsetting force to the destabilization due to support dissolution. The buoyant forces generated by the water will tend to reduce the load applied to the joint surfaces, bedding planes, etc., which allow the independent rock blocks to move. This reduction in shear stresses means that the opening is inherently more stable than before being filled with water. Therefore, the effective safe span, that is, the span which will stand without artificial supports, is increased. This means that most areas of small span (15-25 feet) may well survive relatively unscathed. However, it does not mean that all areas of small span will remain intact. Support is installed for a reason. If it is missing, and it is required, large falls may occur which can block the openings.

In time, after the artificial support has disappeared, the mine will either reach a point of equilibrium, where no further falls occur, or progressive falls will occur which will totally fill the underground operation. At this point it is difficult to say with certainty which will occur. However, given the data which is available at present, without detailed numerical analysis of all possible scenarios, it is assumed that the mine will provide a network of water travel paths for many years into the future.

The ramps which connect the two mining areas are of critical importance to the project. A continuous flow of water will pass through these with any groundfalls hindering this flow. Minor ground falls will likely occur in these ramps as the artificial supports (split-sets) corrode. In order to insure a pathway between the North and South orebodies, these ramps must be filled with a permeable backfill. Ideally, this would be a fill of 4"-6" stream washed cobbles. This would hinder collapse while allowing fluid flow. It will, however, reduce the overall available space available in the mine. It may also tend to blind off (in time) if particles become lodged in the spaces between the cobbles. If that occurs, the amount of fluid which can pass through the ramps may drop below that which is required of the system. In that case, injection wells in the South mining area would be required to supplement the flow.

4.2 Slope Stability

Slope stability will be affected as the pit floods. This is due to the destabilizing effect of the increase in pore pressures as the water rises along the pit walls. Increasing pore pressures can activate old, inactive failures or contribute to causing new failures. As such, sloughing will occur during flooding and slope failures are a possibility. Once the pit has been filled, wall stability should be enhanced over what it is today.

Slope stability impacts on this project as two distinctly different problems. One is the impact of a failure upon pit flooding cutting into, and exposing the underground workings. This is especially possible in the South K area of the workings where stopes are planned to be mined very near the pit wall. The second problem is that of local instability (sloughing) during flooding which will endanger pipelines which feed the water into the underground workings. This is a possibility in the area of the main portal tunnel as the portal tunnel is a lined opening through waste rock. As such its stability must be checked for piezometric conditions that will occur during pit flooding.

Neither scenario has been studied in great detail. The pit slope in the specific area should be examined in detail if this project proceeds past the feasibility stage. Specific locations for pipelines as well as alternative solutions in case of damage due to failure should be included in the detailed design.

4.3 Drill hole Locations

Proposed inflow/outflow holes were located in such a manner as to pierce the underground workings in areas of relative stability. They were also located so that, wherever possible, they have multiple travel paths in various directions. This would assure water passage even if failure occurred in a local area.

4.4 Conclusions and Recommendations

It must be kept in mind that the mine is situated in a natural material. As such, the comments which can be made, and the analyses that can be conducted, are only approximate and will be limited by the constraints nature has provided. Predicting long-term behaviour in rock is difficult. It can be stated with certainty that some failure will occur in the mine over time. Eventually, perhaps tens, hundreds, or thousands of years in the future, the mine will have totally collapsed. The relevant questions to ask at this point is what is required of the mine for this project. Two items come to mind. These are the maintenance of sufficient volume and permeability for the project to be viable long into the future and the prevention of breaches in the system which will allow fluid leakage to, and from, the pit.

There are several areas which require further study. These are:

- Location of argillic/white mica schist areas above active mining areas;
- Mapping of areas of intense support in the underground workings;
- Slope stability analysis of the pit wall above the mining area;
- Stability analysis of the rock fill around the portal tunnel;
- Detailed design of bulkheads, especially for the main portal;
- Evaluation of support methods, efficiency, and requirements for main access ramps.

The location of the rock falls, and type of material within the fall can be better defined. This can be done by using material available to the mine geologists at present. The result would be a better understanding of the areas of instability and how these falls would affect flow paths.

Slope stability of the area must be undertaken before any construction is contemplated. This analysis should include both the rock fill for the portal area as well as the slopes above the North and South mining areas as they will be left upon abandonment. Areas which are likely pathways for downlines to

injection wells should also be examined. Although the slope appears stable, appearances must be checked by analysis before it is acceptable from an engineering standpoint.

The main portal tunnel should be examined to determine which type of bulkhead head is most feasible and a detailed design conducted for this bulkhead. Support requirements for the connecting ramps between the North and South orebodies must be defined. This will be from a system requirements, practicality, and economics viewpoint. At present, these guidelines do not exist.

5.0 WATER MANAGEMENT SYSTEM AND QUALITY

5.1 Conceptual Water Management Plan

5.1.1 Introduction

The water management plan associated with the SRB treatment system should have the following three objectives:

- divert clean runoff to minimize the volume of contaminated water;
- collect contaminated drainage; and,
- provide storage to regulate flow of contaminated water to the SRB treatment system, and to maximize the volume of water treated.

Figure 10 shows the location of facilities for the conceptual water management plan.

The proposed plan for diversion of clean runoff is discussed in Section 5.1.2. Section 5.1.3 describes the facilities associated with the proposed drainage collection systems. Section 5.1.4 provides an estimate of flows for each drainage collection system, and section 5.1.5 provides zinc load estimates. Finally, section 5.1.6 details proposed operating rules for the conceptual water management plan.

5.1.2 Runoff Diversions

The first objective of the conceptual water management plan can, to a large extent, be satisfied with the system of diversion and interceptor ditches already in place. Some modifications will be made to accommodate the decommissioning plan for the Main Pit and to minimize the quantity of leakage through ditches. The following five diversions would be required to control runoff generated by the catchments above the rock piles:

- Faro Creek Diversion Channel;
- Faro Valley Interceptor Ditch;
- Northeast Interceptor Ditch;
- Northwest Interceptor Ditch; and,
- Main Pit Spillway Channel.

The role of the first diversion would be to minimize clean water passing through, and thus becoming contaminated by, the Faro Valley Rock Pile. This diversion would be created by breaching the existing

Faro Creek Diversion Channel at approximately its midway point to allow flows to enter the Main Pit. This re-direction of Faro Creek has been identified as an important measure in maintaining a permanent water cover over the tailings deposited in the Main Pit. Secondly, an impervious liner would be installed from the diversion inlet to the pit discharge point. The existing Faro Creek Diversion Channel leaks. Roughly two-thirds of the flow currently passing through the Faro Valley Rock Pile (or an annual average flow rate of about 20 L/s) can be attributed to leakage from the Faro Creek Diversion Channel. With the volume of leakage being of the same order as the estimated capacity of the SRB treatment system, it is evident that this diversion must be made practically watertight.

The second diversion already exists and, like above, would be used to control the amount of water contacting the Faro Valley Rock Pile. Leakage from the Faro Valley Interceptor Ditch would be much less than presently obtained from the Faro Creek Diversion Channel and, therefore, the need to provide an impervious liner is less clear. An economic analysis could be conducted to balance the cost of installing a liner against the potential savings in treatment costs with reduced leakage.

The lower half of the existing Faro Creek Diversion (renamed Northeast Interceptor Ditch), or that portion that would no longer be required to convey flows on mine closure, could be salvaged to create the third diversion. The diversion would serve to reduce the amount of fresh water passing through the Northeast Rock Piles.

The fourth diversion, the Northwest Interceptor Ditch, would be used to collect clean runoff from a small catchment above the Northwest Rock Piles and divert this flow to Next Creek. No excavation presently exists at the proposed location of the ditch.

The purpose of the final diversion would be to prevent overflow from the Main Pit becoming a significant component of the water passing through the rock piles south of the pit. The spillway overflow section would be located along the southwestern wall of the pit at the entrance of the main haul road. The spillway channel would run in roughly a westward direction from the overflow section to discharge into Next Creek. Spill from the pit will have to be conveyed over the rock piles in a watertight channel. A relatively small leakage from the spillway channel could dramatically increase the volume of water flowing through the rock piles and thus the amount that would have to be treated. In addition, the spillway channel must be designed in such a manner as not to intercept waste rock drainage along its route.

5.1.3 Drainage Collection Systems

The rock piles at the Faro Mine cover a planimetric area of about 3.6 km^2. Collection systems were visualized to collect drainage from almost this entire area. One exception was the portion of the Northwest Rock Piles that drain to Next Creek. Water quality monitoring on Next Creek indicates that

the zinc loading to this stream is relatively small. Future investigations may identify drainage from other portions of the rock piles that also do not warrant collection.

By examination of the mine site topography and the locations of existing mine facilities, such as open pits and diversions, five systems were proposed for the collection of rock pile drainage. These are listed below and have been named after the outlet point for each system:

- Sump A30 System
- Zone II Pit System
- Sidehill Dam System
- Sump 1 System
- Sump 2 System

Figure 11 shows the approximate boundary of the sub-catchment area that would be controlled by each of the five collection systems. The planimetered sub-catchment areas are provided in Table 5.1. Sub-catchment boundaries were derived by superimposing the outlines of rock piles, open pits and diversions on a pre-mining topographical map. Boundaries were drawn assuming that the natural ground underlying the rock piles is the dominant influence on drainage patterns. Surface topography of the rock piles was ignored for this purpose based on the following two assumptions:

- infiltration capacity of the rock pile surface is probably high (i.e., little overland flow occurs)
- drainage through the rock piles would probably be essentially vertical

TABLE 5.1

CHARACTERISTICS OF SUB-CATCHMENTS COMMANDED BY DRAINAGE COLLECTION SYSTEMS

Sub-catchment I.D. (see Fig. 11)	Sub-catchment Outlet	Drainage Area (km^2)	Median Elevation (m)	Mean Annual Runoff (mm)
A	Sump A30	0.57	1300	270
B	Zone II Pit	0.56	1190	250
C	Sidehill Dam	1.94	1170	240
D	Sump 1	0.52	1140	230
E	Sump 2	0.64	1170	240

The Sump A30 System could remain inplace to collect drainage from the Faro Valley Rock Pile. Sump A30 already exists as an integral part of the pit dewatering program during mine operation. This sump is strategically located to intercept a large proportion of the drainage emanating from the Faro Valley Rock Pile. To complete the development of this drainage collection system, a pipeline would be constructed to convey drainage collected in Sump A30 by gravity to the Zone II Pit for storage. Inflows to Sump A30 in excess of the pipeline capacity would overflow to the lake in the Main Pit. The rationale for diverting the flows to the Zone II Pit rather than directly to the SRB treatment system is described in Section 5.1.6. Instability of the northeast pit wall may negate the possibility of running a gravity line from Sump A30 to the Zone II Pit. In this event, flows could be delivered to the Zone II Pit via a pipeline routed above the pit crest. Pumping would, therefore, be required.

The second system comprises the Zone II Pit and the Collector 1 Ditch. The Zone II Pit would be used for storage to attenuate seasonal flows to the SRB treatment system. The interstices of the broken rock that fills this pit will provide a large storage volume. The total capacity for water within the Zone II Pit was estimated to be 0.5 million cubic metres, assuming an average rock porosity of 30%. The Collector 1 Ditch would divert rock pile seepage to the Zone II Pit that would otherwise drain to the Main Pit. This diversion could be created by reinstating an old network of ditches known as the Blue Line Ditches (this network was originally constructed to minimize inflows to the Main Pit and thus reduce pit dewatering costs. Use of the Blue Line Ditches was later discontinued due to water quality considerations in the North Fork of Rose Creek). The implementation of this system would include breaching a diversion known as the Zone II Interceptor Ditch. This diversion controls the drainage from the rock piles to the north of the Zone II Pit.

The Sidehill Dam System commands the largest sub-catchment area and would collect drainage from the Main and Northwest Rock Piles. The system comprises a storage facility and two collector ditches, none of which presently exist. The Sidehill Dam would be located in the old channel of Faro Creek near the toe of the Main Rock Pile. The proposed operating rule, described in Section 5.1.6, was designed to minimize the size of this facility. Much of the water infiltrating through the Main and Northwest Rock Piles would be directed to the old Faro Creek channel and hence to the Sidehill Dam. The two collection ditches, designated No. 6 and 7, would probably capture only a minor amount of the total seepage reporting to the Sidehill Dam.

Drainage collection from the Southeast Rock Piles would be implemented by constructing Sump 1 and two short collector ditches (Nos. 4 and 5). The sump would be located such that it intersects a ravine in the natural ground underlying the rock piles. With this positioning, most of the drainage from the Southeast Rock Piles will discharge directly to Sump 1. The two collector ditches would only marginally increase the inflows to the sump. Sump 1 would also be constructed in the floodplain of the North Fork and, accordingly, would require protection with a dyke.

The role of the Sump 2 system would be to intercept drainage from the East Rock Pile. Examination of topographical maps indicated that a sump would likely be a more cost effective method of providing storage for this system compared to a dam. A potential location for the sump would be on the floodplain immediately opposite the Zone II Pit. A dyke or other form of protection, would have to be constructed to prevent flood waters from the stream inundating the sump. The sump would be fed by two collector ditches (Nos. 2 and 3) skirting the toe of the rock piles. As with the Sidehill Dam, the proposed operating rule was designed to minimize the required storage of Sump 2.

It should be noted that the drainage to be collected by the Sump 1 and Sump 2 Systems currently drains to the North Fork of Rose Creek. These two systems would not have to be constructed unless significant zinc loadings develop. An alternative would be to intercept only selected seeps of poor quality.

5.1.4 Flow Estimates

The drainage collection systems would receive flows from three sources, namely runoff from the catchment areas commanded by the collection systems, seepage from the flooded Main Pit, and ditch leakage from the five diversions. Flow estimates for each of these sources are presented below.

Catchment Runoff

Runoff estimates were made by transposing the flows of regional streamflow gauging stations to the mine site, the assumption being that evaporation from the rock piles is roughly equal to the actual evapotranspiration from the natural ground surface. The technique used to transpose regional streamflow data to the minesite was described in Report 60635 (SRK, April 1991), the decommissioning plan for the Down Valley Tailings Impoundment. A brief description of the technique follows.

Runoff generally increases with increasing elevation. This trend was quantified for the area around the minesite using data from 13 regional streamflow gauging stations of the Water Survey of Canada and of the Department of Indian Affairs and Northern Development. The statistic used to represent runoff was mean annual runoff (MAR) expressed in millimetres (i.e., the depth to which the average annual runoff volume would spread uniformly over the contributing catchment). Elevation was represented by a parameter known as catchment median elevation (ME) expressed in metres (i.e., the contour that divides the contributing catchment area in half). The following linear relationship was interpreted from these data to best represent conditions at the minesite:

$$MAR = 0.273\ (ME) - 80$$

Application of this relationship to the catchments commanded by the five drainage collection systems yields the values listed in Table 5.1. Estimates of MAR range from 230 to 270 mm.

The above estimates of MAR were checked against a record of observed flows at water quality monitoring station X23. This station is located in the old channel of Faro Creek below the toe of the Main Rock Pile and monitors the largest seep emanating from the rock piles. Spot measurements of flow are made at this station on roughly a monthly basis. Over the period of 1986 to 1990, the flow at X23 has averaged about 4 L/s. Given an effective catchment area of about 0.85 km^2, the flow record at X23 indicates a unit MAR for the rock piles of about 150 mm. This is only about 63% of the value obtained using regional analysis. The following three hypotheses have been offered to explain the discrepancy:

- the derived regional relationship between unit MAR and catchment median elevation may not be accurate for low relief catchments in the Valley of Rose Creek (all streamflow gauging stations used to derive this relationship have high relief catchments);

- the rock piles experience a greater evaporation rate than the original land surface (the surface area of the mine rock within the rock piles is large and may act as a large interception storage. The high porosity of the broken rock may allow significant currants of air to pass through the rock piles. The coupled influence of these two factors could serve to enhance evaporation above the level experienced during natural conditions);

- the sporadic record of flows measurements at X23 may underestimate total flow volume. Specifically, peak flows may not have been recorded at X23.

Insufficient data are available to accept or discredit any of the above hypotheses. Accurate determination of the unit MAR is of prime importance in assessing the effectiveness of the SRB treatment system. Resolution of the abovementioned discrepancy could be made by increasing the monitoring frequency at X23. As a conservative measure, the unit MAR values determined by regional analysis have been adopted for this study. If the existing X23 flow records prove to be correct, a significant reduction in seepage volume for treatment will result.

In addition to characterizing the MAR, runoff during wet years was also estimated. The streamflow gauging stations in the region with the three smallest catchment areas were selected for this purpose. A series of annual runoff volumes were extracted from each station's record and fitted to a theoretical frequency distribution to provide estimates of extreme wet events having return periods of 25 and 100 years. To facilitate extrapolation of this information to the mine site, the extreme events were normalized, or expressed as a percentage of MAR. The highest normalized value predicted for the three regional stations was adopted to represent conditions at the minesite. The total runoff volume during a 25-year event was estimated at 143% of that experienced during an average year. This factor increases to 163% for a wet year with a return period of 100 years.

Seepage from Main Pit

Seepage from the Main Pit would be a component of the flows handled by two of the drainage collection systems, namely the Zone II Pit and the Sidehill Dam. Seepage estimates were provided in Appendix A of Kilborn Report 350928 (SRK, June 1991), a water quality assessment for tailings disposal in the Main Pit. Seepage to the Zone II Pit would be dependent on the adopted design for the proposed plug dam located at the high point of a haulroad between the Main and Zone II Pits. For the purpose of the conceptual water management plan, construction of plug dams, with slimes in between, was assumed. This approach should minimize seepage losses to the Zone II Pits,with an estimated seepage rate of about 0.6 L/s. The location of the Main Pit Spillway will also be prone to seepage from the pit via unconsolidated sediments. This potential would be limited during design and construction of the spillway. For the purpose of this assessment, seepage from the pit, which would ultimately enter the Sidehill Dam, was estimated to be about 0.1 L/s.

Ditch Leakage

Leakage from either the Faro Creek Diversion or the Main Pit Spillway would significantly increase the volume of water passing through the rock piles. These two diversions will have to be made as watertight as is practicable. The other three diversions command small catchments and, therefore, leakage is less of a concern. For the purpose of the conceptual water management plan, ditch leakage from all five runoff diversions was assumed to be zero (i.e., all diversions were assumed to be impervious and excavated to a sufficient depth to intercept the interflow component of runoff).

Total Flow

From the above information, the five drainage collection systems were estimated to collect an average annual volume of 1.0×10^6 m^3. This represents an annual average flow rate of about 33 L/s. The annual volume of drainage increases to 1.5×10^6 m^3 (47 L/s) and 1.7×10^6 m^3 (54 L/s) during wet years with return periods of 25 and 100 years, respectively.

5.1.5 Zinc Load Estimates

Zinc loads currently released by the Faro rock piles were estimated from the extensive chemical data base maintained by Curragh Resources Inc. Reference was made to both seep survey and receiving water quality monitoring stations. Application of these data in estimating present-day zinc loads is described below for each of the drainage collection systems. The estimated zinc loads comprise both particulate and dissolved constituents.

Sump A30

The quantity and quality of flow pumped from Sump A30 is monitored at Station X22A. Grab samples are taken on a monthly basis. Flow measurements are somewhat less frequent. Interpretation of these data for the period 1987-1990 generated an average zinc load for the sump of 6.6 tonnes per annum.

Lining of the Faro Creek Diversion and reducing leakage could potentially serve to reduce the zinc loading derived from the Faro Valley Rock Pile. The reduction of leakage would cause a corresponding reduction in flows flushing load from the rock pile. However, no adjustment was made to the estimate above to allow for leakage reduction.

Zone II Pit

The quality of water pumped from the Zone II Pit during 1991 was used to estimate the current level of zinc loading from this source. Four grab samples taken between March and June exhibited zinc concentrations ranging from 47.8 to 60.4 mg/L, with an average of 54 mg/L. The only significant source of inflow to the Zone II Pit during mine operations is assumed to be infiltrating precipitation due to dewatering in the Main Pit. Annual infiltration was estimated to be 240 mm, or 65% of total precipitation. For a pit catchment area of 0.33 km^2, the current average inflow was calculated at 2.5 L/s. The product of the concentration and flow values provided an estimate of the average annual zinc load equal to 4.3 tonnes.

Implementation of the conceptual water management plan will increase the pit catchment area by 0.23km^2. Rock piles located on this additional area will also be sources of zinc to the Zone II Pit. Reference was made to the seep survey network to characterize the average zinc concentration emanating from these rock piles. Four main seeps in the area (Stations 5-8) show an average concentration of about 4 mg/L. Again, assuming that, infiltrating precipitation is the only source of inflow, the additional zinc loading from the expanded catchment area was assessed at 0.2 tonnes per annum.

The short term average annual load to the Zone II Pit collection system was assumed to equal the sum of these two sources, or 4.5 tonnes. This figure excludes contributions from other collection systems diverted or pumped to the Zone II Pit for storage, and from seepage from the Main Pit after closure. The increased inflows to the Zone II Pit may increase the flushing of pit walls and broken rock contained in the pit. This may accelerate the leaching of zinc, and hence the 4.5 tonne estimate may have to be increased.

Sidehill Dam

The largest seep that will enter the Sidehill Dam is monitored at Station X23, located in the old Faro Creek channel near the toe of the Main Rock Pile. The water quality measured at this station was assumed to be representative of all inflows to the Sidehill Dam.

An average monthly inflow hydrograph was estimated for this dam using information contained in Report 60635. Associated average monthly zinc concentrations were derived from the weekly measurements of water quality at X23 over the period 1986-1990. The product of flow and concentration was calculated for each month. Summation of the resulting 12 monthly load values indicated that the average annual zinc loading to the Sidehill Dam will be about 11.6 tonnes. However, the water quality measured at Station X23 may not be a true reflection of the drainage emanating from the rock piles. A component of the flow at Station X23 is thought to originate from dewatering of the Main Pit. The channel used to convey this water is leaky, allowing water to infiltrate through the rock piles to the old Faro Creek channel and hence report to Station X23. The mine water has high zinc concentrations that would bias the concentrations at Station X23. If indeed a significant proportion of pit water is reporting, the estimated loading from the rock piles draining to the Sidehill Dam should be reduced. A further load reduction should also occur either by adjustment of the MAR for the catchment to a lower figure, or by a reduction of the average zinc concentration as a consequence of more dilute flow peaks not being recorded at X23. The 11.6 tonne load estimate should therefore be considered very conservative.

Sumps 1 and 2

Almost all of the rock pile drainage that would be handled by these two collection systems currently drains to the North Fork of Rose Creek. Curragh Resources Inc. operate two receiving water quality stations on the stream in this area. Station R7 is located upstream and, therefore, can be used to estimate background loads. Station X2 is located immediately below the inflows of rock pile drainage. The difference in loads at these two stations has been attributed to the zinc contribution from the rock piles.

Water quality monitoring is conducted once a month at Station X2, and once every three months at Station R7. No long-term records of measured flow are available for this stream to facilitate the calculation of loads. A technique for synthesizing flows at the two stations was adopted involving the extrapolation of the flows of gauged streams to flows in the North Fork of Rose Creek. This is more fully described in Appendix G of Report 60635, April 1991. From the measured concentrations and the synthesized flows, the combined average zinc loading for the Sump 1 and Sump 2 collection systems was calculated at 0.9 tonnes per annum. Insufficient information is available to apportion this total amongst the two collection systems. It should be noted that the Sulphide Cell is located within the drainage basin of the proposed Sump 1. Therefore, there is potential for increased loading to this system in the long term.

Total Zinc Load

The average annual zinc load generated by the Faro rock piles has been estimated at about 24 tonnes based on current water quality data. A summary of the estimated loads for each collection system is presented in Table 5.2. These values are representative of present-day levels of loading. An increasing trend in loading may be observed over the long term. Insufficient information is available to estimate how the annual zinc load from the rock piles fluctuates during dry and wet years.

TABLE 5.2

ESTIMATED AVERAGE ANNUAL ZINC LOAD FOR DRAINAGE COLLECTION SYSTEMS

Drainage Collection System	Average Annual Zinc Load (tonnes)	Average Annual Flow-weighted Zinc Concentration (mg/L)
Sump A30	6.6	43
Zone II Pit	4.5	29
Sidehill Dam	11.6	25
Sumps 1 and 2	0.9	4
TOTAL	23.6	22

Some of the zinc load estimates should be improved in the near future. The estimate for the Zone II Pit will improve with continued monitoring of the quantity and quality of water pumped from the pit. The load estimated at the Sidehill Dam is affected by pit dewatering which influences the water quality recorded at Station X23. Once dewatering ceases in early 1992, the quality measurements at X23 will become more representative of drainage from the Main Rock Piles.

5.1.6 Operating Rules

A set of operating rules were devised to schedule the pumping, diversion and storage of mainly rock pile drainage. Development of these rules was subject to the following constraints and assumptions:

- the capacity of the SRB treatment system was estimated at 33 L/s, constant throughout all seasons of the year;

- the potential live storage within the Zone II Pit was estimated at 500,000 m^3;

- all runoff diversions were assumed to be completely watertight; and,
- all drainage collection ditches and storage facilities were assumed to be completely watertight.

The level of storage required to regulate the flows of contaminated water to the SRB treatment system is dependent on the following three main factors:

- the volume of contaminated drainage generated by the rock piles;
- the capacity of the SRB treatment system; and,
- an acceptable frequency for, and concentration of, spillage of untreated drainage to the environment.

Estimates adopted for this study indicate that the capacity of the treatment system would almost equal the long-term average flow generated by the five collection systems. In such a situation, all rock pile drainage could theoretically be treated and no spillage would occur. However, very large storage capacity would be required to store excess drainage during wet climatic cycles, for subsequent feed to the treatment system during dry cycles. The cost of constructing such large storage could be prohibitive.

A more practical approach might involve providing a sufficient level of storage to ensure that spillage of untreated water only occurs on a very infrequent basis, perhaps during the wettest months of a 10 to 25 year return period wet event. A suitable frequency for spillage would have to be determined. Even with allowance for periodic spillage, the storage requirement could be significant owing to the close similarity between treatment capacity and the long-term drainage yield of the rock piles. A moderate increase in treatment capacity, or decrease in the estimated runoff from the rock piles, would dramatically reduce the level of storage required. Further study of these two factors is required before a more accurate determination of storage requirement is made.

To minimize the impact of any untreated rock pile drainage on the environment, the drainage collection systems which would feed the SRB treatment system would be carefully selected. Higher concentration drainages would be preferentially delivered for treatment and the lowest concentration drainage would be released when storage potential is exceeded. Reference to Table 5.2 shows that the Sump A30 System is currently estimated to provide the highest concentration drainage and therefore would be the priority source of feed for the treatment system. The Zone II Pit would be second, followed by the Sidehill Dam and then Sumps 1 and 2.

Another advantage of making drainage from the Sump A30 and Zone II Pit Systems priority sources of feed for the treatment system is their higher elevation. Pumping would be less costly than from the Sidehill Dam and Sumps 1 and 2.

Based on the constraints and assumptions listed above, and on the strategy to maximize the zinc load delivered to the treatment system operating rules were developed and are explained below.

Sidehill Dam

The operating rule adopted for the Sidehill Dam was largely influenced by the requirement to minimize the construction of new storage capacity. This would be accomplished by pumping rock pile drainage to the treatment system during periods of peak inflow to the dam.

The Sidehill Dam would be equipped with a pump having capacity approximately equal to the maximum output of the treatment system. Pumping would be initiated typically in April just before the arrival of the spring freshet. At this time, water from the Sidehill Dam would become the sole source of feed to the treatment system. The pump would run continuously at a rate equal to the treatment system capacity for an estimated five and a half months of the year, on average. The pump would then be shut down and would remain dormant until the following spring (except if difficulty is experienced in obtaining feed from the other drainage collection systems). Drainage generated during this dormant period would be contained by the storage of the Sidehill Dam. As explained above, this dam may have to regulate long-term year to year fluctuations of drainage inflows, as well as seasonal fluctuations.

Sumps 1 and 2

The operating rule for these sumps has been designed to minimize the construction of new storage capacity. Water from Sumps 1 and 2 would be pumped to the Zone II Pit rather than directly to the SRB treatment system. This would maximize exploitation of the storage available in the Zone II Pit, and minimize the size of the sumps. Such a strategy should be cost effective because development of storage in the Zone II Pit would only involve well installation. This is probably much less expensive than the corresponding additional earthworks required to increase the capacity of the sumps.

Pumping from Sumps 1 and 2 to the Zone II Pit would be conducted continuously during periods of high flow, and intermittently during the remainder of the year. Sizing of the pumps would be a matter of economic analysis. The larger the pump installed, the smaller the storage capacity that would have to be provided.

Sump A30

This has been identified as the priority source of feed water for the treatment system. Accordingly, the operating rule was designed to maximize the delivery of this drainage for treatment. Flows would be directed to the Zone II Pit via the In-pit Diversion Pipeline. The Pipeline would have a capacity. sufficient to convey all outflows from Sump A30, except unusually extreme rainfall and/or snowmelt events. These excesses should have lower zinc concentrations, and controlled spillage to the lake in the Main Pit would occur. Appropriate return periods for such events have yet to be specified. The location of the sump and pipeline would also be reviewed to account for stability concerns.

Diversion of water from Sump A30 directly to the treatment system was rejected for the following two reasons:

- a new storage facility near the inlet portal of the treatment system would be required to regulate drainage flows in excess of the capacity of the treatment system. Otherwise, overflow to the receiving environment may occur;

- flows from Sump A30 during high flow periods would constitute a large proportion of the treatment system capacity. The full treatment capacity could no longer be exclusively reserved for the Sidehill Dam during spring freshet and consequently a larger storage capacity for this dam would be required.

Zone II Pit

In addition to inflows from its own sub-catchment and seepage from the Main Pit, the Zone II Pit will receive inflows from Sump A30, Sump 1 and Sump 2, as described above. Preliminary calculations show that the 0.5 million m^3 of storage available in the Zone II Pit will be adequate to regulate the flow from all four drainage collection systems. It was assumed that a single deep well could practically empty the Zone II Pit if required. The well currently installed is capable of pumping from the Zone II Pit to 14 m from the pit bottom, and perhaps deeper.

The capacity of the pump and the treatment system would be roughly equal. The pump would only operate when abstractions are not being made from the Sidehill Dam. During a typical year, the Zone II Pit would feed the treatment system for about six and a half months.

5.2 Receiving Water Quality

5.2.1 Introduction

A water quality model has been previously developed for the Faro Mine site to predict receiving water quality. The model has been used in this assessment to assess the benefit of employing the sulphate reduction proposal. The model was used previously in the assessment of decommissioning options for the Down Valley Tailings Impoundment (SRK Report 60635, April 1991) and in the assessment of water quality related to the proposed in-pit disposal of tailings in the Main Pit at Faro (Kilborn Report, June 1991). The model consists of a number of elements simulating sources of contaminants on the mine site and runoff catchments, including watercourses. The loadings generated from each source are based on Curragh's water quality sampling program, and therefore the loadings are representative of current water quality. The model is also based on zinc concentrations only, this being the main contaminant.

The modelling conducted for the sulphate reduction system is based on a comparison of the potential receiving water quality if all sources of contaminant loadings are discharged to receiving waters, compared with the receiving water quality that would occur with the implementation of the treatment system. The scenarios and results of the modelling are discussed below.

5.2.2 Scenarios

Scenario 1 can be considered the base case for this assessment and is based on the following conditions:

- the Main Pit has been filled with tailings and discharge from Faro Creek enters and exits via a cutting in the south lip of the pit; with water quality as described in Appendix A of Kilborn Report

- the Down Valley Tailings Impoundment has been decommissioned. The Intermediate Dam remains at elevation 1044.3 metres and contains a small volume of tailings which are flooded; seepage discharge rates and concentrations are as specified in Report 60635, April 1991;

- drainage from rock piles is based on current measurements, although there is no allowance in the model for the loss of contaminant load as a consequence of settling and reaction in the tailings impoundment, as happens at present. Some load loss is still likely to occur but has not been estimated;

- seepage from the Zone II Pit to the North Fork of Rose Creek is a combination of seepage from the Main Pit and infiltration into the Zone II Pit. The Main Pit seepage rate is based on the estimate contained in the Kilborn report. This is 50 m^3/day based on the construction of plug

dams between the two pits consisting of rock berms with tailings slimes deposited between the berms.

It is appreciated that the water level in the Zone II Pit will fluctuate, and that there will likely be a relationship between water level and seepage loss from the pit. Such a relationship has not been developed to date. For the purpose of this assessment, and equilibrium seepage condition has been assumed, with the Zone II seepage loss being equal to the Main Pit seepage estimate.

The total load generated by mine facilities is estimated to be about 25 tonnes/year (total zinc). The resulting receiving water qualities are given for the X14 monitoring station on Rose Creek below the Down Valley Impoundment.

Scenario 2 is the first SRB treatment system simulation and consists of the following deviations from the base case in Scenario 1:

- drainage from the rock piles is directed to the underground treatment system, and discharge quality has a predicted zinc concentration of 0.02 mg/L;
- the flow of drainage to the underground is at an estimated average annual rate of 33 L/s;
- seepage from the Zone II Pit to the North Fork of Rose Creek now comprises only seepage from the Main Pit to Zone II. It is assumed that infiltration to the Zone II Pit is now directed to the underground treatment system. Seepage from the Zone II Pit will occur unless the pit water level is kept at a low elevation.

Scenario 3 consists of the same water management scheme as for Scenario 2. The only difference being that the water being discharged from the underground treatment system is now assumed to have a concentration of 0.1 mg/L. This represents a less than optimal treatment process.

Scenario 4 consists of the same water management system but with a discharge concentration from the underground treatment system of 0.5 mg/L zinc. This represents a further sub-optimal treatment system in operation, and the resulting impact this has on receiving water quality.

5.2.3 Results

The predicted seasonal variation in zinc concentrations at monitoring station X14 for the four scenarios are illustrated in Figures 12 and 13. For comparison purposes, the present-day zinc concentrations at station X14 are also presented in Figure 13. The peak month and average concentrations predicted are summarized as follows in Table 5.3.

TABLE 5.3

RECEIVING WATER QUALITY DATA

Scenario	Peak Monthly (mg/L)	Flow-weighted Average (mg/L)
1	0.45	0.36
2	0.063	0.026
3	0.075	0.027
4	0.14	0.033
Present Day	0.11	0.067

6.0 SULPHATE REDUCTION IMPLEMENTATION

A phased approach is recommended for the implementation of a sulphate reduction system at the Faro mine. This will require the immediate preparation of the underground mine for the implementation of the sulphate reduction system, prior to depositing tailings in the Main pit. Subsequently, once the sulphate reduction system has been inoculated, and is being developed to accommodate the full load of the site runoff, the various sources of contaminated drainage may be phased into the system, depending on the load and immediate environmental impact. Possibly, the Zone II pit drainage may be the primary source to be incorporated into the system, and the secondary and tertiary sources may be identified and phased into the treatment system. However, the development of such a program is beyond the scope of this study. The physical aspects regarding preparation for the implementation of the sulphate reduction system will be addressed in this section.

6.1 Mine Preparation

6.1.1 Inlet and Outlet Well Preparation

Proposed inflow/outflow holes were located in such a manner as to pierce the underground workings in areas of relative stability. They were also located so that they had, if possible, multiple travel paths in various directions. This would help insure water passage even if failure occurred in the local area.

These drillholes must be relatively large diameter in order to accommodate the flows which are required in the workings. However, their very size makes drilling a challenge. Holes are proposed to be at least 8 inch diameter inside the casing. This requires that the drilled holes be approximately 11 inches in diameter.

Several methods of drilling these holes are possible. One is using a large diameter down-the-hole hammer and drilling the hole directly. Down drilling with large diameter rotary drills is also a possibility, as is small diameter down-holes (DTH hammer or core) reamed with a rotary bit.

Rock in the area is of such varying quality, and the holes are so large, that the holes must be cased. Steel casing may well corrode. PVC casing is also an option. Although it does not give the local support to the rock that steel casing does, it will resist corrosion. As such, it may be the casing of choice in this application.

6.1.2 Mine Portal Closure and Bulkhead Design

Bulkheads will be necessary to prevent the entry of tailings into the underground workings. Therefore, any opening which daylights into the pit must have a bulkhead. At present, at least three bulkheads are

necessary: the main portal, south area ventilation raise, and the north area ventilation raise. These are indicated on Figure 8.

The bulkheads will likely require a detailed design once the feasibility of the option is proven. At this point, the bulkheads are envisioned as relatively simple items.

Bulkheads for the ventilation raises should consist of a 10 foot long concrete plug located 20 to 40 feet from the pitwall. This remaining space from the plug to the pit wall would then be filled with loose rock. End-dumping of rock over the portal would complete the seal. Any pipes which must pass through this plug would be installed prior to installation, and protected during rock backfilling procedures.

Two options have been considered for bulkheading the main portal. As the portal is located under backfill, it is lined. This lining will corrode and disappear with time. Therefore, a permanent seal cannot be effected against this steel lining. If the rock is of sufficient quality near the juncture of the north and south area ramps, the liner could be removed and a concrete plug emplaced. This plug should be around 15 feet long and be well anchored to the surrounding rock, either with steel cable or grouted bolts. The remaining space from the plug to the portal would be filled with loose rock, similar to the ventilation raises. If the rock is of insufficient quality to insure good anchorage for the plug, the portal tunnel could be filled with waste rock and concrete pad placed over the portal proper.

Flooding of the mine would be accomplished by placing a 12 inch pipe through the main portal bulkhead into the mine at grade. This should be sufficient to allow the entire flow of the creek into the underground workings as the mine floods. Inlet/outlet drillholes, as well as the north ventilation raise must be left open as air vents as the mine floods. Permanent closure of the flooding pipe can be had by closing the pipe with a simple sliding gate upon completed mine flooding and filling the pipe with cement grout from a pre-installed fitting.

6.2 Preparation for Process Implementation

6.2.1 Substrate Placement

Successful implementation of the sulphate reduction process within a relatively short time span, will require preparation of the underground mine. To ensure the rapid development of an active sulphate reducing bacterial culture in the underground mine it will be beneficial to provide an immediately available substrate within the underground mine. This may be achieved with the placement of hay along the anticipated flowpath of the system, prior to flooding the underground mine. Although it is anticipated that sugar will be used as the primary carbon source, placement of hay will provide additional surface area for bacterial attachment sites once the sulphate reducing bacterial population has developed.

In addition to the placement of the hay, it is recommended that a supply of sugar be placed in the mine to serve as a more readily consumable substrate in order to aid system inoculation and initiation.

6.2.2 System Initiation

Inoculation

As reported by Reynolds *et al.* (1991), cow manure contains active sulphate reducing bacteria. Potentially, this represents a source for the inoculation of the underground system. Fresh cow manure may be shipped in mobile tankers to the site and either be placed in the underground mine with the hay, or alternatively, be pumped underground following flooding of the mine. The latter method will be preferable, since anoxic conditions will have developed in the underground system following flooding, and therefore, the impact on the bacteria which require anaerobic conditions, will be minimal.

A second method of inoculation that may be considered, will require the operation of a pilot scale sulphate reduction system on site. Active biomass may then be extracted directly from this system and introduced into the underground system. It is anticipated that the success rate of this method will be significantly greater than that described previously, since the bacteria will already have been adapted to the water conditions prevailing at the site. The scale of the pilot system will depend on the time scale required for the start-up of the full scale system. Added benefits represented by the operation of a pilot scale system will be the evaluation of the performance anticipated for the full scale system, determining the substrate requirements, sludge generation rates and discharge water quality.

Monitoring

Upon completion of the inoculation program, the underground system should be monitored for sulphate reduction activity. This may be accomplished by monitoring the water quality and the free sulphide generation observed at the inlet, supplemental and outlet wells of the reactor system. In addition, a bacterial sampling program should be conducted to isolate and identify sulphate reducing species.

Start-up

The reduction rate may be estimated by the rate of metal precipitation in conjunction with the rate of free sulphide formation. Once a sulphate reduction rate at or close to the anticipated reduction rate has been established, start-up of the system may be possible.

It is assumed that, concurrent with the start-up of the sulphate reduction system, the retention structures will be constructed. Therefore, as a result of the capacity of these structures, a lag will exist between the

implementation and the full capacity demand on the sulphate reduction system. This lag will be ideally suited to the controlled start-up of the system.

Conceptually, start-up will proceed by initially pumping water to be treated into the system at a rate equal to approximately one third of that anticipated for full-scale operation. Once the bacterial population has acclimatised to the imposed conditions as indicated by the sulphate reduction rate, the pumping rate may be gradually increased. Should an increased pumping rate result in a reduced sulphate reduction rate, it will be possible to reduce the through flow until the bacterial population has adapted to the new conditions imposed. This stepwise increment in the total hydraulic load to the system will ensure the maintenance of an active bacterial population.

6.2.3 Long Term Monitoring

It will be essential to conduct a monitoring program after start-up has been completed to evaluate the system performance. The program will require monitoring of the inlet, the discharge and intermediate (within the reactor itself) water quality to evaluate the reaction kinetics and the efficiency of sulphide utilisation for metal precipitation.

In addition to evaluating discharge water quality objectives, the objectives of the monitoring program will include fine tuning the operating parameters for the system. The operational objective for the system will be to achieve the discharge water quality criteria with a minimum of sulphide overproduction. The rate of sulphide production may be controlled by operating the system under substrate limiting conditions, the rate of substrate supply to the system will be limited to the point where the discharge water quality is at or close to the point of deviating from the discharge objectives. For example, in the baseline case complete removal of zinc was assumed, which may not necessarily be the case. Under such operating objectives, the substrate supply will be decreased stepwise from the theoretical supply rate until an increase in the discharge zinc concentration is observed. This will represent the equilibrium point, and therefore, the substrate supply rate will be increased marginally to ensure the discharge objectives are always met. Economically this will present the most feasible operating approach.

7.0 CONCLUSIONS AND RECOMMENDATIONS

7.1 Conclusions

From the study completed herein, we conclude that:

- sulphate reduction presents a viable water treatment alternative for the effective removal of zinc and other heavy metals from contaminated drainage. The inherent mechanism of sulphate reduction also produces alkalinity, which will buffer the treated water pH at approximately 7.0;

- the Zone II pit at the Faro mine at present does not constitute a viable sulphate reduction reactor, primarily due to the relatively low water temperatures observed for water discharged from this location. Other logistical problems, such as substrate supply and water flowpath management, prohibit the use of the Zone II pit as a sulphate reduction reactor;

- a worst case scenario evaluation for sulphate reduction in the underground section of the Main Pit, with direct short circuiting for a single channel flow from the inlet to the outlet, indicates an underground volumetric utilization of 18,800 m^3. On completion of mining it is anticipated that a void of 251,000 m^3 will remain, of which 100,400 m^3 will be available for the implementation of the sulphate reduction process. Therefore, the base case evaluation represents a utilization of only 18.7% of the available volume.

- a worst case scenario for the underground treatment system assuming direct short circuiting from the inlet to the outlet port for this reactor system, indicates that potentially all site drainage can be treated provided the feed rate is equal to the flow weighted average drainage rate of 33 L/s, and provided adequate storage capacity is developed. Zinc and several other minor heavy metal contaminants will be removed from the water and deposited in the underground workings as a highly insoluble sulphide mineral under stable conditions. While the base case evaluation represents only 18.7 % of the total volume available to the process, the estimated operating conditions indicate that treatment capacity may be enhanced 3 to 4 fold;

- based on the base case volumetric availability for the reactor system, the rate of sludge production will result in an annual retention time reduction of approximately 2 hours, at a capacity of 33 L/s. Calcium carbonate formation as a result of alkalinity production, contribute in excess of 80% (vol/vol) of the sludge generated. Analysis of the base case revealed that sludge formation will not affect system performance for at least 25 to 30 years. Sludge accumulation will not affect the process for at least 360 years if all of the available underground volume is utilised;

- a phased implementation of the system will be beneficial from both economic and operational considerations;

- An analysis of the underground mine stability shows that the mine should provide a network of travel paths well into the future.

- use of the underground treatment system to control the quality of drainage discharging from the mine site should provide adequate protection of receiving water quality.

7.2 Recommendations

Sulphate Reduction

The conclusion that sulphate reduction as a water treatment alternative may be successfully implemented at the Faro mine site is contingent on the experimental confirmation of various factors. It would therefore be prudent to conduct a laboratory study to evaluate the following:

- sulphate reduction rates achievable under conditions closely simulating those expected in the underground mine;

- substrate utilisation

- trace nutrient supplement requirements;

- sludge characterisation and confirmation of generation rates.

In addition, certain site specific assumptions were made to evaluate the process performance. Therefore, to confirm the validity of these assumptions, it is recommended that the following field studies be conducted:

- the temperature profile within the northern section of the underground, which is currently being flooded, should be determined and monitored. This should provide an indication of the accuracy of the underground temperature evaluation.

- once the underground has been flooded, tracer tests should be conducted to evaluate actual underground retention times, and to provide an indication of the flow paths that may be achieved through variations in the inlet-outlet combinations.

Water Quality and Management

A more accurate evaluation of site drainage volume and quality is required. Initially, it is therefore recommended that:

- V-notch weirs be installed at sampling stations X7 and X23, and monitored on at least a daily basis. The stilling basins should be sealed with plastic liners to minimise seepage around and under the weirs. Also, the theoretical rating curves for the weirs should be validated by current meterings or bucket and stopwatch measurements. This will assist in a more accurate evaluation of the loads from the Main Rock Piles;

- integrating flow meters be installed on pipelines from Sump A30 and from the Zone II Pit and monitored on at least a monthly basis. A weir should be constructed at the overflow section of the A30 Sump. The levels in the A30 Sump should be monitored on a daily basis during overflow events.

Underground Stability

Rock instability could impact on both water transfer systems and breaching the underground workings. It is recommended that a monitoring system be considered to indicate areas of collapse in the event that inlet/outlet holes are rendered inoperative, or underground failures occur. This should be done once a specific design has been formulated. A detailed inspection should be completed of all underground workings and the pit to establish and evaluate areas of potential weaknesses.

Engineering Evaluations

Based on the preliminary studies already completed, it is apparent that more detailed engineering evaluations will be required to finalize feasibility design. This should occur in conjunction with the above mentioned field and laboratory studies. A detailed cost estimate for the project should then be completed.

This report, 60643, Sulphate Reduction as a Water Treatment Alternative at the Faro Mine is respectfully submitted by:

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JTC/R-3

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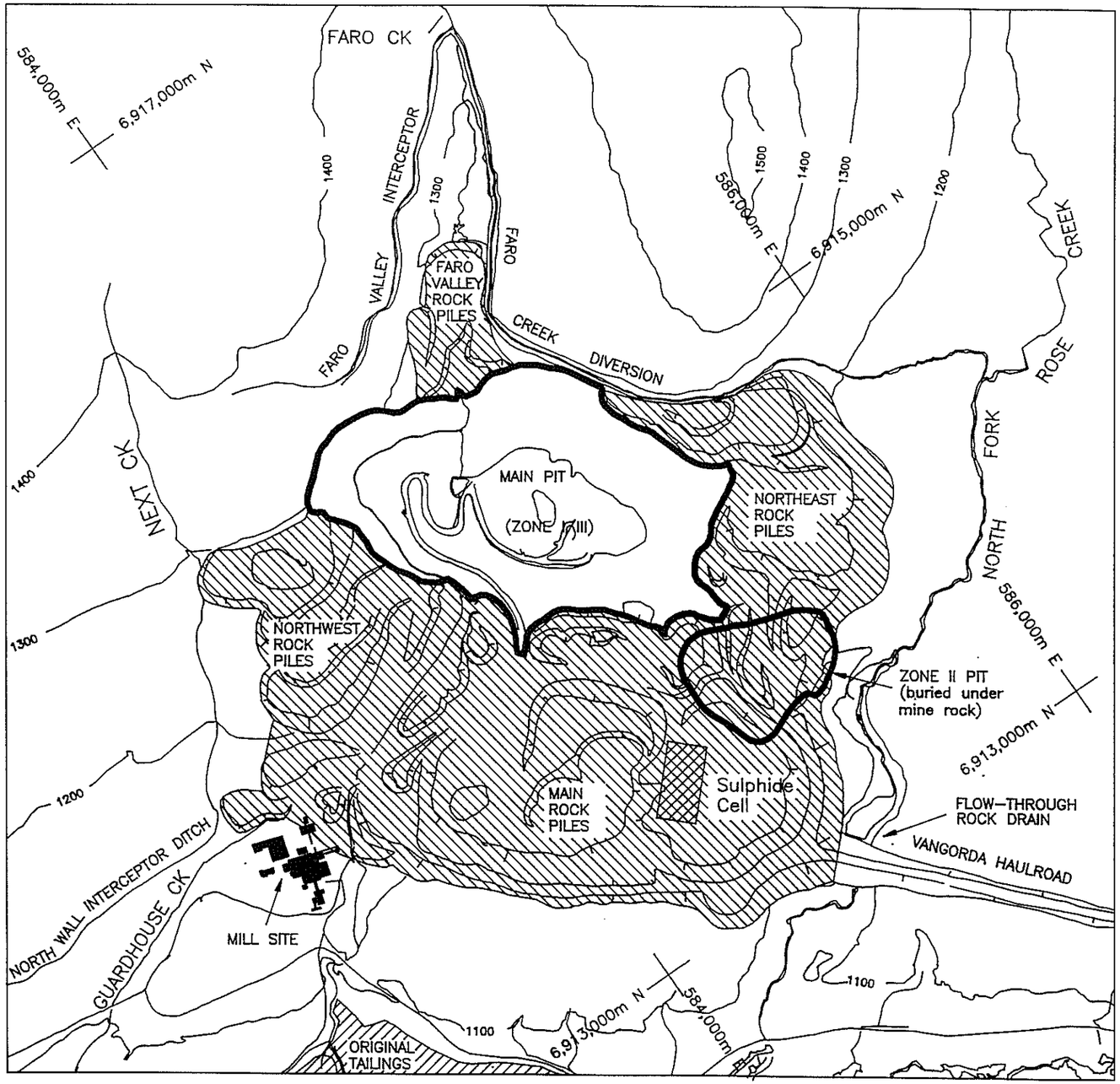
FARO DECOMMISSIONING

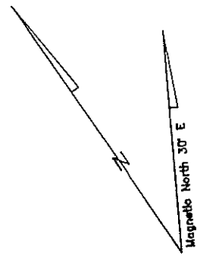
OVERVIEW OF THE ENVIRONMENTAL PLANS

APPENDIX B

Sulphate Reduction as a Water Treatment Alternative at the Faro Mine, Report Number 60643, Steffen, Robertson & Kirsten (B.C.) Inc., Vancouver, B.C., December 1991



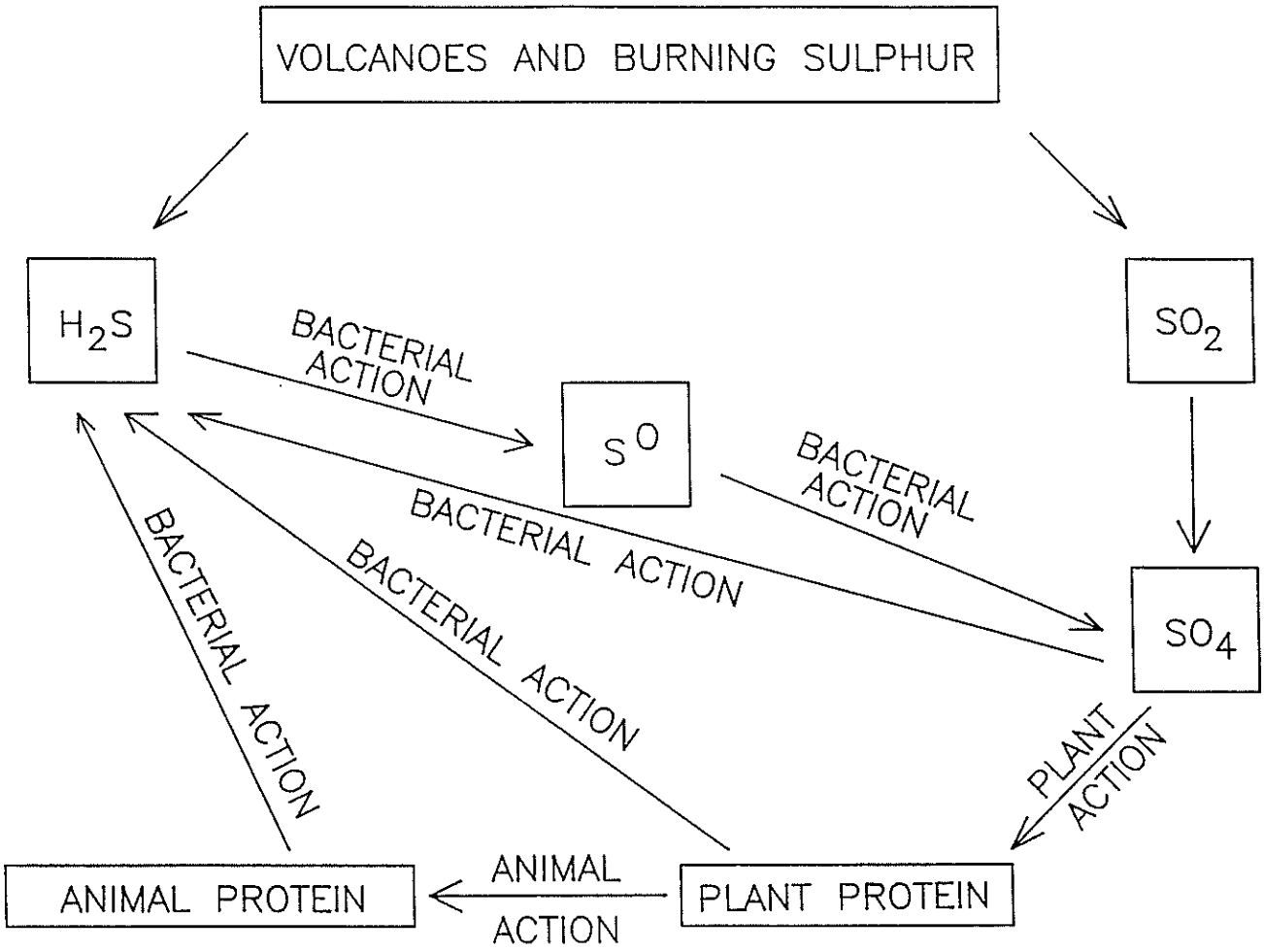
Notes:
Base Map supplied by the Orthoshop, January 1991
North American Datum 1927
Contour Interval 100 m
Universal Transverse Mercator Projection
Map is a composite of 9 basemaps.
It was redigitized to simplify planimetric features.



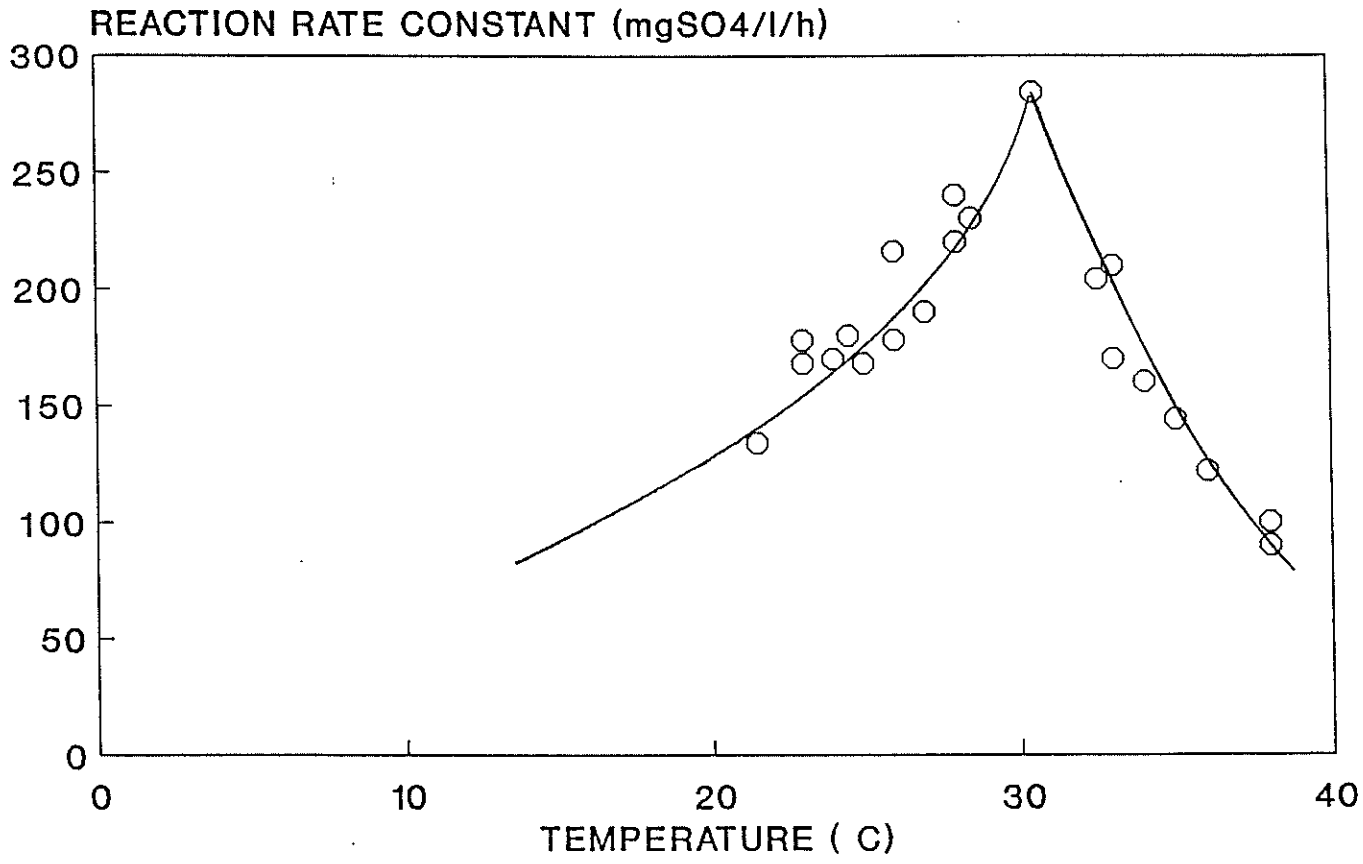
Legend

- Gravel Roads
- Pipeline
- Open Pit
- Ditch
- Waste Dumps
- Sulphide Cell
- Tailings Impoundment

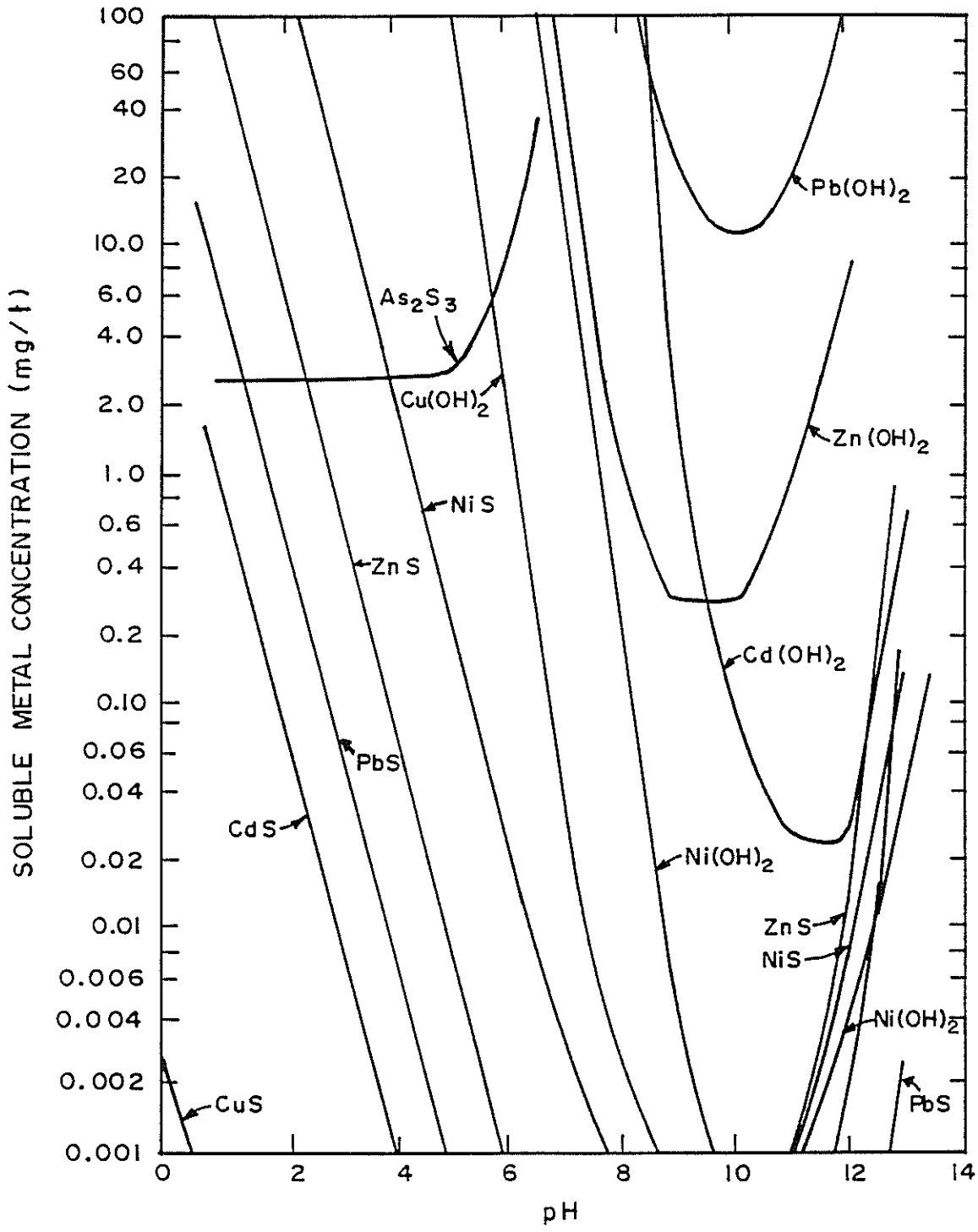
CURRAGH RESOURCES INC.	DATE Nov., 1991
Location of Mine Facilities	PROJ. NO. 60643
	APPROVED
	NO. Fig. 1
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	



Curragh Resources Inc.	Project Number 60643
Figure 2 **Schematic of the Sulphur Cycle**	
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	



Curragh Resources Inc.	Project Number 60643
Figure 3 **Effect of temperature on the biological sulphate reduction rate (extracted from Maree, 1982)**	
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	



Curragh Resources Inc.	Project Number 60643
Figure 4 **Calculated Metal Sulphide and Hydroxide Solubilities (Bhattacharrya Et Al 1981)**	
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	

15000W

14500W

14000W

13500W

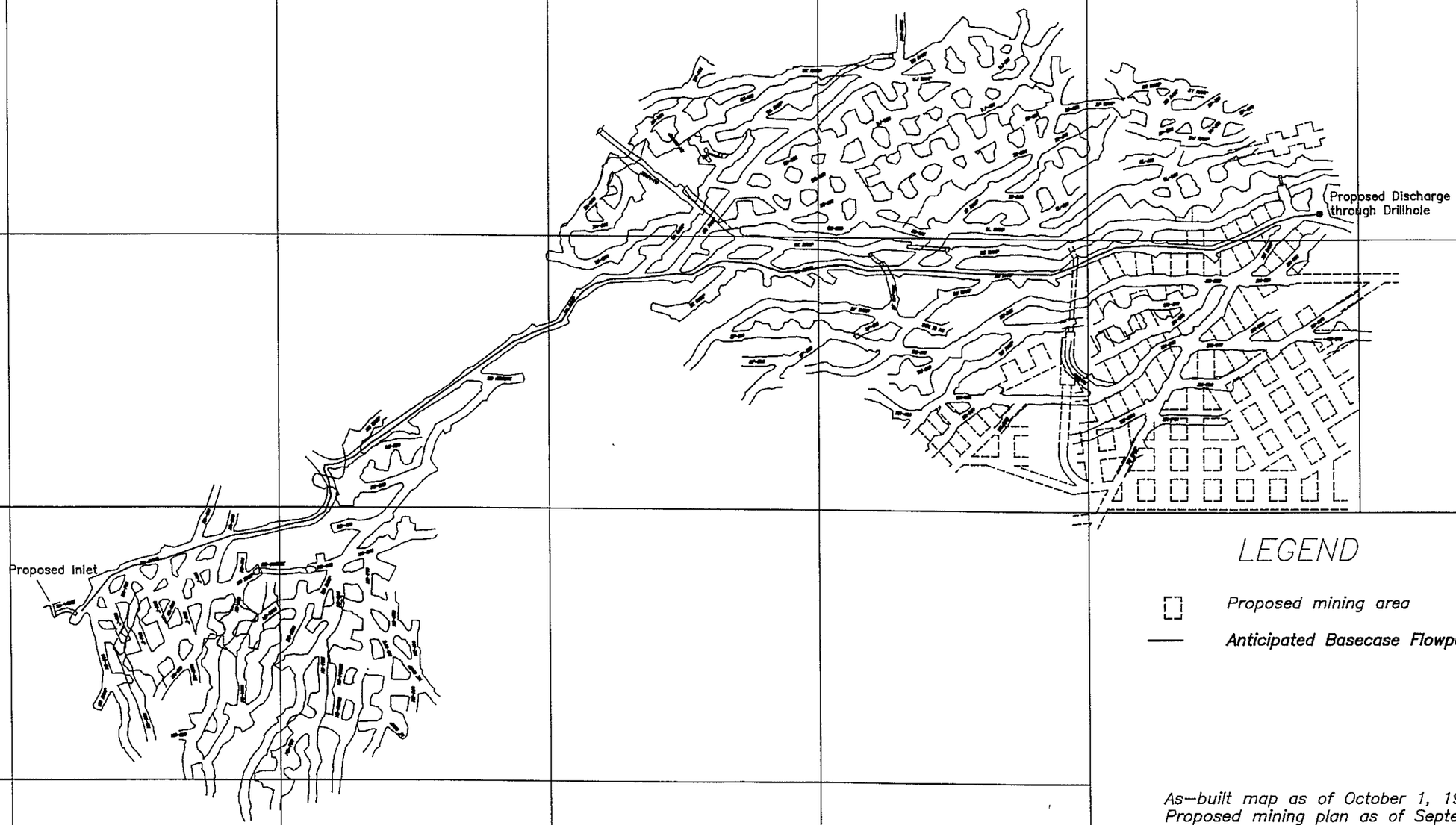
9000N

8500N

8000N

7500N

7000N



LEGEND

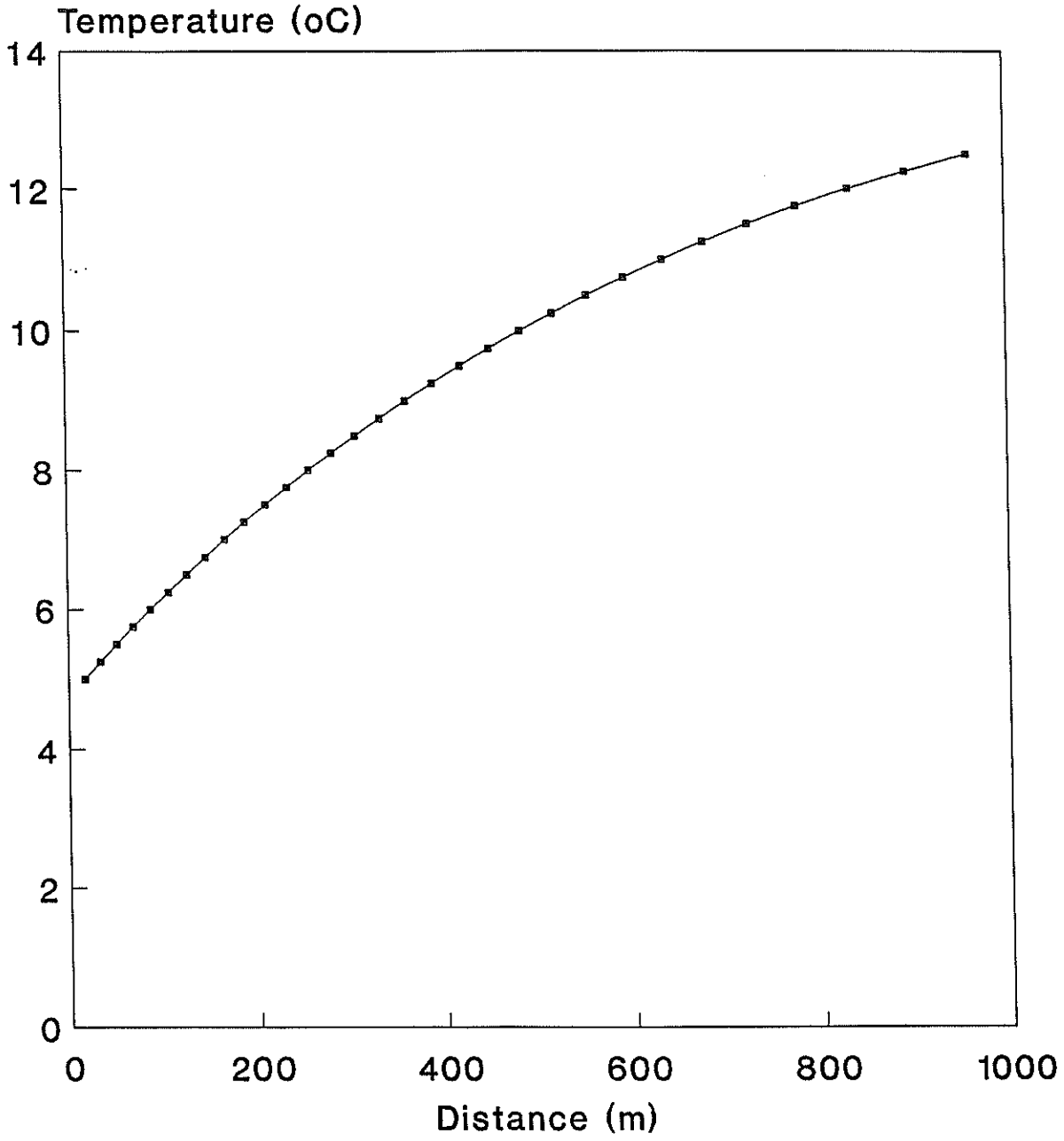
Proposed mining area

Anticipated Basecase Flowpath

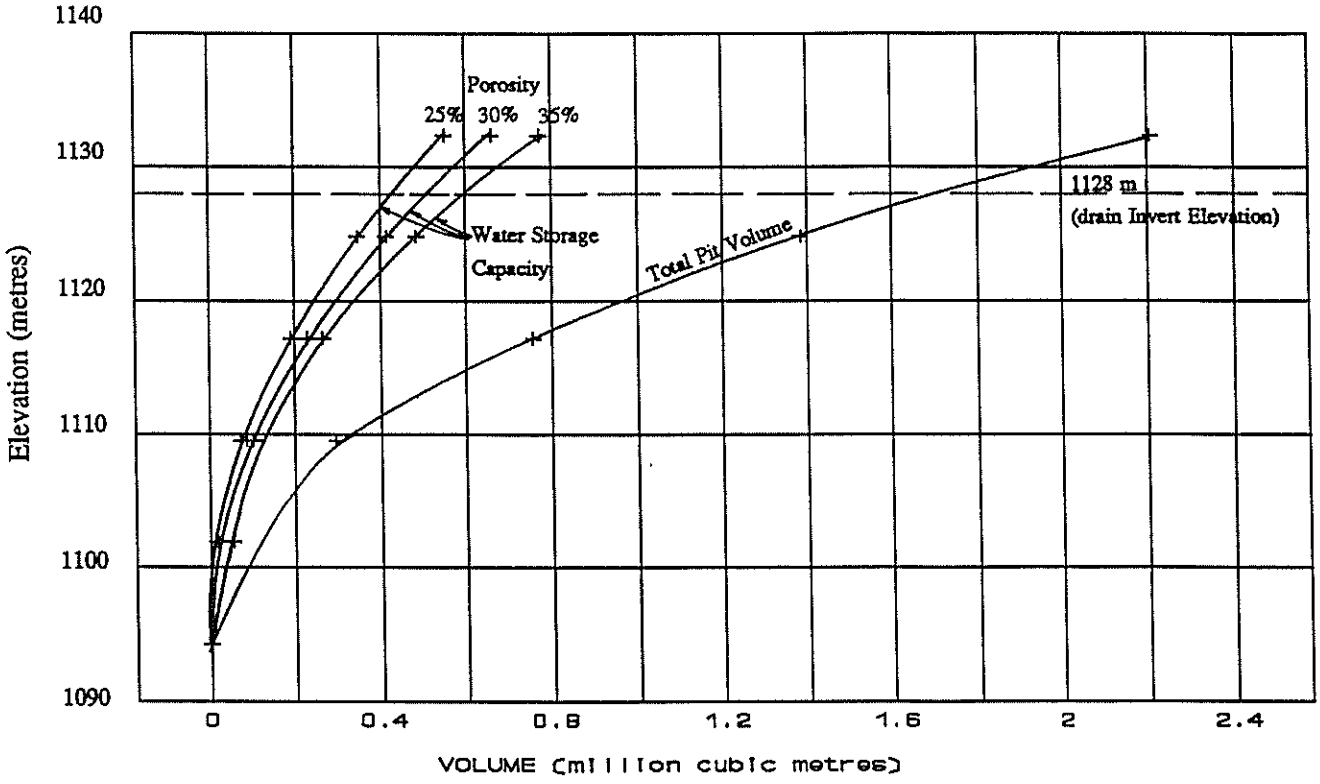
As–built map as of October 1, 1991
Proposed mining plan as of September 1, 1991

NOT TO SCALE

CURRAGH RESOURCES INC.	DATE Nov., 1991
PROPOSED DRILL HOLE LOCATIONS BASELINE SULPHATE REDUCTION EVALUATION	FILE NO. 60643
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	Fig. 5

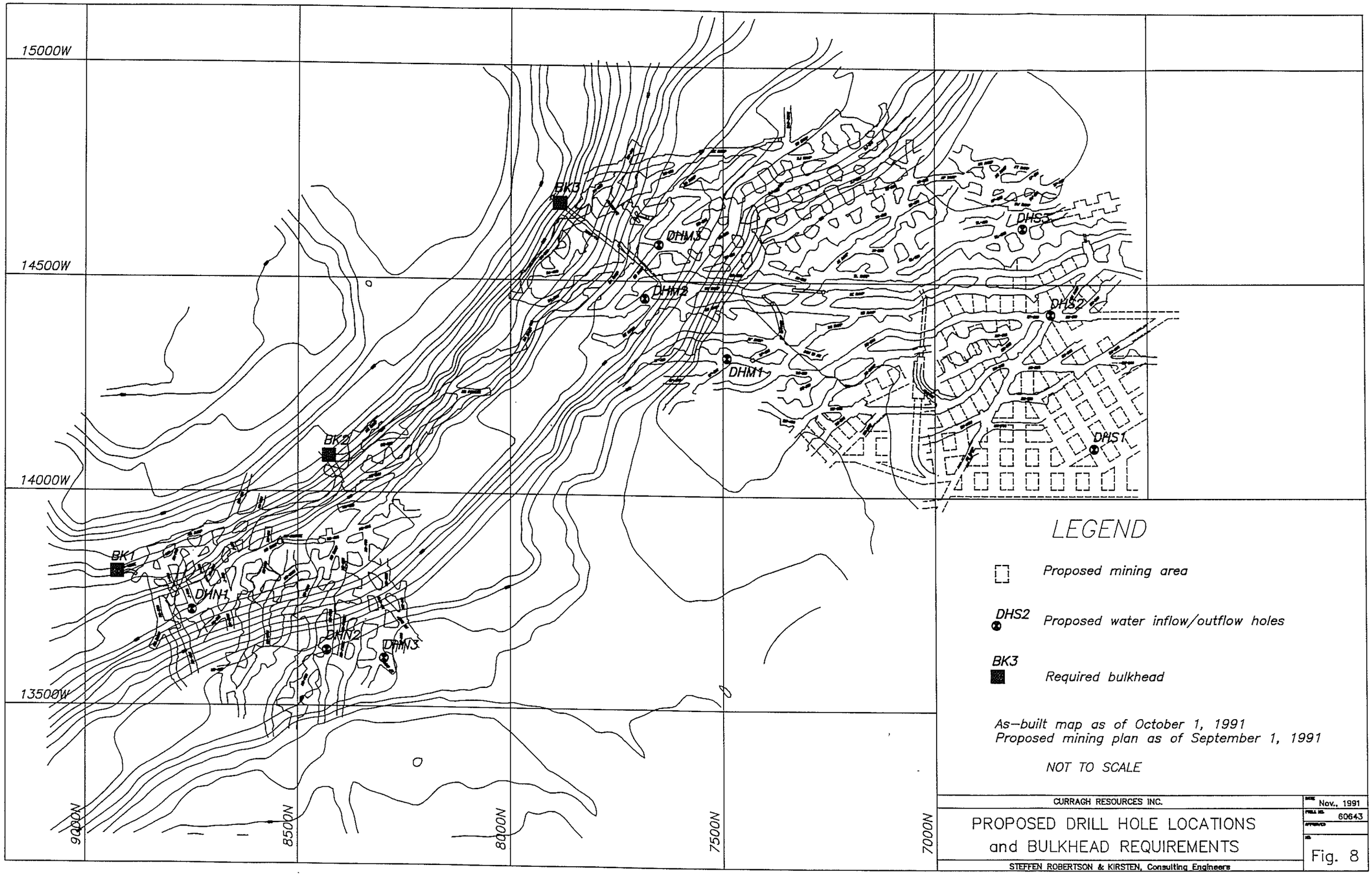


Curragh Resources Inc.	Project Number 60643
Figure 6 **Water temperature as a function of the distance of passage through the underground mine**	
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	



Notes: Total pit volume curve derived from Figure 10, "Development of the Zone 2 Waste Dump", CRI, 1987.
Elevations are relative to North American Datum 1927.

Curragh Resources Inc.	Project Number 60643
Figure 7 **Elevation - Capacity Curve For Zone II Pit**	
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	



LEGEND

- Proposed mining area
- DHS2 — Proposed water inflow/outflow holes
- BK3 — Required bulkhead

As–built map as of October 1, 1991
Proposed mining plan as of September 1, 1991

NOT TO SCALE

CURRAGH RESOURCES INC.

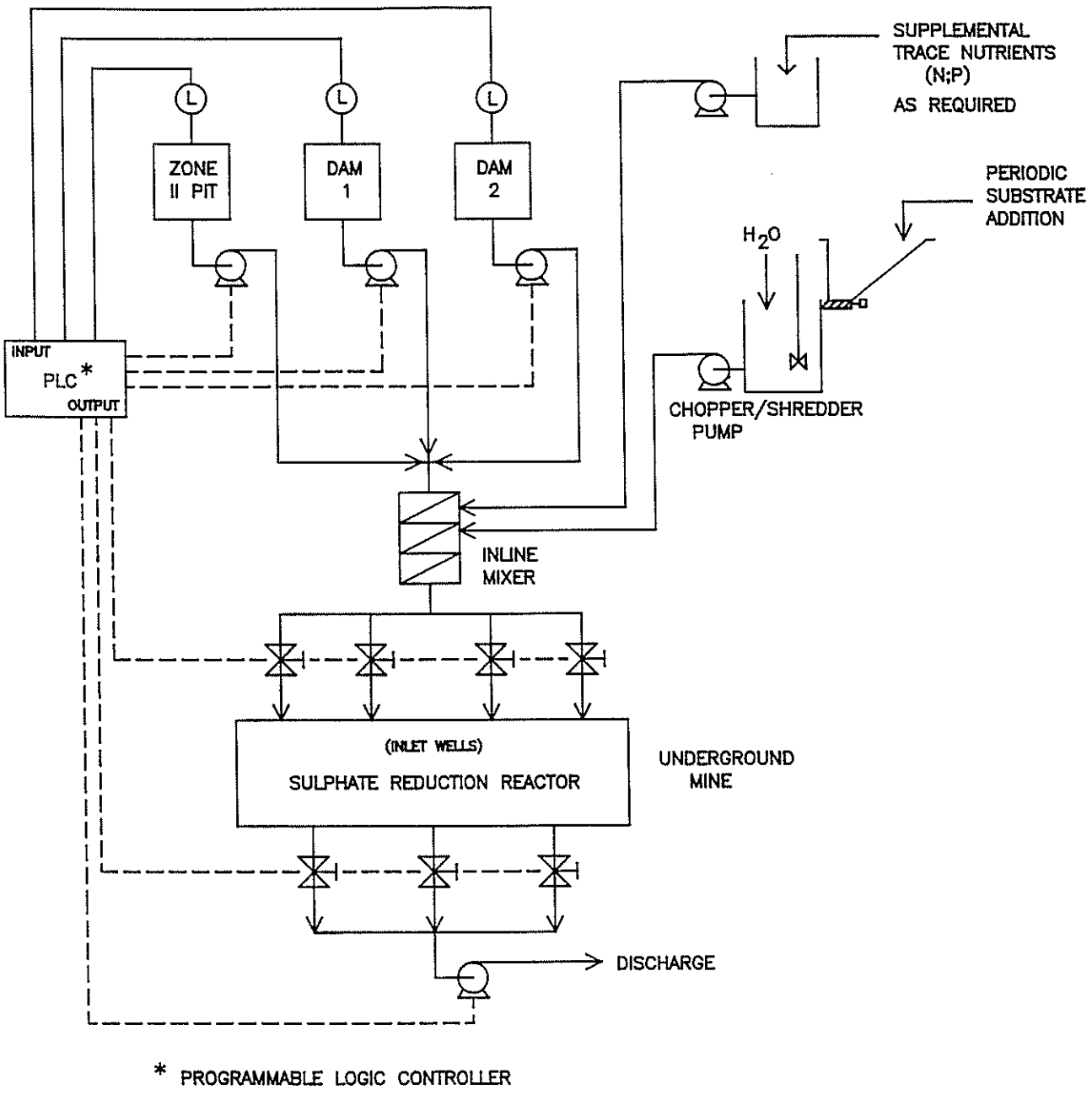
Nov., 1991

60643

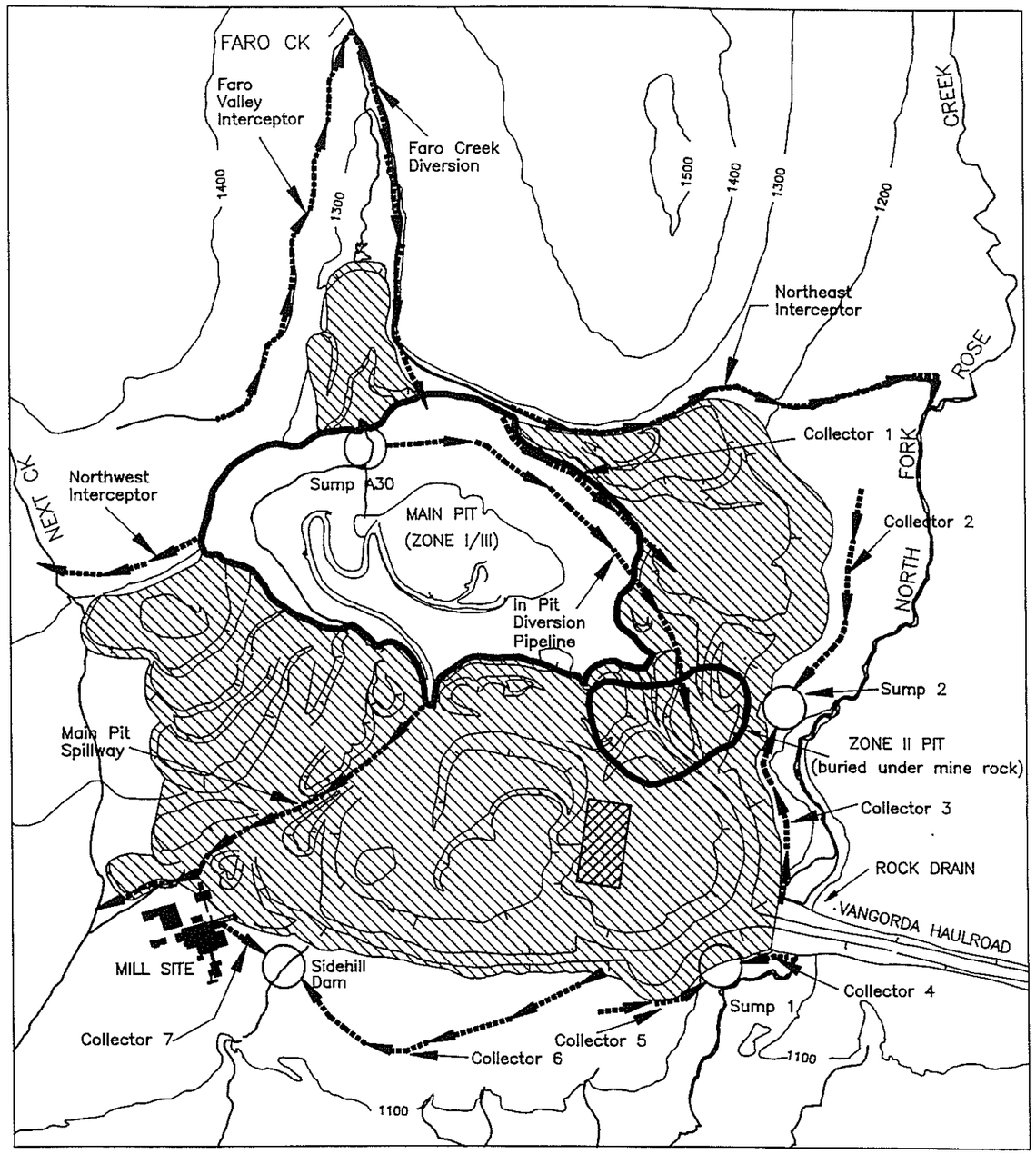
PROPOSED DRILL HOLE LOCATIONS and BULKHEAD REQUIREMENTS

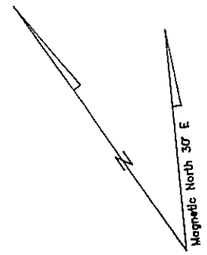
Fig. 8

STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers



Curragh Resources Inc.	Project Number 60643
Figure 9 **Conceptual Process Flow Diagram**	
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	





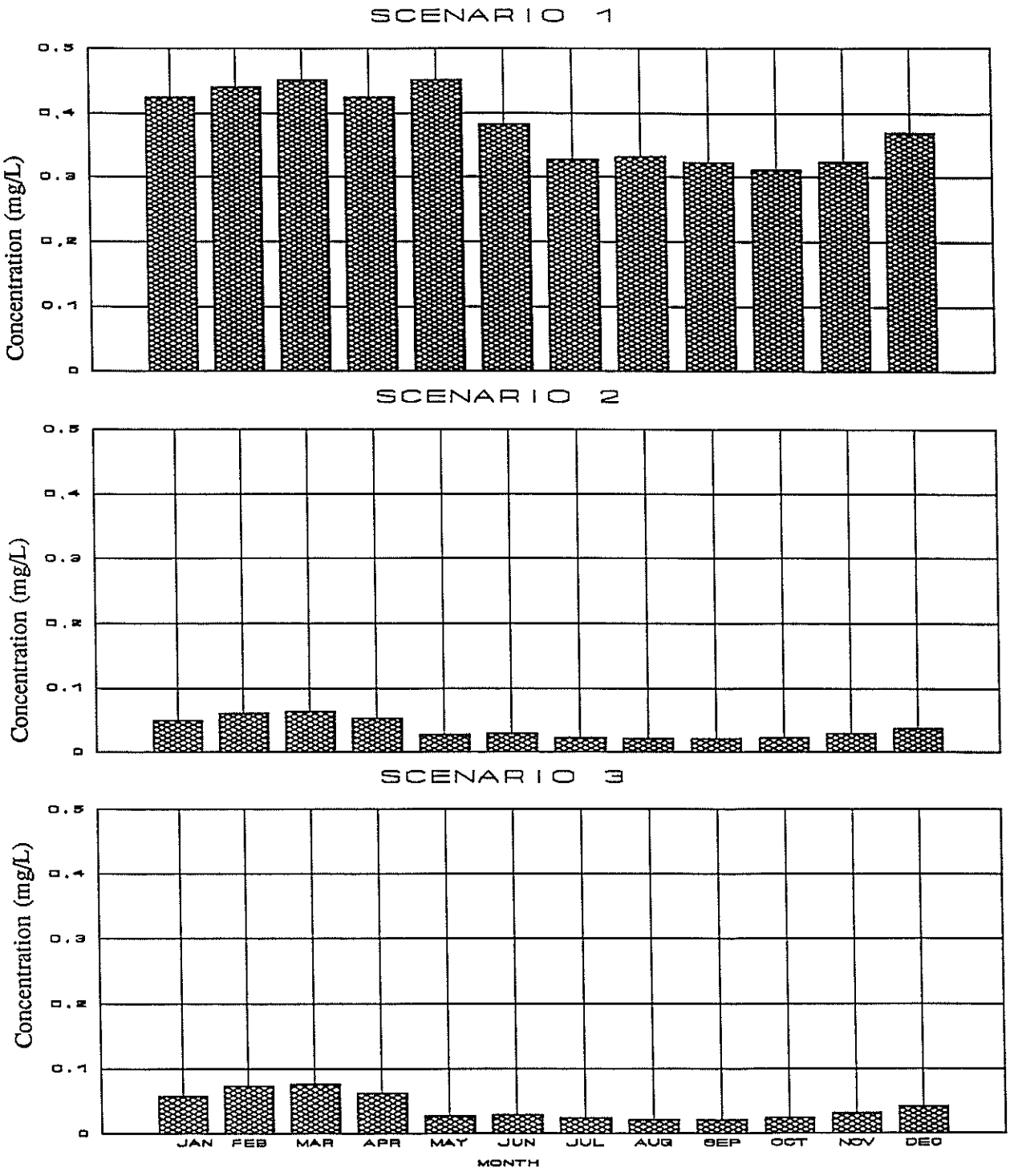
Legend

- Open Pit
- Rock Piles
- Sulphide Cell
- Ditch or Pipeline
- Drainage Collection Point

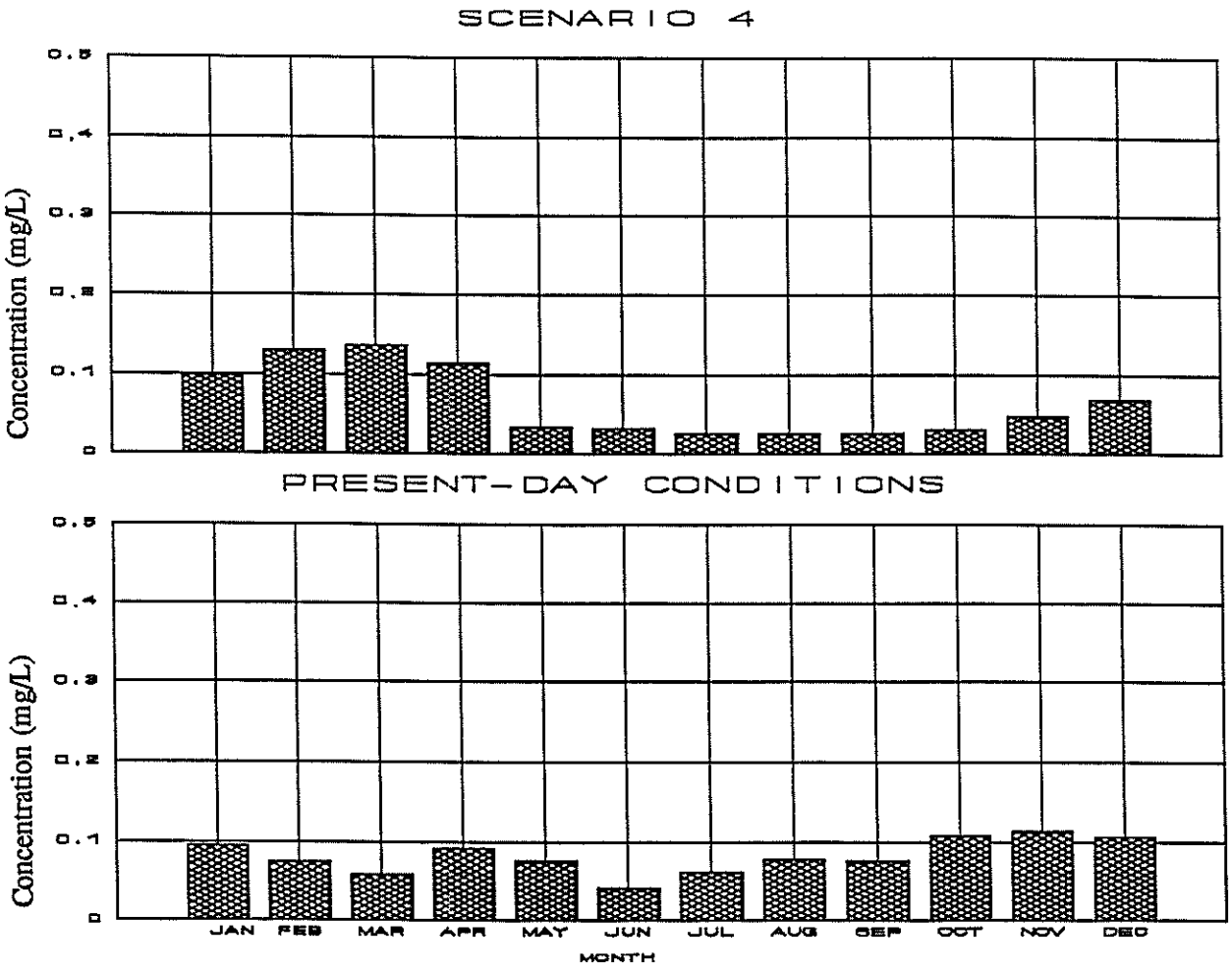
Notes:
Base Map supplied by the Orthoshop, January 1991
North American Datum 1927
Contour Interval 100 m
Universal Transverse Mercator Projection
Map is a composite of 9 basemaps,
It was redigitized to simplify planimetric features.

0 500 1000 1500 metres

CURRAGH RESOURCES INC.	DATE Dec., 1991
Location of Facilities for the Conceptual Water Management Plan	PROJ. NO. 60643
	APPROVED
	NO. Fig. 10
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	



Curragh Resources Inc.	Project Number 60643
Figure 12 **Estimated Average Monthly Zinc Concentrations for Station X14 at Mine Decommissioning Scenarios 1, 2, 3**	
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	



Curragh Resources Inc.	Project Number 60643
Figure 13 **Estimated Average Monthly Zinc Concentrations for Station X14 at Mine Decommissioning Scenario 4 & Present-day Conditions**	
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	

FARO DECOMMISSIONING

OVERVIEW OF THE ENVIRONMENTAL PLANS

APPENDIX C

Curragh Resources Inc., Faro Division, Reprocessing Tailings, Report Number WH91-07, R.F. Downs, P. Eng. and G.W. McDonald, Toronto, Ontario, December 1991

CURRAGH RESOURCES INC.

FARO DIVISION

REPROCESSING TAILINGS

Richard F. Down, P. Eng

G. W. McDonald, V.P. Metallurgy

Report No. WH91-07

December 9, 1991

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INTRODUCTION

This report has been co-authored by Mr. Godfrey McDonald, Vice-President, Metallurgy, Curragh Resources Inc. and Mr. Richard F. Down, P. Eng. of Richard F. Down & Associates Inc., who provides consulting services in mineral processing.

OBJECTIVE

The Faro abandonment plan for the Down Valley Tailings is designed to ensure the final resting place for the tailings is in a low risk environmental location. To accomplish this, the tailings would be stored (submerged) under a two metre water cover so oxidation of the sulphides in the tailings is severely impeded by the lack of readily available oxygen.

The objective of the proposed plan will be to remove all the tailings in the Down Valley Tailings area that is above the 1044.3 metre elevation, to reprocess these tailings to recover the base and precious metals in a marketable bulk concentrate and then to condition the remaining tailings with lime (to precipitate any water soluble metals) before discharging them into the mined-out Faro pit where they will be submerged beneath a 4.25m water cover. The remaining tailings in the Down Valley area, below elevation 1044.3 will be covered with two metres of water so they will be non-reactive.

DESCRIPTION OF THE TAILINGS

There are 37.7 million tonnes of tailings in the Down Valley Tailings area that are above elevation 1044.3. These tailings have the following typical analysis:

Lead	0.79%
Zinc	1.23%
Copper	0.15%
Silver	15 g/t
Gold	0.12 g/t

One tonne of tailings has the following gross value:

Lead @ 30¢ US per lb.	= $ 5.22 US
Zinc @ 60¢ US per lb.	= $16.27 US
Copper @ 90¢ US per lb.	= $ 2.98 US
Silver @ $4.50 US per oz.	= $ 2.17 US
Gold @ $365 US per oz.	= $ 1.41 US
Gross Value	= $28.05 US

METALLURGICAL TEST PROGRAMS

a) Sample Acquisition

The original and the 1974 tailings impoundment areas were sampled by digging 17 holes on a grid plan with a backhoe. Samples representing one metre increments of each grid hole were cut, bagged and stored for subsequent laboratory testwork. A site plan with grid hole location is shown on figure 1. The exposed wall of each grid hole was photographed to illustrate the banding (particle size) witnessed.

b) Initial Laboratory Program

The initial laboratory program was designed to determine the physical and chemical description on each increment sample from three selected grid holes. This laboratory work identified metal distribution in each screen fraction, pH variation as per depth of each grid hole and the element analyses of the supernatant from a water leach of a three grid hole composite.

The screen analyses of each increment tested varied and some samples did indicate a mineral enrichment in the fine fraction. However this factor could not be used as a pre-concentration step for the reprocessing feed because the indication was not universal. The entire tailings would have to be reprocessed.

c) Lakefield Research Testwork

A composite of three grid hole (A2, B2, C2) increments was prepared at Faro and sent to Lakefield for preliminary laboratory flotation testwork. The composite assayed 0.79% Pb, 1.23% Zn, 0.15% Cu, 32.3% Fe, 0.11 g/t Au and 15.2 g/t Ag. The paste pH of this composite was 4.0 to 5.2 which meant an acidic slurry would be present so lime would be required to neutralize.

Mineralogy on the composite sample indicated that some of the sphalerite and galena present in the 20 micron size was liberated while the rest of the mineral worth was in a pyrite-sphalerite-galena middling which will be difficult to recover. The presence of oxidation coatings of sulphates and/or carbonates were observed on some liberated grains of galena. These coatings are detrimental to flotation recovery.

The Lakefield program subjected the composite to differential and bulk flotation procedures on total tailings and sand/fines fractions of tailings. Batch flotation tests on total tailings with a selective lead and zinc concentrate flowsheet produced a lead concentrate assaying 18% lead at a 6% recovery and a zinc concentrate assaying 28% zinc at a 14% recovery. The reagent consumption (not optimized) was very high therefore reagent costs exceeded metal values so the selective flotation approach was

abandoned. Batch flotation tests on the sand and fines fractions of the composite tailings sample were run on a selective lead and zinc concentrate flowsheet. The fines fraction (-43 microns) produced flotation results similar to the total tailings; while the sand fraction (+43 microns) gave minimal metal upgrade or recovery. Reagent consumptions were high so the sand/fines approach was also abandoned.

In the next series of preliminary batch flotation tests at Lakefield the flowsheet and reagent scheme were modified. The total tailings was used in the evaluation of several different collectors to produce a bulk concentrate. The reagents tested (modified R242, AX350 and modified R400) are experimental but can be purchased. The test results were not encouraging at the extremely low reagent consumptions tested so ongoing testwork will investigate increased activator and collector rates.

d) Faro test work

In a second campaign, preliminary batch flotation tests were run in the Faro metallurgical laboratory on a tailings composite from grid hole B7. The test program was designed to produce a bulk concentrate from the total tailings with a simplified Faro reagent scheme. However a preliminary batch flotation test did produce a rougher bulk concentrate assaying 1.7% lead and 7.2% zinc which was upgraded one stage to 2.2% lead and 12.9% zinc at a calculated recovery of 24.5% and 63% respectively. If consideration is given to the mineralogical description of the tailings and the small flotation product weights that were subjected to cleaner flotation, then a marketable bulk concentrate should be possible. Ongoing testwork will optimize concentrate grade and metal recovery.

REPROCESS FLOWSHEET

The hydro-monitored tailings from the Down Valley operation will be pumped to a slurry surge tank (converted fine ore bin) in the mill. Lime will be added to the slurry to raise the pH, then an activator, collector and frother will be added as it is pumped to the feed boxes of six banks of OK38 flotation cells where a rougher-scavenger bulk concentrate will be recovered and the tailings will be pumped to the mined out Faro open pit. The rougher-scavenger concentrate will be subjected to regrinding to further liberate the galena and sphalerite from the pyrite and gangue constituents; then conditioned with the same suite of reagents and subjected to three stages of cleaning. The bulk concentrate from each stage advances to the next, and on through to the third stage where it is final concentrate. The tailings from each cleaner stage, recycles to the feed of the previous stage except the first cleaner tailings which reports directly to the final reprocessing tailings. The final bulk concentrate will be thickened, filtered and thermal dried ready for truck and ship transport to an ISF smelter. All process make-up water will come from the open reclaim water siphon system. Fresh water will be used for reagent mixing. The equipment essential for the reprocessing flowsheet that is in the existing Faro concentrator, will be available for reuse. The flowsheet is shown on figure 2.

EXTRAPOLATED METALLURGICAL BALANCE

The metallurgical balance was extrapolated from preliminary test data, a review of the sample's mineralogy and discussions with laboratory metallurgists (Lakefield, Faro, consultants). As discussed above, further testwork is planned to demonstrate that these results are achievable.

Product	% wt	Assay			Recovery		
		Pb%	Zn%	Ag g/t	Pb%	Zn%	Ag g/t
Bulk concentrate	1.0	14.2	37.1	165	18.0	30.1	11.0
Tailings	99.0	0.65	0.87	13.5	82.0	69.9	89.0
Call. headgrade	100.0	0.79	1.23	15.0	100.0	100.0	100.0

OPERATING CRITERIA

Operating Schedule	hours per day	24
	months per year	6
	shift schedule	2-12
Operating Rate	plant availability	90%
	dry tonnes tailings per hour	1234
	per year	4,800,000
	forecast dry tonnes concentrate per year	48,000

Mobilization and Demobilization - a crew comprising operators and contractors will mobile the equipment before the May 1st start up and they will winterize the equipment after operations cease in the fall.

Rehabilitation - the rehabilitation and grooming of the tailings pond area that is free of tailings will be an ongoing operation.

Operating Crews - The operating crew will be comprised of experienced, perhaps retired managers (mill superintendent, hydromonitoring supervisor, general maintenance foreman, accountant and safety personnel/environment co-ordinator) with the rest of the crews being a mix of former and new, local employees. The crew distribution for reprocessing is as follows:

Management/supervision		9
Operators	}	
Tradesmen	}	20
Surface	}	
Total		29

Caretaker - a couple of employees will be retained on the payroll at the end of each operating season to caretaker the minesite (buildings and equipment). The caretakers will be required to make daily inspections of the minesite.

Electrical power - The electric power contract will have to be negotiated on a seasonal rate (summer usage only) which should be advantageous to the utility company; therefore reasonable rates.

Environmental Monitoring - sampling will be done in the winter by the caretakers and in the summer by a designated operations staff person. During the periods of more intense sampling assistance will be organized by the Curragh Resources Inc. regional office. All data will be collected, recorded and interpreted at the regional office.

CAPITAL COST ESTIMATES

Several modifications will have to be made to the existing mill circuit to accommodate the reprocessing of Down Valley tailings and the pumping of the mill tailings to the mined out Faro open pit for final deposition. The following capital cost estimate does not include the second, tailings pipeline and pumping station or the second reclaim water siphon pipeline from the open pit to the mill, these capital costs are included in the Down Valley hydromonitoring costs prepared by Kilborn:

Rubber line and modify the fine ore bin	$ 30,000
Agitator for the bin	$150,000
Pump bases	$ 3,000
Pipeline modifications	$ 10,000
Refurbish allowance	$250,000
Contingency	$ 57,000
	$500,000

OPERATING COST

The operating costs is based on existing mill cost data and the reprocessing design criteria. Backup detail for the following operating costs is appended:

Area	$/tonne of tailings
Manpower	0.20
Reagents	1.28
Supplies	0.10
Power	0.31
Sub total	$1.89
Hydromonitoring and pumping*	0.39
Total	$2.28 per tonne of tailings

* Detail in the Kilborn report on hydromonitoring and pumping to the mill

BULK CONCENTRATE "NET BACK" VALUE AT THE MINESITE

The production of a bulk concentrate is the recommended process route to follow however the marketing is not as favourable as marketing selective concentrates. Curragh's marketing department believes that 48,000 tonnes per year of bulk concentrate can be marketed.

The following calculated "net back" value to the minesite is based on a projected ISF smelter treatment charge and transportation/marketing cost. The standard ISF smelter contract with a zinc payment of 85% (minimum deduction 7 units) of the contained zinc, a lead payment of 95% (minimum deduction 3 units) of contained lead and a silver payment of 95% (minimum deduction 75 grams) of contained silver. There is no payment for copper. The forecast metal prices used: zinc at $0.60 US per pound, lead at $0.30 US per pound and silver at $4.50 US per ounce. A rate of exchange of 1.14 Canadian was used in the calculation. Details of the calculation are appended.

Bulk Concentrate "Net Back" value to the Mine per tonne of Bulk Concentrate:

Metal Payments	Zinc	$398.40 US	
	Lead	$ 74.09 US	
	Silver	$ 13.02 US	
	Total Metal Payment $485.51 US		$485.51 US
Charges	- smelter treatment rate	$217.30 US	
	- shipping/marketing costs	$ 69.20 US	
	Total Treatment Costs	$286.50 US	(286.50) US
	"Net Back" to the Minesite		$199.01 US

"Net Back" value per tonne of tailings = $1.99 US per tonne tailings
at the exchange rate (1.14) = $2.27 Cdn per tonne tailings

CURRAGH RESOURCES INC.
FARO DIVISION

TAILINGS REPROCESSING
FLOWSHEET

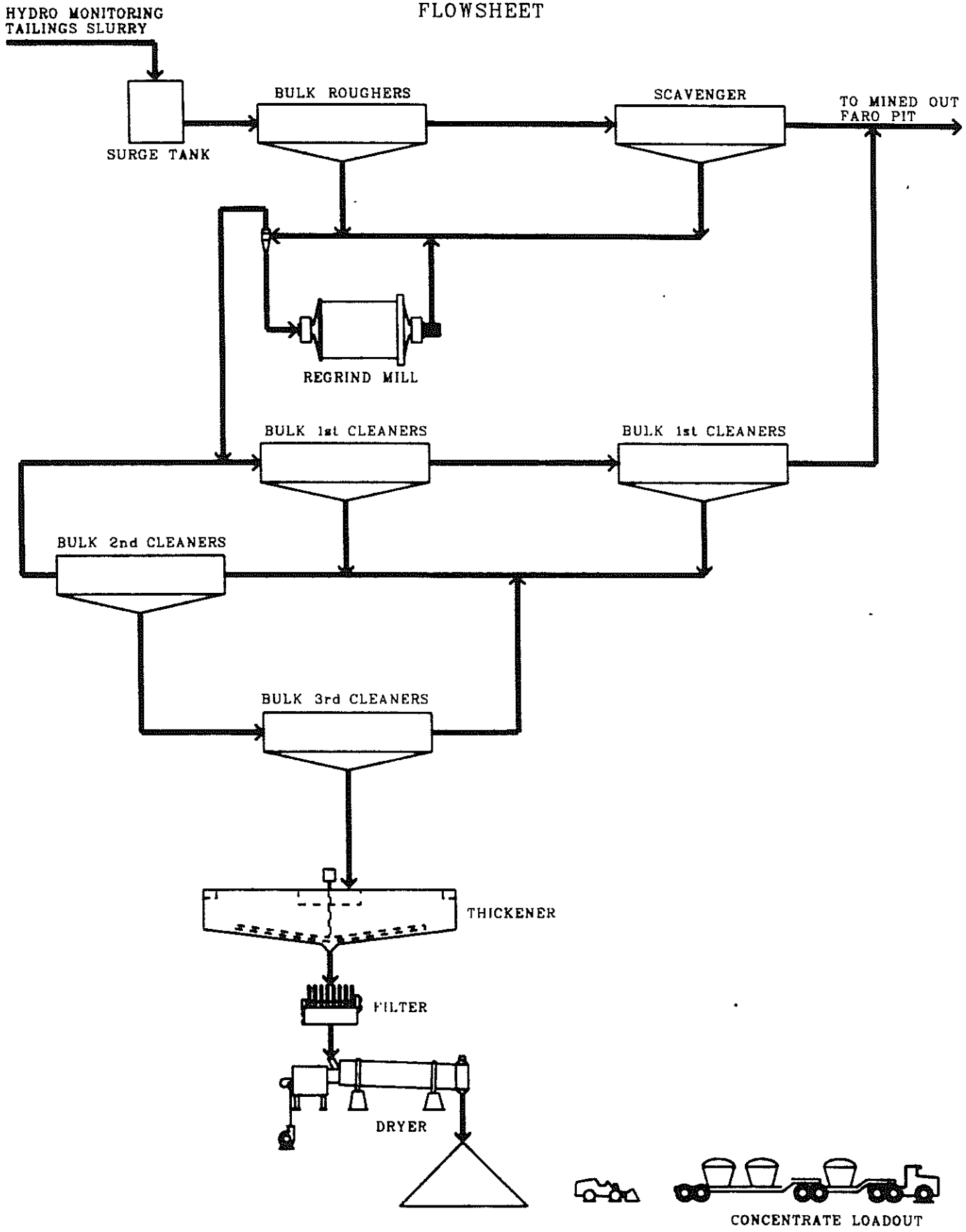


FIGURE 2

APPENDIX A

OPERATING COST DETAIL

CURRAGH RESOURCES INC.
FARO DIVISION
TAILINGS REPROCESSING

OPERATING COST

1)	Flotation/Reagent System*	$/year
	Consumables operating	$ 20,000
	Lubes	$ 12,000
	Consumables Maintenance	$ 32,000
	Mechanical Parts	$150,000
	Electrical Parts	$ 50,000
	Tools	$ 4,000
	Process Control	$ 30,000
	Subtotal	$298,000
2)	Dewatering/Drying*	
	Consumables Operating	$ 1,000
	Lubes and Fuel	$ 40,000
	Consumables Maintenance	$ 1,000
	Mechanical Parts	$ 6,000
	Electrical Parts	$ 4,000
	Subtotal	$ 52,000
3)	Tailings Deposition*	
	Mechanical Parts	$ 30,000
	Electrical Parts	$ 5,000
	Subtotal	$ 35,000

* The current Faro operating budget was used as the base for calculating the operating cost in these compatible areas.

4)	Water - potable	$ 2,000
	- reclaim	$ 1,000
	Building Maintenance	$ 30,000
	Support Equipment	$ 20,000
	Assay/Metallurgical supplies	$ 11,000
	Office Supplies	$ 11,000
	Employment related expenses	$ 13,000
	Subtotal	$ 88,000
5)	Power Consumption	
	Electrical Power	$1,500,000
	Subtotal	$1,500,000
6)	Mobilize/Demobilize**	
	Allowance	$ 200,000
	Subtotal	$ 200,000
7)	Caretaker (during non-operating time)	
	Allowance	$ 100,000
	Subtotal	$ 100,000
8)	Manpower	
	Supervision/staff	$ 177,000
	Operations	$ 329,000
	Burden (at 35%)	$ 176,000
	Subtotal	$ 682,000

** Operating cost assumes one month with full hydromonitoring and milling crew or two months at half the workforce.

REAGENT CONSUMPTION PER TONNE OF TAILINGS

Reagent	kg/t	$/kg	$/t
Lime	6.0	0.10	0.60
$CuSO_4$	0.24	1.595	0.38
Dowfroth	0.03	1.80	0.05
PAX	0.04	1.64	0.06
SD200	0.10	1.95	0.19
Regrind Media	0.015	0.67	0.01
			1.29

Supervision and Staff	No.	$/person
Mill Superintendent	1	55,000
Metallurgist	--	
General Maintenance Foreman	1	48,000
Shift bosses	--	
Assayers	1	40,000
Surveyors	1	40,000
Accountants	1	40,000
Warehouse persons	1	40,000
Safety/personnel }		
Environmental person }	1	35,000
Metallurgical Technician	1	35,000
Clerk	1	25,000
Subtotal	9	353,000
Operations		
Mill Operators	12	400,000
Tradesmen	5	175,000
Surface	3	82,000
Subtotal	20	657,000
Subtotals	29	1,010,000
Burden (35%)		353,000
TOTAL		1,363,000 for 12 month operation

However this operation is 6 months = $682,000

APPENDIX B

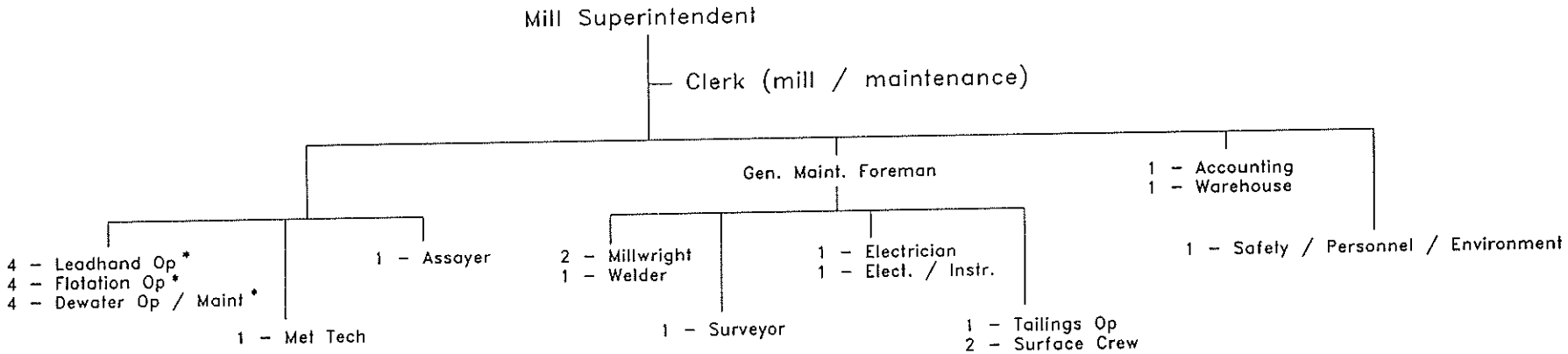
TAILINGS REPROCESSING ORGANIZATION CHART

APPENDIX B

Curragh Resources Inc.

FARO DIVISION

TAILINGS REPROCESSING



* Shift Work

OPERATIONS – 800,000 tonnes per day
24 hours per day
6 months per year

CONTRACTORS – Dam construction
Rehabilitation of original tailing site
Maintenance of the highway from Faro to the minesite
Hauling the concentrate to Skagway and shipping

APPENDIX C

"NET BACK" CALCULATION DETAILS

APPENDIX C

CURRAGH RESOURCES INC.
FARO DIVISION
TAILING REPROCESSING

Bulk Concentrate Production

- concentrate grade and recovery extrapolated from preliminary laboratory testwork at Lakefield Research and Faro
- plan to reprocess 800,000 tonnes of tailing per season
- tailing headgrade Pb 0.79%
  Zn 1.23%
  Ag 15g/t
- Bulk Concentrate Grade Pb 14.2%
  Zn 37.1%
  Ag 165g/t
- Calculated "Net Back" value to the Mine of 1 tonne of Bulk Concentrate.

a) Metal Payment

Pb Value = (2205 x 0.142) - (2205 x 0.03) = 247 lbs. payable Pb
= (247 x $0.30) = $74.09 US

Ag Value = (1 x 165) - (1 x 75) = 90g payable Ag
= (90÷31.1)($4.50) = $13.02 US

Zn Value = (2205 x 0.371) - (2205 x 0.07) = 664 lbs. payable Zn
= (664 x 0.60) = $398.40 US

TOTAL METAL VALUE =$485.51 US

b) Charges

Smelter Treatment Rate	$217.30 US
Shipping and Marketing Costs	$ 69.20 US
TOTAL COSTS	$286.50 US

c) "Net Back" Value to the Mine

Metal Value	= $485.51 US	
Charges	= $286.50 US	
Net Back	= $199.01 US	= $1.99 US/t of tailing
		= $2.27 Cdn/t of tailing

Metal Prices
Pb $0.30 US/lb.
Zn $0.60 US/lb.
Ag $4.50 US/oz
Exchange $1.14 Cdn = $1 US

List of ISF Smelters
31-Oct-91

Smelter	Country	Expected Annual Production Zinc Metal (1000 tonnes) 1990	2000	Comments
C Creek	Australia	73	82	
N Godault	France	89	91	
Berzelius	Germany	73	100	
Chanderiya	India	0	66	New Smelter starting 1991
P Vesme	Italy	70	75	
Hachinohe	Japan	94	100	
Harima	Japan	74	91	
Tomakoma	Japan	0	100	New Smelter for Mid 90's
New ISF	Thailand	0	70	New Smelter for Mid 90's
Avonmouth	UK	93	100	
Titov Veles	Yugoslavia	50	47	
		616	922	

Notes

1. List does not include 3 ISF smelters (Eastern Europe-2 ; Zaire-1)

2. ISF smelters produce both zinc and lead metal and take their feed from lead, zinc and bulk concentrates as well as a small amount of secondary material. Currently ISF purchase 500,000 tonnes of bulk.

3. The main producers of bulk concentrate are Brunswick Mining, Mt Isa, Greens Creek, Hellyer, Brocal, Red Dog and Mae Tong.

4. If possible mines avoid producing bulk concentrate due to lower economic returns.

Note:

Faro Bulk Concentrate= 18,000 tpa Zinc metal
= 2 to 3% of World ISF capacity

A. Long Term Terms
1. Zinc long term treatment charges are $175.00 at $1000 per tonne zinc. Escalation at $.10 per dollar. Zinc is paid at 85% (MD 8 units).

2. Lead long term treatment charges are $145.00 at $.28 per lb. Escallation is $.125 per dollar change in price. Gold is paid at rate of 95% (md 1 gram) and silver is paid at rate of 95% (md 35 grams).

3. Bulk Treatment charges are zinc treatment charges plus $10.00. Zinc payment is 85% (md 7 units) and lead payment is 95% (md 3 units). Silver is paid at rate of 95% (md 75 grams). No payment for copper.

B. Long term prices
1. The 20 year average of zinc prices in 1991 dollars is $.60 US per lb. A good estimate for lead is $.30 per lb. Silver price is expected to be $4.50 per oz.

C. Shipping costs are estimated at $20.00 US/dmt for ocean freight

Bulk Concentrate Treatment Charge:

Zn @ $0.60 US/lb. = 2205 x 0.60	=$1323 US/tonne
Treatment charge	=175 US/tonne
Escalator = (1323 - 1000) (0.10)	=32.30 US/tonne
+ $10/t for Bulk Conc	=10.00 US/tonne
TOTAL	=$217.30 US/tonne

APPENDIX D

FARO METALLURGICAL LAB
BULK FLOTATION TESTWORK

CURRAGH RESOURCES INC.
FARO DIVISION

TAILINGS POND (BULK) FLOTATION

Faro tests were undertaken on sample point B7.

1. Overview

A. Faro has tried a "bulk" route, and as a consequence have produced a zinc concentrate with under 3% Pb, and 25.97% Zn recovery (early tests only).

B. Results

Zn Rougher Grade, %	9.52
Zn Recovery, %	69.05
First Cleaner Grade (Zn)	18.80
Overall Recovery, %	25.97
*Silver in Cleaner, g/t	244

* Note the favourable silver assay

C. Flowsheet

Flowsheet	Bulk
Processed	Unscreened
	First stage rough/cleaning

D. Reagents

Lime, g/t	1000 - 4080
SD-200, g/t	150
NaCN	--
3148A	--

Faro used lower pH at rougher and depressed pyrite in cleaners at higher pH.

As a next step Faro would accept the reasonable rougher that has been obtained, but would focus on better cleaning and higher recovery (at cleaning).

2. Detail

Three tests have been performed on unscreened tailings pond:

a) Test FTA-2 rougher flotation, at low pH (6.0) and low reagent consumption (lime = 1000 g/t, $CuSO_4$ = 250 g/t, PAX = 30 g/t, and Dow 250 = 24 g/t).

b) Rougher flotation with one stage of cleaning.
FTA-3 - with lime (4080 g/t) and pH = 10 - 11

c) As "b" plus:
FTA-4 - with lime (2700 g/t) and pH = 10 - 11 and SD-200 (150 g/t) for pyrite depression

- results shown in Table #1 (FTA-2 and FTA-4).
- good zinc rougher concentrate grade (9.52% Zn) and recovery (69.05%) has been obtained.
- acceptable zinc cleaner concentrate (18.80% Zn, 2.71% Pb and 244 g/t Ag) with 26% Zn recovery has been obtained.
- scavenger concentrate has 5.9% Zn, 1.64% Pb, 127 g/t Ag and 6.15% recovery of Zn.

The table attached "Tailings Pond Testing" gives the main results from Faro.

Lastly the specific results of FTA-2, FTA-3 and FTA-4 are attached.

TAILINGS POND TESTING

Flotation Feed	Unscreened
pH - rougher	Low (6.0)
- cleaner	10 - 11

Reagents g/t

Soda Ash	None
Lime	1000 - 4080
NaCN	None
SD-200	150
$CuSO_4$	250
NOTE - 3418A	None
Xanthate	30
MIBC	None
Dowfroth 250	24
Total Reagents	1464 - 4644 g/t

Flotation Test

Rougher flotation

Zn Grade, %	9.52
Zn Recovery, %	69.05

Cleaner - First stage

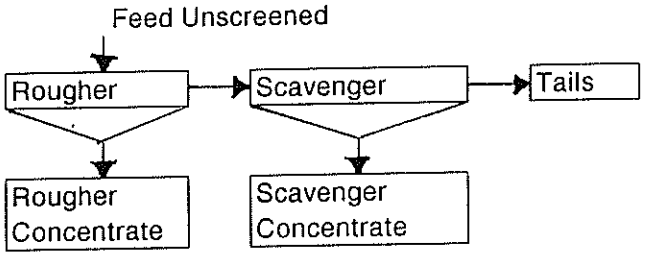
Zn Grade, %	18.8% Zn - 244 g/t Ag
Zn Recovery, %	25.97

Cleaner - Second stage

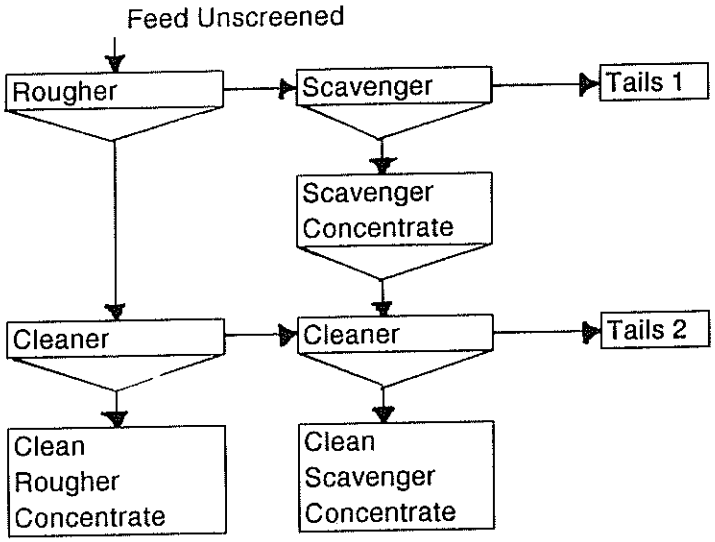
Zn Grade, %	Not done
Zn Recovery, %	Not done

TAILINGS POND – FLOTATION FLOWSHEET
Faro – Metallurgical Lab

Test FTA–2



Tests FTA–3 FTA–4



Test # FTA-2 Test Date: 9-10-91

			ASSAYS			DISTRIBUTION		
	WEIGHT	WT %	% Lead	% Zinc	% Iron	% Lead	% Zinc	% Iron
Feed (Calc)	434.45	100.00	0.73	1.39	24.15	100.00	100.00	100.00
Rougher Conc	43.70	10.06	2.11	9.52	30.10	29.04	69.06	12.54
Scavenger Conc	40.59	9.34	0.98	1.66	34.60	12.53	11.18	13.38
Tails	350.16	80.60	0.53	0.34	22.20	58.43	19.76	74.08

Test # FTA-3 Test Date: 28-10-91

Test Purpose: Pyrite Depression at Higher Rougher and Cleaner pH with no regrinding

			ASSAYS				DISTRIBUTION			
	WEIGHT	WT %	% Lead	% Zinc	% Iron	Ag g/t	% Lead	% Zinc	% Iron	% Silver
Feed (Calc)	440.66	100.00	0.69	1.31	24.17	4.61	100.00	100.00	100.00	100.00
1st Cleaner Conc	25.94	5.89	2.15	12.90	31.40	50.00	18.38	57.92	7.65	63.78
1st Cleaner Tails	25.04	5.68	1.20	1.36	30.00	20.00	9.90	5.89	7.05	24.63
Rougher Conc (Calc)	50.98	11.57	1.68	7.23	30.71	35.26	28.28	63.81	14.70	88.41
Scavenger Conc	23.56	5.35	0.69	1.26	36.00	10.00	5.36	5.14	7.96	11.59
Tails	366.12	83.08	0.55	0.49	22.50	0.00	66.36	31.05	77.34	0.00

Test # FTA-4 Test Date: 31-10-91

Test Purpose: Pyrite Depression at Higher Rougher and Cleaner pH with no regrinding and using SD-200

			ASSAYS				DISTRIBUTION			
	WEIGHT	WT %	% Lead	% Zinc	% Iron	Ag g/t	% Lead	% Zinc	% Iron	% Silver
Feed (Calc)	527.92	100.00	0.58	1.12	29.82	15.00	100.00	100.00	100.00	100.00
1st Cleaner Conc	14.32	2.71	2.25	13.26	31.68	36.09	10.52	32.12	2.88	6.52
1st Cleaner Rougher	8.17	1.55	2.71	14.80	27.88	45.58	7.23	25.97	1.40	4.71
1st Cln Scavenger	6.15	1.16	1.64	5.91	37.90	23.40	3.29	6.15	1.48	1.81
1st Cln Tails	17.94	3.40	1.35	1.78	35.38	1.85	7.91	5.40	4.02	0.42
Rougher Conc (Calc)	32.26	6.11	1.75	6.88	33.69	17.04	18.43	37.52	6.90	6.94
Scavenger Conc	74.42	14.10	0.46	1.52	43.40	7.44	11.17	19.13	20.47	7.00
Tails	421.24	79.79	0.53	0.63	20.10	16.17	70.40	43.35	72.63	86.06

APPENDIX E

FARO METALLURGICAL LAB
SAMPLE CHARACTERIZATION

Tailings Pond

Abandonment Data

C U R R A G H R E S O U R C E S I N C.

Greg L. Rasmussen
Metallurgist

1991

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Sampling Setup

a. Equipment Needed

1. Backhoe	B. Dunn
2. Sample Bags	Warehouse
3. Sample Buckets	Warehouse

b. Locations

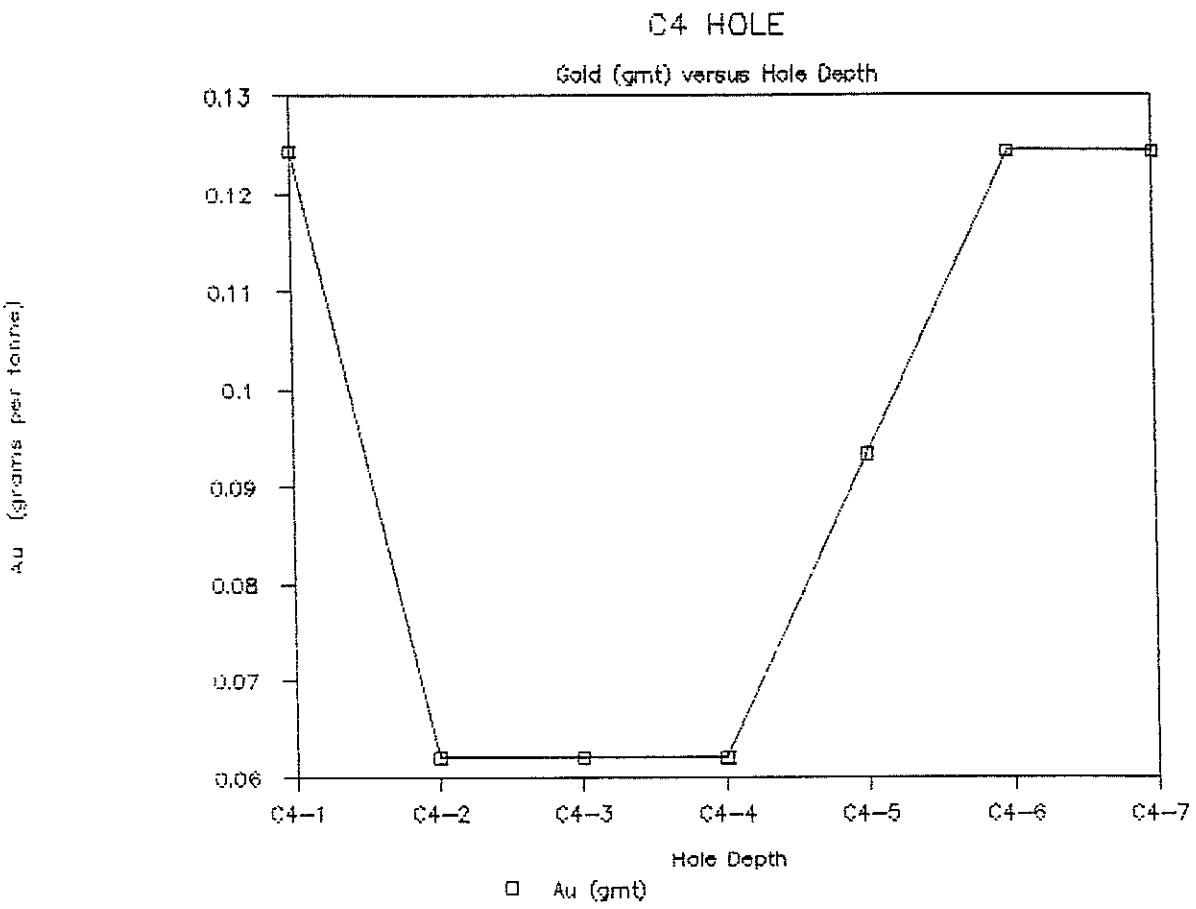
1. 19 Locations - 100 metres apart.
2. 3 Different Line A1 - A5, B1 - B7, & C1 - C7.
3. Take one sample every metre of depth, should be approx. 7 metres.
4. A picture of each hole is to be taken after the samples have been taken.

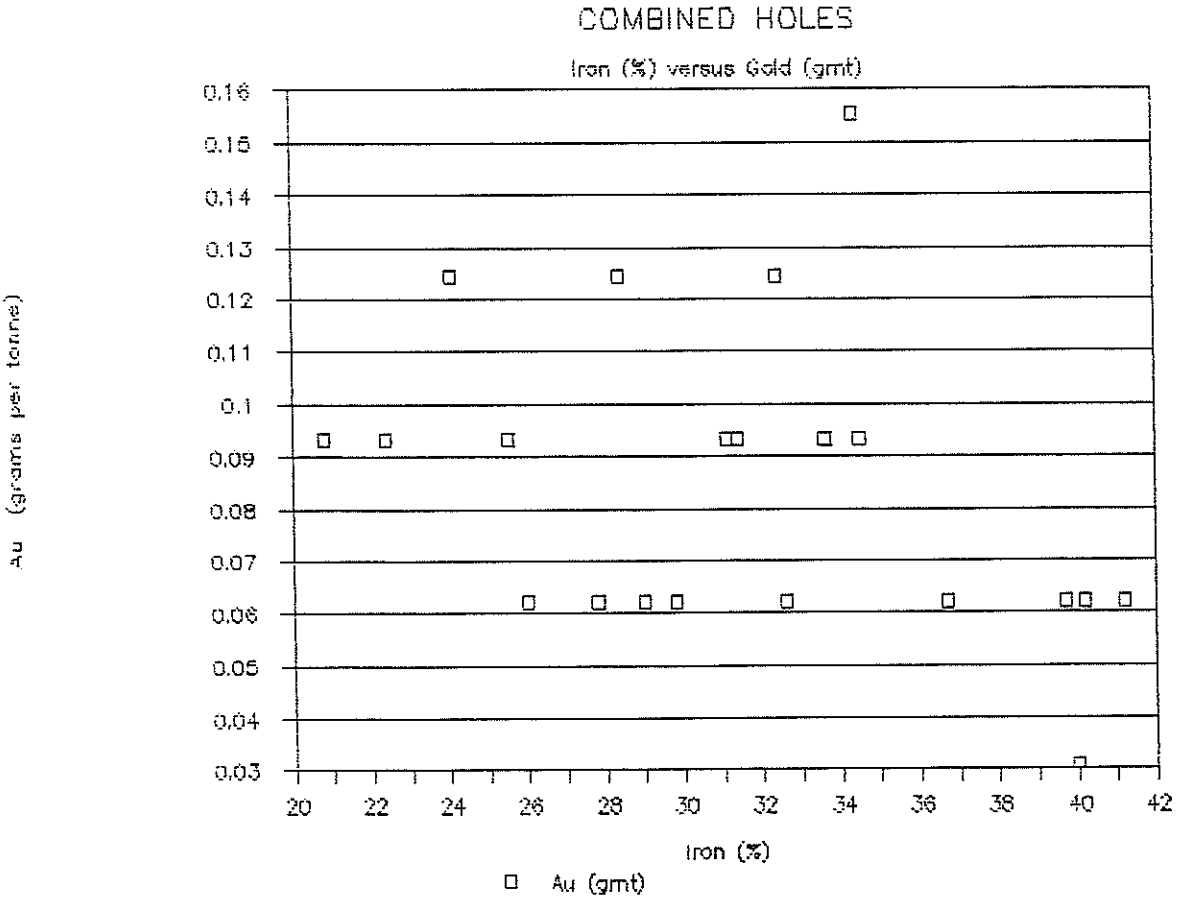
c. Sample Preparations

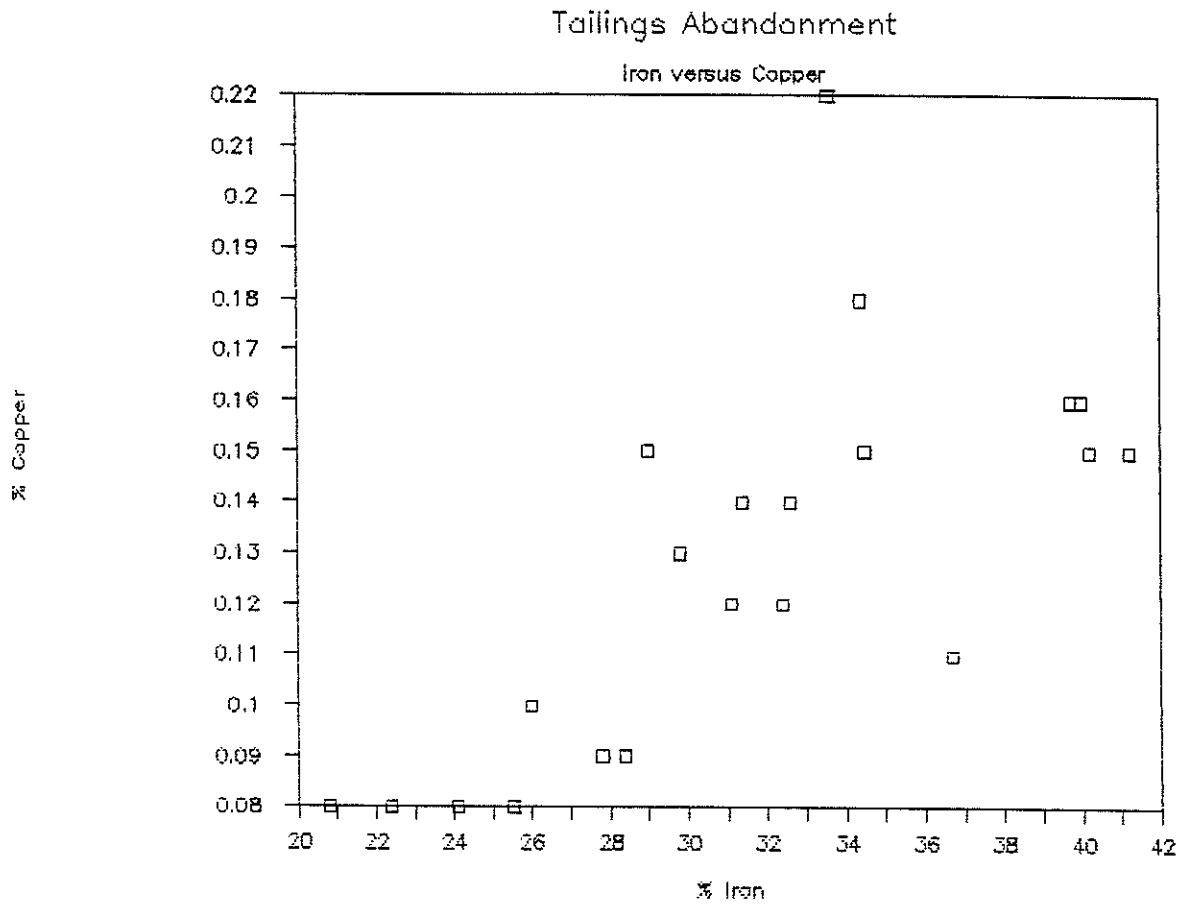
1. If supernate - filter and assay for Pb, Zn, Cu, Fe.
2. Take one hole from each line, basically at random, and take one cut for head assays, one cut for screening for 200# (75 microns) and 325# (45 microns), and one cut for pH.
3. Take approx 250 grams of sample from each meter of depth 100 gms for our assay lab to be assayed for Pb, Zn, Cu, Fe & Ag. 100 gms for Chemex to be fire assayed for Au. 50 gms to be used for natural pH.
4. Take another cut from each depth to be approx 200 grams. This is not to be dried but wet screened through 200 and 325 mesh screens. Each individual screen fraction is then to be assayed at our assay lab for Pb, Zn, Cu, Fe, and Ag.

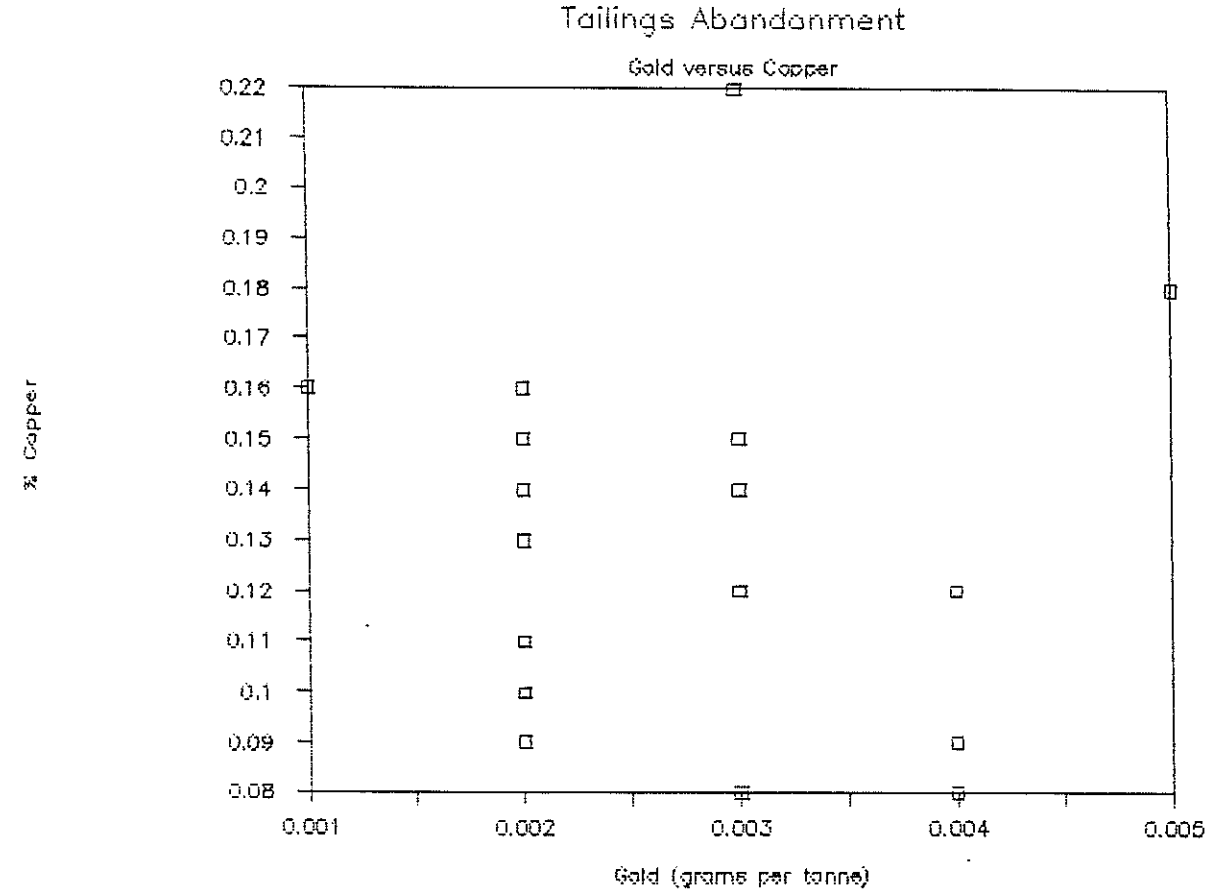
TABLE #1
Basic Hole Data

Hole	Weight (grams)			pH	ASSAYS						
Sample	+200	+325	-325		%Pb	%Zn	%Fe	%Cu	Ag gmt	Au oz/t	Au gmt
A3-1	14.61	46.11	94.06	2.69	0.60	1.57	29.0	0.15	15	0.002	0.062
A3-2	20.49	61.54	120.78	3.63	0.43	0.95	34.5	0.15	10	0.003	0.093
A3-3	22.59	64.65	131.09	3.42	0.58	1.19	34.4	0.18	20	0.005	0.156
A3-4	31.40	49.29	197.68	4.34	0.97	1.57	33.6	0.22	10	0.003	0.093
A3-5	131.20	79.89	82.71	5.19	0.65	1.26	40.0	0.16	20	0.001	0.031
A3-6	142.11	80.82	82.68	5.01	0.70	1.53	40.2	0.15	20	0.002	0.062
A3-7	133.28	91.08	71.75	5.36	0.60	1.20	41.2	0.15	10	0.002	0.062
B7-1	43.01	84.51	118.77	4.45	0.71	1.15	31.1	0.12	20	0.003	0.093
B7-2	61.86	53.15	133.89	4.53	0.76	1.44	20.8	0.08	20	0.003	0.093
B7-3	44.99	66.16	125.35	4.36	0.71	1.40	22.4	0.08	20	0.003	0.093
B7-4	178.97	49.96	37.78	3.91	0.44	1.19	39.7	0.16	10	0.002	0.062
B7-5	245.72	40.47	22.31	4.55	0.55	1.45	32.6	0.14	20	0.002	0.062
B7-6	216.27	42.89	43.56	4.32	0.59	1.48	31.4	0.14	20	0.003	0.093
B7-7	223.54	40.05	54.45	4.49	0.62	1.52	29.8	0.13	20	0.002	0.062
C4-1	121.26	38.71	75.71	4.69	0.65	1.56	32.4	0.12	20	0.004	0.124
C4-2	125.85	74.72	72.13	5.01	0.62	1.44	36.7	0.11	20	0.002	0.062
C4-3	163.62	67.12	43.56	5.45	0.54	1.30	27.8	0.09	10	0.002	0.062
" 4	51.03	67.10	172.61	5.31	0.69	1.18	26.0	0.10	20	0.002	0.062
" 5	96.30	42.31	59.72	4.96	0.59	1.18	25.5	0.08	20	0.003	0.093
C4-6	64.72	44.37	73.11	4.84	0.61	1.06	28.4	0.09	20	0.004	0.124
C4-7	85.53	41.99	34.25	5.04	0.56	1.13	24.1	0.08	20	0.004	0.124


C4 HOLE
Gold (gmt) versus Hole Depth
0.13
0.12
0.11
0.1
0.09
0.08
0.07
0.06
Au (grams per tonne)
C4-1
C4-2
C4-3
C4-4
C4-5
C4-6
C4-7
Hole Depth
Au (gmt)


COMBINED HOLES
Iron (%) versus Gold (gmt)
0.16
0.15
0.14
0.13
0.12
0.11
0.1
0.09
0.08
0.07
0.06
0.05
0.04
0.03
Au (grams per tonne)
20
22
24
26
28
30
32
34
36
38
40
42
Iron (%)
Au (gmt)


Tailings Abandonment
Iron versus Copper
% Copper
0.22
0.21
0.2
0.19
0.18
0.17
0.16
0.15
0.14
0.13
0.12
0.11
0.1
0.09
0.08
20
22
24
26
28
30
32
34
36
38
40
42
% Iron


Tailings Abandonment
Gold versus Copper
% Copper
0.22
0.21
0.2
0.19
0.18
0.17
0.16
0.15
0.14
0.13
0.12
0.11
0.1
0.09
0.08
0.001
0.002
0.003
0.004
0.005
Gold (grams per tonne)

Tailings Abandonment

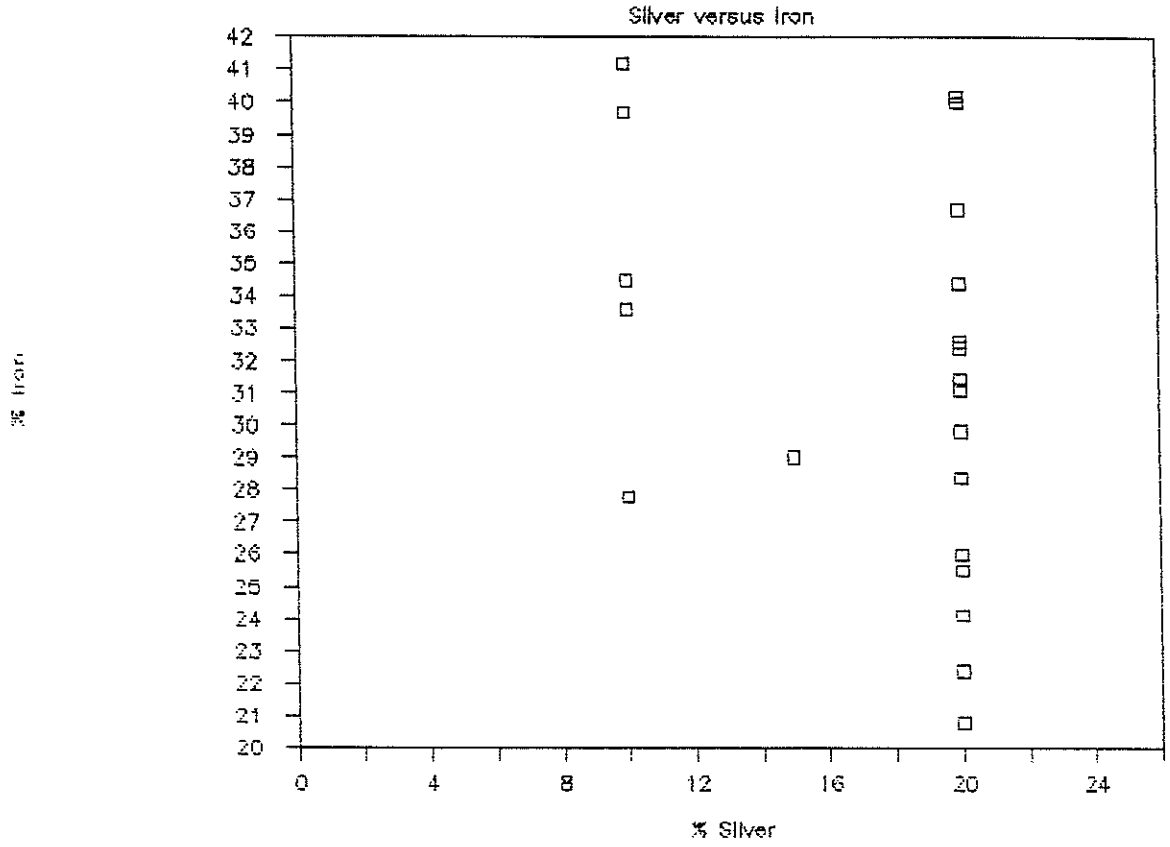
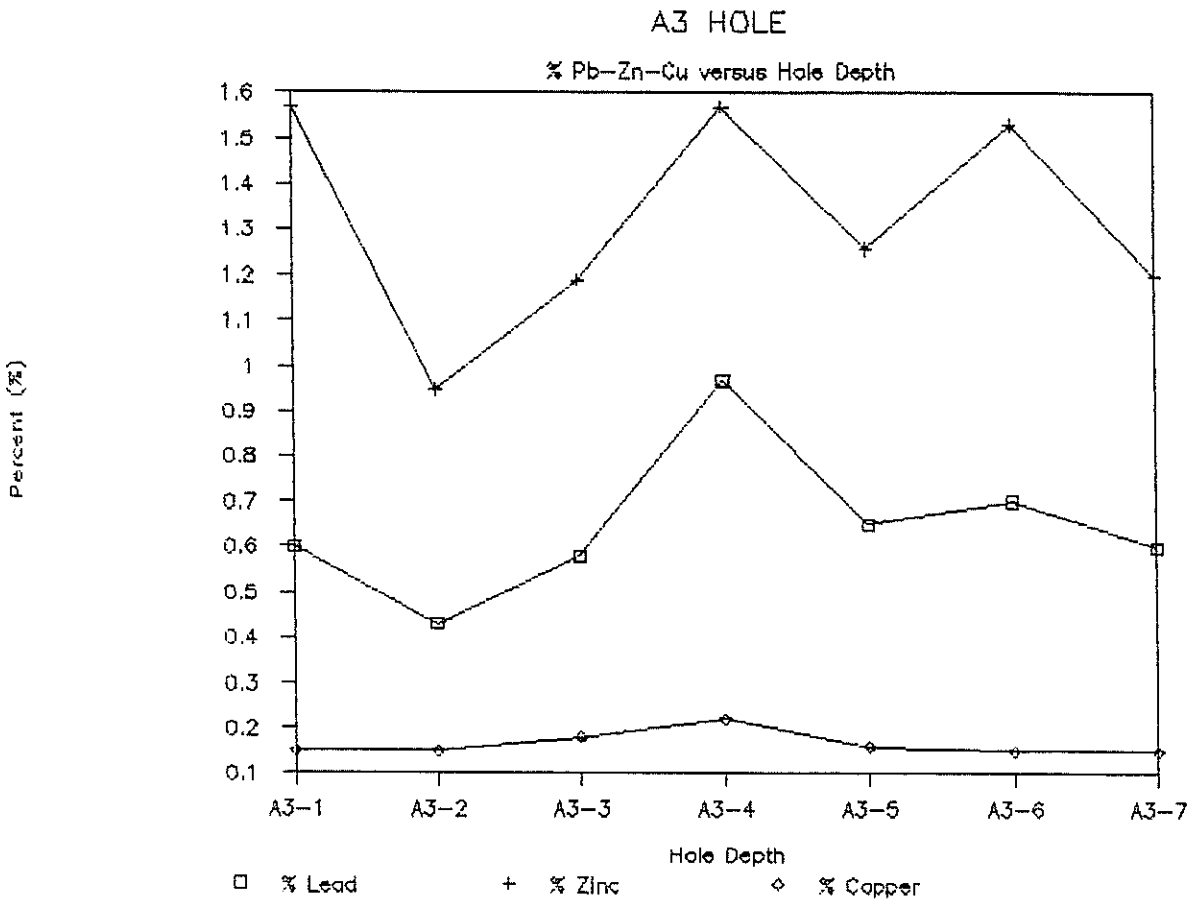
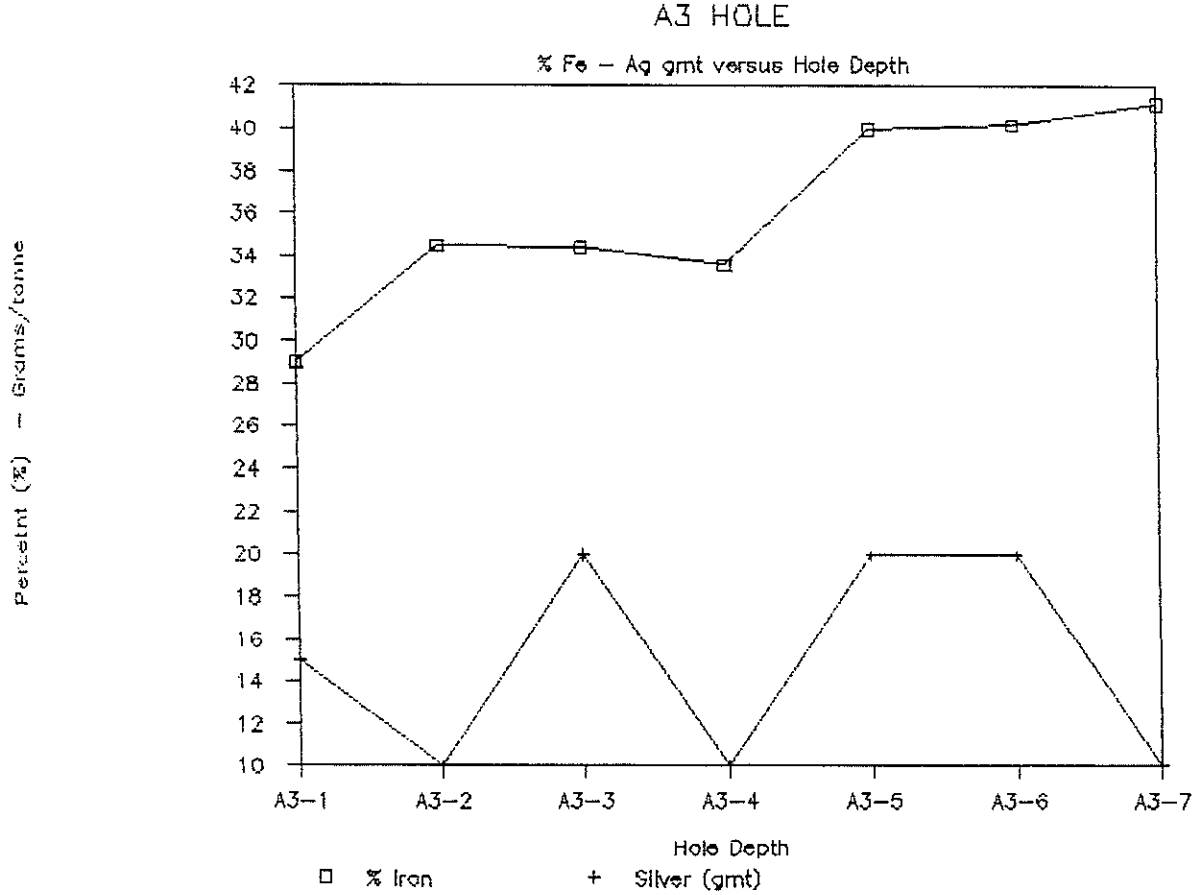
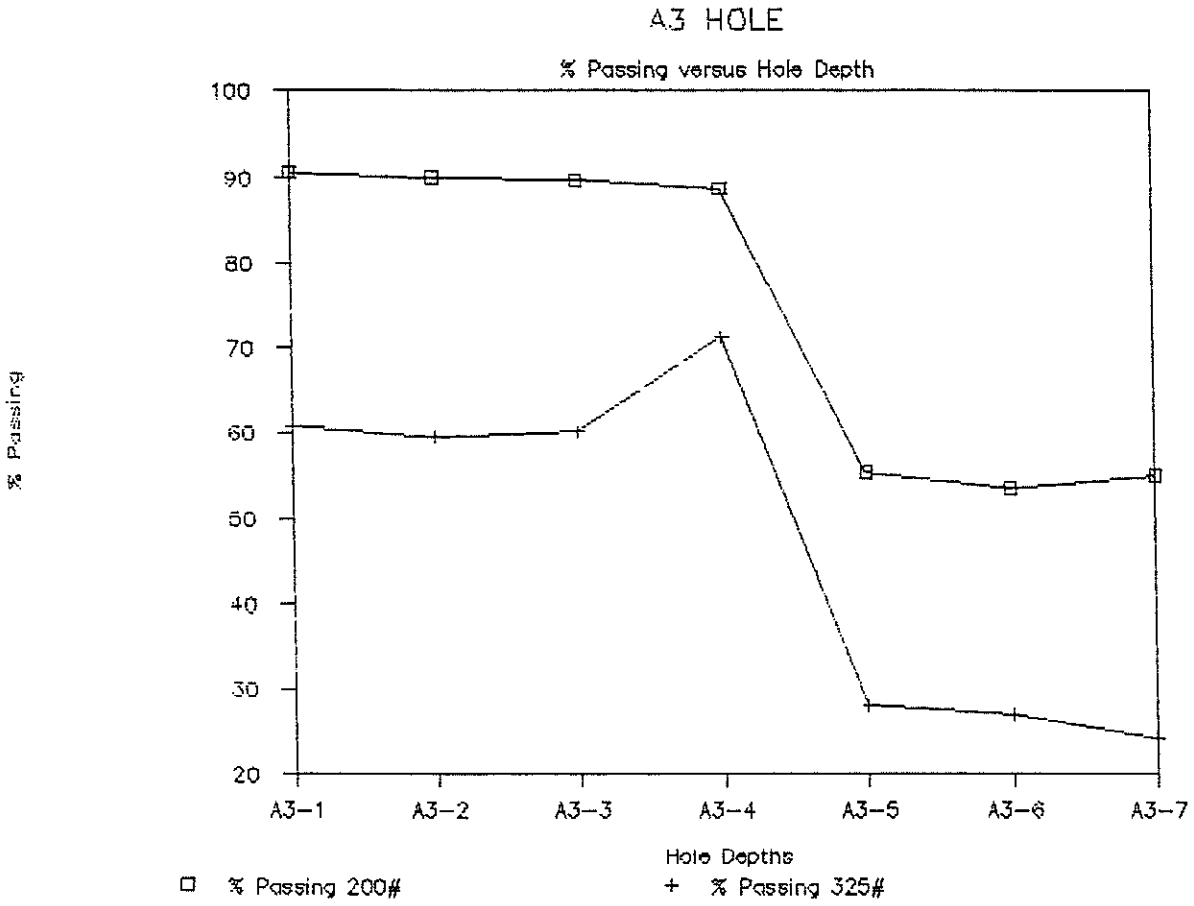


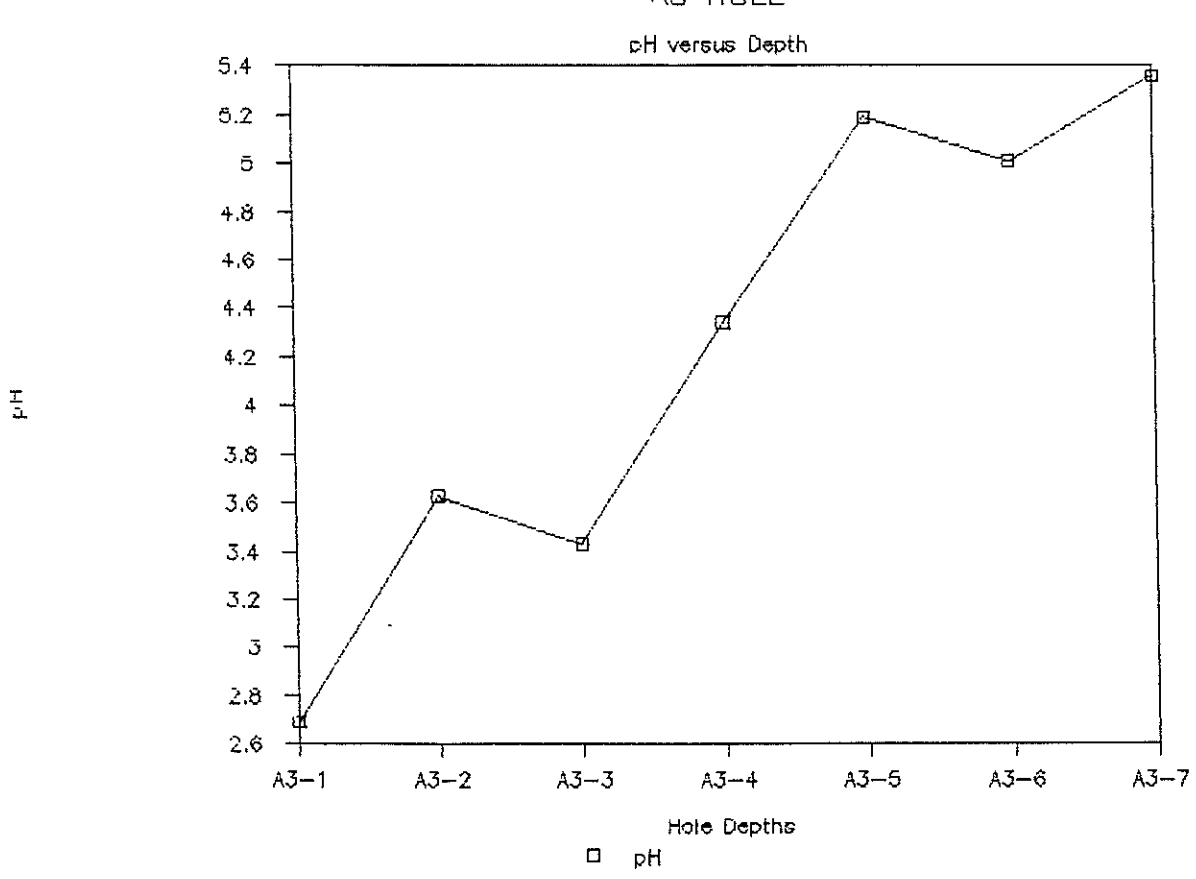
TABLE #2
Screen Analysis Results

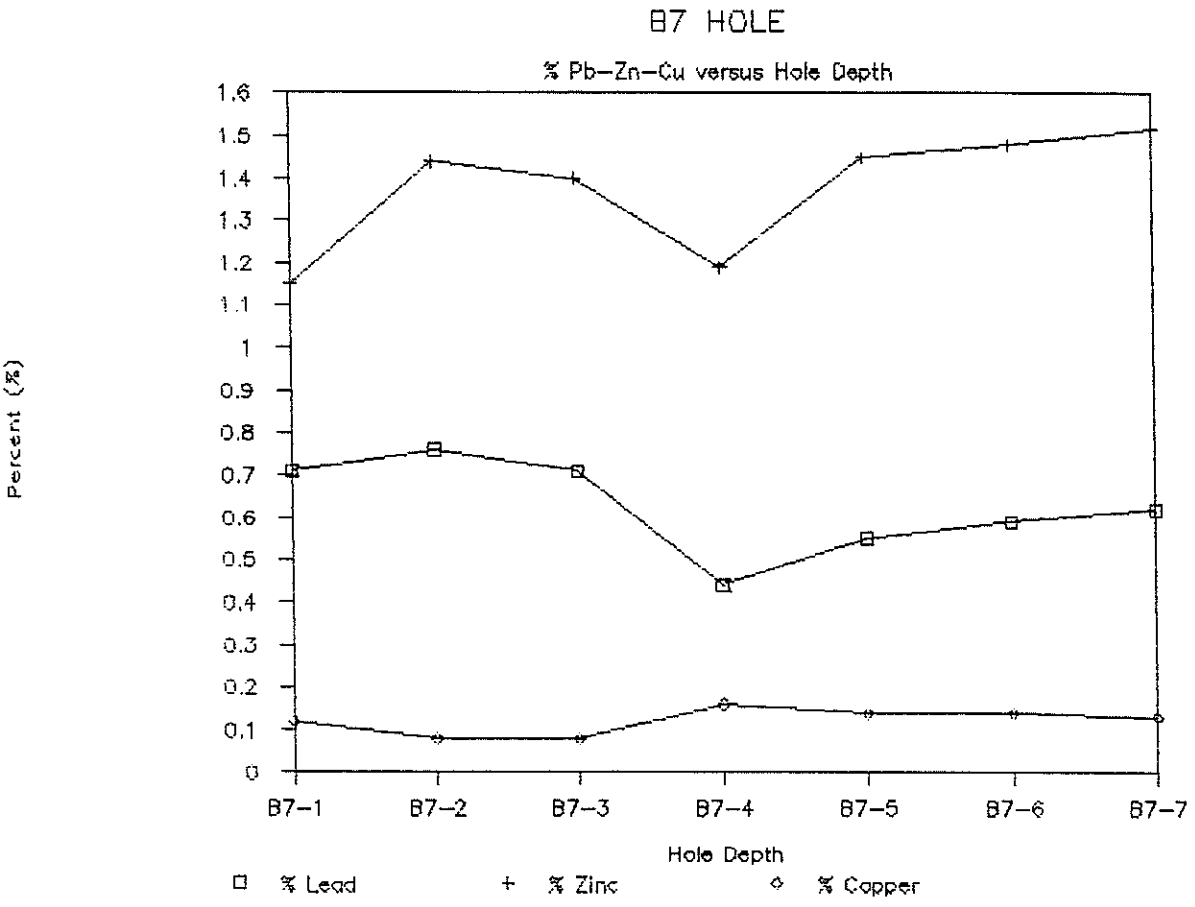
Sample	Weight (grams) +200	+325	-325	% +200	% +325	% -325	% Cumulative % -200	% -325
A3-1	14.61	46.11	94.06	9.44	29.79	60.77	90.56	60.77
A3-2	20.49	61.54	120.78	10.10	30.34	59.55	89.90	59.55
A3-3	22.58	64.65	131.98	10.30	29.49	60.21	89.70	60.21
A3-4	31.40	49.29	137.68	11.32	17.41	71.27	88.68	71.27
A3-5	131.20	79.89	82.71	44.66	27.19	28.15	55.34	28.15
A3-6	142.11	80.82	82.68	46.50	26.45	27.05	53.50	27.05
A3-7	133.28	91.08	71.75	45.01	30.76	24.23	54.99	24.23
B7-1	43.01	84.51	119.77	17.46	34.31	48.22	82.54	48.22
B7-2	61.86	53.15	133.89	24.86	21.35	53.79	75.15	53.79
B7-3	44.09	66.15	125.35	19.02	27.97	53.00	80.98	53.00
B7-4	178.97	49.96	37.78	67.10	18.73	14.17	32.90	14.17
B7-5	245.72	40.47	22.31	79.65	13.12	7.23	20.35	7.23
B7-6	216.27	42.89	43.56	71.44	14.17	14.39	28.56	14.39
B7-7	223.54	40.05	54.45	70.29	12.59	17.12	29.71	17.12
C4-1	121.26	88.71	75.71	42.45	31.05	26.50	57.55	26.50
C4-2	125.85	74.72	72.13	46.15	27.40	26.45	53.85	26.45
C4-3	163.62	67.12	43.56	59.65	24.47	15.88	40.35	15.88
C4-4	51.03	67.10	172.61	17.55	23.08	59.37	82.45	59.37
C4-5	96.30	42.31	59.72	48.56	21.33	30.11	51.44	30.11
C4-6	64.72	44.37	73.11	35.52	24.35	40.13	64.48	40.13
C4-7	85.53	41.99	34.25	52.37	25.96	21.17	47.13	21.17

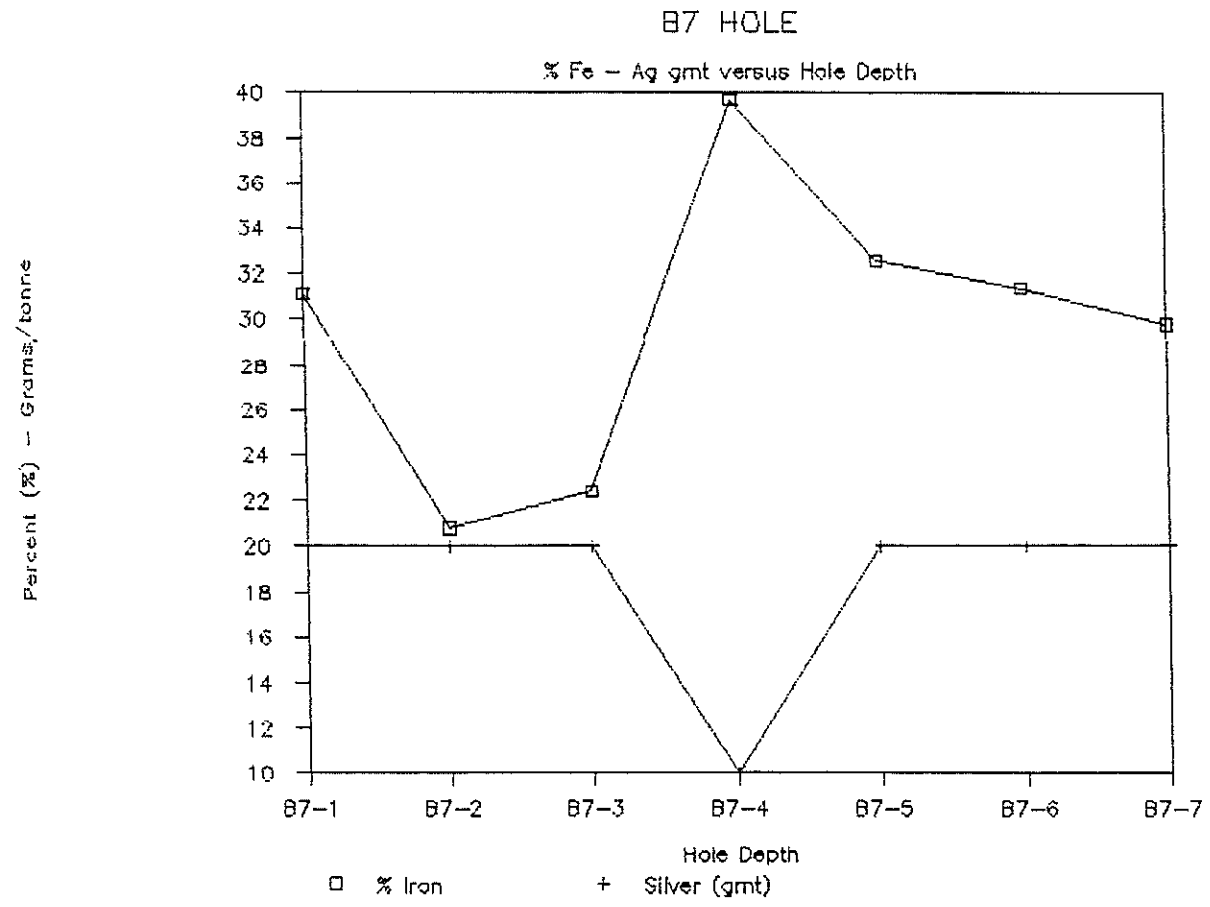

A3 HOLE
% Pb–Zn–Cu versus Hole Depth
Percent (%)
1.6
1.5
1.4
1.3
1.2
1.1
1
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
A3–1
A3–2
A3–3
A3–4
A3–5
A3–6
A3–7
Hole Depth
% Lead
% Zinc
% Copper

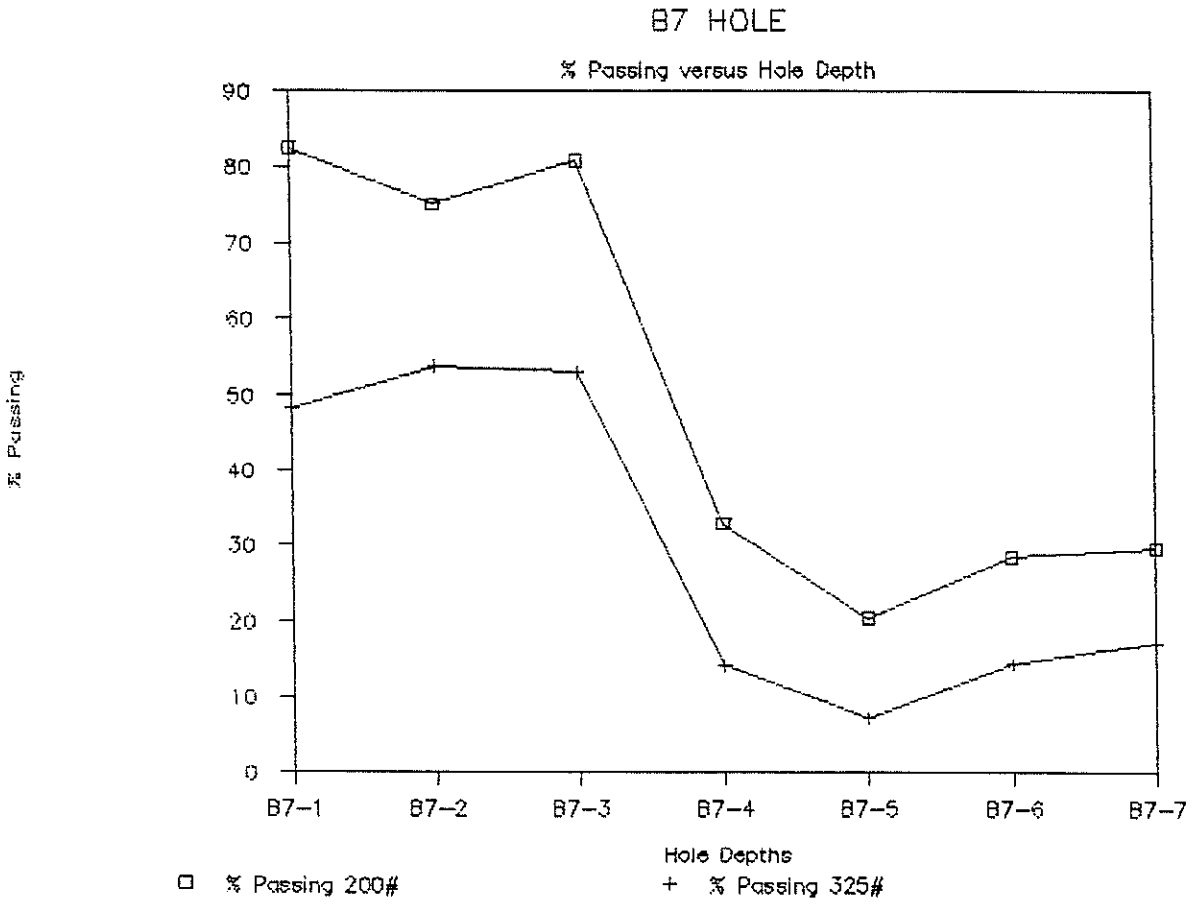

A3 HOLE
% Fe – Ag gmt versus Hole Depth
Percetnt (%) – Grams/tonne
42
40
38
36
34
32
30
28
26
24
22
20
18
16
14
12
10
A3–1
A3–2
A3–3
A3–4
A3–5
A3–6
A3–7
Hole Depth
% Iron
Silver (gmt)

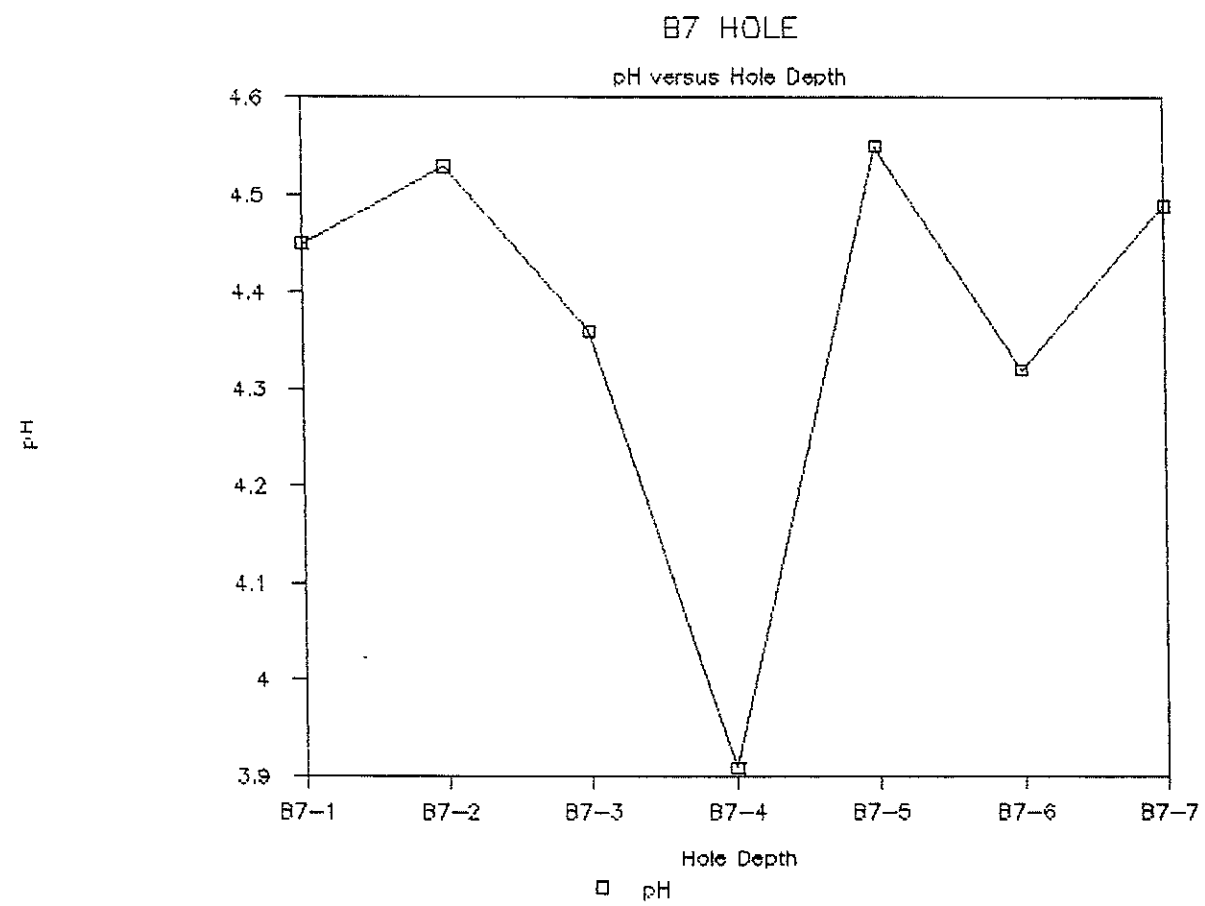

A3 HOLE
% Passing versus Hole Depth
% Passing
100
90
80
70
60
50
40
30
20
A3-1
A3-2
A3-3
A3-4
A3-5
A3-6
A3-7
Hole Depths
% Passing 200#
% Passing 325#


A3 HOLE
pH versus Depth
pH
5.4
5.2
5
4.8
4.6
4.4
4.2
4
3.8
3.6
3.4
3.2
3
2.8
2.6
A3-1
A3-2
A3-3
A3-4
A3-5
A3-6
A3-7
Hole Depths
pH

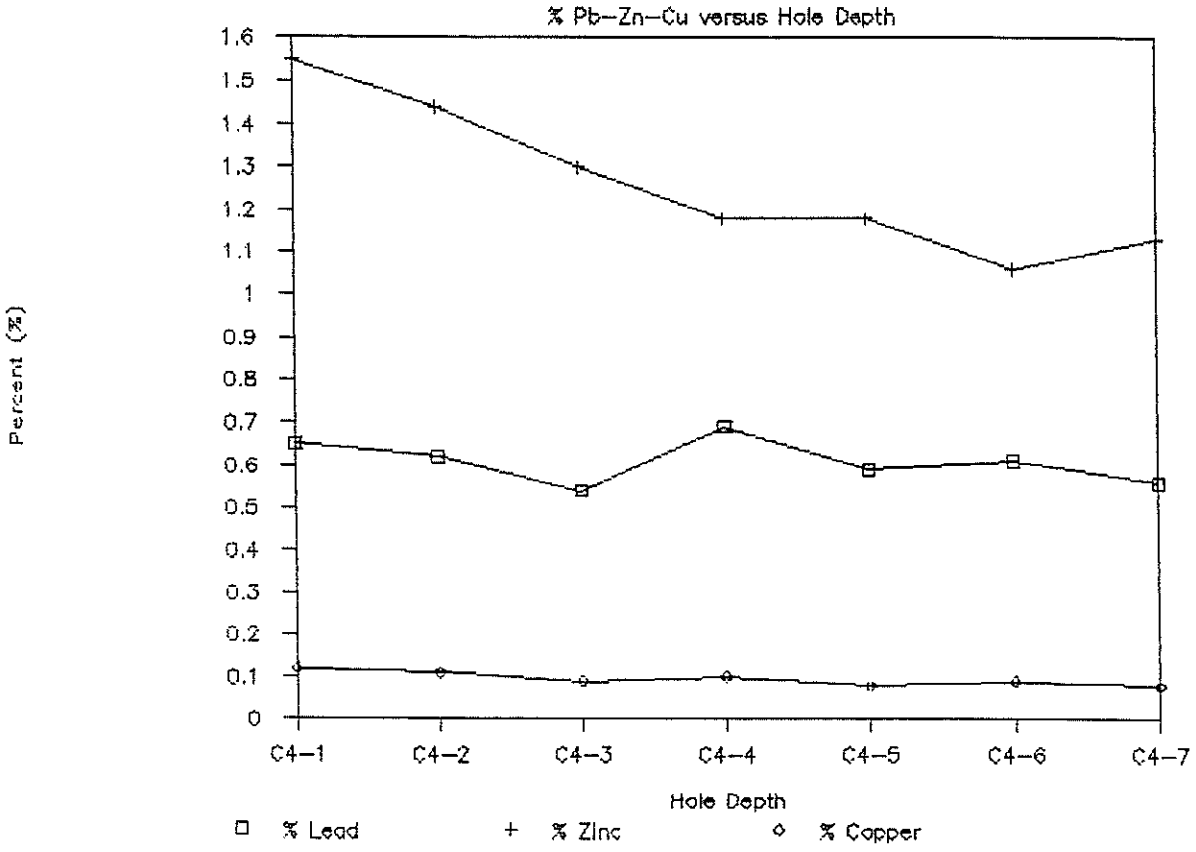

B7 HOLE
% Pb–Zn–Cu versus Hole Depth
Percent (%)
1.6
1.5
1.4
1.3
1.2
1.1
1
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0
B7–1
B7–2
B7–3
B7–4
B7–5
B7–6
B7–7
Hole Depth
% Lead
% Zinc
% Copper


B7 HOLE
% Fe – Ag gmt versus Hole Depth
Percent (%) – Grams/tonne
40
38
36
34
32
30
28
26
24
22
20
18
16
14
12
10
B7–1
B7–2
B7–3
B7–4
B7–5
B7–6
B7–7
Hole Depth
% Iron
Silver (gmt)

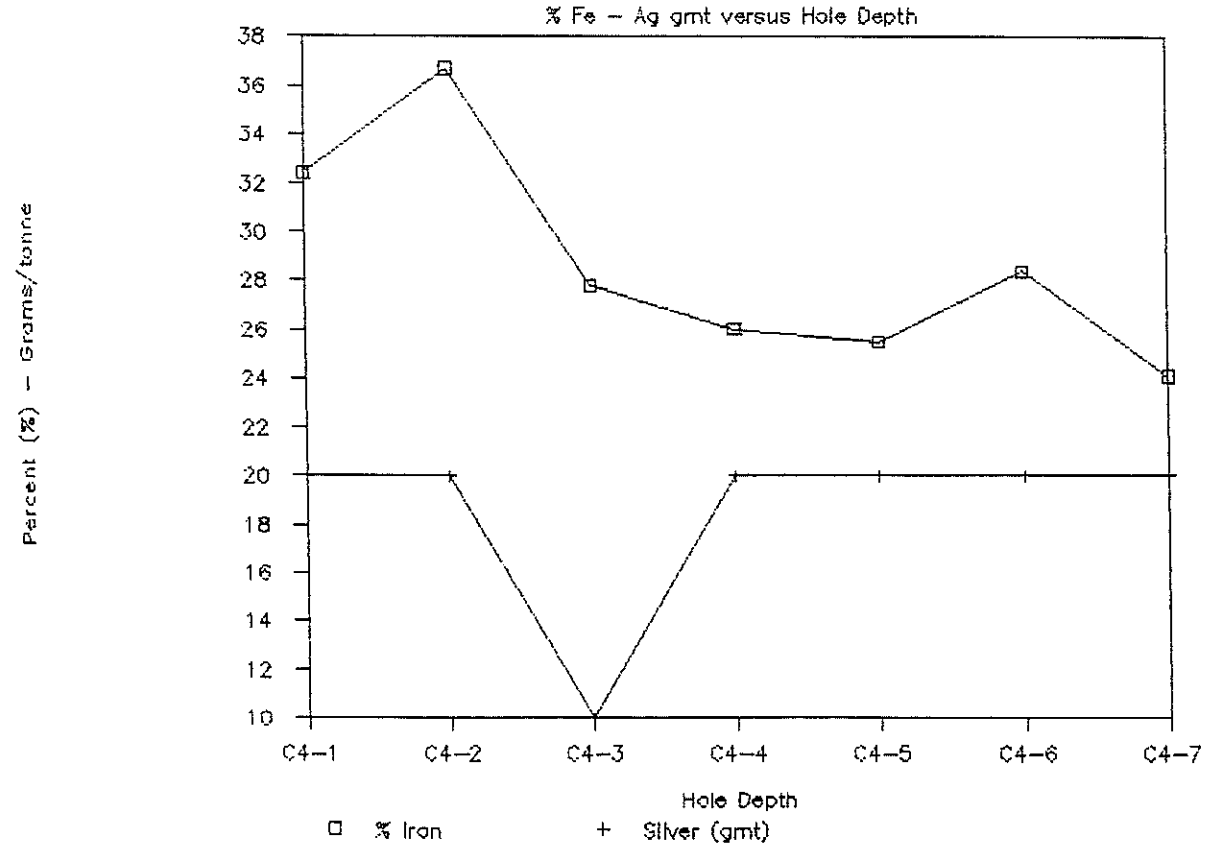

B7 HOLE
% Passing versus Hole Depth
% Passing
90
80
70
60
50
40
30
20
10
0
B7-1
B7-2
B7-3
B7-4
B7-5
B7-6
B7-7
Hole Depths
□ % Passing 200#
+ % Passing 325#

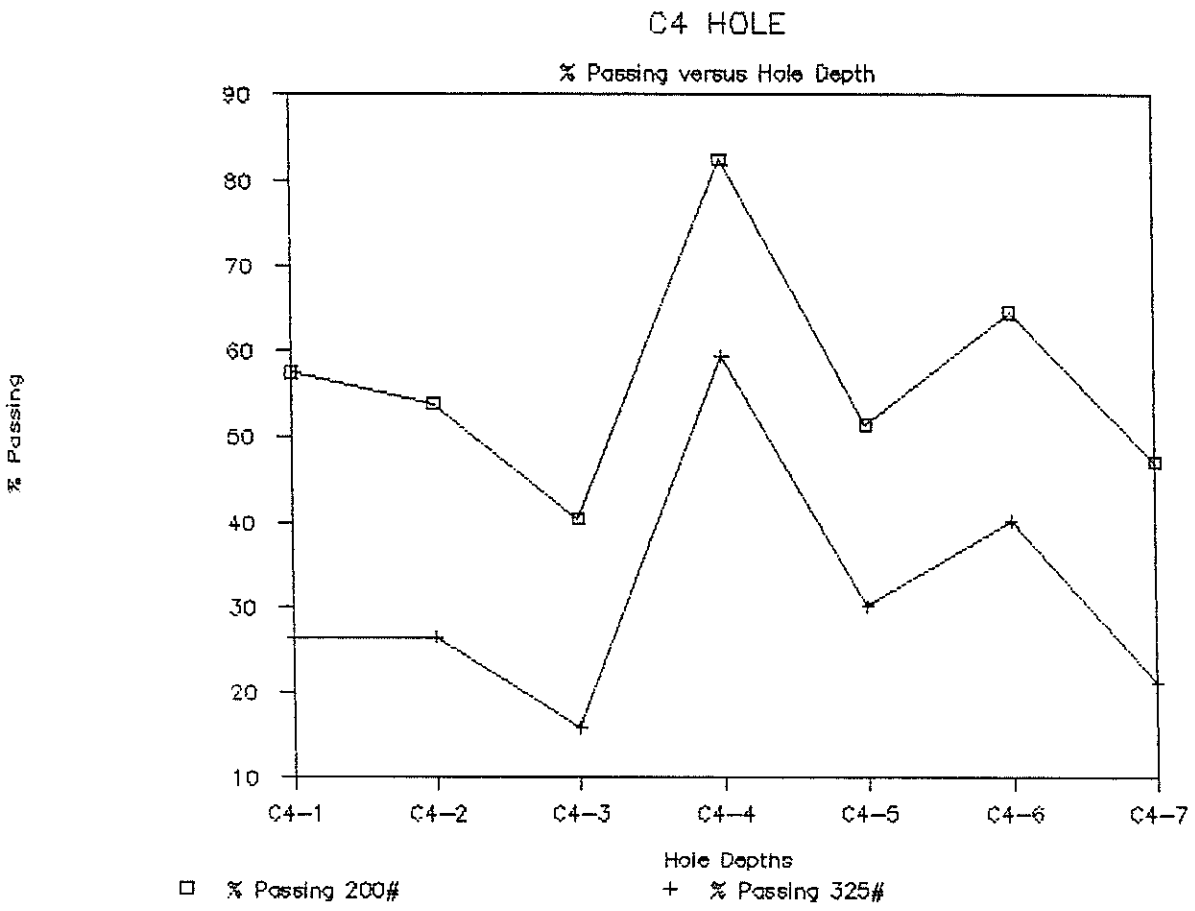

B7 HOLE
pH versus Hole Depth
pH
4.6
4.5
4.4
4.3
4.2
4.1
4
3.9
B7-1
B7-2
B7-3
B7-4
B7-5
B7-6
B7-7
Hole Depth
□ pH

C4 HOLE



C4 HOLE




C4 HOLE
% Passing versus Hole Depth
% Passing
90
80
70
60
50
40
30
20
10
C4-1
C4-2
C4-3
C4-4
C4-5
C4-6
C4-7
Hole Depths
% Passing 200#
% Passing 325#

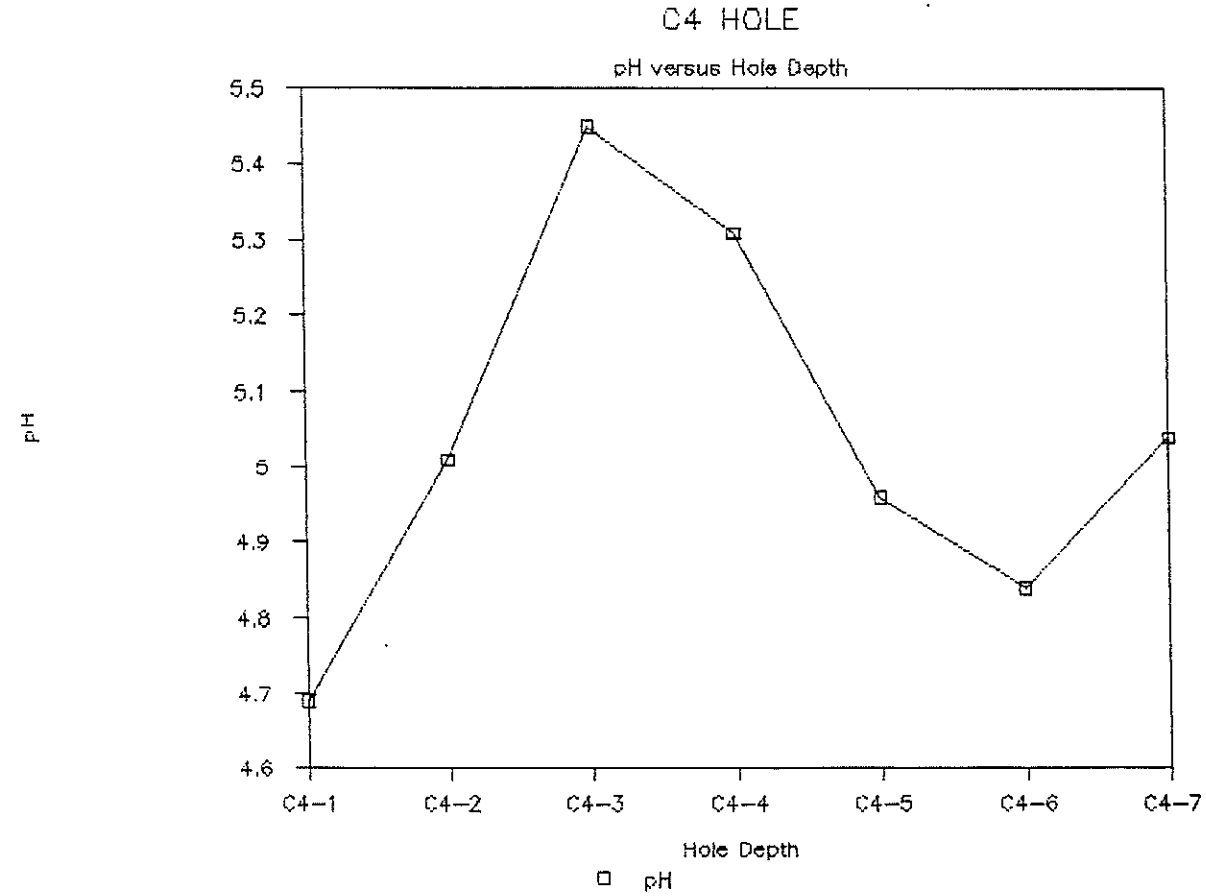

C4 HOLE
pH versus Hole Depth
pH
5.5
5.4
5.3
5.2
5.1
5
4.9
4.8
4.7
4.6
C4-1
C4-2
C4-3
C4-4
C4-5
C4-6
C4-7
Hole Depth
pH

TABLE #3
A3 Hole Metal Distribution

Sample	Screen Size	Weight (gms)	Weight (%)	Calculated Head Assays %Pb	%Zn	%Fe	%Cu	Ag gmt	ASSAYS %Pb	%Zn	%Fe	%Cu	Ag gmt	DISTRIBUTION %Pb	%Zn	%Fe	%Cu	%Ag
A3-1	+200	14.61	9.44	0.57	1.26	20.6	0.14	19	0.18	0.78	10.2	0.06	19	3.01	5.82	3.25	4.00	4.95
A3-1	+325	46.11	29.79						0.29	1.00	30.7	0.13	26	15.29	23.55	30.98	27.35	31.27
A3-1	-325	94.06	60.77						0.76	1.47	32.1	0.16	26	81.71	70.63	65.87	68.66	63.78
A3-2	+200	20.49	10.19	0.45	0.90	34.0	0.16	26	0.22	0.69	18.9	0.10	20	4.98	8.63	5.59	6.45	7.79
A3-2	+325	61.54	30.34						0.32	0.59	34.1	0.13	30	14.96	22.50	30.45	25.17	23.38
A3-2	-325	120.78	59.55						0.60	0.92	36.5	0.19	30	80.06	68.86	63.96	68.39	68.83
A3-3	+200	22.58	10.30	0.61	1.02	33.9	0.17	29	0.37	1.14	20.1	0.10	20	5.21	11.50	6.10	9.57	7.11
A3-3	+325	64.65	29.49						0.32	0.88	34.9	0.14	30	15.37	25.42	30.35	23.99	30.54
A3-3	-325	131.98	60.21						0.90	1.01	35.8	0.19	30	79.43	63.09	63.55	66.44	62.35
A3-4	+200	31.40	11.32	0.87	1.39	33.2	0.21	37	0.49	1.35	21.1	0.13	30	6.39	10.96	7.19	7.00	9.15
A3-4	+325	48.29	17.41						0.46	1.03	37.2	0.14	30	9.21	12.96	19.49	11.60	14.07
A3-4	-325	197.69	71.27						1.03	1.49	34.2	0.24	40	84.41	76.17	73.33	81.40	76.78
A3-5	+200	131.20	44.66	0.66	1.31	38.7	0.15	34	0.66	1.89	37.1	0.14	30	44.74	64.07	42.76	42.09	54.76
A3-5	+325	79.89	27.19						0.50	0.80	41.8	0.13	30	30.64	16.60	39.34	23.80	22.23
A3-5	-325	82.71	28.15						0.81	0.90	38.4	0.19	30	24.62	19.33	27.90	34.11	23.01
A3-6	+200	142.11	46.50	0.72	1.48	39.5	0.14	33	0.73	2.02	30.8	0.14	30	46.94	63.34	46.83	45.58	40.96
A3-6	+325	80.82	26.45						0.51	0.92	41.3	0.12	30	18.65	16.41	37.64	22.22	22.29
A3-6	-325	82.68	27.05						0.92	1.11	37.3	0.17	30	34.42	20.25	25.53	32.20	35.75
A3-7	+200	133.28	45.01	0.68	1.24	39.9	0.11	22	0.64	1.83	39.8	0.13	25	42.23	66.57	44.03	52.46	50.57
A3-7	+325	91.08	30.76						0.47	0.62	42.3	0.07	30	21.19	15.41	32.63	19.30	27.65
A3-7	-325	71.75	24.23						1.03	0.92	38.4	0.13	30	36.58	18.02	23.34	28.24	21.78

TABLE #4

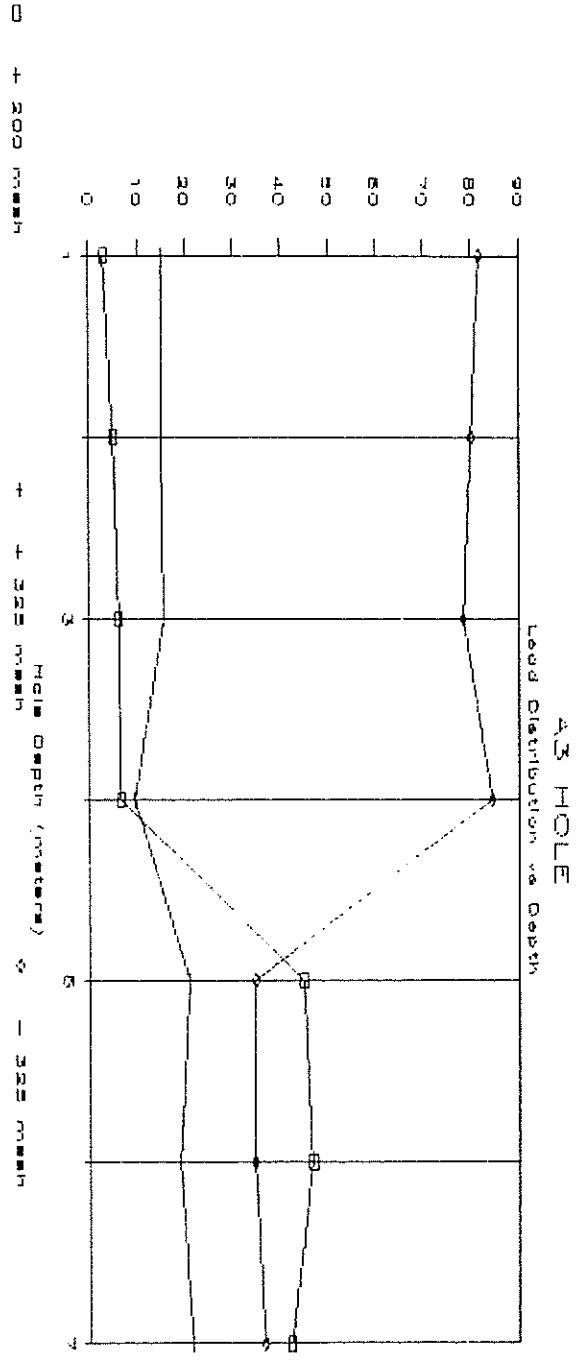
B7 Hole Metal Distribution

Sample	Screen Size	Weight (gms)	Weight (%)	Calculated Head Assays %Pb	%Zn	%Fe	%Cu	Ag gpt	ASSAYS %Pb	%Zn	%Fe	%Cu	Ag gpt	DISTRIBUTION %Pb	%Zn	%Fe	%Cu	%Ag
B7-1	+200	43.01	17.46	0.68	0.88	31.9	0.08	20	0.20	0.00	15.7	0.04	20	7.50	17.82	8.50	8.82	17.46
B7-1	+325	84.51	34.31						0.43	0.89	33.1	0.07	20	21.94	34.53	35.61	30.32	34.31
B7 1	-325	118.77	48.23						0.99	0.87	35.9	0.10	20	70.66	47.56	55.79	60.87	48.22
B7-2	+200	61.96	24.85	0.75	1.34	31.6	0.06	18	0.27	0.93	10.3	0.03	10	8.99	17.20	11.84	13.35	14.19
B7-2	+325	53.15	21.35						0.42	1.13	37.3	0.05	20	11.90	17.96	25.96	10.14	24.38
B7-2	-325	133.89	53.79						1.11	1.62	24.6	0.07	20	79.20	64.95	61.20	67.59	61.43
B7-3	+200	44.99	19.02	0.79	1.30	26.7	0.08	19	0.24	0.83	9.4	0.03	10	5.74	12.14	6.67	7.56	10.51
B7-3	+325	66.15	27.97						0.48	1.30	28.5	0.06	20	16.90	27.06	29.90	22.23	30.91
B7-3	-325	125.35	53.00						1.16	1.47	31.9	0.10	20	77.36	59.80	63.42	70.21	58.58
B7-4	+200	178.97	67.10	0.42	1.14	42.2	0.11	11	0.10	1.33	42.7	0.09	10	38.71	78.58	67.94	48.73	58.78
B7-4	+325	49.96	18.73						0.19	0.55	44.0	0.12	10	9.57	9.07	19.51	20.41	16.41
B7 4	-325	37.79	14.17						1.78	0.99	37.7	0.04	20	60.72	12.35	12.64	30.86	24.82
B7-5	+200	245.72	79.65	0.62	1.43	33.9	0.12	11	0.42	1.63	31.9	0.12	10	64.49	90.62	75.28	80.93	74.29
B7-5	+325	40.47	13.12						0.34	0.60	43.0	0.10	10	8.60	5.49	16.71	11.11	12.23
B7-5	-325	22.31	7.23						1.03	0.77	37.4	0.13	20	26.91	3.90	8.01	7.96	13.49
B7-6	+200	216.27	71.44	0.61	1.55	34.1	0.11	10	0.49	1.77	34.2	0.11	20	56.37	81.33	71.59	69.62	74.87
B7 6	+325	42.89	14.17						0.35	0.99	38.7	0.11	15	9.15	7.29	16.06	13.91	11.02
B7-6	-325	43.56	14.39						1.50	1.23	29.3	0.13	20	35.48	11.38	12.35	16.57	14.92
B7-7	+200	223.54	70.29	0.54	1.37	29.9	0.11	12	0.45	1.60	29.5	0.11	10	60.21	82.17	69.32	71.10	60.01
B7-7	+325	40.05	12.59						0.31	0.66	39.7	0.10	10	7.22	6.07	16.71	11.58	10.75
B7-7	-325	54.45	17.12						1.02	0.94	24.4	0.11	20	32.52	11.76	13.97	17.32	29.24

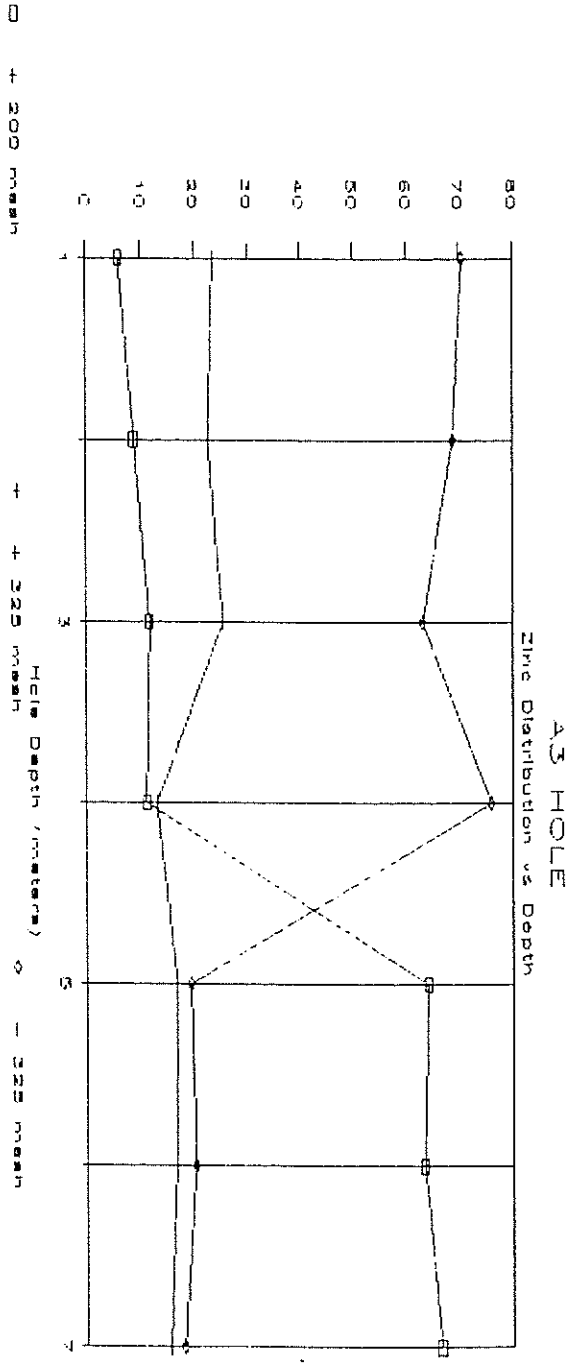
TABLE #5
C4 Hole Metal Distribution

Sample	Screen Size	Weight (gms)	Weight (%)	Calculated Head Assays %Pb	%Zn	%Fe	%Cu	Ag gpt	ASSAYS %Pb	%Zn	%Fe	%Cu	Ag gpt	DISTRIBUTION %Pb	%Zn	%Fe	%Cu	%Ag
C4-1	+200	121.26	42.45	0.67	1.45	33.2	0.11	23	0.40	1.83	27.6	0.09	20	25.23	53.51	36.35	35.92	37.48
C4-1	+325	88.71	31.05						0.46	1.20	37.3	0.10	20	21.22	25.67	35.94	29.20	27.42
C4-1	-325	75.71	26.50						1.36	1.11	33.7	0.14	30	53.55	20.81	27.71	34.89	35.10
C4-2	+200	125.85	46.15	0.68	1.44	28.2	0.10	19	0.34	1.64	19.8	0.08	10	22.92	52.64	32.35	36.43	25.60
C4-2	+325	74.72	27.40						0.41	1.25	37.1	0.10	20	16.41	33.82	35.99	27.04	30.30
C4-2	-325	72.13	26.45						1.57	1.28	33.8	0.14	30	60.67	33.55	31.65	36.54	44.01
C4-3	+200	163.62	59.65	0.57	1.23	27.2	0.05	22	0.36	1.56	19.5	0.05	20	37.93	75.83	42.76	54.16	55.25
C4-3	+325	67.13	24.47						0.33	0.68	40.6	0.05	20	13.93	13.56	36.52	22.34	22.67
C4-3	-325	43.56	15.88						1.73	0.82	35.5	0.09	30	48.24	10.61	20.72	23.20	22.07
C4-4	+200	51.03	17.55	0.70	1.12	33.2	0.06	28	0.39	1.27	13.3	0.03	20	9.76	19.94	10.04	9.39	12.43
C4-4	+325	67.10	23.08						0.48	1.02	25.9	0.04	30	15.79	20.96	25.61	16.47	24.51
C4-4	-325	172.61	59.37						0.88	1.12	25.2	0.07	30	74.46	59.20	64.35	74.14	63.06
C4-5	+200	96.30	48.56	0.58	1.16	37.5	0.05	33	0.46	1.48	29.3	0.04	30	38.45	62.31	35.79	37.96	44.13
C4-5	+325	42.31	21.33						0.42	0.95	32.9	0.05	30	15.42	17.46	25.40	20.85	19.39
C4-5	-325	59.72	30.11						0.89	0.78	35.5	0.07	40	46.13	20.23	38.81	41.19	36.49
C4-6	+200	64.72	35.62	0.57	0.97	28.8	0.06	36	0.40	1.25	17.7	0.05	30	24.75	46.07	21.80	28.63	29.24
C4-6	+325	44.37	24.35						0.34	0.80	34.1	0.05	40	14.43	20.05	28.89	19.63	25.73
C4-6	-325	73.11	40.13						0.87	0.82	35.5	0.09	40	60.82	33.87	49.40	51.74	44.04
C4-7	+200	85.53	52.87	0.53	1.01	31.9	0.05	40	0.42	1.12	12.6	0.04	40	42.06	58.74	30.36	45.63	52.87
C4-7	+325	41.99	25.96						0.43	0.95	32.2	0.04	40	20.65	24.46	38.09	22.40	25.96
C4-7	-325	34.25	21.17						0.93	0.80	32.7	0.07	40	37.29	16.80	31.55	31.97	21.17

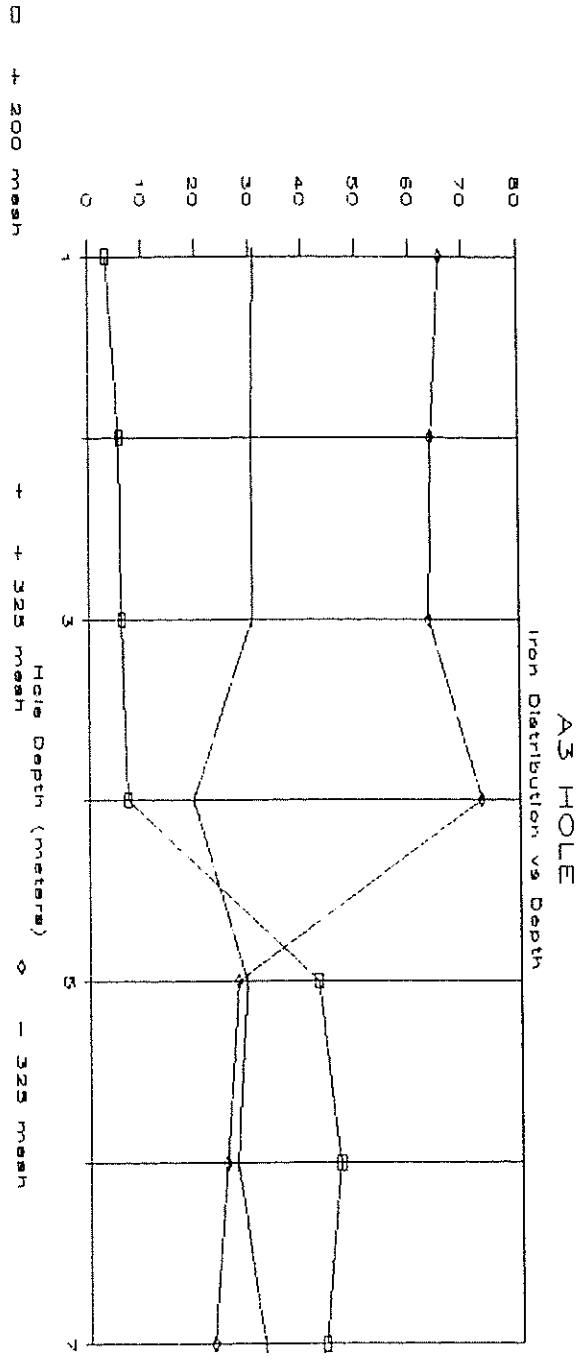
A3 Hole % Lead Distribution



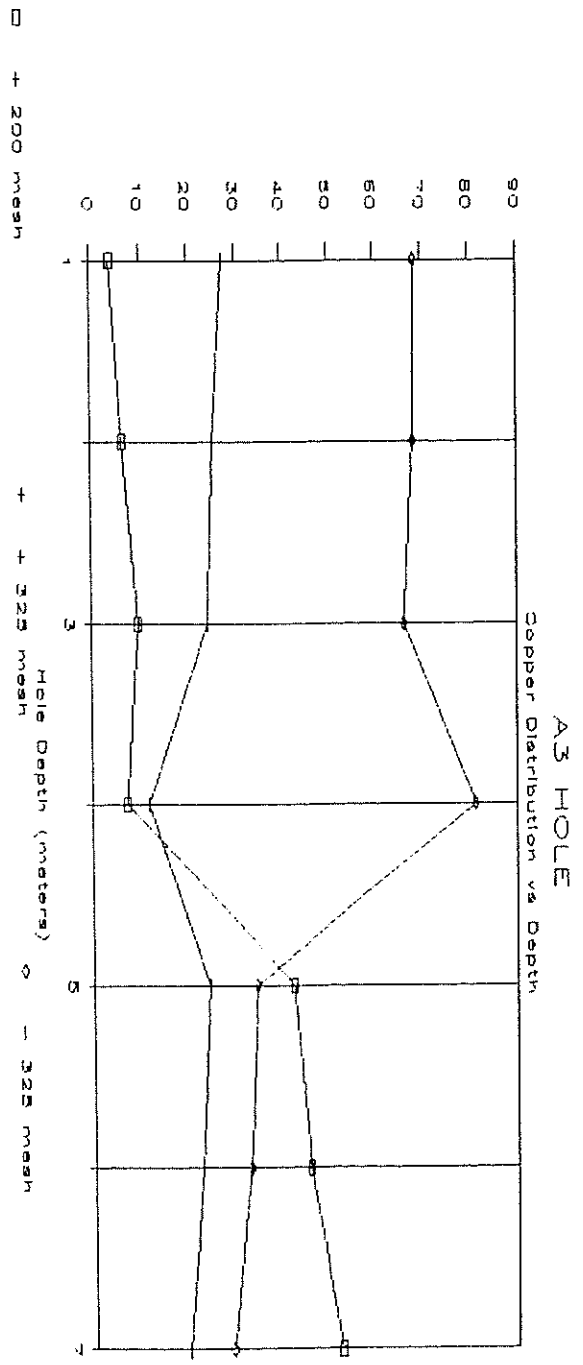
A3 Hole % Zinc Distribution



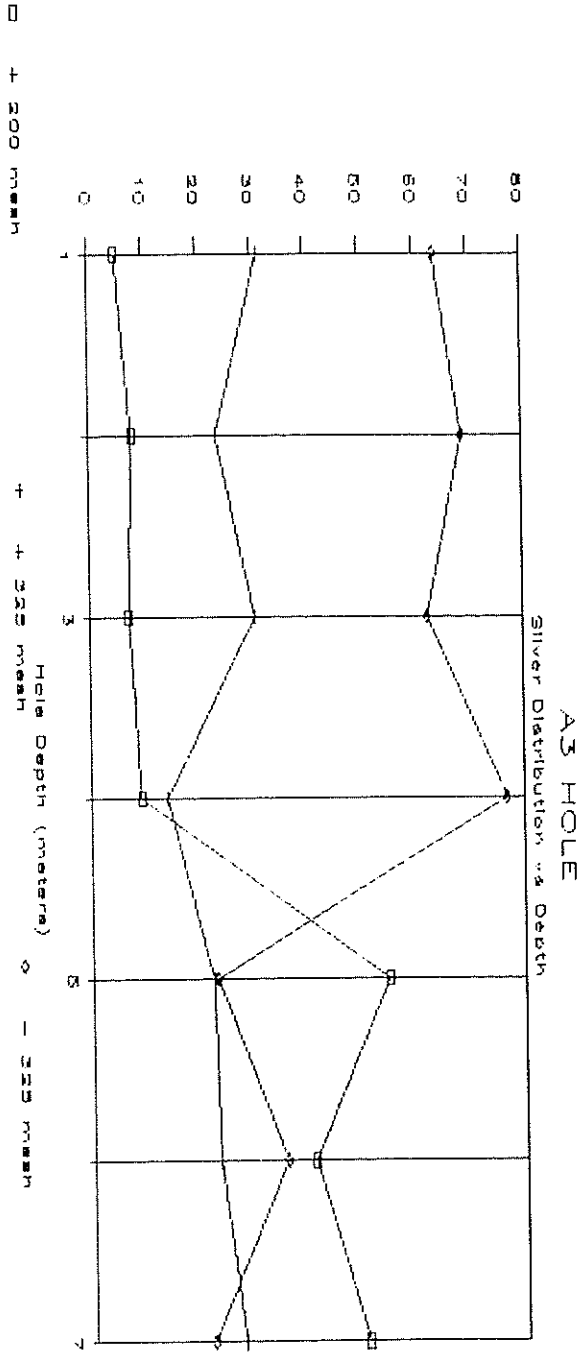
A3 Hole % Iron Distribution



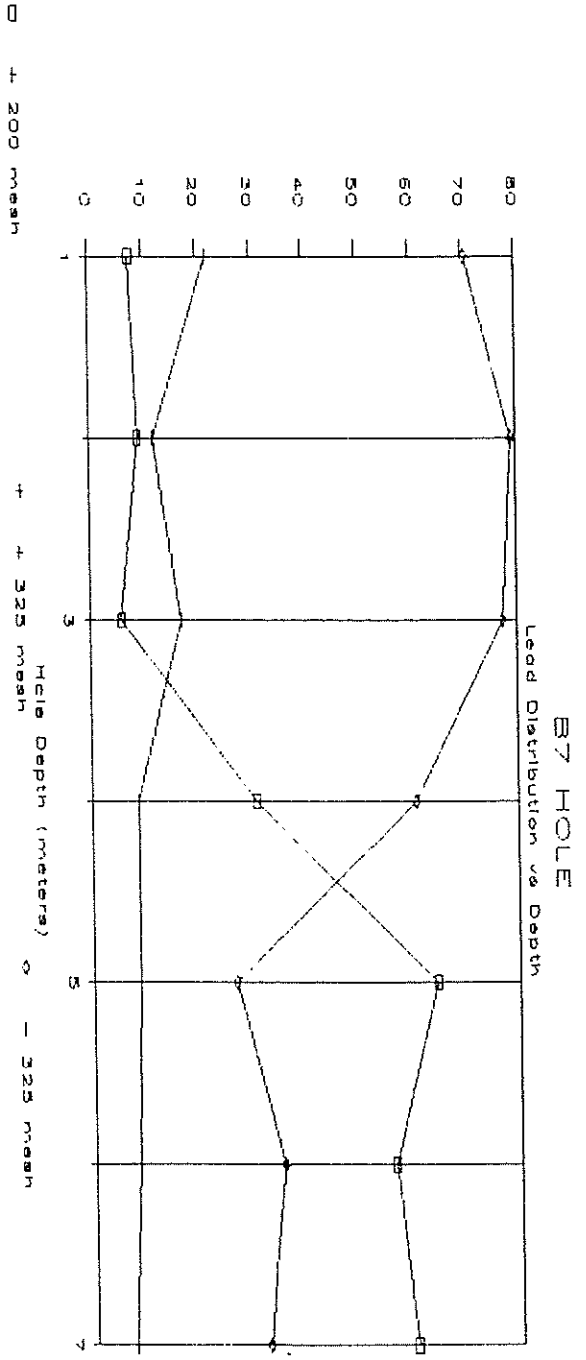
A3 Hole % Copper Distribution



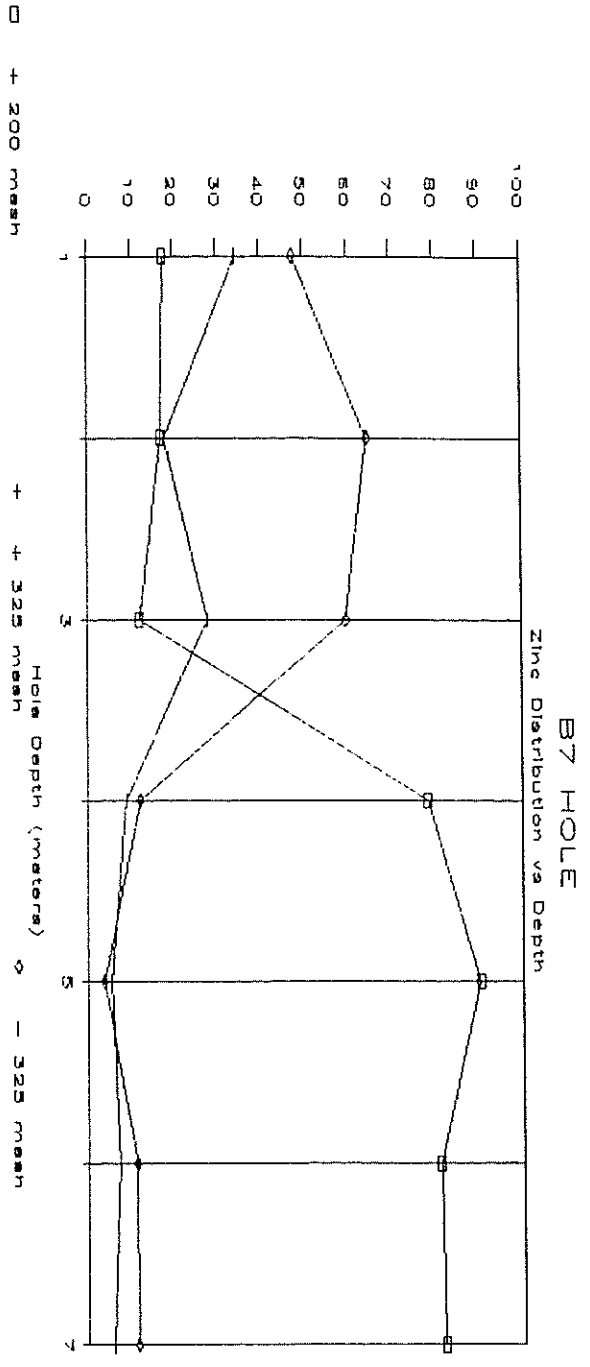
A3 Hole % Silver Distribution



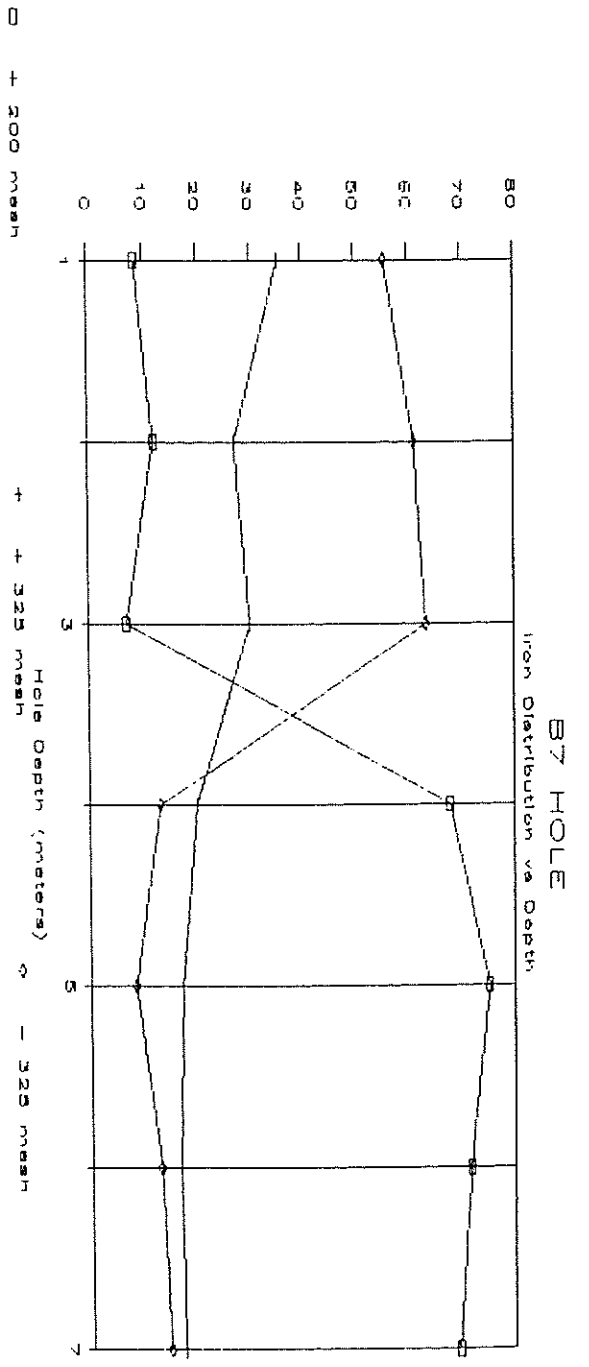
B7 Hole % Lead Distribution



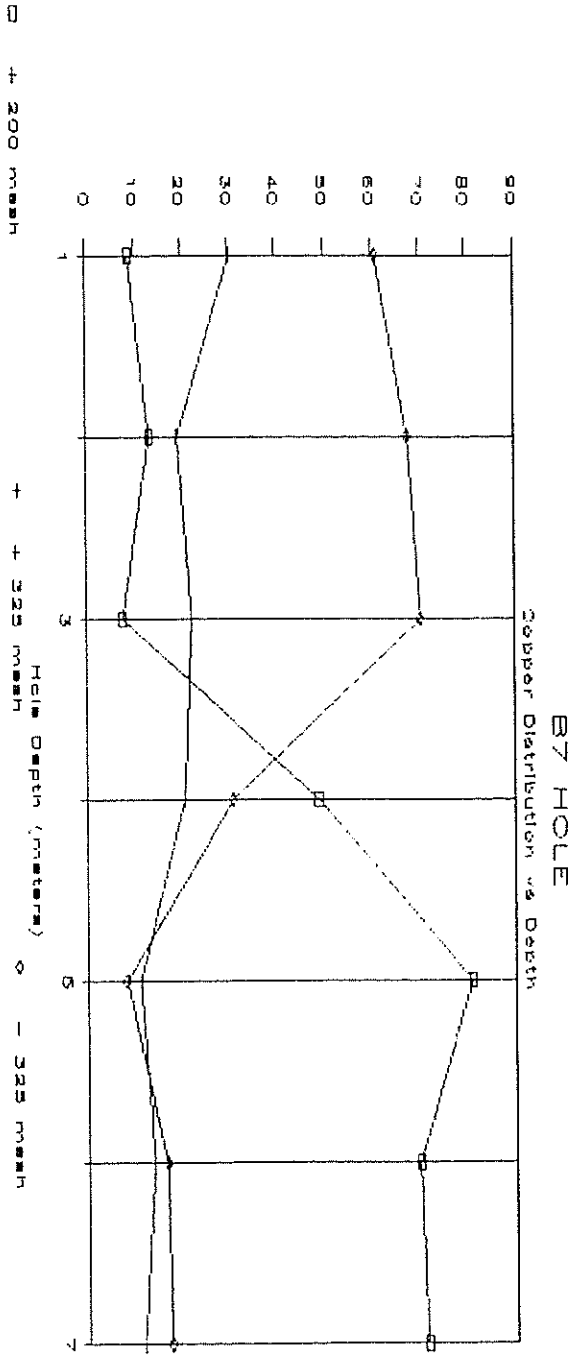
B7 Hole % Zinc Distribution



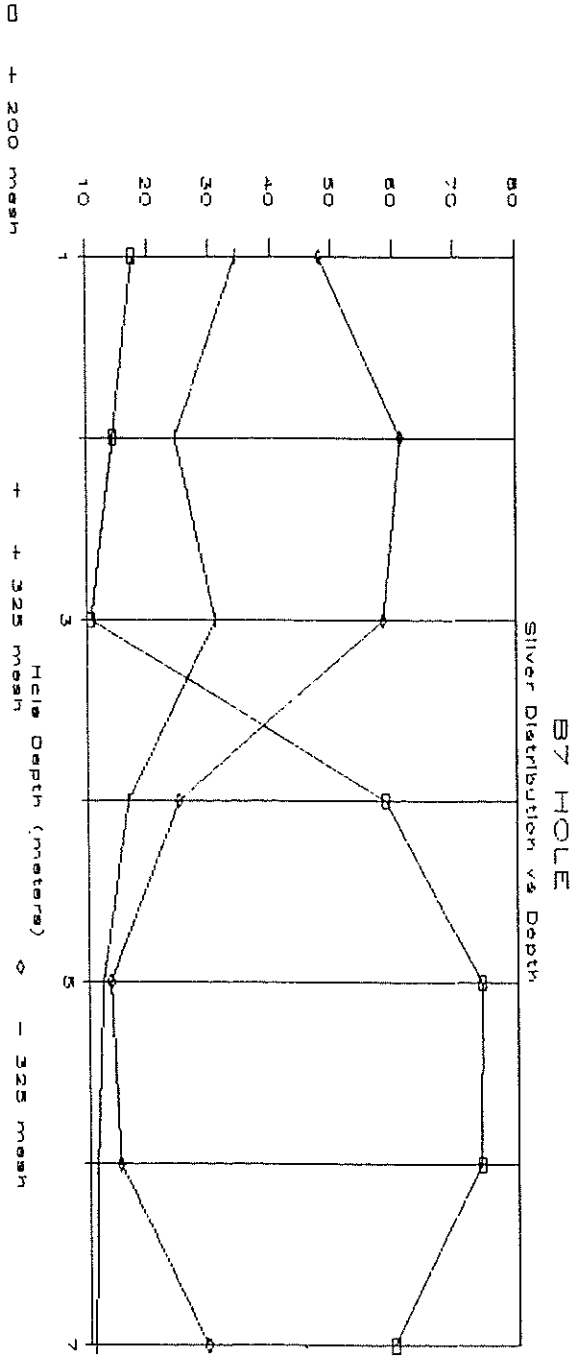
B7 Hole % Iron Distribution



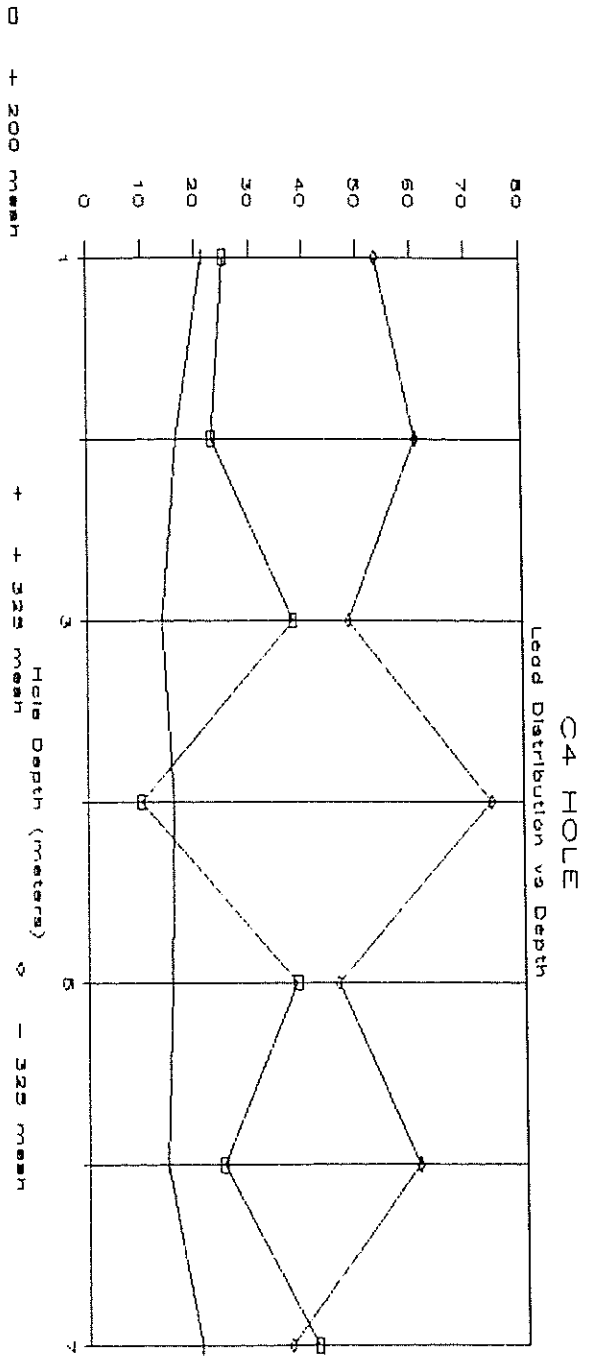
B7 Hole % Copper Distribution



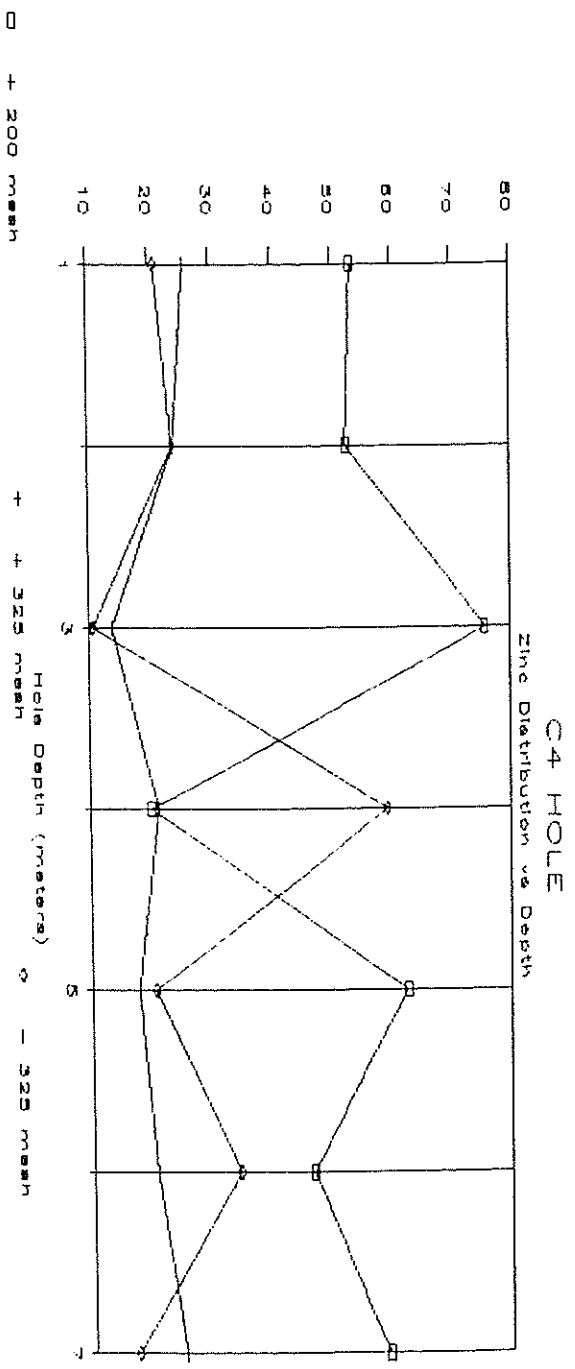
B7 Hole % Silver Distribution



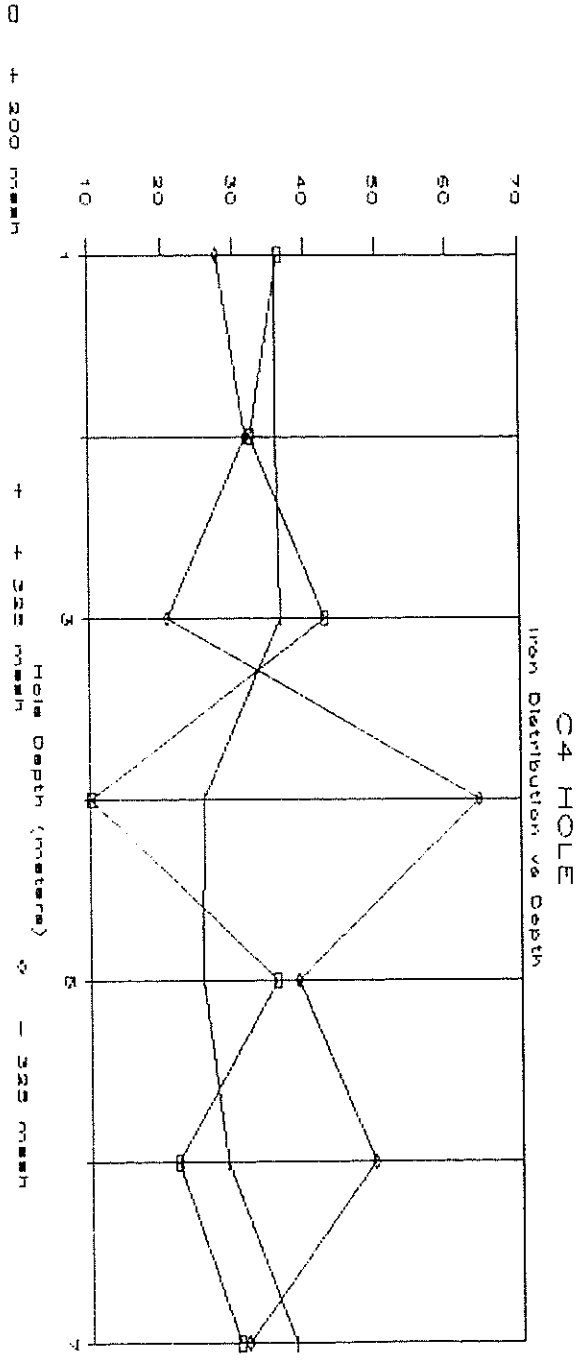
C4 Hole % Lead Distribution



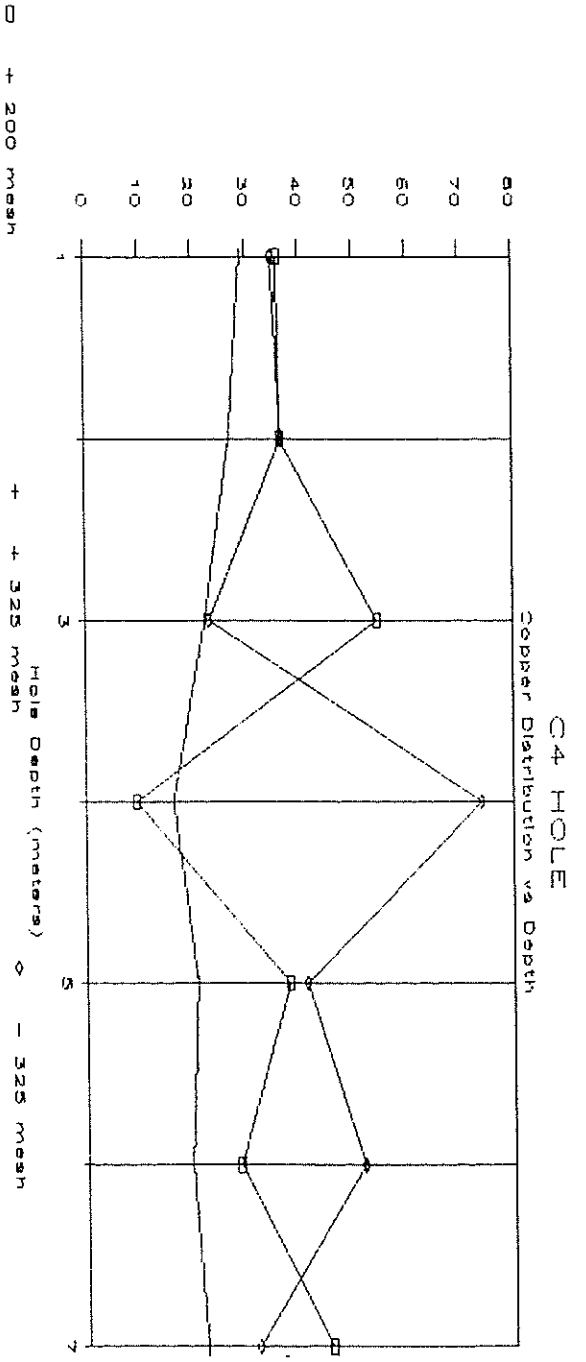
C4 Hole % Zinc Distribution



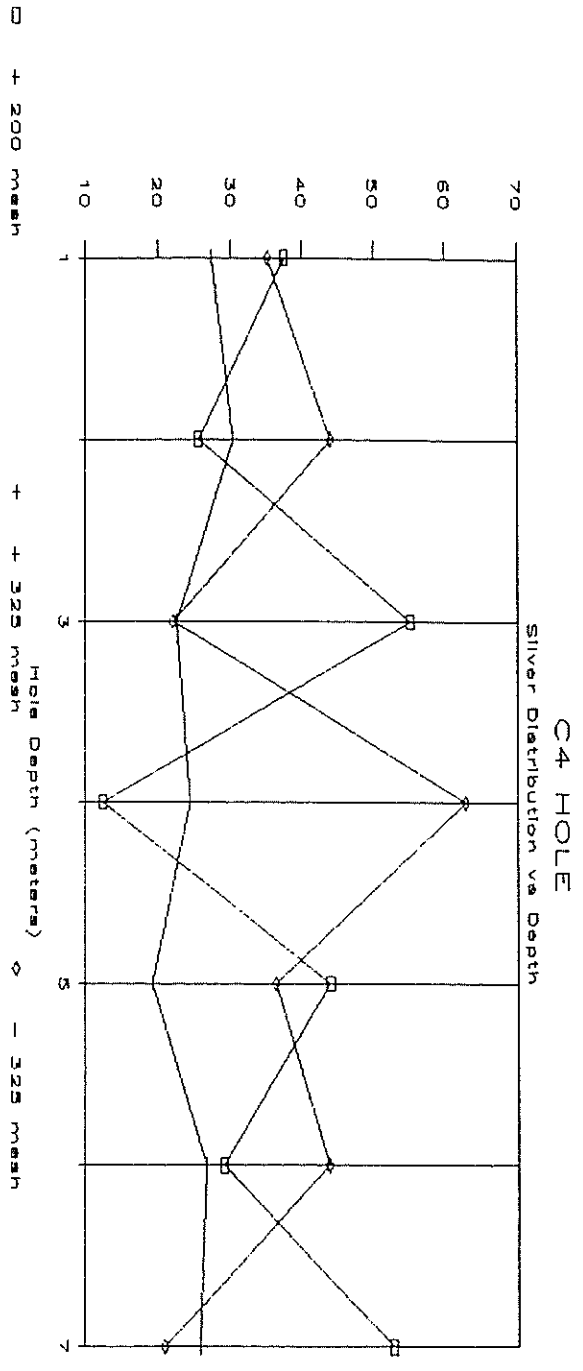
C4 Hole % Iron Distribution



C4 Hole % Copper Distribution



C4 Hole % Silver Distribution

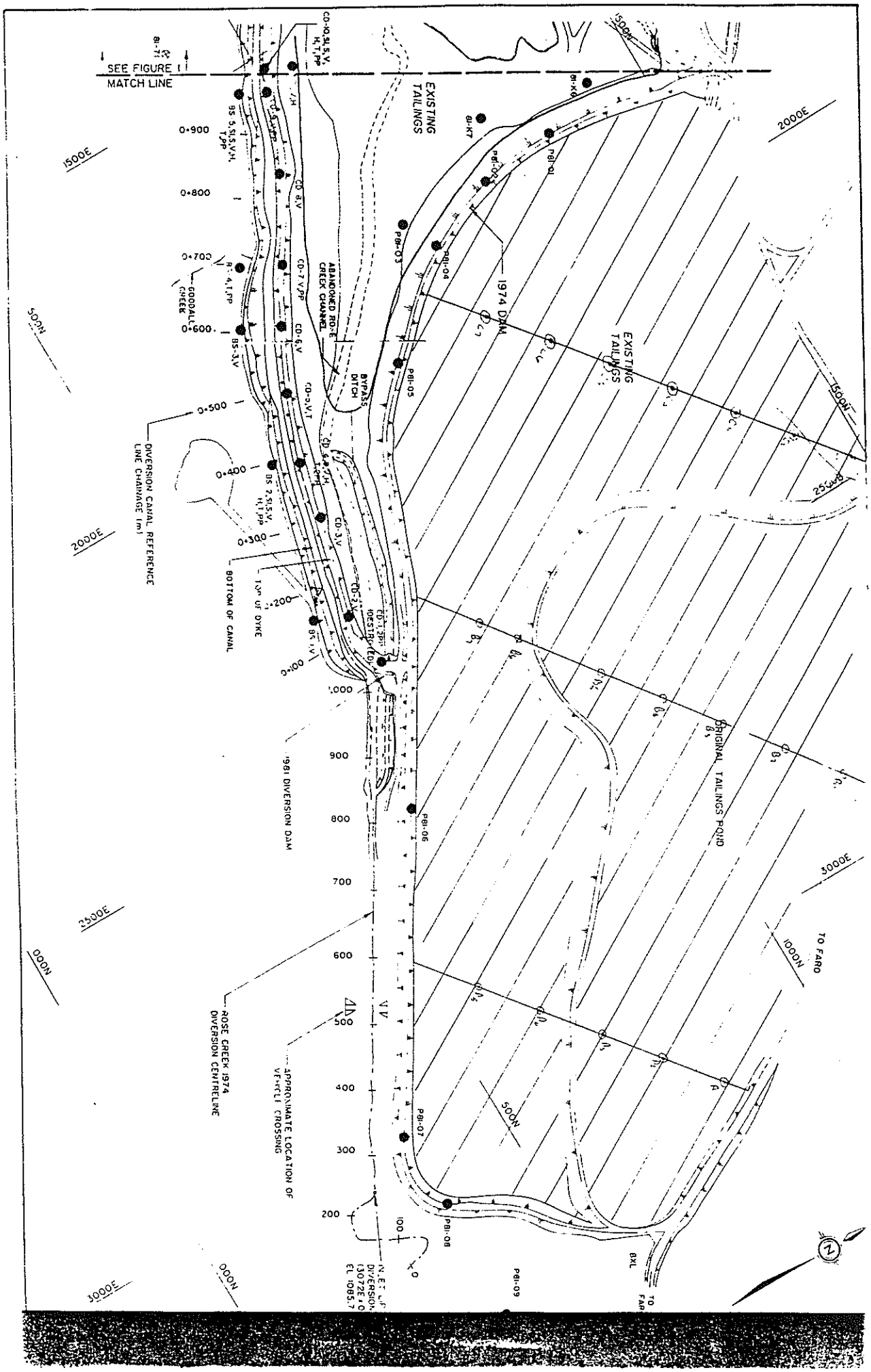


A P P E N D I X # 1

Appendix #1
Field Data During Sampling

Hole	Depths	Observations
A1	1 - 5	Hit bottom after 5 meters.
A2	1 - 7	Lots of clay.
A3	1 - 7	Lots of clay.
A4	1 - 7	Top is very badly oxidized.
A5	1 - 5	Very muddy and lots of clay, too muddy hole collapsing in only could go to five meters.
B1	1 - 7	Very soft on the top layer but, harder lower down.
B2	1 - 7	Hole collapsing back in at 6 meters but still able to get all the samples.
B3	1 - 4	Very very muddy only could get to 4 meters, the hole was collapsing back in.
B4		Too muddy to get to hole, no samples taken.
B5		Too muddy to get to hole, no samples taken.
B6	1 - 5	Hit bottom at edge of dike at 5 meters.
B7	1 - 7	Lots of pyrite at 4 meters.
C1	1 - 7	7 meters from creek, grey sand at 4 and 5 meters.
C2	1 - 7	Mud at 4, 5, & 6 meters, grey sand at 2 to 6 meters.
C3	1 - 7	Mud at 4 and 5 meters and grey sand at 5 meters.
C4	1 - 7	Mud at 4 and 5 meters and grey sand at 5 meters.
C5	1 - 7	All dry throughout the hole.
C6	1 - 7	Mud at 5 through to 7 meters.
C7	1 - 7	Very wet mud at 5 through to 7 meters, there was also a creek 10 meters away.

A P P E N D I X # 2


SEE FIGURE 1
MATCH LINE
EXISTING TAILINGS
1974 DAM
EXISTING TAILINGS
ABANDONED RO-E CREEK CHANNEL
BYPASS DITCH
GOODALL CREEK
DIVERSION CANAL REFERENCE LINE CHAINAGE (m)
BOTTOM OF CANAL
TOP OF DYKE
1981 DIVERSION DAM
ORIGINAL TAILINGS POND
TO FARO
ROSE CREEK 1974 DIVERSION CENTRELINE
APPROXIMATE LOCATION OF VEHICLE CROSSING
1500E
2000E
2500E
3000E
500N
000N
1000N
1500N
2500N
0+900
0+800
0+700
0+600
0+500
0+400
0+300
0+200
0+100
1,000
900
800
700
600
500
400
300
200
100
0
PBI-01
PBI-02
PBI-03
PBI-04
PBI-05
PBI-06
PBI-07
PBI-08
PBI-09
BXL

APPENDIX F

LAKEFIELD RESEARCH
FLOTATION TESTWORK

An Investigation into

FLOTATION OF FARO TAILINGS

for

CURRAGH RESOURCES INC.

Progress Report No. 1

Project No. L.R. 4208

NOTE:

This report refers to the samples as received.

The practice of this Company in issuing reports of this nature is to require the recipient not to publish the report or any part thereof without the written consent of Lakefield Research.

LAKEFIELD RESEARCH
A DIVISION OF FALCONBRIDGE LIMITED
December 9th, 1991

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DETAILS OF TESTS	15
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2. Testwork Results	

INTRODUCTION

This report details mineralogical, flotation and gravity testwork on tailings from Curragh Resources Inc., Faro tailings enpoundment. The objective of this testwork was to have a preliminary look at the recovery of lead, zinc and precious metal from this tailings sample.

A secondary objective was to study flotation reagent regimes that would be cost effective for flotation.

This work was requested by Mr. Godfrey McDonald of Curragh Resources. Results were given to him and consultant Mr. Dick Down when they became available. During the program a couple of meetings were held between Lakefield Research and the clients representatives to discuss testwork direction.

LAKEFIELD RESEARCH

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Senior Metallurgist

Report Preparation by: S. McKenzie

SUMMARY

1. Sample Description

Two samples were used for testwork. Both samples were composited from locations A2, B2, C2 of the Faro tailings pond. These locations refer to grid map of the tailings pond, supplied by Curragh Resources Inc. to Lakefield Research. A copy of this grid map is enclosed with the Testwork Details section of this report.

A second sample was taken because the first sample was used up in testwork. The first sample was used for Tests 1-9 and the second sample was used for Tests 10-20.

As both samples were tailings no sample preparation size reduction was necessary. However, both samples contained nodules of hardened material. These were broken up by hand. The samples were then riffled, mixed for blending and charges for testwork cut out of the blend.

Given in Table 1 is a summary of head analysis and specific gravity for samples 1 and 2, including average heads calculated from testwork.

TABLE NO. 1 :
Head Analysis

	Element													
Sample	%Cu	%Pb	%Zn	%Fe	Au g/t	Ag g/t	%Pb oxide	%Zn oxide	%Cu oxide	%As	%Bi	%Cd	%Sb	Hg ppm	S.G.
Sample 1	0.14	0.79	1.23	32.3	0.11	15.2	0.12	0.15	0.032	0.057	<0.002	0.002	<0.002	71	3.91
Sample 1 - Testwork Average		0.74	1.07		0.15	15.9									
Sample 2		0.64	1.16		0.28										
Sample 2 - Testwork Average		0.65	0.95			12.7									

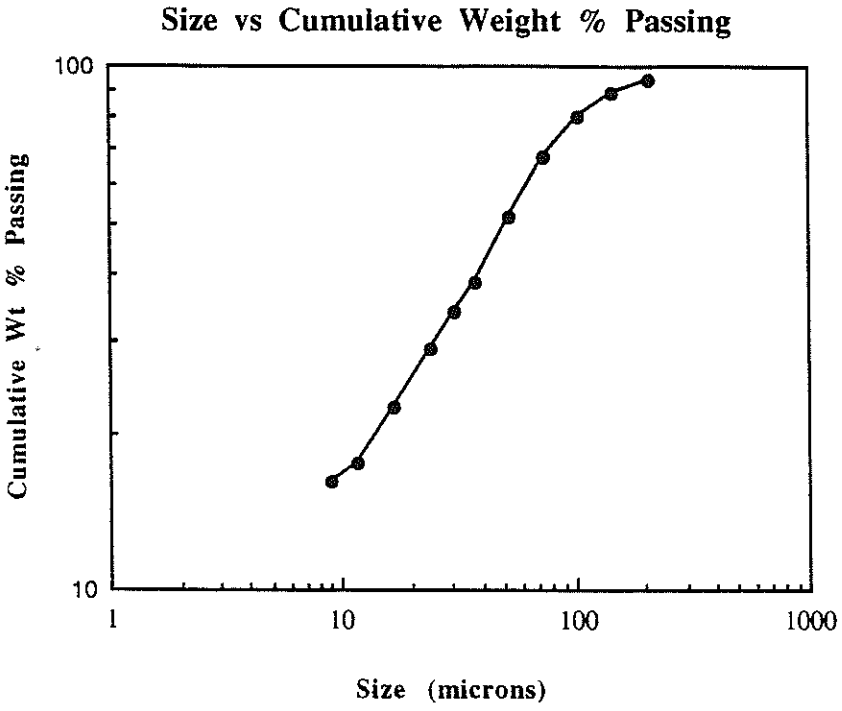
A head sample of Composite Sample 1 was sent for assay by size fraction analysis. These results are shown in Table 2 and given graphically in Figures 1, 2 and 3.

TABLE NO. 2 :
Sample 1 Assay by Size Fraction Analysis

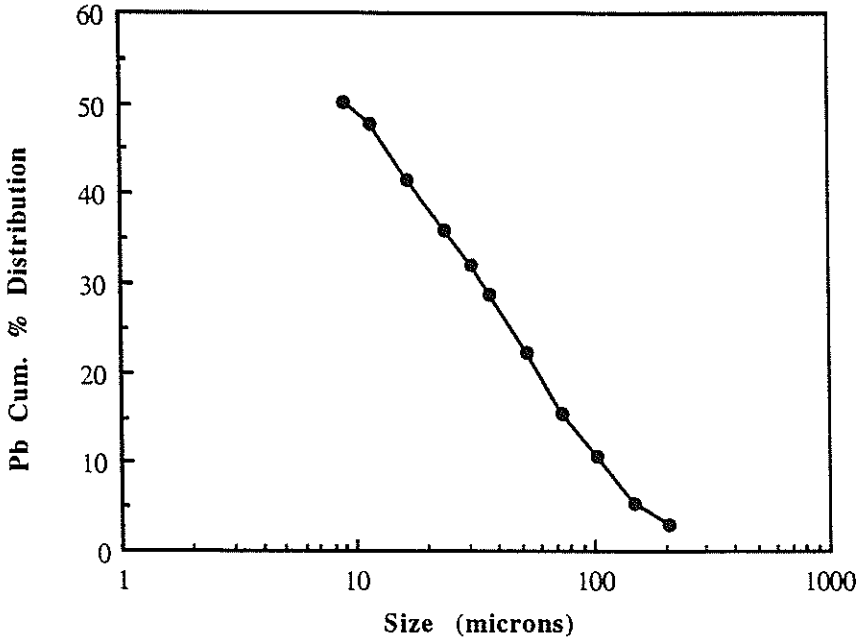
Product	% Weight		Assays %		% Distribution			
					Individual		Cumulative	
	Ind.	Cum. % Passing	Pb	Zn	Pb	Zn	Pb	Zn
+65 mesh	6.0	94.0	0.35	0.82	2.9	4.9	2.9	4.9
+100 mesh	5.0	89.0	0.35	1.13	2.4	5.7	5.2	10.6
+150 mesh	8.7	80.3	0.46	1.11	5.5	9.7	10.7	20.3
+200 mesh	12.7	67.6	0.28	1.00	4.8	12.7	15.5	33.0
+270 mesh	15.8	51.8	0.32	0.94	6.9	14.9	22.4	47.8
+400 mesh	13.2	38.6	0.35	0.87	6.3	11.5	28.7	59.3
+30.8 µm	4.5	34.1	0.53	0.61	3.3	2.7	32.0	62.1
+23.9 µm	5.2	28.9	0.54	1.08	3.8	5.6	35.8	67.7
+16.7 µm	6.4	22.5	0.65	0.92	5.7	5.9	41.5	73.6
+11.5 µm	5.0	17.5	0.94	0.90	6.4	4.5	47.9	78.1
+8.9 µm	1.4	16.1	1.21	0.89	2.3	1.2	50.2	79.4
-8.9 µm	16.1	-	2.27	1.28	49.8	20.6	100.0	100.0
Head (calc.)	100.0	-	0.73	1.00	100.0	100.0	-	-

These results show a definite increase in Pb content at very fine size, but the Zn content is very consistent over the complete size range.

FIGURE 1:

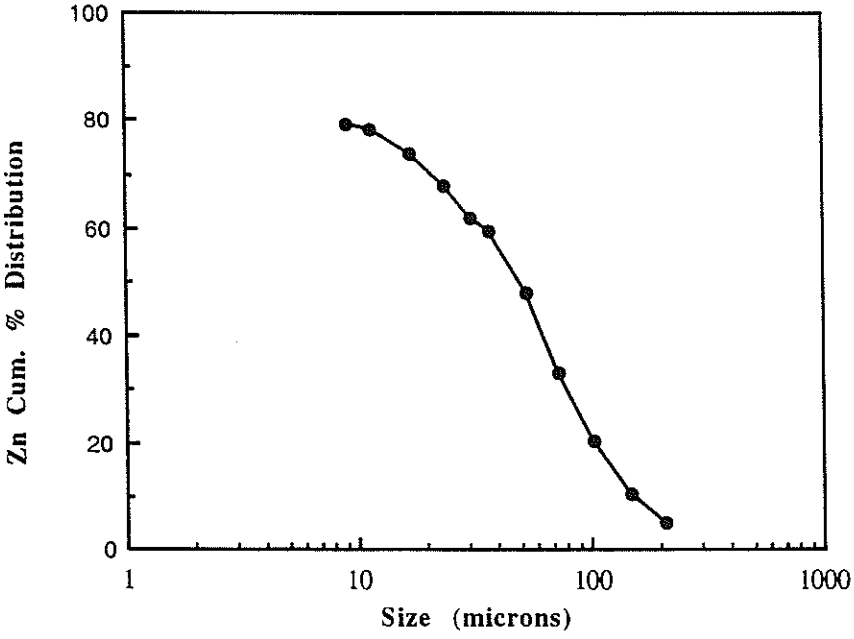

Size vs Cumulative Weight % Passing
100
10
Cumulative Wt % Passing
1
10
100
1000
Size (microns)

Size vs Pb Cumulative % Distribution



- Figure 2 -

Size vs Zn Cumulative % Distribution



- Figure 3 -

1.1. Head Sample Mineralogy

A portion of Composite Sample 1 was submitted to the mineralogical laboratory for a preliminary evaluation of the type(s) of Zn-Pb-Cu minerals, their liberation character and textural/mineralogical associations.

A polished section, PS#3399, was prepared and examined under the ore microscope. The frequency, surface area coverage, and mineral distribution by association, were reported for sphalerite, galena and chalcopyrite.

RESULTS:

The following observations were noted. The detailed mineral distribution by association is given below.

Mineral Liberation, Associations - Composite Sample 1

Mineral	Association	Area % Distribution
Sphalerite	liberated	55.7
	binary with pyrite	15.5
	binary with gangue	22.4
	binary with galena	5.0
	binary with chalcopyrite	0.8
	binary with pyrrhotite	0.6
Galena	liberated	21.5
	binary with pyrite	8.4
	binary with gangue	37.3
	binary with sphalerite	31.4
	binary with chalcopyrite	0.2
	binary with pyrrhotite	1.2
Chalcopyrite	liberated	58.0
	binary with pyrite	21.5
	binary with gangue	13.7
	binary with galena	0.7
	binary with sphalerite	5.8
	binary with pyrrhotite	0.3

Sphalerite occurred primarily in liberated particles (~56% area %) ranging from <20 to 200 micrometers, with lesser binary associations, chiefly with pyrite and gangue minerals. The binaries commonly displayed simple intergrowths, the sphalerite often associated with grain boundaries typically interstitial to subidiomorphic pyrite. Peripheral exposure, due to intergranular breakage, was common, particularly surrounding pyrite.

Galena occurred as fine-grained blebs or stringers, commonly as intergranular mineralization, of less than 10 to 20 micrometers. Liberated galena comprised only 21.5 area % with binary associations with gangue minerals and sphalerite contributing approximately 69 area %. Minor, coarse-grained galena to 50 micrometers was noted with or enclosed by silicate gangue minerals. Only 5.4% of the galena was reported with pyrite, typically as encapsulated blebs.

Chalcopyrite reported 58% liberation commonly in blocky grains of 30-75 micrometers. Binary associations comprised intergranular textures with pyrite and fine-grained blebs hosted by grain margins and cleavage of sphalerite.

The sphalerite should report to the zinc concentrate with copper and lead reporting from binaries and inclusions of chalcopyrite and galena, respectively. Binaries with gangue/pyrite will be carried to the zinc rougher concentrate, in part due to peripheral surface exposure of sphalerite. Galena should report as liberated grains, though a number exhibit possible oxidation rims of anglesite or cerussite which may inhibit recovery. Binaries with sphalerite will report to the flotation concentrate though poor liberation from pyrite and gangue minerals will contribute to low recovery.

1.2. Preliminary Testwork

A sample of Composite Sample 1 was taken and washed with distilled water. 100 g of sample was washed with 350 g of water (to represent 22.2% solids by weight slurry density). The wash water was retained and submitted for analysis of metals. The water analysed 15.2 mg/L Cu, 161 mg/L Fe, 3.41 mg/L Pb and 429 mg/L Zn. These results show quite a high content of metal ions in solution. It would be expected that this amount of metal ions in solution would influence flotation chemistry.

A sample of moist composite Sample 1 was tested for pH; being pH of the tailings paste. It was difficult to get an exact pH because of the inability to satisfactorily contact the tailings paste with the pH electrode. However, several determinations of this gave pH in the range 4.0-5.2.

The results of the sample washing and paste pH determination, indicates some oxidation of sulphides has occurred, putting metal ions into soluble form and generating some acid.

2. Flotation Testwork

2.1. Preliminary Testwork

Five preliminary tests were run on samples of Composite Sample 1. The objective of these tests was to apply fairly standard Pb/Zn flotation reagent regimes in order to gauge the response of this sample.

Test 1 was with the Sample 1. To prepare sample for Tests 2-5 a 10 kg sample of Composite Sample 1 was wet screened on 325 mesh. Tests 2 and 4 were run on -325 mesh material from Sample 1 while the +325 mesh material was tested in Tests 3 and 5. A size analysis of this screen gave 58% of the weight as +325 mesh and 42% of the weight as -325 mesh. The products from wet screening were then filtered and "wet cakes" were cut out for feed samples to flotation. It is important to note this, as this sample preparation technique effectively washed the sample before flotation. Thus some of the soluble metal ions would have been removed by this process, before flotation.

Given in Table 3 are the conditions for Tests 1-5 and the results from this testwork.

TABLE NO. 3 :
Preliminary Flotation Testwork

Test No.	Sample No.	Test Description		Final Concentrate Grade		Final Concentrate Recovery	
				%Pb	%Zn	%Pb	%Zn
1	1	Full feed Pb circuit - soda ash, Aerophine 3418A Zn circuit - lime, copper sulphate and Xanthate 343	Pb Conc	17.90	5.38	6.1	1.2
			Zn Conc	3.49	40.20	1.4	10.3
2	1	-325 mesh sample Pb circuit - soda ash, Aerophine 3418A Zn circuit - lime, copper sulphate and Xanthate 343	Pb Conc	14.00	10.20	10.8	10.0
			Zn Conc	2.29	29.50	1.2	20.4
3	1	+325 mesh sample Pb circuit - soda ash, Aerophine 3418A Zn circuit - lime, copper sulphate and Xanthate 343	Pb Conc	0.14	0.23	2.6	1.5
			Zn Conc	0.11	0.26	0.8	0.7
4	1	-325 mesh sample Pb circuit - soda ash, SD100, Aero 3418A Zn circuit - copper sulphate, lime and Xanthate 343	Pb Conc	9.35	22.10	15.3	44.6
			Zn Conc	1.84	8.76	1.6	9.3
5	1	+325 mesh sample Pb circuit - soda ash, SD100, Aero 3418A Zn circuit - copper sulphate, lime, and Xanthate 343	Pb Conc	0.42	0.48	22.5	10.6
			Zn Conc	0.28	4.95	2.9	20.9

These results show:

- There is potential to make saleable concentrate from this tailing sample. The zinc concentrate from Test 1 (3.5% Pb; 40.2% Zn) is close to being saleable. The bulk (Pb) concentrate from Test 4 (9.4% Pb, 22.1 % Zn) could possibly be upgraded to a level that would be saleable as a bulk concentrate.

- Testwork with the +325 mesh fraction was very poor compared with tests on the -325 mesh fraction. Concentrates that were produced in Tests 3 and 5 were highly pyritic. These tests indicate that most of the recoverable Pb and Zn in this sample as received (no regrinding), was in the -325 mesh fraction.

- The use of a light regrind in Test 5 increased recovery of Pb and Zn but still to a very pyritic concentrate.

- Consumption of pH modifiers soda ash and lime was high, but washing the sample in Tests 2-5, did reduce the demand for pH modifiers somewhat.

Based on these results it was decided to continue all future testwork on the full sample rather than a size split sample.

It should be noted that with Tests 1-5 no attempt was made to optimize or reduce reagent consumption. The objective of these tests was to study metallurgical response and the effect of flotation of different size feeds.

Zn Rougher Tail Mineralogy

A mineralogical examination was carried out upon a portion of the zinc rougher tailings from Test 3. The purpose was to characterize liberation data and mineral distribution by association.

Two polished sections, PS#3411 and PS#3412, were prepared and examined under the ore microscope. The procedures were described with the mineralogy of the head Composite Sample 1.

RESULTS:

The following comments describe the mineralogical observations of the zinc rougher tailings. The detailed mineral distributions by association are described in the accompanying tabulation.

Mineral Liberation, Associations - Zn Rougher Tail

Mineral	Association	Area % Distribution
Sphalerite	liberated	69.0
	binary with pyrite	17.7
	binary with gangue	9.8
	binary with galena	0.5
	binary with chalcopyrite	2.5
	binary with pyrrhotite	0.5
Galena	liberated	31.9 / 27.5
	binary with pyrite	26.4 / 36.4
	binary with gangue	25.8 / 22.2
	binary with sphalerite	14.2 / 12.2
	binary with chalcopyrite	0.8 / 0.7
	binary with pyrrhotite	1.0 / 0.9
Chalcopyrite	liberated	52.0
	binary with pyrite	18.6
	binary with gangue	13.7
	binary with galena	0.7
	binary with sphalerite	14.2
	binary with pyrrhotite	0.8

Sphalerite continued to report as abundant liberated grains (~69 area %) ranging from <20-150 micrometers in diameter. Binaries occurred primarily as interstitial associations within pyrite or intergranular textures hosted by silicate gangue.

Galena ranged from <5-80 micrometers in diameter with approximately 30% liberation. Two data values are reported in the preceding tabulation due to the effects of a single very coarse-grained particle of galena-pyrite (300X 400 micrometers). Pyrite and gangue minerals report as the dominant binary associations, commonly with galena as fine inclusions fully encapsulated, and lesser exposed grains. Oxidation rims on liberated and exposed binary galena were common.

Chalcopyrite displayed liberation data and textures similar to the head composite with approximatley 52% fully liberated and binaries dominated by pyrite and gangue minerals. Grains of <5-150 micrometers were noted.

2.2. Rougher Reagent Screening Testwork

Fourteen tests were run with rougher flotation only, in order to screen a large range of reagent conditions (particularly collector type) with the tailing samples. These tests are summarized in Table 4.

TABLE NO. 4 :
Rougher Flotation Reagent Screening Testwork

Test No.	Sample No.	Test Description	Rougher Concentrate Grade		Rougher Concentrate Recovery	
			%Pb	%Zn	%Pb	%Zn
6	1	Collector TH2552A	0.92	2.69	16.3	36.8
7	1	Collector TH2752A/Acrisol A3 mixture	1.10	2.47	23.1	30.9
8	1	Collector PAX/V4083 mixture	1.64	3.06	19.1	28.7
9	1	Collector R242/Minerec 748/V4083 mixture plus 100gpt copper sulphate	2.09	3.70	23.3	28.6
10	2	Collector modified R242 plus 500 gpt copper sulphate	0.47	1.39	10.4	20.6
11	2	Collector modified R242	1.32	2.96	20.8	32.1
12	2	Collector modified R400	0.65	1.85	15.8	29.3
13	2	Collector modified R400 at pH 6 with lime	1.24	2.91	14.9	21.6
14	2	Collector modified R242 at pH 8 with lime	1.29	2.53	26.7	30.1
15	2	Collector modified R242 at pH 4.5 with lime	1.54	3.94	19.1	34.7
16	2	Collector modified R242 with Na2S/NaCN 50 gpt - 50/50 mixture	2.07	2.82	28.7	27.0
17	2	Collector modified R242 with Na2S 150 gpt	1.12	2.35	18.1	27.4
18	2	Collector modified R242 with Na2S 300 gpt	0.96	1.50	16.9	18.9
19	2	Collector modified R242 with thiourea 150 gpt and R3477 collector 20 gpt	2.25	1.43	27.0	12.2

The following observations are made with the results from Tests 6-19.

- none of the tests gave significant Pb/Zn recovery. Pb recoveries were between 10-29% with grades from 0.9-2.1% Pb and Zn recoveries were 12-37% with grades from 1.4-3.9% Zn.

- no particular reagent regrime stands out as preferable with many of the results very similar. Test 15 gave the best overall Zn result.

These tests suggest more work is required to come up with an appropriate rougher flotation reagent regime.

3. Gravity Testwork

One test, Test 20, was a gravity test, using a Wilfley and Mozley table to concentrate Pb/Zn minerals. The result of this test is given in Table 5.

TABLE NO. 5 :
Gravity Testwork Results

Product	Weight %	Conc. Grade		Conc. Recovery	
		%Pb	%Zn	%Pb	%Zn
Mozley Table Concentrate	37.0	0.38	0.89	21.5	36.5
Wilfley Table Concentrate	46.6	0.38	0.84	26.7	43.3
Wilfley Table Tail	53.4	0.90	0.96	73.3	56.7
Head (calc.)	100.0	0.66	0.90	100.0	100.0

This result shows no successful concentration of Pb/Zn to a gravity concentrate.

CONCLUSIONS

Testwork was conducted on tailings from Faro, Yukon for Curragh Resources Inc. The sample tested assayed 0.79% Pb, 1.23 % Zn, 15.2 g/t Ag and 0.11 g/t Au (preliminary head analysis).

Testwork indicated that the material was partially oxidized with a significant level of Cu, Pb, Zn and Fe metal ions in solution when washed. Mineralogy indicated that a significant amount of the sphalerite was liberated but most of the galena was unliberated.

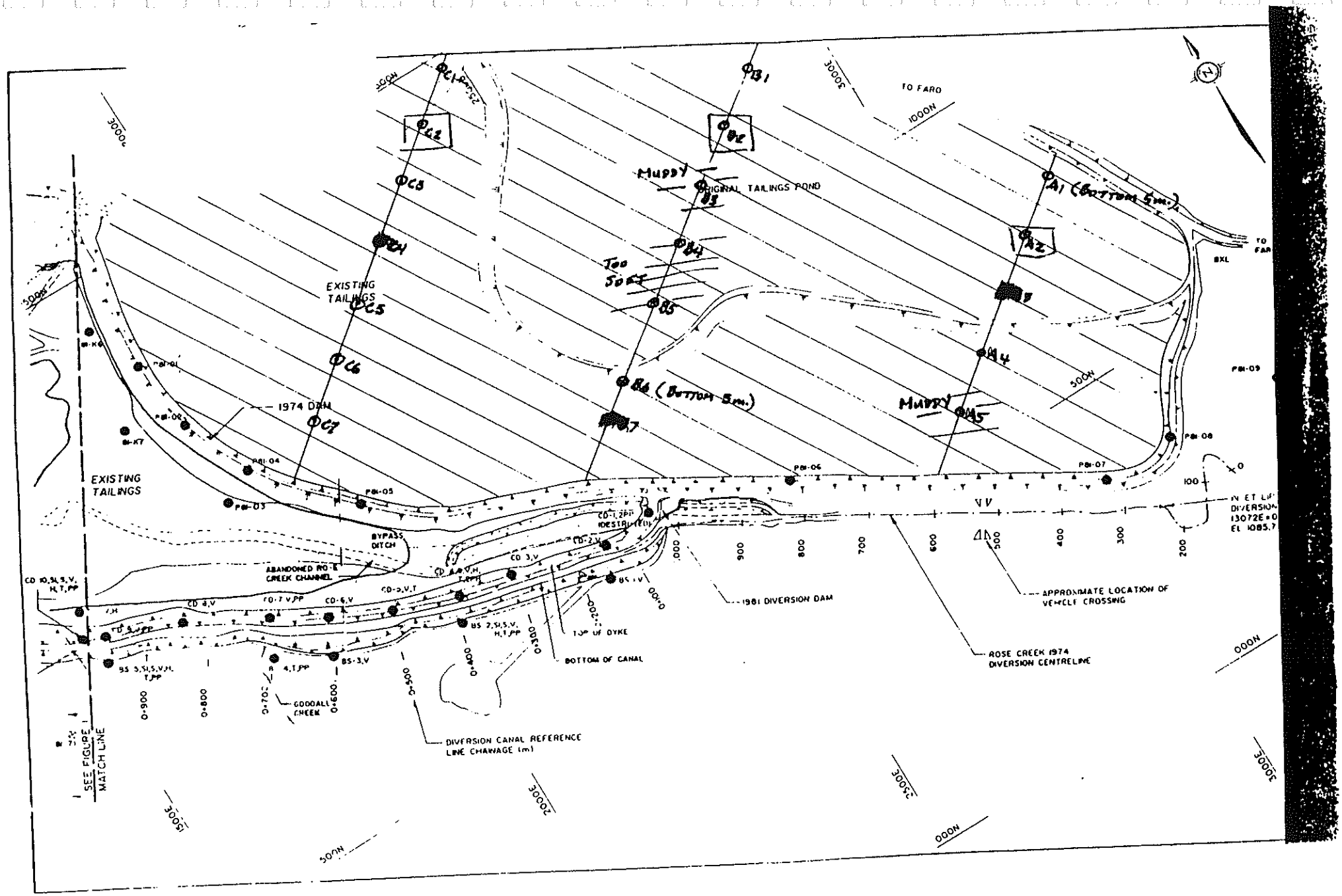
Flotation testwork showed that using a conventional Pb/Zn reagent regime a saleable Zn or Pb/Zn concentrate could probably be produced, albeit at low recovery. Testwork aimed at establishing a rougher reagent regime did not indicate a preferable route. More testwork is required in this area.

DETAILS OF TESTWORK

1. Sample Description

   Map showing location of sample points A2, B2, C2

2. Testwork Results



TO FARO
ORIGINAL TAILINGS POND
MUDDY
TOO SOFT
EXISTING TAILINGS
1974 DAM
EXISTING TAILINGS
BYPASS DITCH
ABANDONED RO-8 CREEK CHANNEL
GOODALL CREEK
TOP OF DYKE
BOTTOM OF CANAL
1981 DIVERSION DAM
APPROXIMATE LOCATION OF VEHICLE CROSSING
ROSE CREEK 1974 DIVERSION CENTRELINE
DIVERSION CANAL REFERENCE LINE CHAINAGE (m)
SEE FIGURE 1 MATCH LINE

Test No. 1 Project No. 4208 Operator: LP Date: 18/10/90

Purpose: To investigate the flotation response of the Faro tailings.

Procedure: As outlined below.

Feed: 2000 grams Faro tailings
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne							Time, minutes			
	Na2CO3	3418A	MIBC	CuSO4	Ca(OH)2	Ax343	NaCN	Grind	Cond.	Froth	pH
Condition	6000	-	-	-	-	-	-		5		4.0-7.0
Pb Rougher	-	10	10	-	-	-	-		2	2	6.7
	-	6.7	-	-	-	-	-		1	2	
Zn Rougher	-	-	-	200	-	-	-		5		
	-	-	-	-	4800	-	-		1		6.8-11.1
	-	-	-	-	-	15	-		1		
	-	-	10	-	-	-	-		1	3	10.3
	-	-	-	-	775	-	-		1		9.9-11.1
	-	-	10	-	-	8	-		1	2	
	-	-	5	-	-	-	-		1	2	
Pb Cleaner	340	-	-	-	-	-	-		2		7.0-9.4
	-	-	-	-	-	-	100		5		
	-	5	-	-	-	-	-		1	3	
Zn 1st Cleaner	-	-	5	-	210	-	-		2	3	9.6-11.6
Zn 2nd Cleaner	-	-	7.5	-	185	-	-		5	1	10.3-11.5
	-	-	5	-	-	-	-		1	1	

Stage	Rougher									
Flotation Cell	500 g D-1									
Speed: r.p.m.	1500									
% Solids										
%- mesh										

Test No. 1

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Pb Cl Conc	4.3	0.2	17.9	5.38	283	1.14	6.1	1.2	3.8	1.4
2. Pb Cl Tail	27.5	1.5	1.74	3.73	32.2	0.14	3.8	5.1	2.8	1.1
3. Zn 2nd Cl Conc	5.1	0.3	3.49	40.2	142	5.01	1.4	10.3	2.3	7.1
4. Zn 2nd Cl Tail	5.3	0.3	3.43	15.6	110	1.08	1.4	4.1	1.8	1.6
5. Zn 1st Cl Tail	41.0	2.3	2.16	2.78	45.7	0.27	7.1	5.7	5.9	3.1
6. Zn Ro Tail	1706.3	95.4	0.59	0.86	15.6	0.18	80.1	73.6	83.4	85.7
Head (calc.)	1789.5	100.0	0.70	1.11	17.8	0.20	100.0	100.0	100.0	100.0
Calculated Grades and Recoveries										
Pb Ro Conc (Prod 1+2)		1.8	3.93	3.95	66.1	0.28	9.9	6.3	6.6	2.4
Zn 1st Cl Conc (Prod 3+4)		0.6	3.46	27.7	126	3.01	2.9	14.4	4.1	8.7
Zn Ro Conc (Prod 3-5)		2.9	2.42	7.81	61.9	0.82	9.9	20.1	10.0	11.8
Comb Conc (Prod 1-5)		4.6	3.00	6.34	63.5	0.61	19.9	26.4	16.6	14.3

Test No. 2 Project No. 4208 Operator: BW Date: 18/10/90

Purpose: To investigate the flotation response of the -325 mesh Faro tailings.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings - 325 mesh
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. Approximately 10 kg of sample was wet screened on a 325 mesh screen. The plus and minus fractions were filtered. 1 kg charges were cut from the fractions. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne							Time, minutes			
	Na2CO3	3418A	MIBC	CuSO4	Ca(OH)2	Ax343	NaCN	Grind	Cond.	Froth	pH
Condition	5000	-	-	-	-	-	100		5		8.5-7.2
Pb Rougher	-	20	20	-	-	-	-		2	2	7.2
	-	10	20	-	-	-	-		1	2	
Zn Rougher	-	-	-	200	-	-	-		5		7.1
	-	-	-	-	3100	-	-		1		11.0
	-	-	-	-	-	15	-		1		10.1
	-	-	10	-	-	-	-		1	3	
	-	-	10	-	-	8	-		1	2	
Pb Cleaner	900	-	-	-	-	-	-		2		9.5-9.4
	-	-	-	-	-	-	50		5		
	-	5	10	-	-	-	-		1	3	
Zn Cleaner	-	-	-	-	1480	-	-		5		12.0
	-	5	-	-	-	4	-		1	2	

Stage	Rougher									
Flotation Cell	500 g D-1									
Speed: r.p.m.	1500									
% Solids										
%- mesh										

Test No. 2

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Pb Cl Conc	9.8	1.0	14.0	10.2	204	3.10	10.8	10.0	9.7	22.3
2. Pb Cl Tail	47.5	4.7	2.19	3.5	39.2	0.17	8.2	16.5	9.0	5.9
3. Zn Cl Conc	6.9	0.7	2.29	29.5	90.5	1.50	1.2	20.4	3.0	7.6
4. Zn Cl Tail	22.1	2.2	2.20	2.53	58.5	0.59	3.8	5.6	6.3	9.6
5. Zn Ro Tail	929.5	91.5	1.04	0.51	16.0	0.08	76.0	47.5	72.0	54.6
Head (calc.)	1015.8	100.0	1.25	0.98	20.3	0.13	100.0	100.0	100.0	100.0
Calculated Grades and Recoveries										
Pb Ro Conc (Prod 1+2)		5.6	4.21	4.62	67.4	0.67	19.0	26.5	18.7	28.2
Zn Ro Conc (Prod 3+4)		2.9	2.22	8.95	66.1	0.81	5.1	26.0	9.3	17.2
Comb Conc (Prod 1-4)		8.5	3.54	6.07	67.0	0.72	24.0	52.5	28.0	45.4

Test No. 3 Project No. 4208 Operator: LP Date: 18/10/90

Purpose: To investigate the flotation response of the +325 mesh Faro tailings.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings +325 mesh
Sample Preparation: As for test 2.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne							Time, minutes			pH
	Na2CO3	3418A	MIBC	CuSO4	Ca(OH)2	Ax343	NaCN	Grind	Cond.	Froth	
Condition	1520	-	-	-	-	-	-		5		5.3-9.5
Pb Rougher	-	20	20	-	-	-	-		2	1	8.7
Zn Rougher	-	-	-	200	-	-	-		5		7.7
	-	-	-	-	1280	-	-		1		11.1
	-	-	-	-	-	15	-		1		
	-	-	10	-	-	-	-		1	0.5	

Stage	Rougher									
Flotation Cell	500 g D-1									
Speed: r.p.m.	1500									
% Solids										
%- mesh										

Test No. 3

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Pb Ro Conc	67.1	6.3	0.14	0.23	3.40	0.08	2.6	1.5	1.7	5.2
2. Zn Ro Conc	27.4	2.6	0.11	0.26	3.10	0.07	0.8	0.7	0.6	1.9
3. Zn Ro Tail	963.5	91.1	0.36	1.08	13.9	0.10	96.5	97.9	97.7	93.0
Head (calc.)	1058.0	100.0	0.34	1.00	13.0	0.10	100.0	100.0	100.0	100.0
Calculated Grades and Recoveries										
Comb Conc (Prod 1+2)		8.9	0.13	0.24	3.31	0.08	3.5	2.1	2.3	7.0

Test No. 4 Project No. 4208 Operator: LP Date: 23/10/90

Purpose: As for test 2 with the addition of SD-100.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings - 325 mesh
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. Approximately 10 kg of sample was wet screened on a 325 mesh screen. The plus and minus fractions were filtered. 1 kg charges were cut from the fractions. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne								Time, minutes			pH
	SD100	Na2CO3	3418A	MIBC	CuSO4	Ca(OH)2	Ax343	NaCN	Grind	Cond.	Froth	
Condition	-	5740	-	-	-	-	-	-		5		4.9-9.0
High Speed Cond.	200	-	-	-	-	-	-	-		10		
Pb Rougher	-	-	20	20	-	-	-	-		2	2	8.0
		-	10	20	-	-	-	-		1	2	
Zn Rougher	-	-	-	-	200	-	-	-		5		
	-	-	-	-	-	2400	-	-		1		7.7-11.1
	-	-	-	-	-	-	15	-		1		
	-	-	-	10	-	-	-	-		1	3	10.4
	-	-	-	10	-	-	8	-		1	2	
Pb 1st Cleaner	-	360	-	-	-	-	-	-		2		9.6
	-	-	-	-	-	-	-	50		5		
	-	-	5	10	-	-	-	-		1	3	9.4
Pb 2nd Cleaner	-	200	-	-	-	-	-			1	2	8.5-9.9
Zn Cleaner	-	-	-	-	-	260	-	-		5		9.4-11.5
	-	-	5	-	-	-	4	-		1	2	11.0

Stage	High Speed Cond.	Roughers	Cleaners						
Flotation Cell	500 g D-1	500 g D-1	250 g D-1						
Speed: r.p.m.	2000	1500	1200						
% Solids									
%- mesh									

Test No. 4

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Pb 2nd Cl Conc	19.4	1.9	9.35	22.1	192	1.34	15.3	44.6	21.2	10.4
2. Pb 2nd Cl Tail	7.5	0.8	3.23	1.69	70.9	0.66	2.0	1.3	3.0	2.0
3. Pb 1st Cl Tail	50.4	5.0	1.79	0.46	27.2	0.23	7.6	2.4	7.8	4.6
4. Zn Cl Conc	10.2	1.0	1.84	8.76	61.7	2.66	1.6	9.3	3.6	10.8
5. Zn Cl Tail	57.0	5.7	1.71	0.55	25.3	0.32	8.2	3.3	8.2	7.3
6. Zn Ro Tail	853.6	85.5	0.91	0.44	11.6	0.19	65.3	39.1	56.3	64.8
Head (calc.)	998.1	100.0	1.19	0.96	17.6	0.25	100.0	100.0	100.0	100.0
Calculated Grades and Recoveries										
Pb 1st Cl Conc (Prod 1+2)		2.7	7.64	16.4	158	1.15	17.3	45.9	24.2	12.4
Pb Ro Conc (Prod 1-3)		7.7	3.83	6.01	72.8	0.55	24.9	48.4	32.0	17.0
Zn Ro Conc (Prod 4+5)		6.7	1.73	1.80	30.8	0.68	9.8	12.6	11.8	18.1
Comb Conc (Prod 1-5)		14.5	2.85	4.05	53.3	0.61	34.7	60.9	43.7	35.2

Test No. 5 Project No. 4208 Operator: LP Date: 23/10/90

Purpose: As for test 4 with a primary grind.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings + 325 mesh
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. Approximately 10 kg of sample was wet screened on a 325 mesh screen. The plus and minus fractions were filtered. 1 kg charges were cut from the fractions. The sample was stored moist.

Grind: 7 minutes/kg at 50 percent solids in a lab ball mill

Conditions:

Stage	Reagents added, grams per tonne								Time, minutes			pH
	SD100	Na2CO3	3418A	MIBC	CuSO4	Ca(OH)2	Ax343	NaCN	Grind	Cond.	Froth	
Grind	-	-	-	-	-	-	-	-	7			
Condition	-	2000	-	-	-	-	-	-		5		6.0-9.2
High Speed Cond.	200	-	-	-	-	-	-	-		10		
Pb Rougher	-	-	20	20	-	-	-	-		2	0.5	7.3
Zn Rougher	-	-	-	-	200	-	-	-		5		
	-	-	-	-	-	1140	-	-		1		7.6-11.1
	-	-	-	-	-	-	15	-		1		
	-	-	-	10	-	-	-	-		1	0.5	10.1

Stage	High Speed Cond.	Roughers								
Flotation Cell	500 g D-1	500 g D-1								
Speed: r.p.m.	2000	1500								
% Solids										
%- mesh										

Test No. 5

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Pb Ro Conc	216.7	21.0	0.42	0.48	8.70	0.19	22.5	10.6	13.4	28.1
2. Zn Ro Conc	41.5	4.0	0.28	4.95	31.0	0.31	2.9	20.9	9.1	8.8
3. Zn Ro Tail	772.8	75.0	0.39	0.87	14.1	0.12	74.6	68.5	77.5	63.2
Head (calc.)	1031.0	100.0	0.39	0.95	13.6	0.14	100.0	100.0	100.0	100.0
Calculated Grades and Recoveries										
Comb Conc (Prod 1+2)		25.0	0.40	1.20	12.3	0.21	25.4	31.5	22.5	36.8

Test No. 6 Project No. 4208 Operator: LP Date: 30/10/91

Purpose: To investigate the effect of TH2752A on the flotation response of the Faro tailings.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	CuSO4		TH2752A		MIBC		Grind	Cond.	Froth	
Condition	50		-		-			5		3.9
Rougher 1	-		-		10			1	2	
Rougher 2	-		8		-			1	2	
	-		-		10			1	2	
Rougher 3	-		8		10			1	2	

Stage	Rougher								
Flotation Cell	500 g D-1								
Speed: r.p.m.	1500								
% Solids									
%- mesh									

Test No. 6

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	10.5	1.1	4.70	10.1	89.4	0.41	6.1	10.1	6.0	4.0
2. Ro Conc 2	48.8	5.0	1.22	3.53	29.3	0.19	7.3	16.4	9.1	8.6
3. Ro Conc 3	84.1	8.6	0.28	1.27	11.3	0.12	2.9	10.2	6.1	9.4
4. Ro Tail	838.4	85.4	0.81	0.79	14.7	0.10	83.7	63.2	78.8	78.0
Head (calc.)	981.8	100.0	0.83	1.07	15.9	0.11	100.0	100.0	100.0	100.0
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		6.0	1.84	4.69	39.9	0.23	13.4	26.6	15.1	12.6
Ro Conc 1-3 (Prod 1-3)		14.6	0.92	2.69	23.1	0.17	16.3	36.8	21.2	22.0

Test No. 7 Project No. 4208 Operator: LP Date: 30/10/91

Purpose: To investigate the effect of the TH2752A/Acrisol A3 mixture on the flotation response of the Faro tailings.

Procedure: As outlined below. .

Feed: 1000 grams Faro tailings
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	CuSO4		*TH2752A/ Acrisol A3		MIBC		Grind	Cond.	Froth	
Condition	100		-		-			5		3.9
Rougher 1	-		-		20			1	2	
Rougher 2	-		4		-			1	2	
	-		-		10			1	2	
Rougher 3	-		4		10			1	2	
	-		4		-			1	1	
	-		4		-			1	1	
	-		4		-			1	1	
	-		4		-			1	1	
	-		10		-			1	1	
	-		10		10			1	1	
	-		10		-			1	1	
	-		10		-			1	1	

Stage	Rougher	*Mixture ratio 80:20 TH2752A:Acrisol A3.							
Flotation Cell	500 g D-1								
Speed: r.p.m.	1500								
% Solids									
%- mesh									

Test No. 7

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	11.7	1.2	3.54	10.9	75.1	0.32	6.6	12.0	5.2	2.8
2. Ro Conc 2	18.3	1.9	1.96	4.20	47.9	0.22	5.7	7.3	5.1	3.0
3. Ro Conc 3	102.1	10.6	0.66	1.20	23.4	0.19	10.8	11.6	14.0	14.3
4. Ro Tail	831.2	86.3	0.58	0.88	15.5	0.13	76.9	69.1	75.7	79.9
Head (calc.)	963.3	100.0	0.65	1.10	17.7	0.14	100.0	100.0	100.0	100.0
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		3.1	2.58	6.81	58.5	0.26	12.3	19.3	10.3	5.7
Ro Conc 1-3 (Prod 1-3)		13.7	1.10	2.47	31.4	0.21	23.1	30.9	24.3	20.1

Test No. 8 Project No. 4208 Operator: LP Date: 30/10/91

Purpose: To investigate the effect of the Ax350/V4083 mixture on the flotation response of the Faro tailings.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	CuSO4		*Ax350/ V4083		MIBC		Grind	Cond.	Froth	
Condition	100		-		-			5		3.9
Rougher 1	-		-		20			1	2	
	-		-		20			1	2	
Rougher 2	-		4		-			1	2	
	-		4		10			1	2	
Rougher 3	-		4		10			1	2	
	-		4		-			1	2	

Stage	Rougher	*Recipe:	1 gram V4083 in 100 mL 0.2 % NaOH						
Flotation Cell	500 g D-1		solution with 3 gram Ax350.						
Speed: r.p.m.	1500								
% Solids									
%- mesh									

Test No. 8

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	31.1	3.2	2.72	5.95	35.4	0.27	10.8	18.9	9.1	5.5
2. Ro Conc 2	32.2	3.3	1.06	2.02	24.8	0.18	4.3	6.7	6.6	3.8
3. Ro Conc 3	28.4	2.9	1.10	1.08	38.1	0.18	4.0	3.1	8.9	3.4
4. Ro Tail	882.6	90.6	0.72	0.79	10.4	0.15	80.9	71.3	75.5	87.3
Head (calc.)	974.3	100.0	0.81	1.00	12.5	0.16	100.0	100.0	100.0	100.0
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		6.5	1.88	3.95	30.0	0.22	15.1	25.6	15.6	9.4
Ro Conc 1-3 (Prod 1-3)		9.4	1.64	3.06	32.5	0.21	19.1	28.7	24.5	12.7

Test No. 9 Project No. 4208 Operator: LP Date: 30/10/91

Purpose: To investigate the effect of the R242/Minerec 748/V4083 mixture on the flotation response of the Faro tailings.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	CuSO4		*R242/V4083/ Minerec 748		MIBC		Grind	Cond.	Froth	
Condition	100		-		-			5		3.9
Rougher 1	-		-		20			1	2	
	-		-		20			1	2	
Rougher 2	-		4		-			1	2	
	-		4		10			1	2	
Rougher 3	-		4		10			1	2	
	-		4		-			1	2	

Stage	Rougher	*Mixture ratio 1:1:1 R242/V4083/Minerec 748.								
Flotation Cell	500 g D-1									
Speed: r.p.m.	1500									
% Solids										
%- mesh										

Test No. 9

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	29.9	3.1	2.98	6.20	58.9	0.27	12.5	18.0	11.8	6.4
2. Ro Conc 2	30.5	3.1	1.70	2.53	31.3	0.21	7.3	7.5	6.4	5.1
3. Ro Conc 3	19.0	1.9	1.33	1.66	28.3	0.20	3.5	3.1	3.6	3.0
4. Ro Tail	895.6	91.9	0.61	0.82	13.1	0.12	76.7	71.4	78.3	85.5
Head (calc.)	975.0	100.0	0.73	1.05	15.4	0.13	100.0	100.0	100.0	100.0
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		6.2	2.33	4.35	45.0	0.24	19.8	25.5	18.1	11.5
Ro Conc 1-3 (Prod 1-3)		8.1	2.09	3.70	41.0	0.23	23.3	28.6	21.7	14.5

Test No. 10 Project No. 4208 Operator: LP Date: 05/11/91

Purpose: To examine different collector on bulk flotation.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings (Sample 2)
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	Ca(OH)2		Modified R242	Ax350	MIBC		Grind	Cond.	Froth	
Condition	500		-	-	-			5		4.0
Rougher 1	-		2	5	-			1	3	
Rougher 2	-		2	-	-			1	3	
Rougher 3	-		4	-	-			1	3	

Stage	Rougher								
Flotation Cell	500 g D-1								
Speed: r.p.m.	1500								
% Solids									
%- mesh									

Test No. 10

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	24.0	2.4	0.60	1.74	14.3	-	2.3	4.3	2.7	-
2. Ro Conc 2	21.2	2.1	1.17	1.46	19.6	-	3.9	3.2	3.2	-
3. Ro Conc 3	96.8	9.6	0.28	1.29	8.50	-	4.2	13.0	6.4	-
4. Ro Tail	867.5	85.9	0.66	0.88	13.0	-	89.6	79.4	87.7	-
Head (calc.)	1009.5	100.0	0.63	0.95	12.7	-	100.0	100.0	100.0	-
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		4.5	0.87	1.61	16.8	-	6.1	7.6	5.9	-
Ro Conc 1-3 (Prod 1-3)		14.1	0.47	1.39	11.1	-	10.4	20.6	12.3	-

Test No. 12 Project No. 4208 Operator: LP Date: 05/11/91

Purpose: To repeat the conditions of test 11 but with modified R-400.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings (Sample 2)
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
			Modified R400		MIBC		Grind	Cond.	Froth	
Condition			-		-			3		3.5
Rougher 1			2		-			1	3	
Rougher 2			4		-			1	4	
Rougher 3			4		-			1	4	

Stage	Rougher								
Flotation Cell	500 g D-1								
Speed: r.p.m.	1500								
% Solids									
%- mesh									

Test No. 12

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	55.8	5.5	1.09	3.54	36.6	-	10.1	21.4	13.4	-
2. Ro Conc 2	47.8	4.7	0.57	1.17	37.8	-	4.5	6.0	11.8	-
3. Ro Conc 3	43.2	4.3	0.17	0.41	5.90	-	1.2	1.9	1.7	-
4. Ro Tail	859.7	85.4	0.59	0.76	13.0	-	84.2	70.7	73.1	-
Head (calc.)	1006.5	100.0	0.60	0.92	15.2	-	100.0	100.0	100.0	-
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		10.3	0.85	2.45	37.2	-	14.6	27.4	25.2	-
Ro Conc 1-3 (Prod 1-3)		14.6	0.65	1.85	28.0	-	15.8	29.3	26.9	-

Test No. 11 Project No. 4208 Operator: LP Date: 05/11/91

Purpose: To investigate the flotation of bulk concentrate using modified collector R242.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings (Sample 2)
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
			Modified R242		MIBC		Grind	Cond.	Froth	
Condition			-		-			5		3.5
Rougher 1			2		-			1	3	
Rougher 2			4		2			1	4	
Rougher 3			4		2			1	4	

Stage	Rougher								
Flotation Cell	500 g D-1								
Speed: r.p.m.	1500								
% Solids									
%- mesh									

Test No. 11

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	23.1	2.3	2.35	6.45	60.8	-	8.5	16.0	7.4	-
2. Ro Conc 2	30.3	3.0	1.26	3.24	25.9	-	6.0	10.5	4.1	-
3. Ro Conc 3	47.7	4.8	0.85	1.09	18.0	-	6.4	5.6	4.5	-
4. Ro Tail	902.6	89.9	0.56	0.70	17.7	-	79.2	67.9	84.0	-
Head (calc.)	1003.7	100.0	0.64	0.93	19.0	-	100.0	100.0	100.0	-
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		5.3	1.73	4.63	41.0	-	14.5	26.5	11.5	-
Ro Conc 1-3 (Prod 1-3)		10.1	1.32	2.96	30.1	-	20.8	32.1	16.0	-

Test No. 13 Project No. 4208 Operator: LP Date: 05/11/91

Purpose: To repeat the conditions of test 11 but at pH 6.0

Procedure: As outlined below.

Feed: 1000 grams Faro tailings (Sample 2)
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	Ca(OH)2		Modified R242		MIBC		Grind	Cond.	Froth	
Condition	700		-		-			3		6.0
Rougher 1	-		2		-			1	3	
Rougher 2	-		4		-			1	4	
Rougher 3	-		4		-			1	4	

Stage	Rougher
Flotation Cell	500 g D-1
Speed: r.p.m.	1500
% Solids	
%- mesh	

Test No. 13

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	20.7	2.0	2.35	6.22	68.4	-	7.6	12.4	9.8	-
2. Ro Conc 2	12.9	1.3	1.39	3.08	32.5	-	2.8	3.8	2.9	-
3. Ro Conc 3	43.5	4.3	0.66	1.29	14.7	-	4.5	5.4	4.4	-
4. Ro Tail	938.0	92.4	0.58	0.87	12.7	-	85.1	78.4	82.8	-
Head (calc.)	1015.1	100.0	0.63	1.03	14.2	-	100.0	100.0	100.0	-
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		3.3	1.98	5.01	54.6	-	10.4	16.2	12.8	-
Ro Conc 1-3 (Prod 1-3)		7.6	1.24	2.91	32.1	-	14.9	21.6	17.2	-

Test No. 14 Project No. 4208 Operator: LP Date: 05/11/91

Purpose: To repeat the conditions of test 11 at pH 8.0.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings (Sample 2)
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	Ca(OH)2		Modified R242		MIBC		Grind	Cond.	Froth	
Condition	1400		-		-			3		8.0
Rougher 1	-		2		2			1	3	
Rougher 2	-		4		-			1	4	
Rougher 3	-		4		-			1	4	

Stage	Rougher
Flotation Cell	500 g D-1
Speed: r.p.m.	1500
% Solids	
%- mesh	

Test No. 14

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	66.0	6.4	1.08	2.44	23.8	-	10.5	13.6	9.9	-
2. Ro Conc 2	48.5	4.7	1.45	2.90	27.2	-	10.4	11.9	8.3	-
3. Ro Conc 3	26.1	2.5	1.51	2.06	24.0	-	5.8	4.6	3.9	-
4. Ro Tail	888.0	86.3	0.56	0.93	13.9	-	73.3	69.9	77.8	-
Head (calc.)	1028.6	100.0	0.66	1.15	15.4	-	100.0	100.0	100.0	-
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		11.1	1.24	2.63	25.2	-	20.9	25.5	18.2	-
Ro Conc 1-3 (Prod 1-3)		13.7	1.29	2.53	25.0	-	26.7	30.1	22.2	-

Test No. 15 Project No. 4208 Operator: LP Date: 05/11/91

Purpose: To repeat the conditions of test 11 but increase pH to 4.5.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings (Sample 2)
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	Ca(OH)2		Modified R242		MIBC		Grind	Cond.	Froth	
Condition	400		-		-			3		4.5
Rougher 1	-		2		2			1	3	
Rougher 2	-		4		-			1	4	
Rougher 3	-		4		-			1	4	

Stage	Rougher									
Flotation Cell	500 g D-1									
Speed: r.p.m.	1500									
% Solids										
%- mesh										

Test No. 15 Project No. 4208 Operator: LP Date: 05/11/91

Purpose: To repeat the conditions of test 11 but increase pH to 4.5.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings (Sample 2)
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	Ca(OH)2		Modified R242		MIBC		Grind	Cond.	Froth	
Condition	400		-		-			3		4.5
Rougher 1	-		2		2			1	3	
Rougher 2	-		4		-			1	4	
Rougher 3	-		4		-			1	4	

Stage	Rougher								
Flotation Cell	500 g D-1								
Speed: r.p.m.	1500								
% Solids									
%- mesh									

Test No. 16 Project No. 4208 Operator: LP Date: 08/11/91

Purpose: To investigate the effect of Na2S/NaCN and modified collector R242 on the flotation response of the Faro tailings.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings (Sample 2)
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	Na2S/NaCN 50/50		Modified R242		MIBC		Grind	Cond.	Froth	
Condition	-		-		-			3		3.3
	50		-		-			3		
Rougher 1	-		12		2			1	2.5	
Rougher 2	-		6		-			1	3	
Rougher 3	-		6		-			1	3	

Stage	Rougher
Flotation Cell	500 g D-1
Speed: r.p.m.	1500
% Solids	
%- mesh	

Test No. 16

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	38.9	3.9	2.39	3.03	43.6	-	14.6	12.8	13.1	-
2. Ro Conc 2	30.3	3.0	1.97	2.21	32.9	-	9.4	7.3	7.7	-
3. Ro Conc 3	19.1	1.9	1.58	3.37	30.4	-	4.7	7.0	4.5	-
4. Ro Tail	909.8	91.2	0.50	0.74	10.6	-	71.3	73.0	74.7	-
Head (calc.)	998.1	100.0	0.64	0.92	12.9	-	100.0	100.0	100.0	-
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		6.9	2.21	2.67	38.9	-	23.9	20.0	20.8	-
Ro Conc 1-3 (Prod 1-3)		8.8	2.07	2.82	37.1	-	28.7	27.0	25.3	-

Test No. 17 Project No. 4208 Operator: LP Date: 08/11/91

Purpose: To investigate the effect of Na2S and modified collector R242 on the flotation response of the Faro tailings.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings (Sample 2)
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	Na2S		Modified R242		MIBC		Grind	Cond.	Froth	
Condition	150		-		-			10		3.7
Rougher 1	-		-		5			1	3	
Rougher 2	-		6		-			1	3	
Rougher 3	-		6		-			1	3	

Stage	Rougher									
Flotation Cell	500 g D-1									
Speed: r.p.m.	1500									
% Solids										
%- mesh										

Test No. 17

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	41.7	4.2	1.83	3.78	42.8	-	11.6	17.3	13.8	-
2. Ro Conc 2	35.4	3.5	0.65	1.81	21.1	-	3.5	7.0	5.8	-
3. Ro Conc 3	29.4	2.9	0.67	0.97	16.6	-	3.0	3.1	3.8	-
4. Ro Tail	895.3	89.4	0.60	0.74	11.1	-	81.9	72.6	76.7	-
Head (calc.)	1001.8	100.0	0.66	0.91	12.9	-	100.0	100.0	100.0	-
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		7.7	1.29	2.88	32.8	-	15.1	24.3	19.5	-
Ro Conc 1-3 (Prod 1-3)		10.6	1.12	2.35	28.4	-	18.1	27.4	23.3	-

Test No. 18 Project No. 4208 Operator: LP Date: 08/11/91

Purpose: To repeat test 17 but increase Na2S addition.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings (Sample 2)
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	Na2S		Modified R242		MIBC		Grind	Cond.	Froth	
Condition	300		-		-			10		3.9
Rougher 1	-		-		-			1	3	
Rougher 2	-		6		-			1	3	
Rougher 3	-		6		-			1	3	

Stage	Rougher
Flotation Cell	500 g D-1
Speed: r.p.m.	1500
% Solids	
%- mesh	

Test No. 18

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	24.8	2.5	2.34	3.09	47.9	-	8.7	8.2	9.8	-
2. Ro Conc 2	25.4	2.5	1.24	2.06	31.7	-	4.7	5.6	6.6	-
3. Ro Conc 3	67.0	6.7	0.34	0.70	15.0	-	3.4	5.0	8.3	-
4. Ro Tail	889.5	88.4	0.62	0.85	10.3	-	83.1	81.1	75.3	-
Head (calc.)	1006.7	100.0	0.66	0.93	12.1	-	100.0	100.0	100.0	-
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		5.0	1.78	2.57	39.7	-	13.5	13.8	16.4	-
Ro Conc 1-3 (Prod 1-3)		11.6	0.96	1.50	25.6	-	16.9	18.9	24.7	-

Test No. 19 Project No. 4208 Operator: LP Date: 08/11/91

Purpose: To investigate the effect of thiourea and modified collector R242 on the flotation response of the Faro tailings.

Procedure: As outlined below.

Feed: 1000 grams Faro tailings (Sample 2)
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Conditions:

Stage	Reagents added, grams per tonne						Time, minutes			pH
	Thiourea		Modified R242	R3477	MIBC		Grind	Cond.	Froth	
Condition	150		-	-	-			10		3.5
Rougher 1	-		-	-	-			1	3	
Rougher 2	-		6	-	10			1	3	
Rougher 3	-		-	10	-			1	2.5	
Rougher 4	-		-	10	-			1	3	

Stage	Rougher								
Flotation Cell	500 g D-1								
Speed: r.p.m.	1500								
% Solids									
%- mesh									

Test No. 19

Metallurgical Balance

Product	Weight		Assays %, g/t				% Distribution			
	(g)	(%)	Pb	Zn	Ag	Au	Pb	Zn	Ag	Au
1. Ro Conc 1	20.8	2.1	2.75	1.91	47.0	-	8.7	4.3	7.7	-
2. Ro Conc 2	17.9	1.8	2.17	1.20	32.2	-	5.9	2.3	4.6	-
3. Ro Conc 3	26.0	2.6	1.96	1.19	33.5	-	7.7	3.3	6.9	-
4. Ro Conc 4	14.1	1.4	2.16	1.47	37.4	-	4.6	2.2	4.2	-
5. Ro Tail	924.1	92.1	0.52	0.88	10.5	-	73.0	87.8	76.7	-
Head (calc.)	1002.9	100.0	0.66	0.92	12.6	-	100.0	100.0	100.0	-
Calculated Grades and Recoveries										
Ro Conc 1+2 (Prod 1+2)		3.9	2.48	1.58	40.2	-	14.6	6.6	12.3	-
Ro Conc 1-3 (Prod 1-3)		6.5	2.27	1.42	37.5	-	22.3	10.0	19.2	-
Ro Conc 1-4 (Prod 1-4)		7.9	2.25	1.43	37.5	-	27.0	12.2	23.3	-

Test No. 20 Project No. 4208 Operator: LP Date: 18/11/91

Purpose: To conduct a gravity separation test on the mesh Faro tailings.

Procedure: The sample was treated by gravity separation on a 1/8th size Wilfley table. The concentrate and tailings were collected. The table concentrate was further upgraded on a Mozley mineral separator. The table tailings, the Mozley concentrate and Mozley tail were filtered and assayed.

Feed: 2000 grams Faro tailings
Sample Preparation: Three samples A2, B2, C2 were mixed thoroughly and any lumps were broken down by hand. The sample was stored moist.

Grind: nil

Test No. 20

Product	Weight		Assays %		% Distribution	
	g	%	Pb	Zn	Pb	Zn
1. Mozley Table Conc	364.1	37.0	0.38	0.89	21.5	36.5
2. Mozley Table Tail	94.4	9.6	0.36	0.64	5.3	6.8
3. Wilfley Table Tail	524.9	53.4	0.90	0.96	73.3	56.7
Head (calc)	983.4	100.0	0.66	0.90	100.00	100.00
Combined Products						
Wilfley Table Conc		46.6	0.38	0.84	26.7	43.3

FARO DECOMMISSIONING

OVERVIEW OF THE ENVIRONMENTAL PLANS

APPENDIX D

Letter Report on Geotechnical considerations for Faro Pit Decommissioning, File #85-80413, Alan F. Stewart, P. Eng., Piteau & Associates Engineering Ltd., December 13, 1991

PITEAU ASSOCIATES
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December 13, 1991

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Mr. Gerry Acott
Manager, Environmental Affairs
Curragh Resources Inc.
117 Industrial Road
Whitehorse, Y.T.
Y1A 2T8

Dear Mr. Acott:

Re: Faro Pit Decommissioning

As per your fax messages of December 5, 9 and 10, 1991, and telephone conversations of December 9 and 11, 1991, it is understood that a number of geotechnical questions have been raised regarding the planned decommissioning of the Faro Pit. The following summarizes our understanding of the project decommissioning scheme and, based on the information that we have available, our opinions with regard to the geotechnical questions that have been raised.

SUMMARY OF PROPOSED DECOMMISSIONING PLANS

It is understood that the Faro Pit is to be used for tailings disposal beginning in March 1992. To accelerate the mine's ability to use recycled water in the milling process (i.e. of the Vangorda and Grum ore), the flow of Faro Creek will also be diverted into the pit at that time. Recycling of water will commence when water levels reach 3850 ft UTM (anticipated to be about August 1994), at which time a siphon will be activated to deliver water to the mill.

Tailings will be deposited in the pit through to the end of mine life (potentially including about 40,000,000 tonnes of tailings from the reclaimed Down Valley Tailings Impoundment Area), resulting in a tailings level approximately 14 ft below the final water elevation of 3850 ft. To prevent water at final elevation from outflowing toward the Zone II pit, a "plug dam" will be constructed at the south end of the pit, to a final elevation of 3855 ft. Apparently, the "plug dam" will have a crest width of at least 60m.

It is further understood that the mine is considering using the underground workings for water treatment through sulphate reduction, and that to accomplish this, the underground portal and vent raises would be sealed to prohibit tailings entry and maintain a flooded anaerobic environment. At all times, there would be no head differential between the pit and the underground workings (i.e. the pit and underground workings would be flooded at the same time, and to the same level). During operation of the water treatment scheme,

about 450 igpm would be treated, and all underground workings would be below the water table.

RESPONSE TO GEOTECHNICAL QUESTIONS

1. Stability of Northeast Wall During Flooding of Pit

Instability on the northeast wall of the pit during mining has been an ongoing concern for a number of years. This instability has been summarized and discussed in previous reports, and will not be repeated herein. To briefly evaluate the relative stability of this wall during flooding of the pit, a relatively simplistic sensitivity analysis, comparing a non-flooded (i.e. end of mining) configuration with a flooded configuration, was conducted. In this assessment, the stability of an approximate slope profile through the northeast wall, immediately after completion of mining and prior to flooding, was evaluated by assigning friction only strength parameters to the rock mass. Approximately the bottom one-third of the slope was assumed to be saturated. Both of these conditions are considered to be conservative in the context of comparing the non-flooded versus flooded pit wall stability. Following this, a flooded condition was modelled, wherein it was conservatively assumed that the pit was filled with water (i.e. with a specific gravity of 1.0 and no strength) rather than a denser mixture of tailings and water that would provide a greater degree of buttressing support, and thus a greater degree of stability.

The results of the above sensitivity assessment indicate that the northeast wall would be least stable prior to being flooded. During filling, the stability of the wall would improve slightly, reaching a maximum on completion of flooding to the 3850 ft level. While the exact increase in stability has not been quantified, it is likely to be relatively slight (i.e. possibly about a 5% increase in Factor of Safety over the non-flooded configuration).

2. Stability of the Southwest Wall During and After Pit Flooding

As Piteau Associates has no specific data from the southwest area of the pit with which to evaluate the stability of the southwest wall and/or the potential of a failure which would shear through the underground workings, it is not possible to offer a definitive assessment of these matters. However, it is noted that this pit wall has performed very well over the years, with no major stability problems. While the

flooding of the underground workings will, in time, likely result in the decay and failure of the artificial support (i.e. split sets, steel strapping, etc.) installed in the underground workings, and may subsequently result in failures within the underground openings, it does not necessarily follow that such failures will result in the collapse of the pit wall into the underground mine. However, the possibility of such an event cannot be completely ruled out.

3. Non-Catastrophic Failure of Northeast Wall

As has been observed and documented for some time, movement of the Northeast Wall is greatest in response to climate and mining activity. That is, movement has slowed down in winter months and reached a maximum during and immediately after the spring thaw, when surface water and groundwater flow are highest. It has also been documented that movement can be accelerated by mining at the toe of the slope.

Following completion of mining, it is likely that non-catastrophic movement of the Northeast Wall will continue for some time, most likely in the form of creeping movement, with rockfalls and ravelling being generated by the slide mass. However, as the pit will not be deepened and tailings will act to buttress the slope somewhat, it is anticipated that this movement will decrease with time, with the failed material likely forming a talus slope along the wall. The volume of material that could fail into the pit below the final water level is difficult to estimate. However, as much of the Northeast Wall above the 3850 ft elevation is already at a relatively flat angle, it is anticipated that the volume of material that will fail into the flooded pit will be sufficiently small such that a water cover will be maintained over the tailings.

4. Protection of "Plug Dam"

While the occurrence of a failure large enough to generate a wave in the tailings pond that could overtop the "plug dam" at the south end of the pit is not anticipated, the possibility of such an event cannot be completely ruled out. Thus, it is recommended that the exposed faces of the dam be protected with a layer of rip-rap of appropriate size. Consideration could also be given to raising the dam to provide additional freeboard.

Mr. Gerry Acott
Manager, Environmental Affairs
Curragh Resources Inc.

December 13, 1991

I hope the above is sufficient for your needs at this time. If you have any questions concerning the above, please do not hesitate to contact me.

Yours very truly,

PITEAU ASSOCIATES ENGINEERING LTD.

Alan F. Stewart, P.Eng.

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
ALAN F. STEWART

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