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ACID ROCK DRAINAGE
REMEDIATION MEASURES
AT FOUR NORWEGIAN MINES

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This project was funded by the Resource Management Branch of the
British Columbia Ministry of Energy, Mines and Petroleum Resources
and by the Norwegian State Pollution Control Authority

November 1991

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REMEDIATION MEASURES
AT FOUR NORWEGIAN MINES

STEFFEN, ROBERTSON AND KIRSTEN (B.C.) INC.
Suite 800 - 580 Hornby Street
Vancouver, British Columbia
Canada, V6C 3B6

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ARD REMEDIATION MEASURES AT FOUR NORWEGIAN MINES

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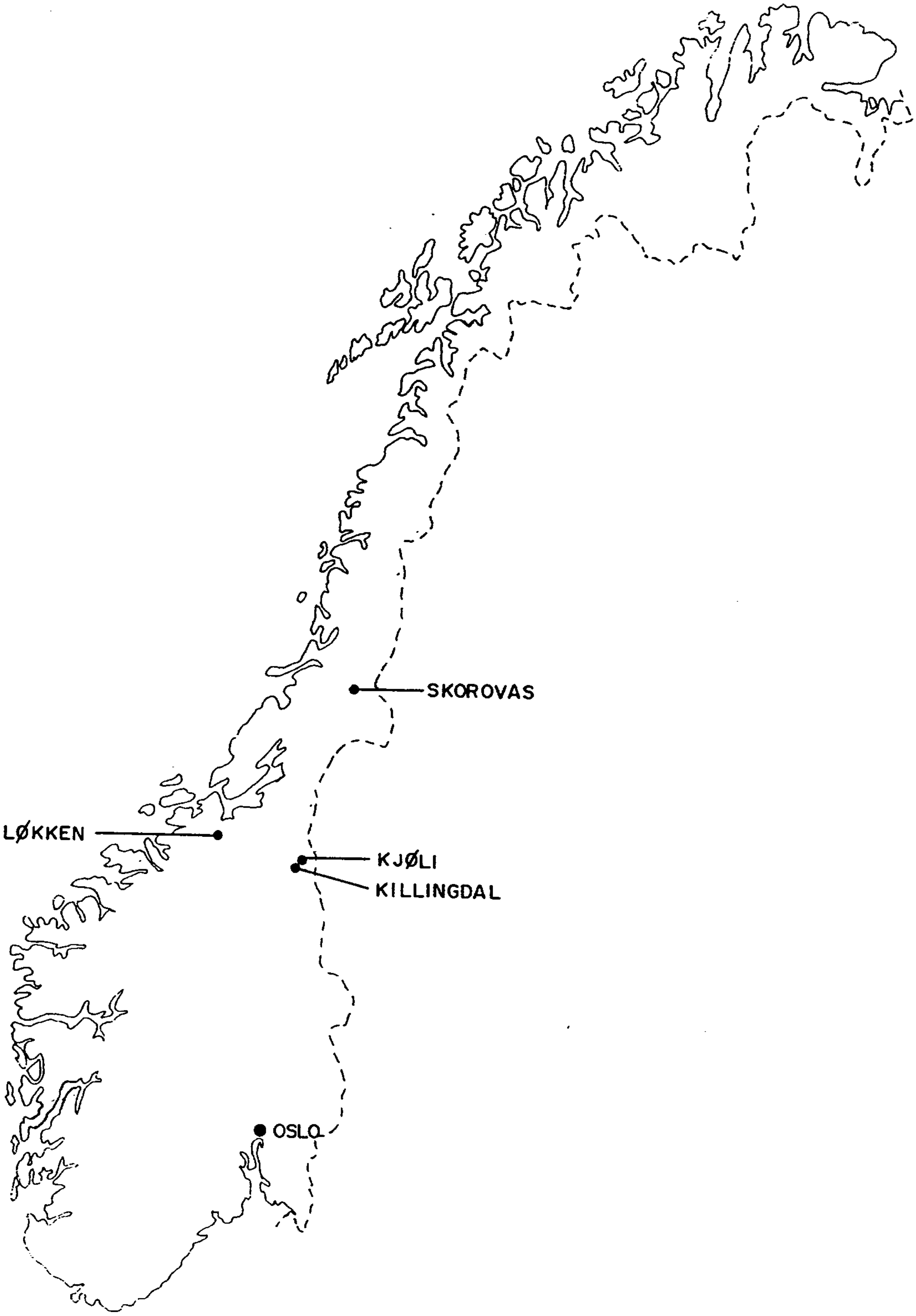

NORWAY
LOCATIONS OF MINE SITES

REPORT ON
ARD REMEDIATION MEASURES AT FOUR NORWEGIAN MINES

EXECUTIVE SUMMARY

The purpose of this review and report was to provide a transfer of recent Norwegian experience and technology on the remediation of acidic rock drainage (ARD) at four sulphide mines in Norway to Canada, and to provide an opinion as to the immediate and long term effectiveness of the remediation measures to the Norwegian State Pollution Control Authority (SFT).

The sites visited and remediation measures reviewed were:

- The consolidation of mine rock at the Kjøli mine and the placement of a geosynthetic membrane cover to control infiltration, and hence migration of ARD from the mine rock pile. The composite geosynthetic cover comprised a sequence of a filter fabric, a 2mm HDPE liner, geonet drainage layer, a filter fabric and a till cover. Rip rap was placed for erosion control and the till cover was revegetated. The lower drainage adit was plugged resulting in the flooding of about 70% of the underground mine workings.

- The excavation of mine rock from the original rock pile site at the Skorovas mine and its placement in a nearby lake under a permanent water cover of 2 m. The mine rock was excavated with a back hoe and transported in dump trucks to the new site. The original site was cleaned by hand to minimise the migration of stored ARD products from residual mine rock. Placement of the old mine rock in the lake required temporary lowering of the water level by pumping to allow 'placement in the dry', construction of a retention berm for stability purposes, and placing a till cover over the mine rock. During mine rock placement, the water quality in the lake water was controlled by adding lime. On completion of mine rock placement, a concrete dam was constructed at the lake outlet to permanently raise the water level in the lake to 2 m above the mine rock level.

- At the Killingdal mine site, the mine rock was consolidated and covered with a till cover. A seepage collection ditch was constructed to divert ARD into the underground workings, with the objective of utilizing the evaporative action of the natural convective air flow through the mine to evaporate the ARD as it trickled down through the workings, and thus deposit salts in the mine.

- At the Løkken underground mine dewatering has been discontinued and the mine allowed to flood. Natural improvement of the water quality in the Wallenberg shaft has been observed. Sulphate reducing bacteria have been isolated from the deep anaerobic zones of the water filled shaft. The presence of the sulphate reducing bacteria, and the observed natural improvement in the water quality observed, presents an *in situ* ARD treatment alternative. It is planned to divert ARD from mine rock in surface piles into the mine and to rely on the natural processes occurring in the mine to improve water quality.

The report describes the implementation techniques applied at the respective sites, and where possible provides a performance evaluation based on the site water quality monitoring data. Summaries of the cost incurred for the implementation of the various measures are provided.

The geosynthetic membrane cover was found to be effective in reducing the overall metal load transported from the Kjøli mine rock pile through the prevention of water infiltration. Large reductions in the measured oxygen concentrations indicate that the cover also served to reduce, but not eliminate, the entry of oxygen into the rock pile. The decreasing temperature profiles of the pile after covering shows that the oxidation process is considerably reduced.

Since relocating the mine rock to a below water location at Skorovas mine it has been possible to discontinue lime treatment without an increase in downstream contaminant discharges. This represents a very large decrease in acidity and contaminant release from the mine rock. Water quality monitoring in the lake appears to indicate that long term pH values of 4.5 will be achieved. The old mine rock was not treated with lime prior to placement in the lake and this results in a pH of about 2.5 in the water in the submerged pile. Stored oxidation products were therefore not neutralized and precipitated prior to submergence. Migration of this low pH water to the lake could result in lake water qualities that are less than what may have been achieved if such neutralization had been completed.

The costs for all mine rock remediation at each of the Kjøli and Skorovas mine sites, including placement of mine rock under a permanent water cover or providing a geosynthetic cover, are comparable at approximately 59 NOK per tonne mine rock (CAD$10.00). Costs for individual remediation measures are summarized in the report.

The effectiveness of placement of a till cover only, as at the Killingdal mine, could not be evaluated from available data. Substantial quantities of low pH seepage was observed to be occurring from below the collection ditch. It is not known if this seepage originates from seepage flow in the bedrock below the ditch or from the ditch itself. Although the concept of collection and treatment in the mine by evaporation is unique, its effectiveness is dependant on the effectiveness of the collection system. The collection ditch was incomplete at the time of the site visit.

REPORT ON
ARD REMEDIATION MEASURES AT NORWEGIAN MINES

1.0 INTRODUCTION

1.1 Objectives and Scope

The purpose of this review and report was to provide a transfer of recent Norwegian experience and technology on the remediation of acidic rock drainage (ARD) at four sulphide mines in Norway to Canada, and to provide an opinion as to the immediate and long term effectiveness of the remediation measures to the Norwegian State Pollution Control Authority (SFT).

The study was co-sponsored by the B.C. AMD Task Force and the Norwegian State Pollution Control Authority.

The initial scope of the study required visiting three mine sites in Norway: the Kjøli and Skorovas Mines, where geosynthetic membrane cover and underwater placement remediation technologies have been applied to reduce ARD; and the Løkken mine, where *in situ* sulphate reduction may provide natural water treatment. At the request of SFT a fourth mine site, the Killingdal mine, where a unique approach of ARD collection and evaporation in the underground workings was implemented, was included in the review.

At the Kjøli mine a 2mm HDPE membrane composite cover was placed over acid generating mine rock during the summer of 1989. The rock pile has been instrumented for oxygen and temperature monitoring. The scope of the study provided for a review of the construction procedures, costs and effectiveness and to evaluate the applicability of this technology to Canadian mines.

A mine rock pile at the Skorovas Mine site, containing aged mine rock with stored acidic oxidation products, was moved during the summer of 1990 and placed under water in a nearby lake. Contaminant release during the replacement of the mine rock was successfully controlled. The study provided for a review of the construction and contaminant control procedures, difficulties, and costs, and effectiveness of this remediation measure.

At the Løkken mine site, the scope of the study called for the evaluation of water quality results and local conditions observed in the mine, to verify the occurrence of natural *in situ* sulphate reduction. A verification of sulphate reduction occurring at this site, and an understanding of the processes, aids the evaluation of the potential of this technology for the long term treatment of ARD from the rock piles if introduced into the mine. It also enables the potential for the application of this form of remediation at other mines to be assessed.

In the case of the Killingdal mine, mine rock was consolidated in 1990 and covered with a till cover. A collection ditch was constructed to drain the seepage from the rock pile into the underground mine workings. The two major shafts of this mine are connected, with a large differential in altitude of the portals. The natural convection through the mine provides a means for evaporation, resulting in a net water loss from the mine and the accumulation of the oxidation products in the mine. No data on downstream water quality was available for this site and the review for this mine is limited to the observations made on site.

1.2 Norwegian ARD History

Pyritic ore mining and processing has been practiced in Norway for more than 300 years. In a review document by NIVA (1987) 39 pyritic ore bodies are listed, of which the majority have been mined.

Due to this long history of mining, several areas exist in Norway where acidic drainage has impacted severely on the environment. Several underground mines and associated mine rock piles exist that have been generating acidic runoff for extended periods. Since these mines were developed prior to there being an appreciation of the cause and effect of ARD, little or no control measures were implemented and ARD discharges are large. At some mine locations the downstream environmental impacts on the aquatic environment is severe for many kilometres.

Norwegian mining companies and regulatory authorities have been actively seeking and implementing appropriate remediation measures. Some early attempts, such as at the Kjøli mine (in 1981) where mine rock piles were consolidated, contoured to a gentle, uniform slope and 100 tonnes of hydrated lime was sprayed onto the 2 ha surface of the dump, were not successful. During 1987 some of these mine sites were visited by SRK at the request of SFT to evaluate the ARD, and potential remediation options. The observations and conclusions of these brief site visits and evaluations are reported in:

> Report 64701/1 - ALTERNATIVE MEASURES FOR ACID MINE DRAINAGE ABATEMENT AT NORWEGIAN MINES. November, 1987.

In this report the potential ARD control options of a geosynthetic membrane for Kjøli mine and under water placement at Skorovas mine were reviewed.

Additional evaluations of appropriate alternative ARD control options by both Norwegian mine owners (including Elkem A/S at Skorovas mine) and regulatory authorities (Bergvesenet and SFT) led to the selection of a variety of different options to suit site specific conditions.

To reduce the impacts of ARD, remediative actions have been taken at the sites mentioned and the experience gained at these sites is of considerable benefit for other mines in similar geological and climatic circumstances, such as Canada.

1.3 Norwegian Objectives for Remediation

At many of the old Norwegian mine sites the impact of ARD on the downstream aquatic environment is fully developed. Any remediation of ARD therefore improves the situation. There are almost no new mines under consideration and therefore little need for the consideration of ARD prevention. The emphasis in Norway is therefore on remediation and not on prevention of ARD.

This emphasis difference is important when comparing potential ARD control options. Options available for prevention, i.e. before the mine development starts, may not be applicable, practical or economic for developed mines. Given a particular mine development and the consequential ARD that has occurred, it is difficult and considerably more costly to achieve the same discharge standards as could be considered for new mine development.

The original developers of many of the mines are no longer in existence, and the burden of remediation cost must be borne by the State. Where owners exist, their resources are often limited and, severe remediation standards may be beyond their fiscal capability.

It is therefore necessary to find effective remediation technologies that will reduce the impact on the environment to within acceptable levels and be cost effective. Site specific water quality objectives are generally established in Norway to achieve a reduction in current contaminant discharge, taking into account the previous level of discharge and the down stream impacts.

Typically these water quality objectives are considerably less stringent than would apply in Canada for new mine development. This lower level of ARD control is an important consideration in assessing the potential applicability of experience in Norway for potential application to new mines in Canada. Such experience is, however, of direct relevance to the abandonment of old mine sites in Canada.

1.4 Field Visit and Review Process

A visit to Norway and the mine sites was made during the period 2nd to 9th September, 1991. To achieve the objectives of this study, it was necessary to meet with the various organizations involved with the investigation, design, evaluation, implementation and monitoring of the remediation measures at the various mine sites. The following meetings were held:

- SFT (Mr G. Jordfald, Mrs. Kari Kjønigson, Ms Siri Sorteberg) to discuss all four sites and the results of the remediation from a regulatory viewpoint.

- Miljøplan A/S (Mr. P. Liseth and Mr. P. Å. Beck) to discuss Kjøli and Skorovas mines for which they were responsible for the investigation, monitoring and elements of design.

- NIVA (Dr. Merete Johannessen, Dr. Morten Laake, Mr Eigil Iversen and Mr. Bjørn Christensen) to discuss monitoring of downstream water quality at all four sites and particularly the investigation and monitoring of water quality and sulphate reduction processes in the mine.

- Bergvesenet (Mr. H. Ese) to discuss construction measures, difficulties, costs, controls and effectiveness at the Kjøli mine and the Killingdal mine, for which they were responsible.

- Løkken Verk A/S (Mr. K. Brøndbo) to discuss site conditions, ARD remediation and water quality observations at the mine site.

During the meetings the following issues as well as site specific issues were addressed:

- Implementation techniques;
- Construction difficulties;
- Effectiveness of remediation measures and improvements in discharge quality; and,
- Costs (construction, materials, contracting etc.)

Available relevant documentation was obtained from each of the entities to aid the review process.

2.0 LØKKEN MINE

2.1 History and Setting

The Løkken area, located approximately 50 km south east of Trondheim, has been mined since 1654 when the first copper bearing pyrite was discovered. The mine was worked by Løkken Verk A/S & Co., which is owned jointly and equally by Orkla Industrier A/S and Outokumpu Oy of Finland.

The Løkken mine is situated on the steep slope of the western embankment of the Raubekken stream, and is located near the village of Løkken Verk. It is the remnant of Norways' largest copper-zinc sulphide deposit. The Løkken area lies within the Norwegian Caledonides and consists of an inverted and folded succession of metavolcanic rocks with younger sediments. The metavolcanics comprise of two subgroups which are mainly alkaline in composition. The ore deposits occur in the lowest section of the upper metavolcanic subgroup.

A cross section of the mine workings is illustrated in Figure 1. The underground mine workings are accessed by four shafts. Three of the shafts were sunk in this century. The Fearnley shaft was sunk in 1911, the Wallenberg shaft in 1916 and the Astrup shaft in 1972. Mining methods included sub-level stoping with long hole drilling, and room and pillar methods. Løkken ore is a very hard, fine grained pyritic material, and generally roof conditions are good. In the older section of the mine wood was used as a roof support medium. Wood consumption in the underground workings during the 1950's and the 1960's ranged between 100 and 300 m^3 per year. The wood was preserved using conventional arsenic compound impregnation techniques.

After depletion of ore reserves and termination of mining activities in the older section of the mine, the Astrup shaft workings were isolated from the rest of the mine by the installation of 3 concrete plugs at the locations shown in Figure 1. Previously mine water drained into the Astrup workings at a rate of 500 000 m^3/yr. While mining operations continued, mine water from the Astrup workings was pumped to the isolated section of the mine through the concrete plug at the 340 level at a rate of approximately 25 000 m^3/yr. Mining activities at the Astrup workings were terminated in 1987.

The Løkken ore comprises 70-75% pyrite with minor quantities of pyrrhotite, 6% chalcopyrite, 3% sphalerite plus some magnetite. Gangue minerals include quartz, calcite and chlorite. The fine grained ore consists of pyrite grains between 20-30 µm and about 5% of the chalcopyrite occurs as small particles of 1-2µm in the pyrite crystals. On average the ore contains 41% sulphur, 37% iron, 2% copper, 2% zinc, 0.04% arsenic and 0.01% cadmium.

The locations of mine rock and tailings are illustrated in Figure 2 (mine wastes are indicated by shaded areas). Mine rock has been placed at various locations on the mine site, with the majority of the mine rock piles dating to pre 1920. The total mine rock placed is estimated at 2 000 000 tonnes. The copper

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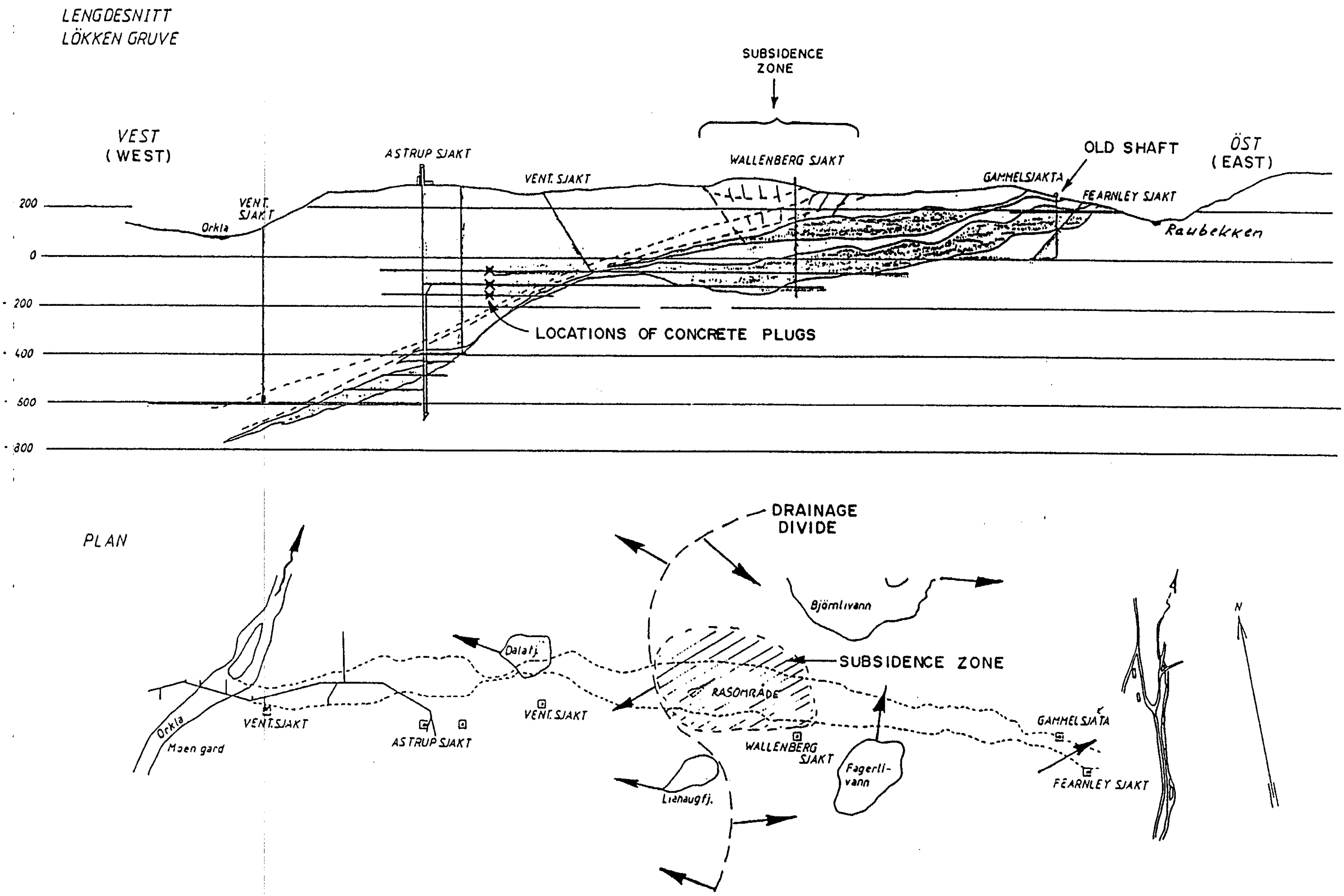


FIGURE 1 LØKKEN MINE—CROSS-SECTION AND PLAN OF UNDERGROUND MINE
ARROWS SHOW GENERAL DRAINAGE DIRECTION

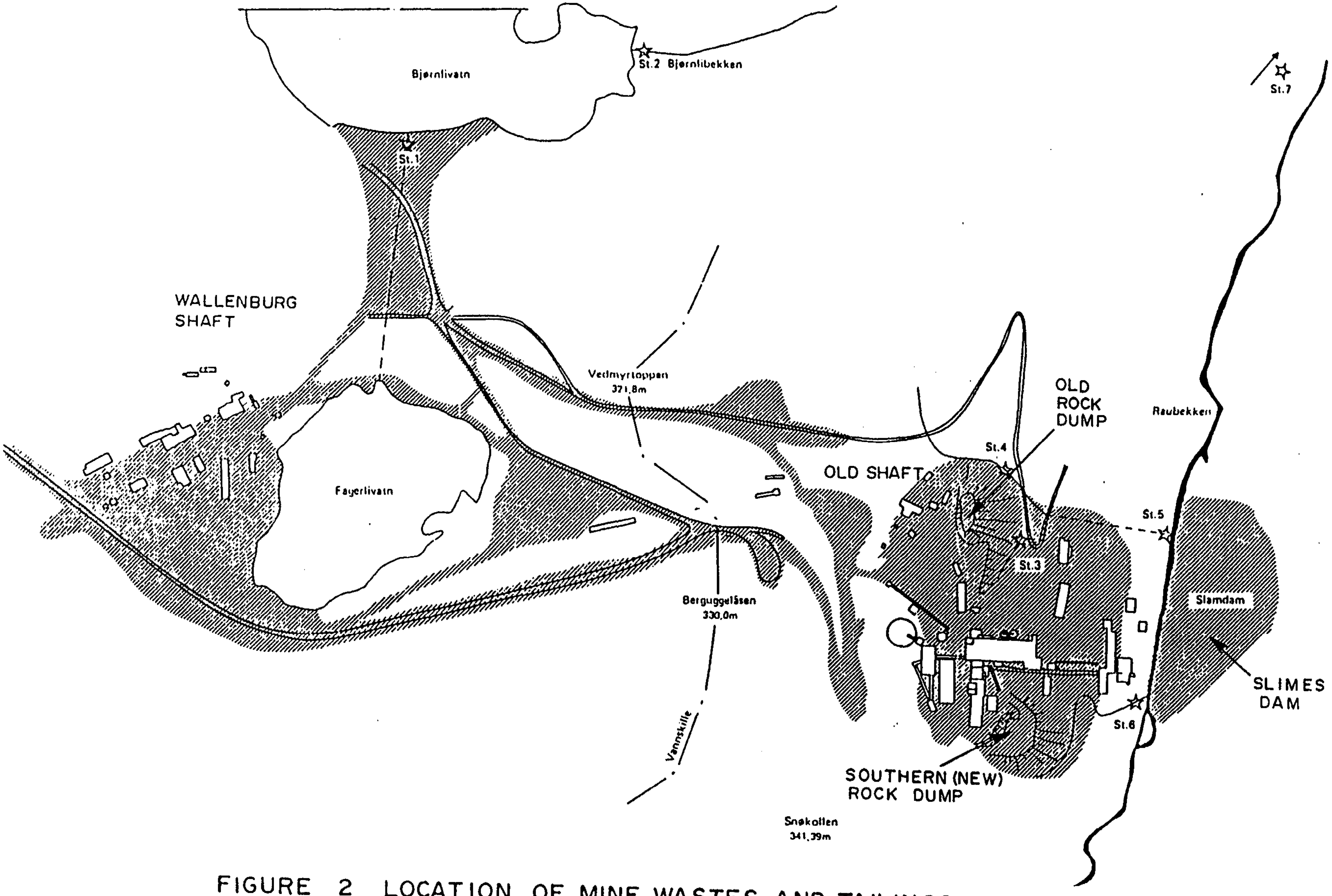


FIGURE 2 LOCATION OF MINE WASTES AND TAILINGS AT LØKKEN VERK (NEW TAILINGS DAM NOT SHOWN) SHADED AREAS SHOW LOCATIONS OF MINE WASTES

2.2 ARD

According to NIVAs' observations, the southern piles are oxidising more rapidly than the old pile. Between 1975 and 1978 the old pile was covered with a 0.5 m till cover. Since then, various collection ditches have been constructed and covers placed on the smaller piles. Other remediative actions taken at the mine site included:

- Providing a permanent water cover for the tailings dam;
- Placement of rip rap on the slimes dam to the east of the Raubekken (for prevention of dust and erosion);
- Covering various acid generating mine rock piles with 0.5 m uncompacted sandy till covers;
- Revegetating the till covered mine rock piles; and,
- Diversion ditching around subsidence zones in 1988 and 1989

Although this has not been a complete solution, some improvement of the water quality has been observed. The water quality in the Raubekken, as measured at Station No. 7 - Figure 2, monitored from 1975 to 1986, is summarised in Table 1.

TABLE 1
Raubekken Water Quality Measured at Station no. 7

YEAR	pH	SO_4^{2} mg/L	Fe mg/L	Cu mg/L	Zn mg/L	Cd µg/L	Pb µg/L
1975	3.6	251	17.0	3.07	8.46	-	-
1976	3.3	416	24.3	4.13	11.20	-	-
1977	3.4	335	20.3	2.95	6.67	-	-
1978	3.2	336	27.6	3.70	7.69	30.0	-
1979	3.5	327	21.4	2.96	6.67	21.7	-
1980	3.3	289	26.5	3.27	6.20	23.4	-
1981	3.4	383	20.9	3.02	5.42	19.1	-
1982	3.4	476	28.5	3.51	6.07	19.1	1.03
1983	3.9	255	19.4	2.22	3.58	9.54	0.92
1984	3.4	318	22.5	2.59	4.45	15.7	1.38
1985	3.4	427	24.5	2.13	3.63	12.4	0.48
1986	3.2	421	25.0	2.49	3.94	11.9	1.45

The average annual flow rate for the Raubekken is estimated at approximately 1 000 L/s. NIVA estimated that during 1980, approximately 73 tonnes of copper and 133 tonnes of zinc were transported from the

Løkken mine site, via the Raubekken, to the Orkla river. Currently, the copper load is estimated (by NIVA) to be approximately 48 tonnes per year.

The Raubekken is diverted to the Svorkmo power station, through a tunnel, where it is mixed with the water from the Svorka river. Sufficient dilution occurs following entry of the water into the Orkla such that no apparent impact on the downstream aquatic life is observed. During winter, as a result of the low flow rates, copper concentrations of up to 250 µg/L and zinc concentrations of up to 700 µg/L are observed in the Orkla river. Fish deaths occurred in the Orkla in 1981 as a result of this effect. It is therefore desirable to reduce the overall copper load originating from the Løkken mine site.

An analysis of the copper load contribution to the Raubekken indicates approximately 42 of the 48 tonnes per year originate from the rock piles. It is therefore necessary to concentrate initial rehabilitation measures on the control of ARD from these piles.

2.3 Proposed Remediation

The target of the decommissioning plan is for a 60% reduction in the annual copper load transported to the Orkla river.

The results of a water sampling program conducted on the water depth profile in the Wallenberg shaft indicated a natural improvement in the water quality with time. 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 Fearnley shaft. Simultaneously, water will be pumped and discharged from the Wallenberg shaft. It is envisaged that the natural improvement in the water quality which occurs in the mine will be sustained thus reducing copper concentrations in the ARD as it flows through the mine.

In the decommissioning plan it is estimated that approximately 60 % of the seepage will be intercepted before it enters the Raubekken, and will be transferred into the mine. Should the natural remediation in the mine improve water quality to the level currently experienced in the Wallenberg shaft, it is anticipated that the overall copper transport to the Orkla will be reduced by between 50 and 52 %.

The processes responsible for water quality improvement in the mine is poorly understood. 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.

If the short and long term effectiveness of the natural treatment processes occurring in the mine is to be demonstrated then it is essential that the nature of the processes and the factors that sustain them must be understood.

To develop a basis for the evaluation of the potential for long term sulphate reduction in the mine, the requirements for sustained bacterial activity are first reviewed.

2.4 Sulphate Reduction and Mine Water Quality

2.4.1 Sulphate Reduction

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). These bacteria are all gram negative, heterotrophic organisms and have a respiratory metabolism in which sulphates, sulphites and/or other reducible sulphur compounds serve as the final electron acceptors, with the resulting production of hydrogen sulphide. The organic substrates for these bacteria are generally short chain acids such as lactic and pyruvic acid. In nature these substrates are provided through fermentative activities of anaerobic bacteria on more complex organic substrates. Thus when oxygen is depleted where organic matter is available in the presence of sulphate, hydrogen sulphide production can be expected. During anaerobic respiration sulphate reducing bacteria utilise lactate to produce acetate according to the following reaction:

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

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:

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

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 activity of these sulphate reducing bacteria is important, not only due to the production of hydrogen sulphide, but also due to the subsequent metal interaction with the hydrogen sulphide. Hydrogen sulphide readily reacts with solubilised copper at low pH values to form the insoluble sulphide mineral. Similarly, the precipitation of zinc is possible, provided suitable conditions exist, for example a solution pH of greater than 3.5-4.0 is required (the theoretical solubility curves for various metal sulphides and hydroxides are provided in Appendix A - Figure A-1).

Since the sulphate reducing bacteria occur naturally, sulphate reduction can be utilised as an *in situ* processing option for the treatment of acidic rock drainage provided the correct conditions can be maintained for sustained bacterial activity.

The potential for achieving a sustained sulphate reduction process in a mine is best evaluated from our understanding of the requirements determined in the development of controlled process (commercial) reactors. Recent sulphate reduction technology developments in South Africa and in Canada have been directed at the commercialization of an integrated process, as a commercially viable processing of metal and high sulphate contaminated waste waters. The process involves the recovery of metals in solution as saleable metal sulphide concentrates, and converting excess hydrogen sulphide to elemental sulphur or other saleable sulphurous compounds.

Thus far fixed bed reactors have been evaluated. Stirred type reactors with a suspended solid medium are being investigated with the objective of achieving higher reduction rates through improved operating conditions and reactor utilisation. However, conventional fixed bed type reactors do hold the advantage in that no moving parts (and associated seals) are required and thus require a lower level of maintenance. An *in situ* application such as that anticipated for the Løkken mine may be analogous to the fixed bed reactor type.

For the commercial process, the generated hydrogen sulphide (H_2S) is stripped from solution in the anaerobic biological reactor. The hydrogen sulphide is then utilised in a pretreatment step where metals are selectively removed as sulphides. Due to the differences in reaction kinetics and conditions under which the metals in solution react with the hydrogen sulphide, it is possible to selectively 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.

For the *in situ* process it is anticipated with the correct management of the water flow path through the mine, and with the long retention times inherent of the large mine volumes, that it would be possible to 'create' reaction zones where the metals will react with the hydrogen sulphide generated to form sulphide minerals, maintaining low metal concentrations in the bacterial zone. This will be analogous to the pre-treatment stage described for the commercial approach.

2.4.2 Analysis of Water Quality Data and Observations

Factors that have been identified that may contribute to the chemical gradients that exist in the water column within the Wallenberg shaft are as follows:

Surface Water Quality

The density difference between the highly contaminated water at depth (S.G. estimated at approximately 1.038 - extrapolated from water quality) and the cleaner surface water (S.G. estimated at approximately 1.001) will arguably maintain the existing water column, provided there are no external influences that will cause a turnover in the water column.

The surface water quality is illustrated in Figures 3 and 4 (day 500 on the time scale corresponds to Jan. 1988). Two significant increases in the sulphate and metal contents of the surface water of the column were observed at the end of 1987 and early 1990. The water level at the location of these marked increases coincided with the intersections of the main mine tunnels at the 300 m and 200 m levels. These increases in the concentrations are likely as a result of the 'washing' action of the water entering the connecting portals from the main body of the mine, resulting in the solubilization of stored oxidation products in the tunnels, and transport into the Wallenberg Shaft.

Deep Water Quality

During flooding of the mine the acidic products on the exposed mine walls, broken rock and backfill will have been taken up in the water column. The water quality measurements made as of 1987 are therefore the result of this initial uptake and the subsequent water quality changes. Unfortunately these early changes were not monitored and deductions as to the natural water treatment processes that may be occurring must be based only on the recent observations.

On the basis previously discussed, the water at depth (490 level) is expected to remain stagnant, provided there are no external forces to cause a turnover in the water column. It is recognised that diffusion will provide a mechanism for the reduction in the chemical gradient of the water column, but in the case of the 490 level, this mechanism is expected to be insignificant. The water quality data for the 490 level is

LØKKEN WATER MONITORING
SURFACE

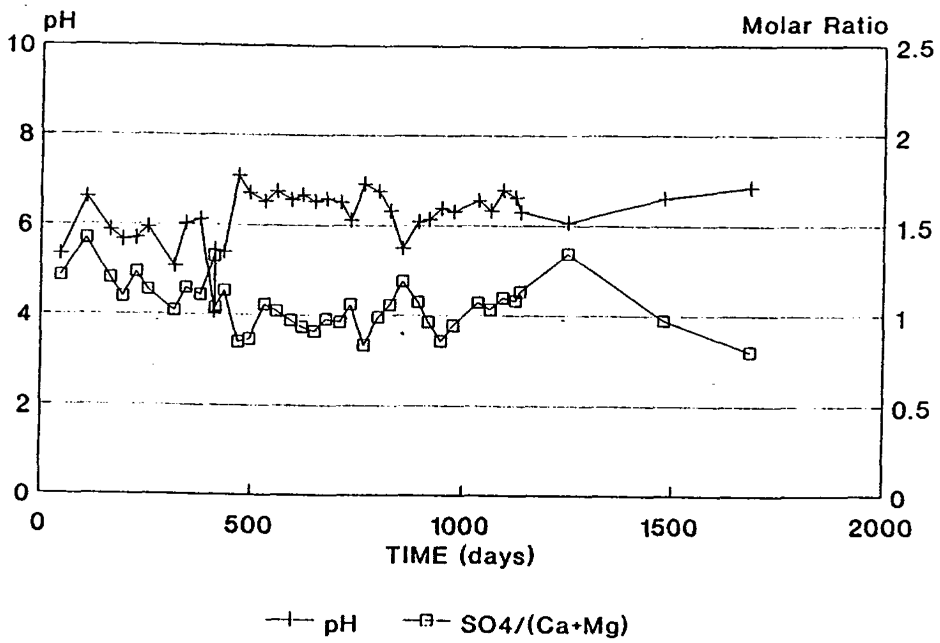


FIGURE 3

LØKKEN WATER MONITORING
SURFACE

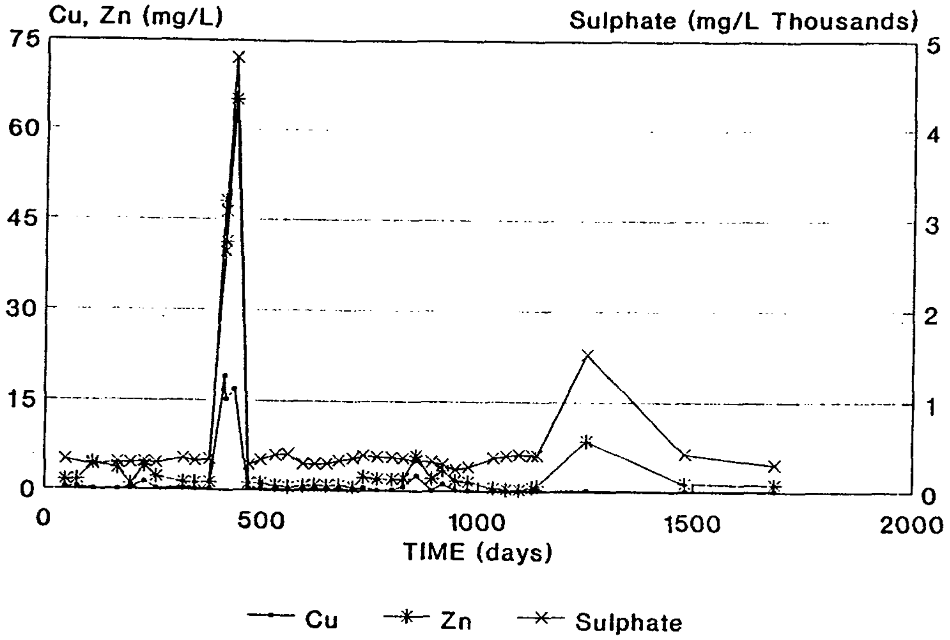


FIGURE 4

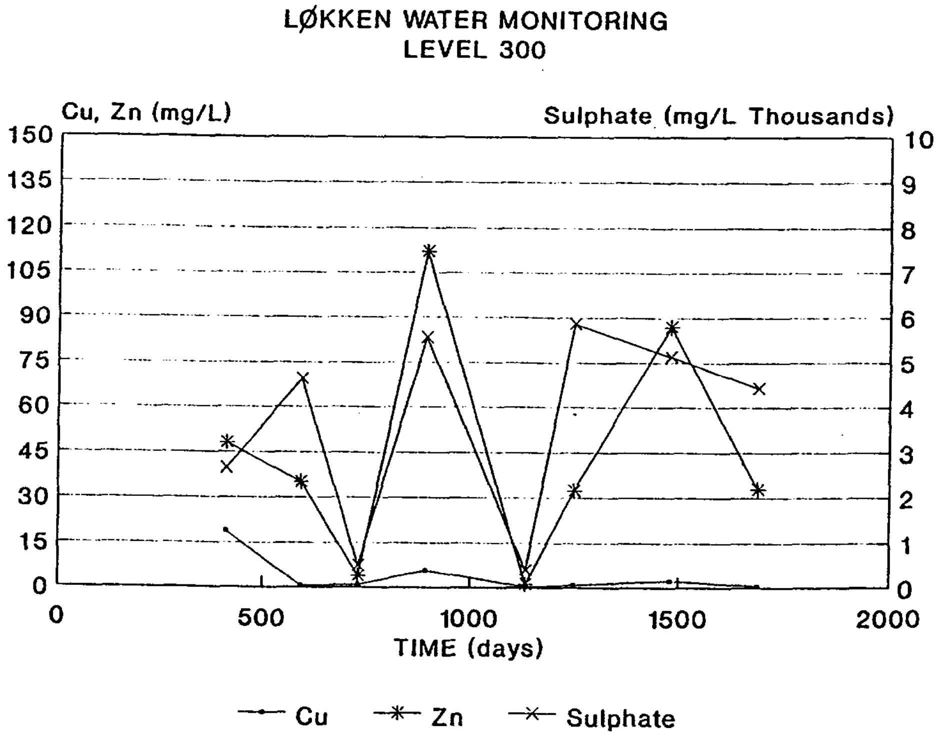


FIGURE 5

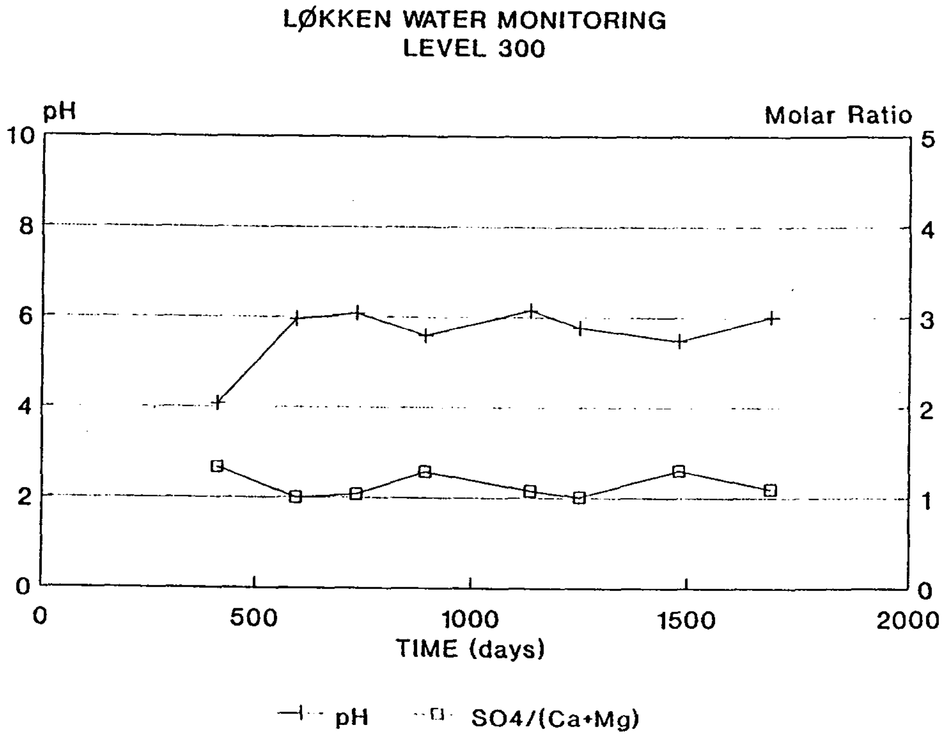


FIGURE 6

LØKKEN WATER MONITORING
LEVEL 340

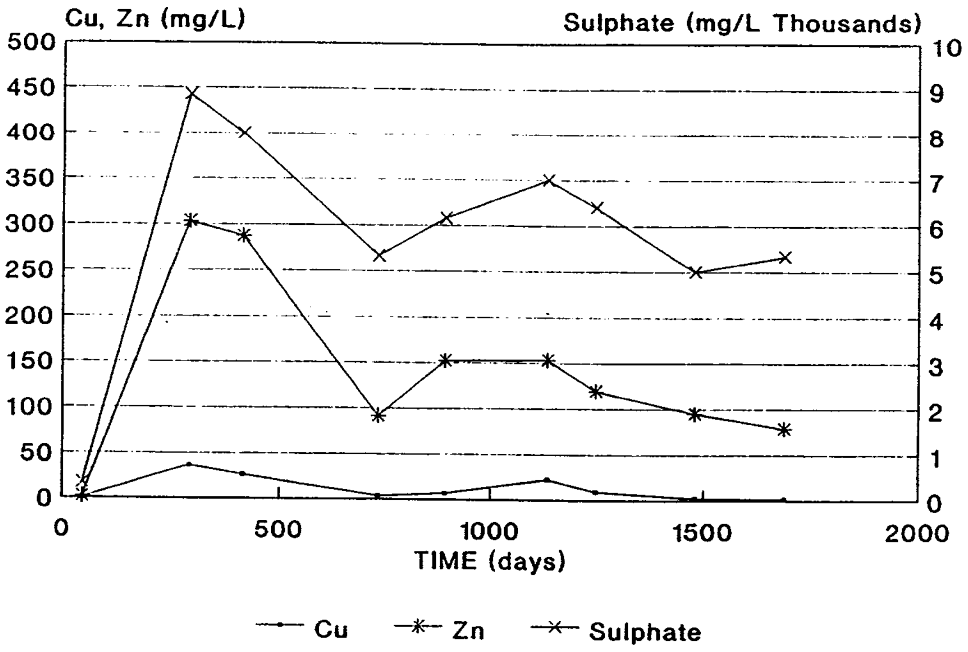


FIGURE 7

LØKKEN WATER MONITORING
LEVEL 340

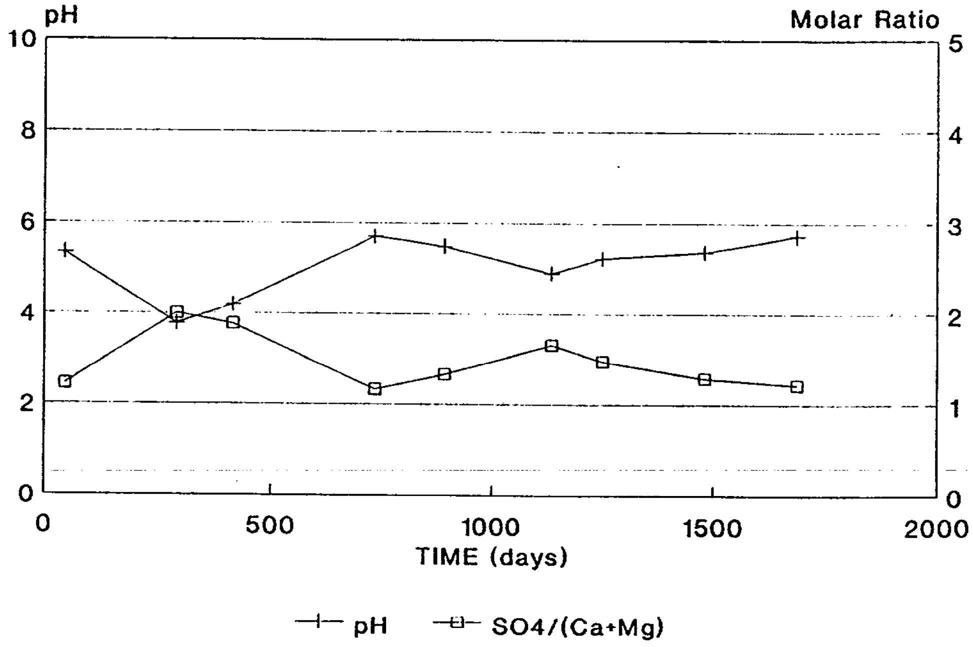


FIGURE 8

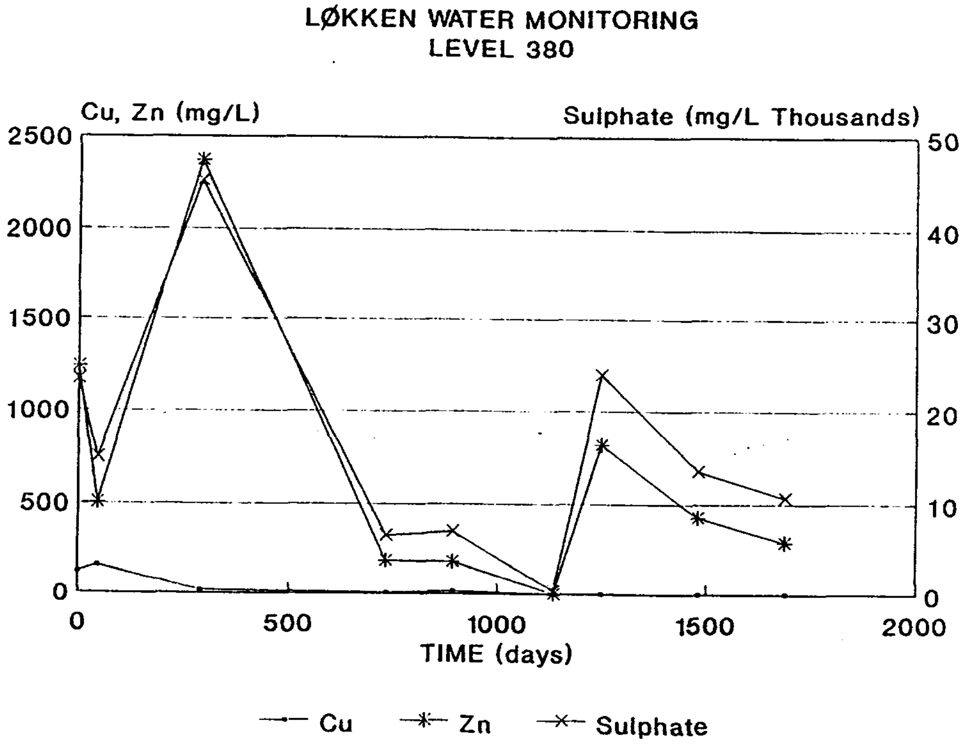


FIGURE 9

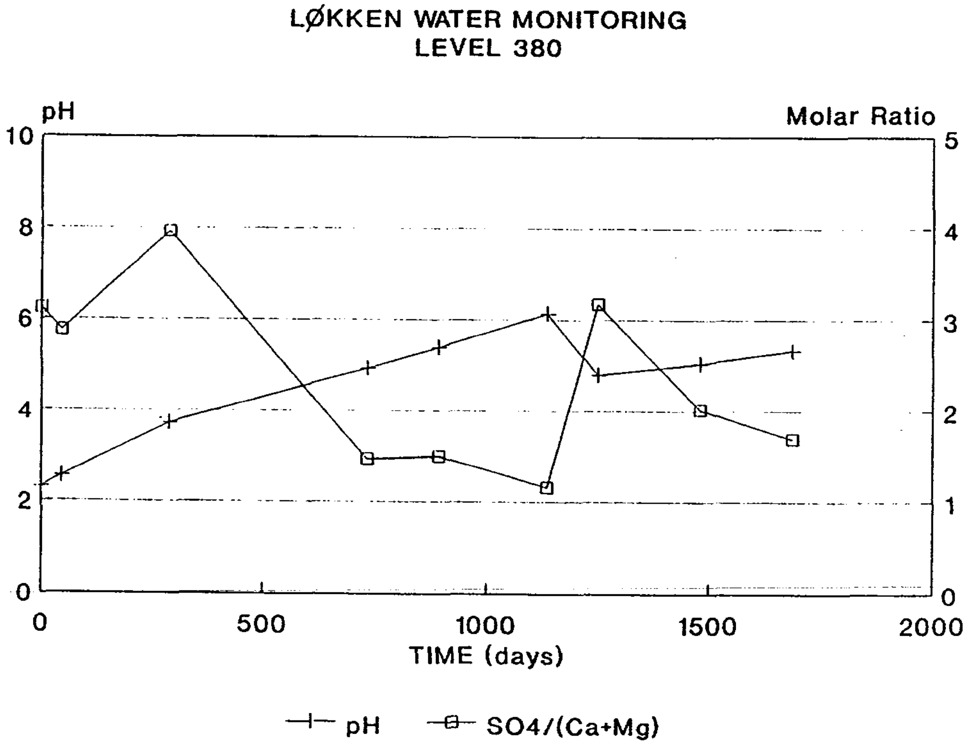
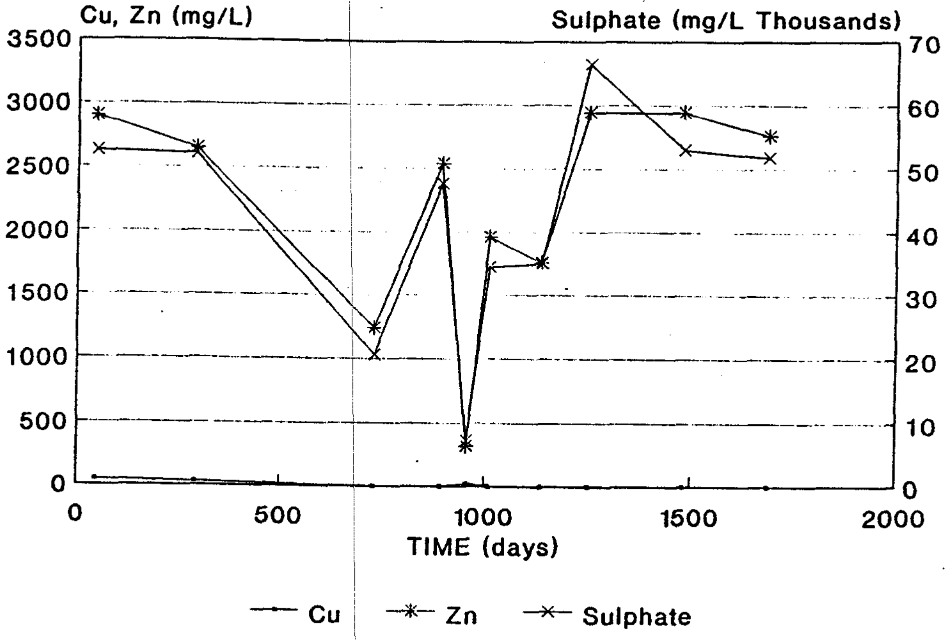
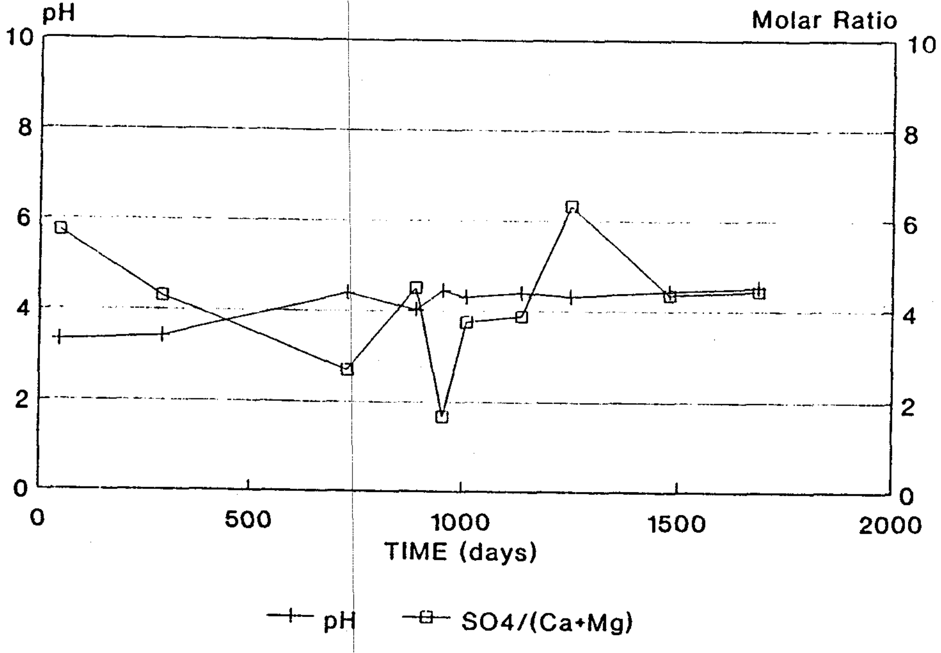


FIGURE 10



LØKKEN WATER MONITORING
LEVEL 430

FIGURE 11



LØKKEN WATER MONITORING
LEVEL 430

FIGURE 12

LØKKEN WATER MONITORING
LEVEL 490

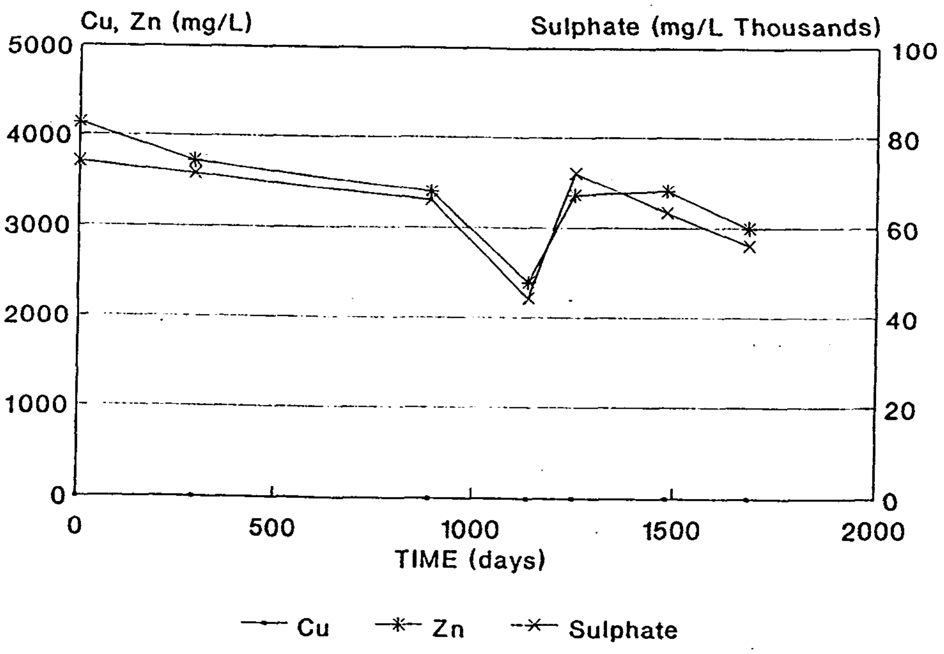


FIGURE 13

LØKKEN WATER MONITORING
LEVEL 490

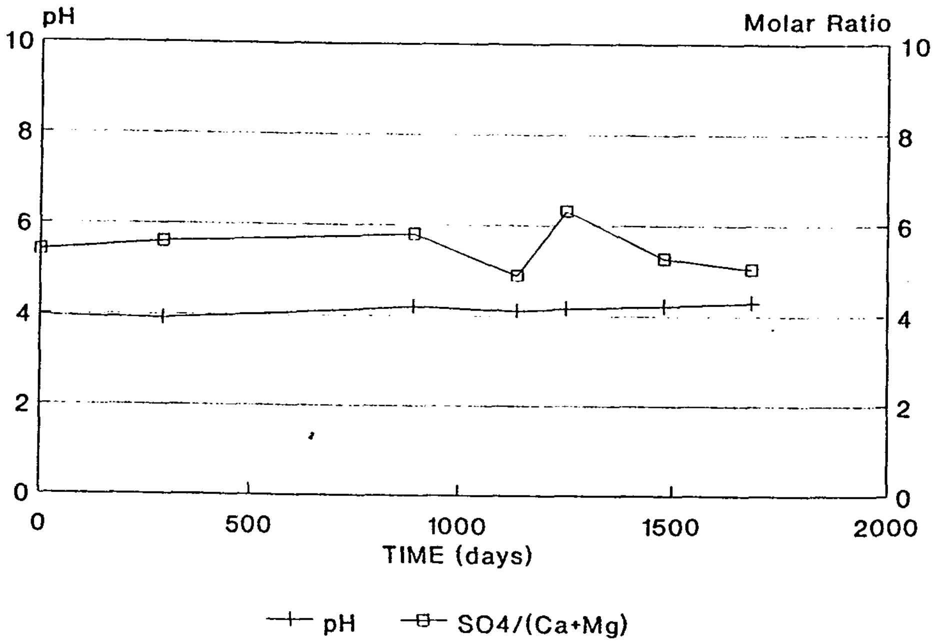


FIGURE 14

illustrated in Figures 13 and 14. Similarly, the water quality data for the 430, 380, 340 and 300 levels are illustrated in Figures 5 to 12. With the exception of one data point the 490 level displays relatively stable curves, confirming the relative isolation of this section of the mine. The large deviation of the single exception, suggests a sampling or possibly an analytical error. Therefore, it may be fair to assume that there are no external impacts on the water quality for the level 490 location.

Neither the copper nor the zinc concentrations will be affected by pH at the prevailing conditions (i.e through the precipitation of hydroxides). However, for the initial period of monitoring a slow reduction in the sulphate concentration is observed. For the more recent data the rate of reduction is observed to increase. A similar trend is observed for the zinc concentration.

If diffusion of the ionic species was the predominant mechanism for the reduction in the sulphate and zinc concentrations, the concentration profiles would show a rapid initial rate of decrease (at the time of the highest concentration differential and thus the greatest driving force). In time this rate will steadily decrease and approach zero as equilibrium in the water column is reached. Therefore a diffusion mechanism will result in rates of change opposite to that observed.

Similarly, if neutralisation (reaction with alkaline minerals) is the primary mechanism, the greatest change in the pH and concentration profiles will be observed at the time of the highest concentrations. The same be said for the uptake of metals by organic matter. A high initial rate when concentrations are high, would be observed, decreasing as saturation is approached.

However, a lag phase is observed prior to the change in the concentration for the water quality profiles. When this is considered in conjunction with the isolation of sulphate reducing bacteria at this level, it is evident that sulphate reduction through the activity of sulphate reducing bacteria may be the principal mechanism present. The initial lag phase observed may be due to the time required to:

i) establish anaerobic conditions, and,
ii) establish an active sulphate reducing bacterial population.

The decrease in the zinc concentration is probably as a result of zinc sulphide formation, following the reaction with hydrogen sulphide formed during sulphate reduction.

In discussions with Mr. Brøndbo of Løkken Verk A/S, it was established that copper concentrations of up to 50 mg/L have been observed in the water column prior to the initiation of the NIVA monitoring program. The present copper concentration at this level (490) was observed to be relatively low (< 1 mg/L). Amongst others, two possible mechanisms exist for the reduction in the copper concentration:

i) electroplating (cementation) on metallic iron (total *e.m.f.* = +0.78 V), and,
ii) sulphide precipitation.

However, it should be noted that the cadmium concentration profiles decreased in phase with the copper concentration, and, since the electroplating of cadmium on metallic iron is less likely (total *e.m.f.* = +0.04V) than for copper, sulphide precipitation is more likely to have caused the concentration decrease (also note the solubility curves for cadmium and copper sulphides and hydroxides in Appendix A, Figure A-1).

Carbon Dioxide

There are two possible reasons for the elevated carbon dioxide concentrations at depth:

- Calcite and dolomite reacting with the acidic water will generate carbon dioxide. However, since the magnesium concentration has remained approximately constant at 2 500 to 3 000 mg/L, and is significantly lower than the saturation concentration of approximately 60 000 mg/L, there is no indication of dolomite participating in the reaction. Since the calcium concentration has remained at or close to the saturation level, it is not possible to establish this as a continuing neutralization source (i.e. calcite); and,

- One of the possible fermentation products of simple sugars is carbon dioxide. The sugars for the fermentation process may be released through the hydrolysis of wood (see Section 2.4.4).

At present, with the limited data available, it is not possible to exactly establish the origin of the elevated carbon dioxide concentration for the water samples taken at depth.

2.4.3 Assessment of Future Performance

Sulphate Reduction

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 possibly reliant on a diffusion mechanism.

The sulphate reducing bacteria may rely on any one, or a combination of various respiration processes, as mentioned in Section 2.4.1. Based on the observations at the site, the following is noted:

- The surface runoff entering the mine has a high natural organic (humic acid) 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 solution of iron could 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. Basically all wood species can be hydrolysed. The rate of hydrolysis is primarily dictated by the temperature and the strength of the acidic solution. 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 60 to 70 mg SO_4^{2-}/L/h 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 a sulphate reduction rate of approximately 1.6 mg/L/h. 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.

In establishing the future performance of the sulphate reducing bacteria, it is essential to establish the nutrient and substrate sources currently utilised. The performance of the sulphate reducing bacteria may be enhanced by supplementing nutrients and substrates as required.

Expected Hydraulic Load

The pumping station for transferring the collected drainage to the underground workings has already been designed based on the local hydrological evaluation. The pumping station will consist of a total of four pumps, two on standby, with a configuration of two in series. The pumping capacity of a single two in series configuration is 15.5 L/s (i.e. 1340 m^3/day or 488 808 m^3/year). At this rate of water entry to the mine, the retention time is approximately 20 years, assuming there is no short circuiting and all of the mine voidage is available to the process.

However, since the Wallenberg shaft (point of exit) is not located to the extreme opposite side of the mine compared with the Fearnley shaft (point of entry), the actual retention time in the underground mine is considerably less, and has to be evaluated more accurately.

2.5 Conclusions and Recommendations for Evaluation and Optimisation of Sulphate Reduction

The concept of utilising the processes occurring within the underground workings of the mine for the treatment of the acidic drainage has great potential, provided a thorough understanding of the overall process is developed. As described previously, the processes occurring within the mine represent complex interactions. An analysis of the available water quality data indicates that sulphate reduction may be one of the primary mechanisms affecting the water quality in the Wallenberg shaft.

Whether conditions which currently encourage sulphate reduction will be maintained when acidic rock pile water is introduced to the mine is essentially unknown. In particular it is not known whether:

i) flows through the mine will short circuit and the volume of the mine contributing to the reduction processes along the flow path will be reduced;

ii) acidic conditions developed as a result of ARD injection will inhibit sulphate reduction to the extent that the processes are inadequately slow;

iii) organic substrate resources will be sufficient to sustain sulphate reduction at the required levels; and,

iv) changes in the mine flow patterns as a consequence of mine collapse and/or sludge deposition may alter the processing potential.

To fully utilise the potential of the sulphate reduction process, it is necessary to fully understand the limitations of the process, as they currently exist in the mine, and as they may develop over time, and to define possible methods of enhancing the kinetics of the process. Three distinct areas of study are identified for the development and optimization of *in situ* sulphate reduction technology, and can be summarised as follows.

2.5.1 Monitoring

An enhanced monitoring program will aid the evaluation and definition of all the processes contributing to the change in the water quality observed for the Wallenberg shaft. Such a monitoring program should include:

- Additional monitoring of the water quality in the Wallenberg shaft to confirm sulphate reduction and to collaborate the mechanism of metal removal. This monitoring program should incorporate the following:
    - Dissolved oxygen profile with depth.
    - Total organic carbon content and species profile, including COD.
    - Bacterial speciation other than sulphate reducing.
    - Gas phase monitoring for hydrogen, methane and carbon dioxide.
- Site water monitoring for water quality and organic carbon content.
- Water quality monitoring at other accessible locations in the mine, including organic content.
- An accurate water and mass balance evaluation is required to obtain an evaluation of the current loads to the mine, and to obtain the current performance of the system.
- Tracer tests should be designed and performed on the underground mine workings to establish the flow path through the mine, effective retention times and the possibility of engineering, or utilising the flow path optimally to maximise the sulphate reduction performance.
- Sediments and precipitates should be collected and analyzed for metal sulphides.

2.5.2 Laboratory Simulation

A laboratory simulation program should be initiated to define the processes occurring in the mine. This will provide a means of answering questions such as what the current organic substrate source being utilised by the bacteria is, and what the current supply and availability of this carbon source is. The laboratory scale system may also be used to evaluate alternative organic substrates that may be available at the mine site, or in close proximity.

Based on the definition of the current processes active in the mine, the program can be expanded either to investigate alternative carbon sources, or supplemental sources to enhance the process performance to cope with the expected sulphate and metal loads. Similarly, the overall process performance may be evaluated and the results from the laboratory program may be used to predict the long term performance of the process.

It is essential that the test program be conducted at conditions closely related to the conditions expected within the mine. In order to evaluate the storage capacity and thus the life expectancy for the process, it is necessary to estimate the rate of sludge generation, while utilising the organic source expected to be applied to the system.

2.5.3 Mine Performance Evaluation and Enhancement

Since the mine will be used both as an anaerobic bio-reactor and as a storage unit for the organic and the mineralised products from the sulphate reduction process, it is essential to establish the flow path, capacity, physical behaviour and life expectancy of the mine.

As previously mentioned, tracer tests should be completed in the mine to aid in determining the flow paths and transfer rates.

The geometry of the flow paths will depend on the geometry of the mine workings as originally developed and modified by the filling and collapse that has occurred to date. Changes in this geometry will depend on future stability, given that both timber and rockbolt support will deteriorate. This requires a basic mine development review and rock mechanics assessment.

Accumulation of sludge will depend on the rate of generation and the location and physical properties of deposition. Investigation of these conditions should include a pilot study to provide a large scale simulation of the actual in-mine conditions. A pilot study will also provide an indication of the in-mine rates that may be expected under non-ideal conditions, and will allow a study of the sludge formation and stability, bacterial species, populations and interaction and substrate requirements.

3.0 KJØLI MINE

3.1 History and Setting

Mining for copper occurred at the Kjøli mine from 1766 to 1798 and from 1857 to 1868, while pyrite was exported from this location from 1896 to 1907 and 1910 to 1920. The location and layout of the mine is shown in Figure 15. The mine rock piles were established from 1886 to 1920 and constituted a total volume of approximately 80 000 m^3 (NIVA 1987) and covered an area of approximately 2 ha. Rock samples, taken from test pits (TP - Figure 15) and from the drill holes excavated for the monitoring equipment (DH - Figure 15), were analyzed for the sulphur and metal contents and are summarised in Table 2. The primary sulphide minerals in the mine rock are pyrite, chalcopyrite and sphalerite. Water from the mine site drains via the Storbekken to the Gaula river.

TABLE 2
Kjøli Mine Rock Pile - Sulphur and Metal Analyses

Location	S (%)	Cu (%)	Zn (ppm)	Fe (%)	Ca (%)
TP 1 - 2m	1.73	0.14	50	11.8	0.39
TP 3 - 2m	2.17	0.28	50	13.2	0.20
TP 4 - 2m	2.22	0.07	50	13.7	0.79
DH 1 - comp.	5.66	0.53	69	12.9	0.30
DH 3 - comp.	5.08	0.29	271	13.3	0.20
DH 4 - comp.	2.01	0.10	50	12.2	0.10

3.2 ARD

Prior to any remediative action, acidic drainage originated from the underground mine workings and the rock piles. The results from the 1977 and 1978 (NIVA 1979b) water quality sampling and analyses conducted by NIVA, indicated that the water from the mine contributed approximately 20% of the total ARD metal and sulphate loads from the mine site.

In 1981 Bergvesenet undertook some abatement measures in an attempt to improve the water quality issuing from the rock piles. The mine rock was consolidated, substantially into one large and two smaller piles and recontoured to gentle smooth surface slopes, as shown in Figure 15 and Plate 1. Approximately 100 tonnes of hydrated lime was sprayed onto the surface of the pile. It was anticipated that the smooth contouring of the pile slopes would encourage run-off and that the lime would assist in improving the quality of water infiltrating through the surface and seeping through the pile. The results of water quality draining from the site following implementation of these abatement measures are reported in NIVA 1986. Unexpectedly, the acid drainage from the piles increased each year between 1981 and 1985, as indicated by the average yearly copper loads provided in Table 3

TABLE 3

Annual Materials Transport from the Kjøli Mine

Year	Ca (t/yr)	SO_4^{2-} (t/yr)	Al (t/yr)	Fe (t/yr)	Cu (t/yr)	Zn (t/yr)	Q (m^3/yr)
1977-8	-	-	-	-	4.2	-	578,512
1981	3.5	176	-	29.2	4.9	0.10	458,054
1982	7.7	271	-	49.0	5.7	0.21	311,766
1983	6.1	418	-	60.5	10.9	0.23	711,431
1984	7.1	333	17.4	51.1	13.1	0.22	453,513
1985	16.3	797	48.7	149.6	15.2	0.48	402,296
1986	11.6	540	31.3	93.0	9.7	0.30	385,042
1987	11.3	691	48.5	118.0	12.4	0.34	430,566
1988	12.7	603	38.9	102.2	11.0	0.30	430,572
1989	13.2	430	25.0	71.8	8.1	0.22	567,697
1990	13.2	172	14.5	18.3	2.3	0.13	547,197

It has been suggested (SRK, 1987) that the increase in the load was primarily due to the disturbance of the old mine rock, resulting in the establishment of new infiltration and seepage paths, which allows accumulated ARD products to be transported from the rock piles. Similarly, the disturbances resulting in new air and water pathways will have exposed additional surfaces for increased oxidation to occur. The surface application of hydrated lime was ineffective compared to the increment in the overall ARD caused by the disturbance, as indicated by the increase in the metal transport loads subsequent to the remediation effort.

The results of a seep survey in the vicinity of the pile in 1988 is shown on Figure 15. The pH measurements clearly indicate the pile as the major source of ARD. Note that the drainage in the ditch leading from the mine portal is at pH 3.5 while the drainage from the southern end of the rock pile is at pH 2.2 (see Table 3 for metals transport).

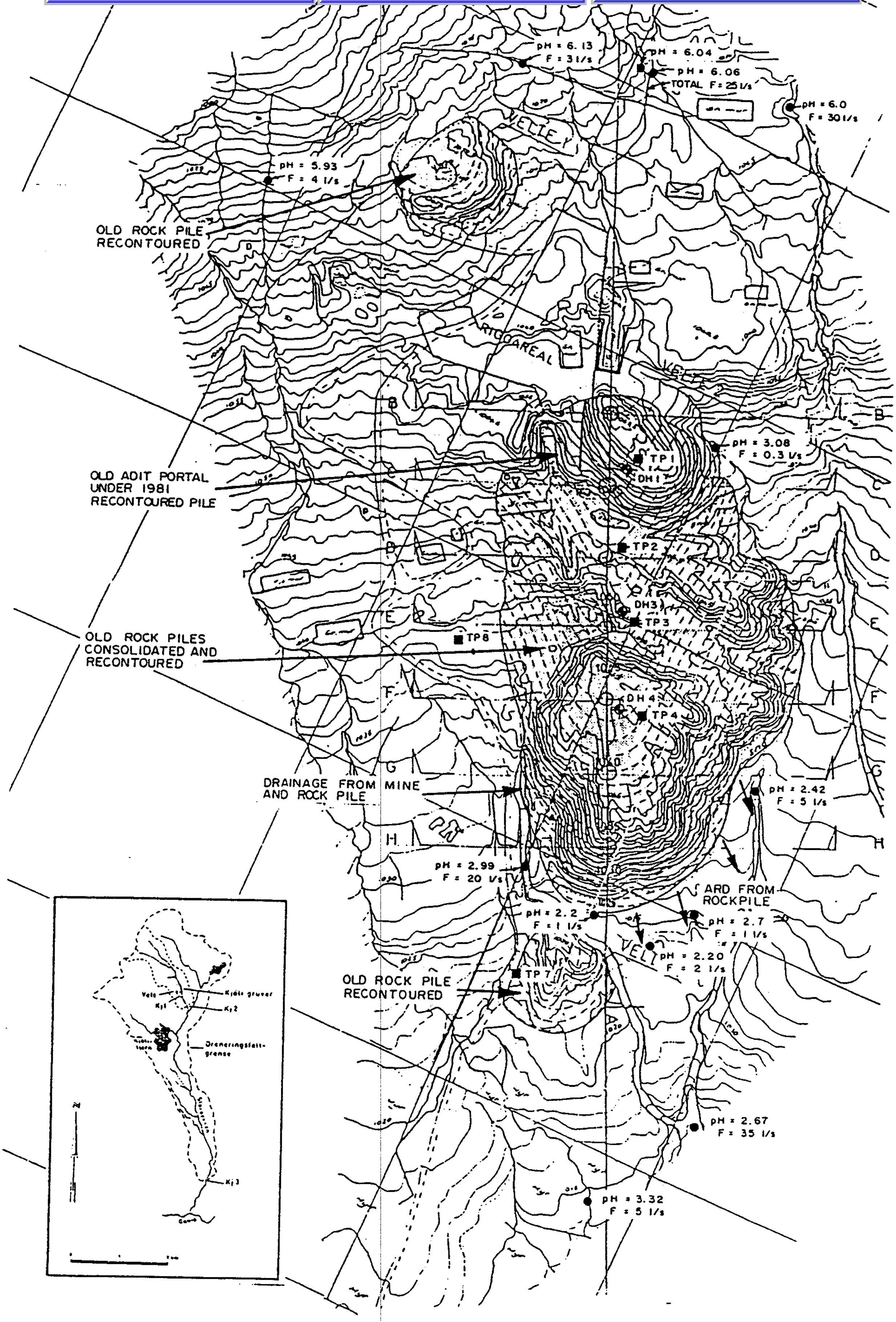


FIGURE 15 PLAN OF 1981 CONSOLIDATION OF KJØLI MINE ROCK PILES

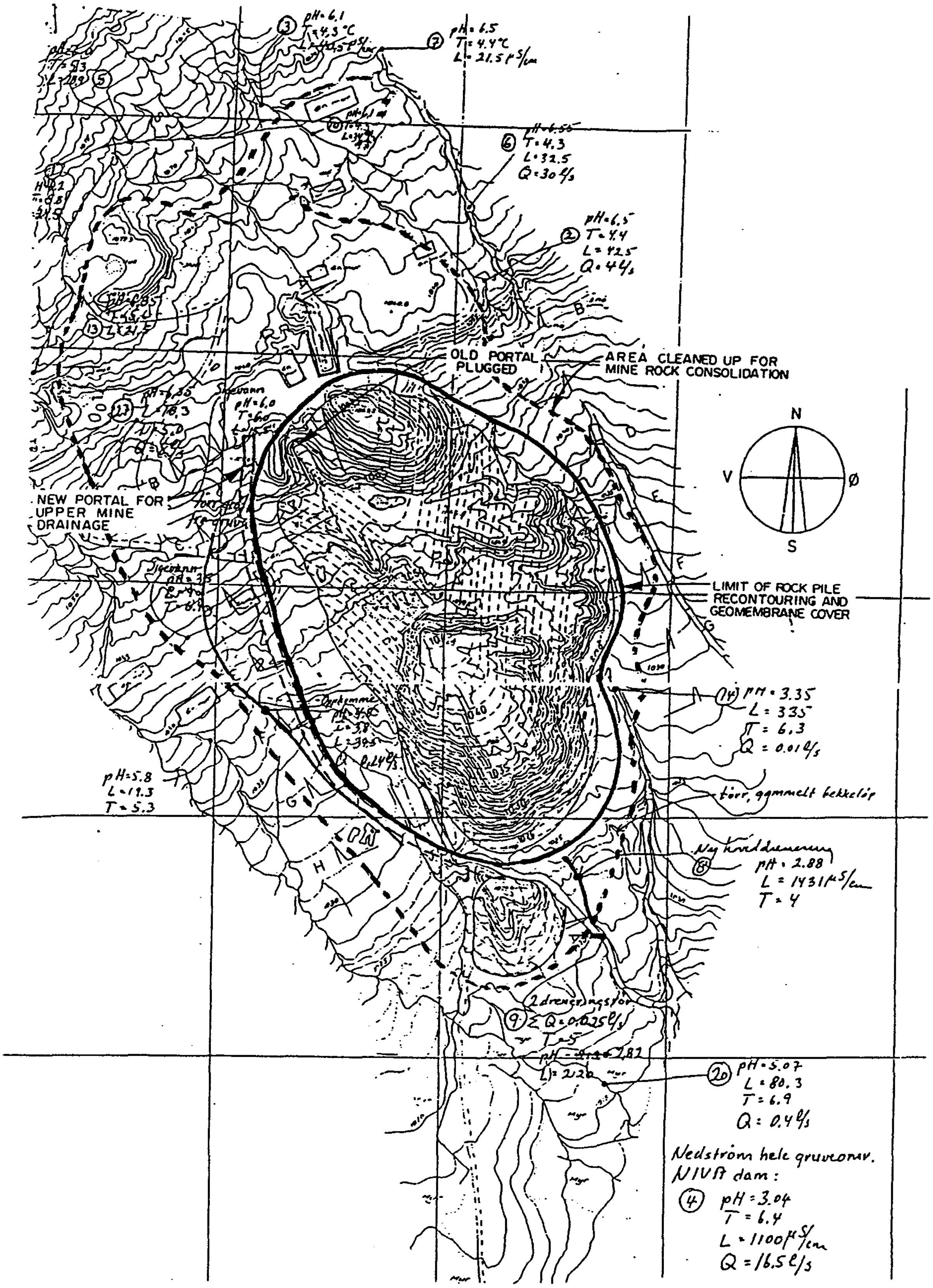


FIGURE 16 KJØLI ROCK PILE REMEDIATION DURING 1989 PROGRAM ; SEEP SURVEY RESULTS FOR SEPT. 20/90

Plate 1: Kjøli - Main rock pile after 1981 remediation program

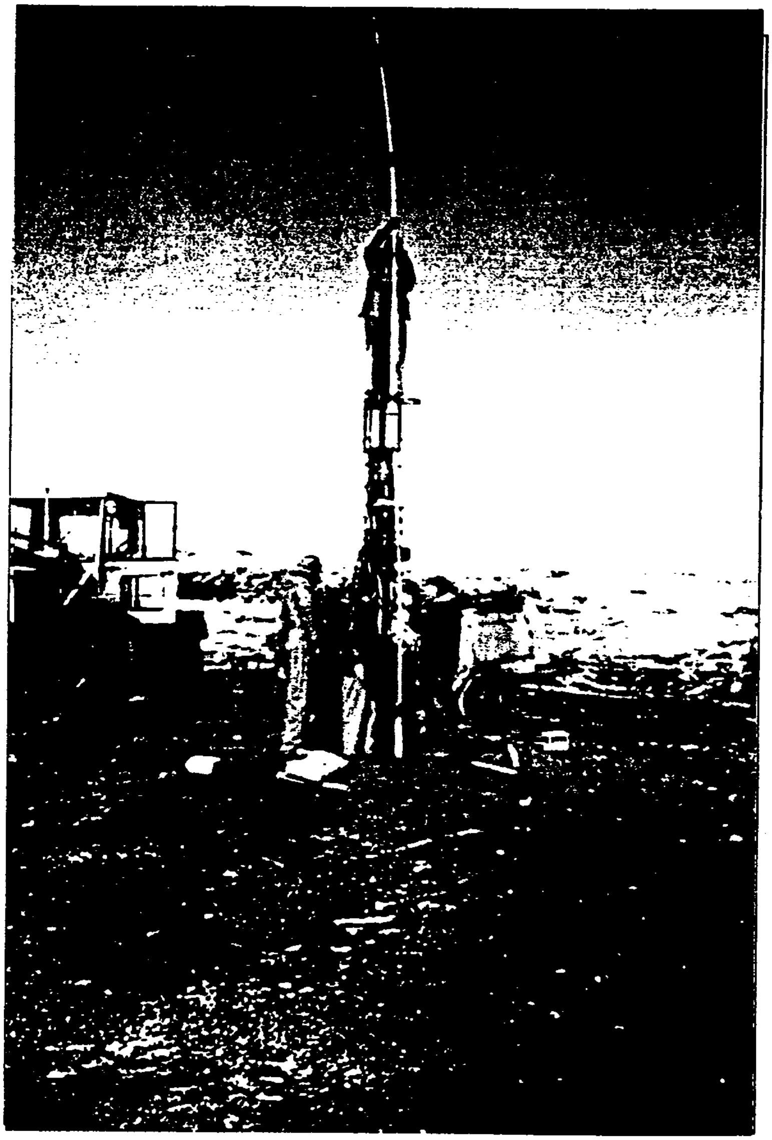


Plate 2: Kjøli - Instrumentation installation

3.3 Remediation

Underground mine

Remediation of the Kjøli site was undertaken in 1989 and for the underground workings consisted of plugging the lower adit of the mine, allowing approximately 60% of the underground workings to be flooded. Since the upper mine entrance emerged under the rock pile, a tunnel diversion was prepared to provide an exit to the west of the pile, as shown in Figure 16. A discharge ditch was prepared with a v-notched overflow gauge for water flow measurement. The mine has filled within two years and was overflowing at the time of the site visit.

Rock Piles

In 1988, prior to any remediation, seepage surveys were conducted around the rock piles and three holes were drilled into the main rock pile, at the locations shown on Figure 15, and instrumented (Plate 2). Instrumentation comprised an open standpipe for water sampling and water level measurements at the base of the pile, and thermistors and pore gas sampling tubes distributed through the vertical profile.

In the summer of 1989 the minor mine rock piles and remaining mine rock over the site area were excavated and consolidated into the main pile which was recontoured and covered with a 2 mm HDPE synthetic membrane composite cover to minimise infiltration. The extent of remediation is shown on Figure 16.

Construction

During the consolidation, the mine rock distributed on the site was excavated with a backhoe (Plate 3) and placed on the main pile. The areas where the mine rock was collected from were not hand cleaned but were covered with a 1 m layer of till (Plate 4). The consolidated rock pile was recontoured to provide a maximum gradient of 3.5 : 1 horizontal to vertical.

A ditch was blasted into bedrock around the mine rock pile to a depth of approximately 2.0 m to allow the geomembrane cover to be sealed into bedrock (Plate 5). Before placing the liner, since no bedding layer was placed between the mine rock and the HDPE liner, the mine rock was compacted with a vibrating roller to smooth out the surfaces in order to minimise potential damage to the geomembrane (Plate 5). A geofabric layer was placed between the mine rock and the geomembrane (Plate 6). The colour of the liner was selected to be white as this would show up damage during placement.

The final composition of the composite liner is illustrated in Figure 17, and consisted of filter cloth placed directly on the mine rock followed by the 2 mm HDPE geomembrane. A geonet layer was placed on top of the geomembrane, to provide a drainage layer, and covered with a geofabric to serve as a filter. This drainage layer was considered necessary for the stability of the steeper parts of the slopes. Two 0.5 m layers of dam quality till were then placed to provide a physical protective cover and vegetation supporting layer (Plates 7 and 8). Finally, rip rap was placed in the areas of the slope with a steep gradient to minimise erosion. The exposed till cover was fertilized and revegetated.

The seal around the mine rock pile was constructed by extending the geomembrane, as illustrated in Figure 17, to the base of the ditch constructed around the perimeter of the rock pile, and refilling the ditch with till.

Quality assurance of the liner connections was achieved through double welding and air testing.

Difficulties

Some damages to the HDPE membrane occurred during construction as a result of heavy transport vehicle movement during placement of the till covers. This damage was easily repaired by conventional patching techniques. Generally, little difficulty was experienced with the implementation of the remediation measures.

Costs

The cost for the remediation completed at the Kjøli mine, excluding the cost for the program completed in 1981, is summarised in Table 4. A total surface area of 27 000 m^2 was covered.

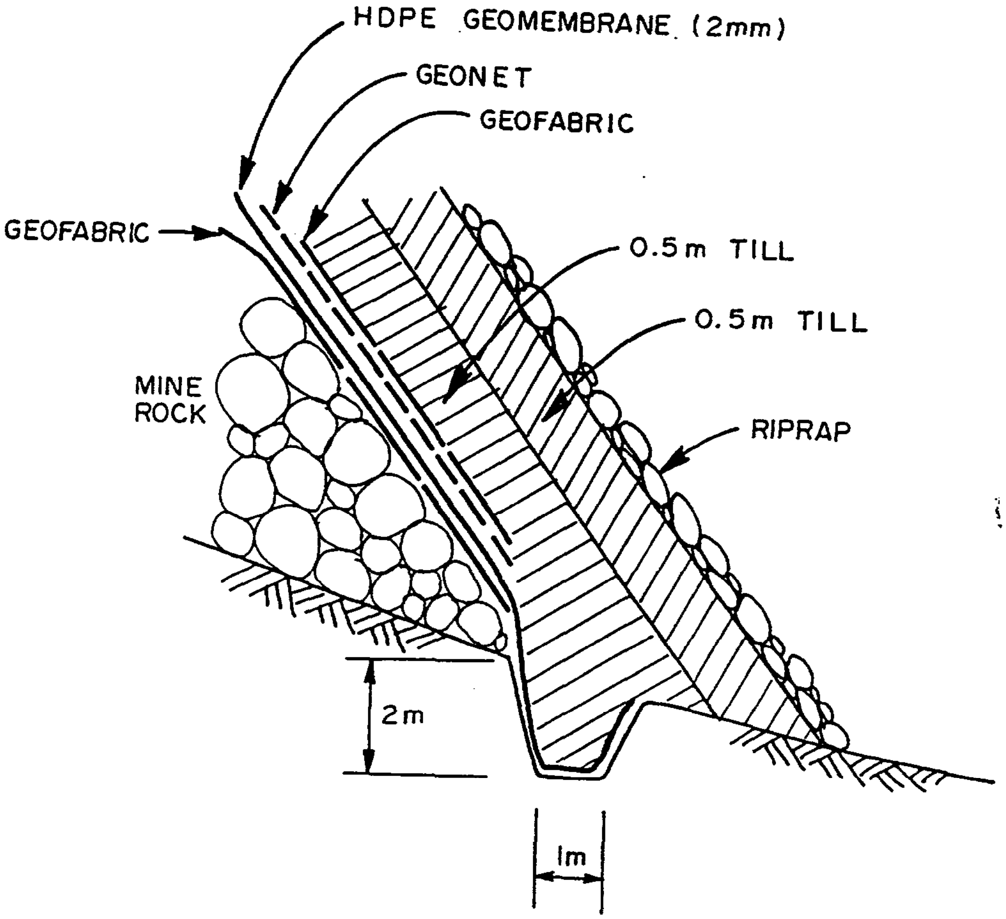


FIGURE 17 COVER COMPOSITION AND PERIMETER SEAL

Plate 3: Kjøli - Old Mine rock stripped to bedrock by backhoe

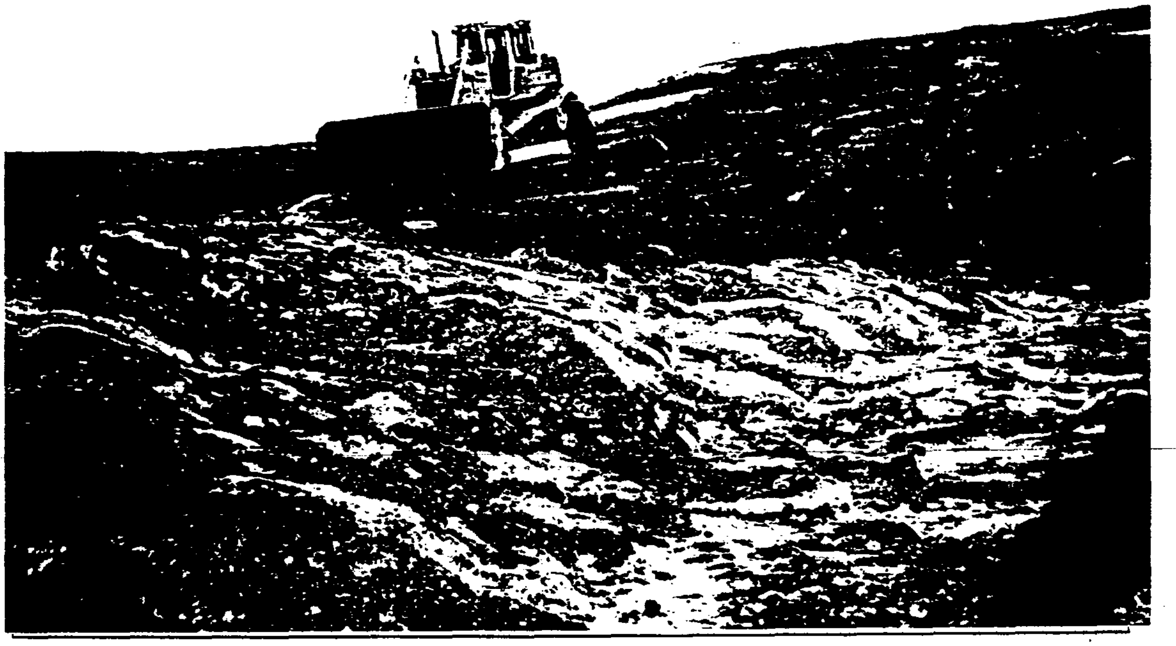


Plate 4: Kjøli - Till being spread over area from which mine rock has been removed

Welcome Screen Search Report List

Plate 5: Kjøli - Ditch at edge of rock pile into which geomembrane cover was sealed

Plate 6: Kjøli - Installation of lower geofabric layer and geomembrane

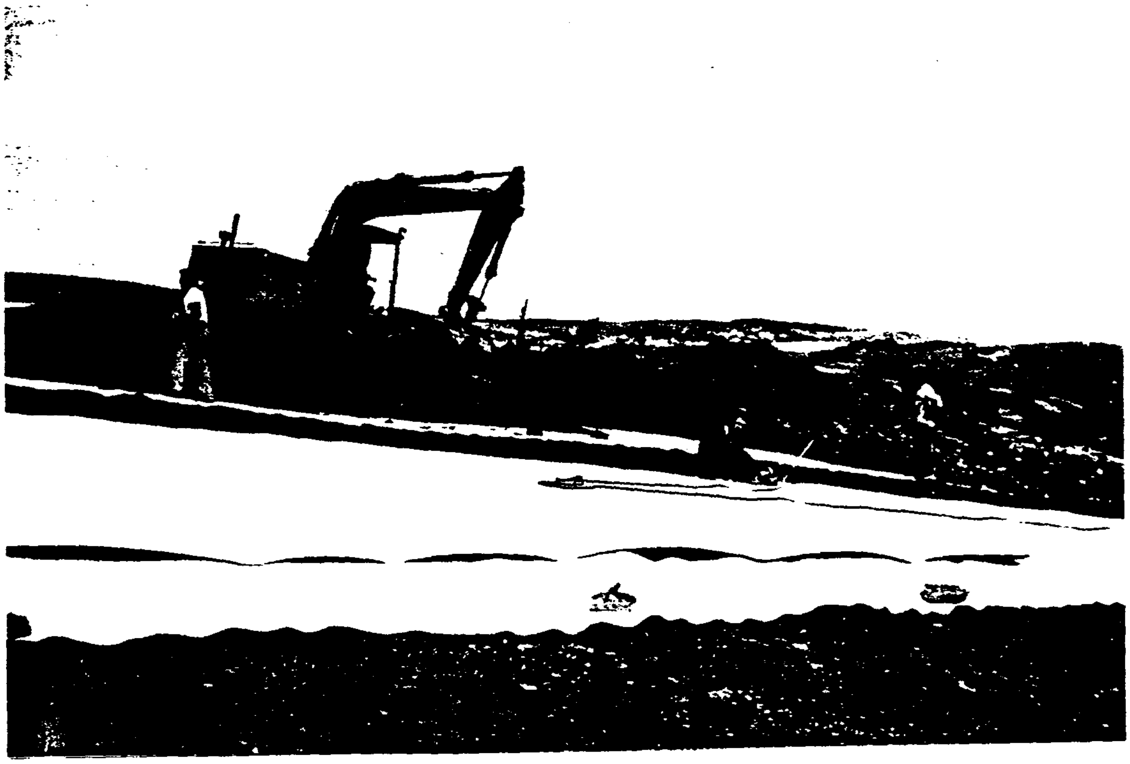


Plate 7: Kjøli - Various layers of cover being placed

Plate 8: Kjøli - General view of rock pile with cover being placed

TABLE 4
Remediation Costs at the Kjøli Mine Site

ITEM	COST	
	Norwegian Krone	Canadian Dollars
Consulting	388 000	65 960
Control and Quality Assurance	464 000	78 880
Access Roads	536 000	91 120
Additional Roads	28 000	4 760
Closing of Adit (Concrete Plug)	124 000	21 080
Injection of Grout	7 000	1 190
Construction of new tunnel	185 000	31 450
Safety Measures	78 000	13 260
Contouring	75 000	12 750
Additional Ditch	21 000	3 570
Measuring point	112 000	19 040
Transport of Heap #1	147 000	24 990
Cleaning	489 000	83 130
Contouring	128 000	21 760
Perimeter Ditch Construction	381 000	64 770
Drainage Net - Purchase (partial coverage)	730 000	124 100
Installation	88 000	14 960
HDPE Membrane - Purchase	1 666 000	283 220
Installation	332 000	56 440
Filter Cloth - Purchase	483 000	82 110
Installation	172 000	29 240
Till Covers	1 251 000	212 670
Rip Rap (Quarry and placement)	883 000	150 110
Perimeter Ditch Refill	306 000	52 020
Lime	280 000	47 600
Revegetation Consultant	6 000	600
Revegetation	200 000	34 000
TOTAL	9 560 000	1 625 200

(Note: Pyrite was found in the original quarry - costs include covering and relocating of the quarry site)

Therefore, the cost for mine rock consolidation and placing the cover is approximately CAD $10.00 per tonne. The cost of cover placement, per square meter of cover, was about $5.13 for contouring and ditching, $21.85 for the supply and installation of all the geosynthetic layers and $13.44 for the placement of the till and rip-rap, for a total of $40.42 per square meter.

3.4 Monitoring and Performance

The 3 monitoring wells installed in 1988 were extended through the cover. The results of the monitoring program are enclosed in Appendix B and the temperature and oxygen concentration profiles are illustrated in Figures 18 to 35.

Oxygen Profiles

The oxygen concentration in most of the rock pile remained stable at approximately 20% for the 12 months after the installation of the instruments but prior to the placement of the cover, as observed for drill hole locations 1 and 4 at all depths. However, drill hole location 3 showed lower initial oxygen concentrations (6.2%) close to the surface while at depth concentrations close to 20 % were observed. This hole is located in the materials which were moved during the 1981 recontouring and may contain a greater percentage of fines in the near surface layers which may be more acid generating and oxygen consuming. It is concluded that acid generation was not oxygen limited at any location in the pile prior to cover placement.

Following installation of the cover in the summer of 1989 (twelve months after the instruments were installed), rapid oxygen depletion was observed for all locations at all depths. However, it should be noted that for drill hole locations 1 and 3 the extent of depletion was greater at or close to the surface than at depth. This suggests that oxygen may be entering the system at the base of the rock pile. Since convective flow through the cover is inhibited it is concluded that the oxygen entry is due to one or a combination of 'barometric breathing' flows bypassing the edge seals or convective flows along the base of the pile from points of the sealing ditch which are at lower elevations to points of the ditch at higher elevations. Regardless of the oxygen source it is apparent that while oxygen concentrations are considerably reduced there is still significant concentrations (5 to 10%) in drill holes 1 and 3. There is still sufficient oxygen entering the pile for acid generation to be maintained.

It should also be noted from the initial data that the oxygen profile indicates an inverse relationship to the temperature profile, i.e. when a low in the temperature is observed, the oxygen concentration is at a high and visa versa. This may be related to a 'breathing' action where, during cooler temperatures a low pressure is created within the mine rock pile, thus promoting air entry into the rock pile, while at warmer temperatures a high pressure is generated resulting in a net exit from the pile.

Temperature

The temperature profile for the mine rock close to the surface to a depth of approximately 3.0 m accurately reflects the seasonal temperature variation. At depths in excess of 4.0 m the temperature profiles are less affected by the seasonal variation. The relatively high temperatures observed at depth indicate continued oxidation. In general a downward trend in all the temperature profiles are observed

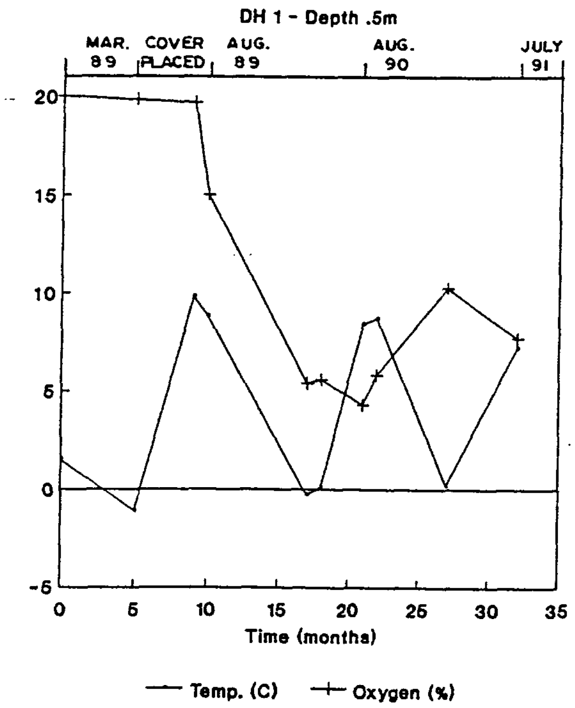


Figure 18

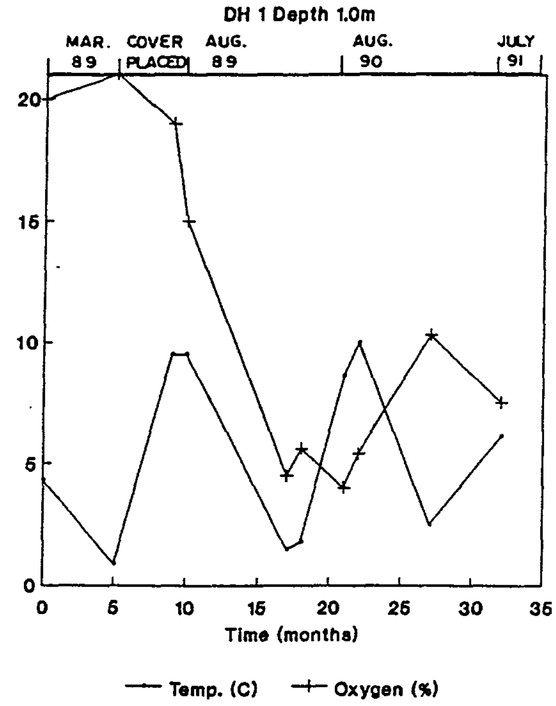
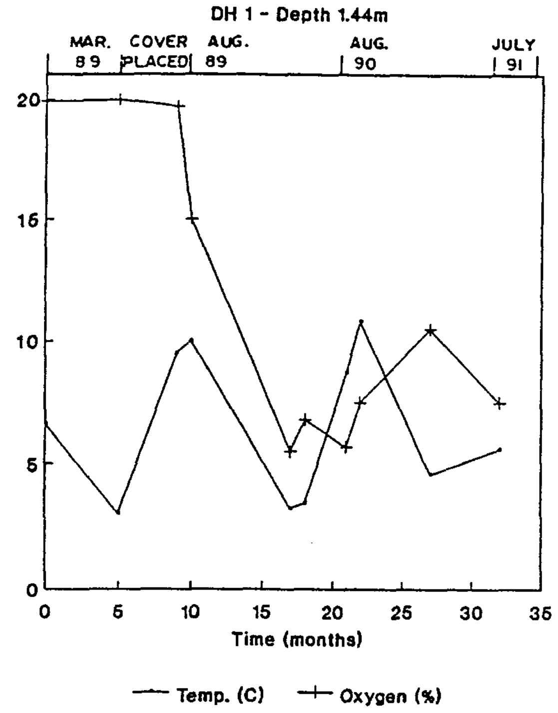


Figure 19



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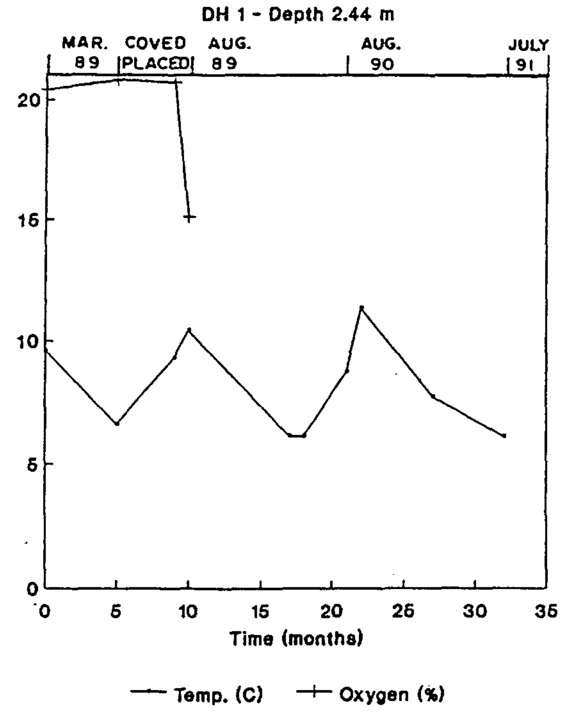

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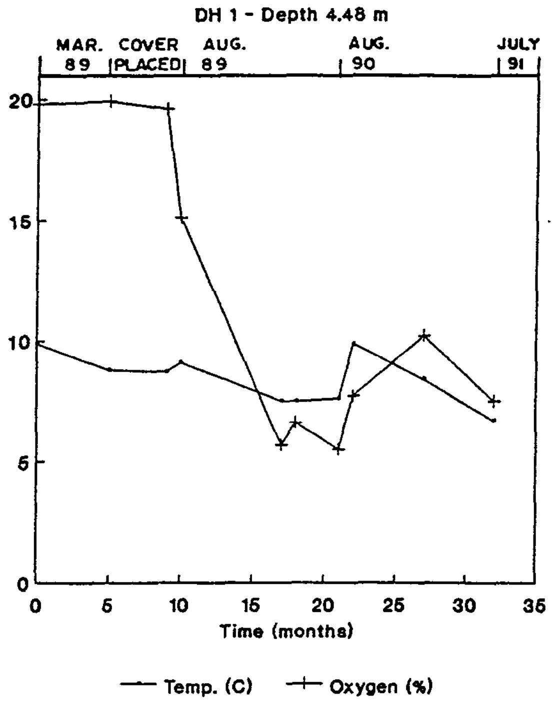

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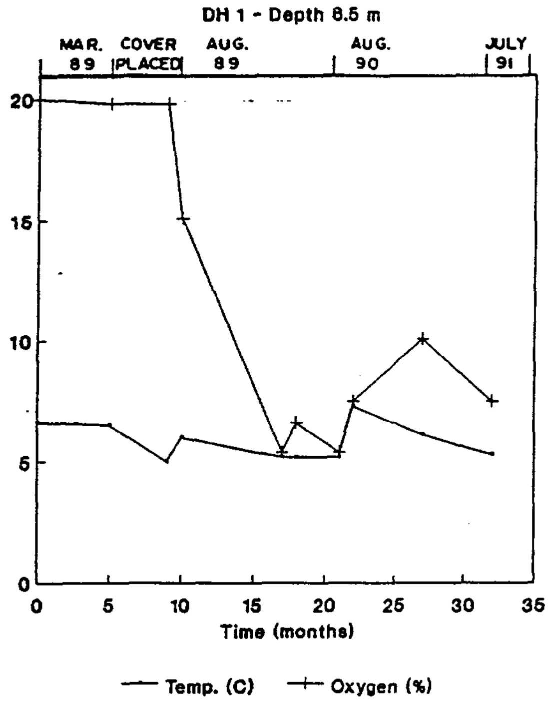

Figure 23

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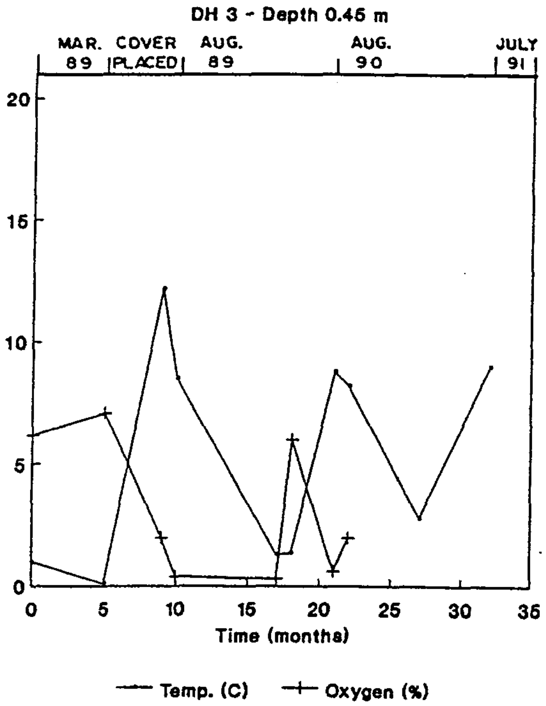

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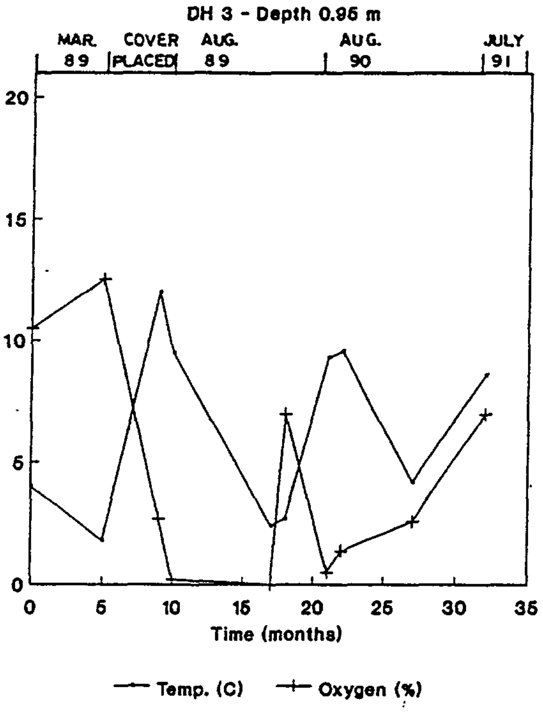

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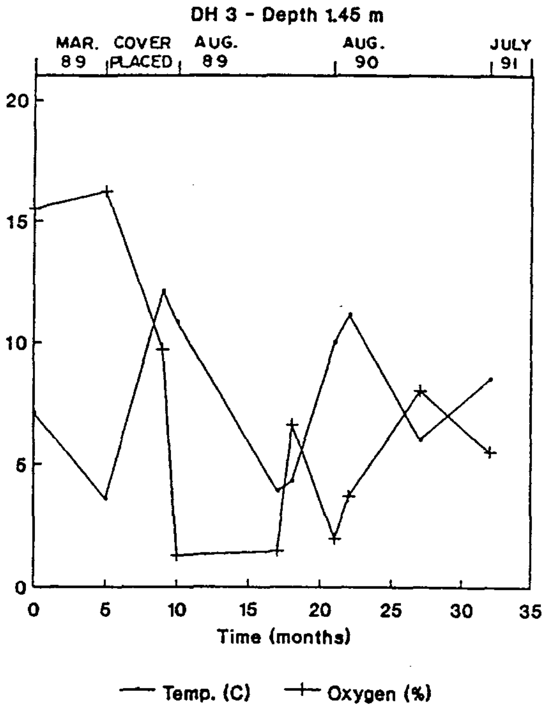

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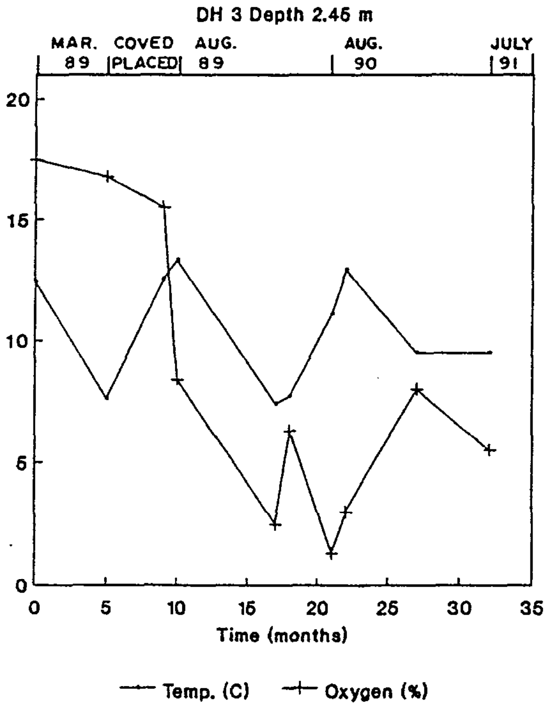

Figure 27

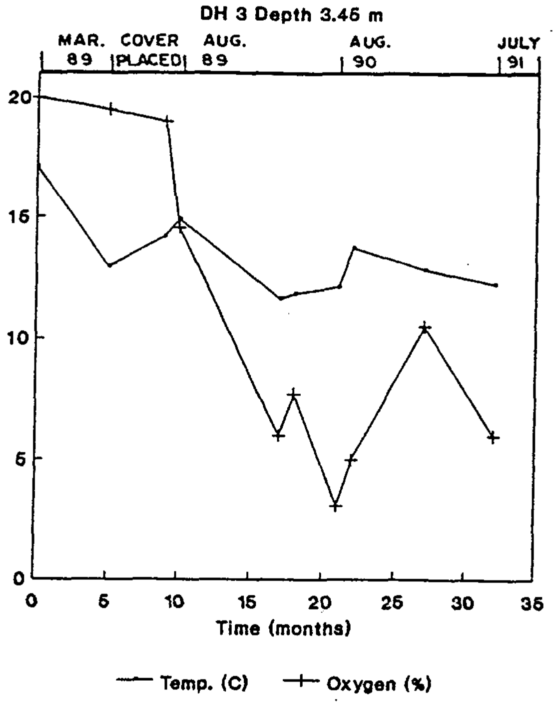

Figure 28

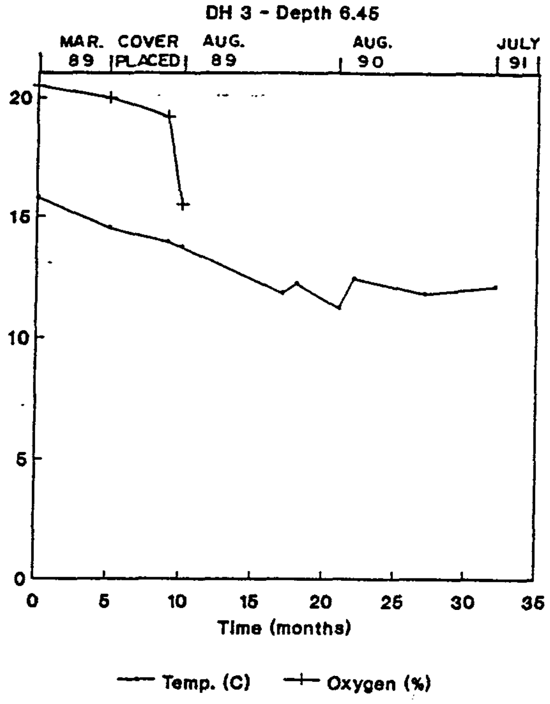

Figure 29

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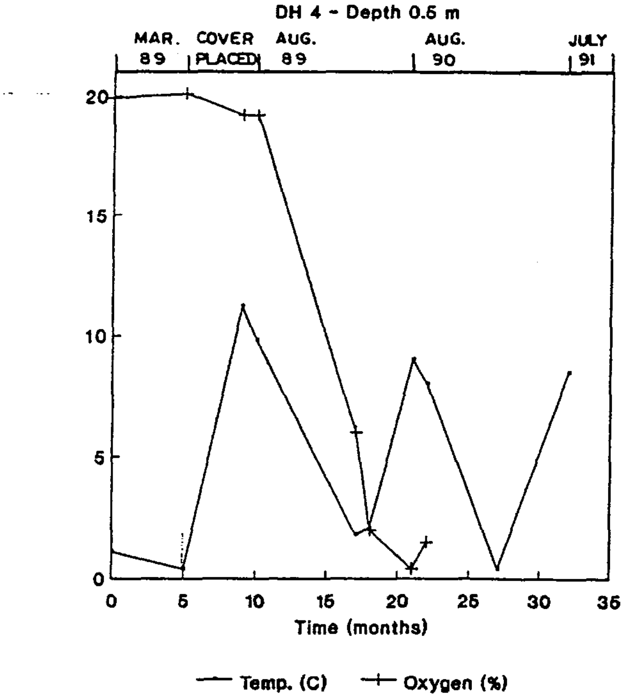


Figure 30

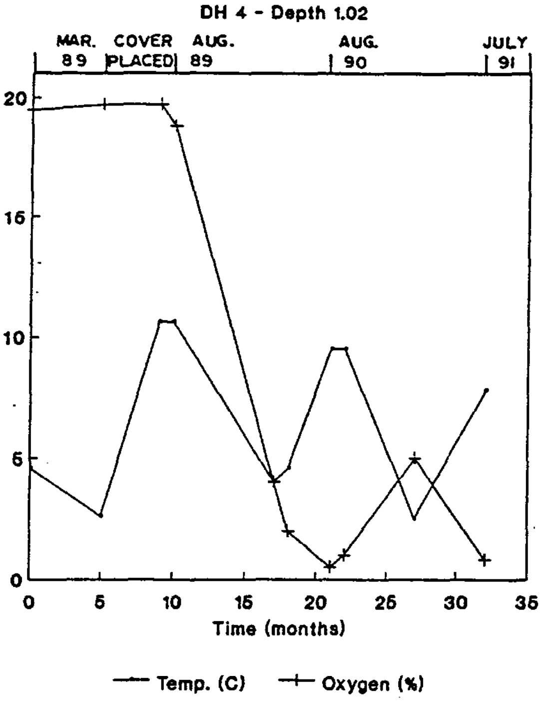


Figure 31

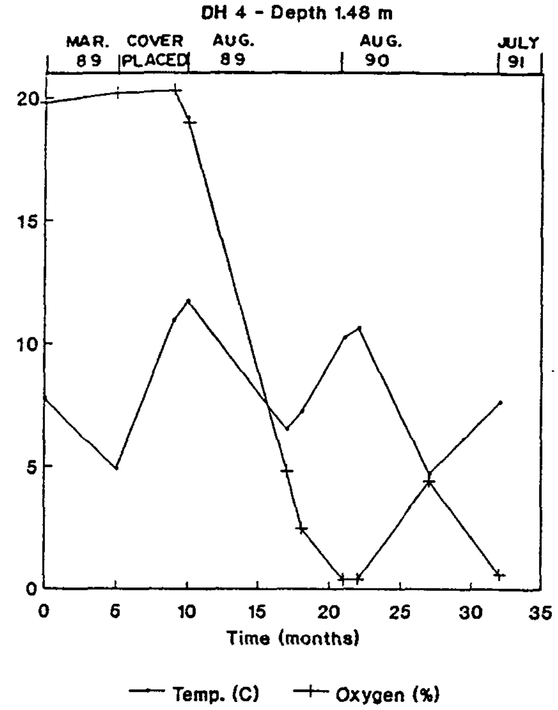


Figure 32

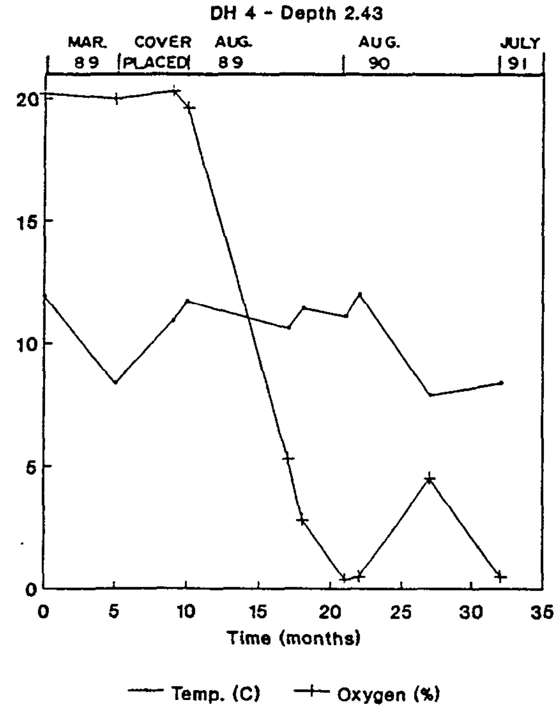


Figure 33

Figure 34

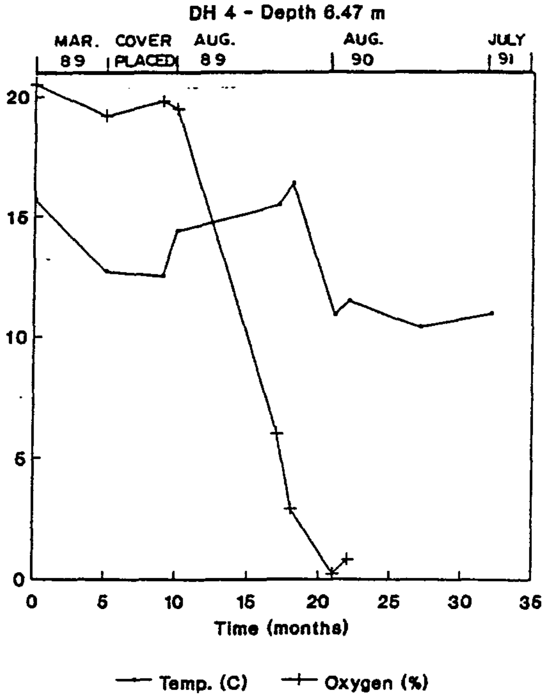


Figure 35

at depth since the placement of the cover. This trend is well illustrated by the drill hole 3 location at a depth of 6.45 m. It is concluded that the rate of oxidation in the rock pile has been significantly reduced as a result of the cover placement.

Water Quality

Water issuing from the rock pile is still at a relatively low pH. At the time of the site visit a pH of 3.5 was observed for the seepage from the southern limit of the rock pile, up from 2.2 at the time of the 1988 seep survey. The results of a seep survey in September, 1990, is shown on Figure 16. The pH observed at the southern end of the rock pile in the 1990 survey is 2.88 midway between the 1988 and 1991 observations. Such a slow improvement may be attributed to a slow decline in the volume of the contaminated seepage from the pile as it drains to its field moisture content.

During the site visit the water flow rate observed issuing from the southern end of the rock pile was considerable. It was raining lightly at the time of the site visit and no surface runoff was observed on the cover. Water may be infiltrating through the till to the drainage net and migrating along the geomembrane to appear as a seepage from the pile, which may explain the relatively high flow rate at this point. This dilution of the reduced volume contaminated seepage with the fresh water migrating along the geomembrane could account for the increase in pH measurements.

It is reported that moist air venting occurs at the points where the instrumented borehole casings penetrate the cover. This suggests another moisture migration mechanism within the covered rock pile. Some of the water observed as seepage from the pile may be generated as a result of an evaporation - condensation mechanism: water at depth (in the mine rock) in close proximity of the elevated temperature zone may be evaporated, and condensed at or near the cooler cover. The accumulated water at the top of the pile migrates downward carrying with it the oxidation products.

The cover has effectively reduced the ARD from the mine site. This is most evident in the reduction of the annual metal transport from the site. While the total copper transport from the site was reduced from 8.3 tonnes per year in 1989 to 2.3 tonnes per year in 1990, the total copper transport from the site for 1991, at the time of the review, was estimated to be less than 1 tonne. The reduction in load from the site, results in an equivalent copper concentration reduction from 160 µg/L to less than 20 µg/L in the Gaula river.

However, it should be noted that the mine water exiting from the underground workings, was at a pH of approximately 2.5 at the time of the visit, and a significant flow rate was observed at this source. This discharge had apparently not been previously observed indicating that it had taken about two years for the mine workings to fill to the overflow point.

It was previously estimated that the mine water contributed only 15 - 20% of the total copper load prior to remediation. During flooding the acid products which would have accumulated in the mine workings will have been dissolved and the discharge water quality would reflect this. Since the contribution of the mine water to the ARD draining from the site would have been absent during the period of mine filling there may now be some increase in the copper load. This increased copper load would be expected to decrease as the stored acid load in the mine is flushed out.

At the time of the visit the water flow measuring weir located in the diversion ditch for the water exiting from the underground mine workings was ineffective. The drainage disappeared into the ground before reaching the weir, and emerged downstream from the weir. It is understood that this weir will be relocated to circumvent this problem. Measurements of flows and concentrations at the weir will be required to separate the effects of the mine drainage from the rock pile drainage.

3.5 Conclusions and Recommendations

The execution of the remediation plan has been completed successfully. The contractor should be commended on the quality of the site work and an excellent attention to detail.

The placement of rip rap in layers and berms has been very successful as an erosion control measure. Very little evidence of erosion was found on the cover or at adjacent ditching and recontoured surfaces. Revegetation of the till cover also appears to be developing successfully. It is apparent that the site has been returned to a land form and use equivalent to what existed prior to mining. The remaining environmental damage is the acidity and metal concentrations in the site drainage, which has been considerably abated by the remediation plan.

The current temperature and oxygen monitoring program is providing very useful and unique information on the performance and effectiveness of a geomembrane covered mine rock pile. As equivalent information cannot be obtained anywhere else in the world, it provides the best source of such data for the evaluation and modelling of geomembrane cover performance internationally. It is recommended that the present frequency of instrument monitoring be maintained at four episodes per year for a further two years and then at one episode per year for another 3 years. This information should be published to allow its use by others for the development of appropriate modelling and assessment techniques, such as the Canadian MEND sponsored Mine Rock Model Development Program.

4.0 KILLINGDAL

4.1 History and Setting

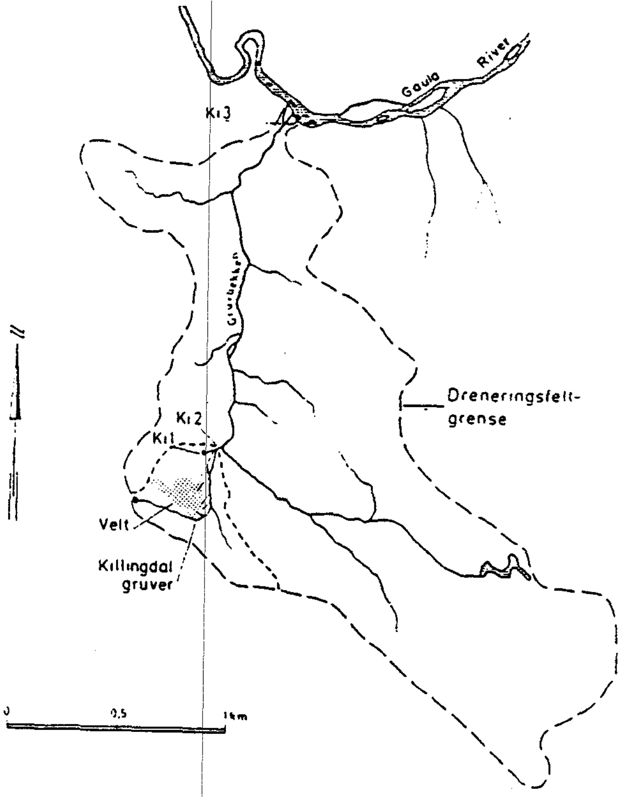
The Killingdal mine is located in the same region as the Kjøli mine at the location shown on Figure 36. There are two principal mining areas associated with the Killingdal mine site; the Nodre mine was established in 1674 and abandoned in 1968 and is situated at the top of the mountain, and the Sondre mine was started in 1793 and decommissioned in 1986. The two mines are connected and extend to a depth of about 1500 m, as illustrated in Figure 36. The ore mined graded between 1 and 2 % copper, 3 to 8% zinc and 20 to 50 % sulphur. From 1953 to 1986, the ore mined at the site was transported via rail to a treatment plant in Trondheim.

The typical mine development consists of an incline shaft at about 30 degrees in the hanging wall with cross-cuts to the stope. The stopes are mainly of open room and pillar construction. However, mine rock has been placed in some of the older workings. Approximately 150 000 m^3 of mine rock has been placed near the upper (old) mine entry.

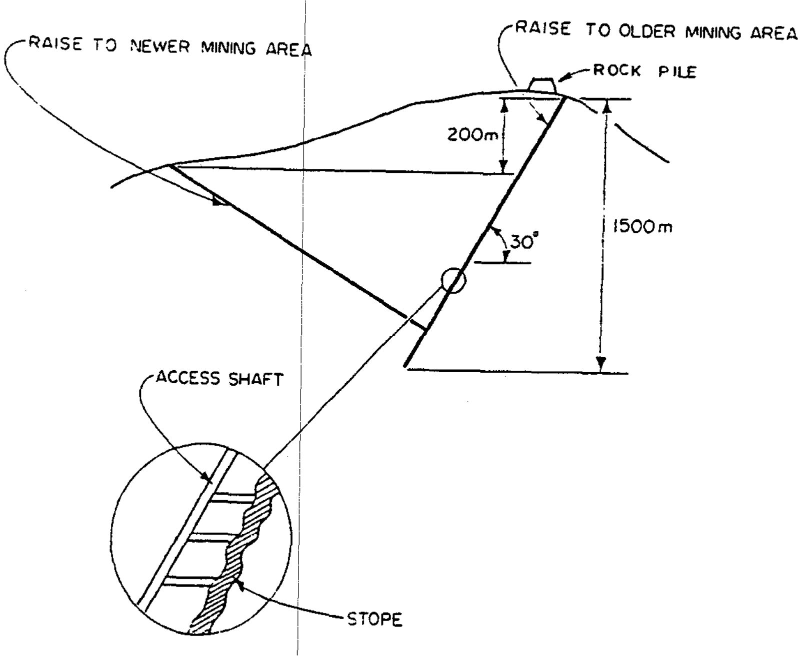
4.2 ARD

The primary source for ARD from the site is from the rock piles placed at the upper mine entry (Plate 9). This source is responsible for approximately two thirds of the total copper transported from the site. The mine rock varies considerably in nature, including coarse and fine rock in discrete piles, burned or roasted rock and newer coarse mine rock.

Secondly, mine drainage used to be pumped from the underground workings which contributed one third of the total copper transported from the site. Pumping of this mine water has been discontinued. Mine water draining from an adit just below the entry to the upper mine contributed marginally to the overall ARD from the site. The copper and zinc loads from these sources are summarised in Table 5.



MINE LOCATION



SCHEMATIC SECTION

FIGURE 36 KILLINGDAL MINE

TABLE 5
Annual Copper and Zinc Loads from the Killingdal Mine

SOURCE	CONCENTRATION (mg/L)	LOAD (t/a)
COPPER		
i) Mine Rock Pile	40	2.5
ii) Pumped mine water	625	1.2
iii) Adit	80	0.2
ZINC		
i) Mine Rock Pile	175	12
ii) Pumped mine water	7550	13
iii) Adit	385	1

ARD from the site drains into Gruvebekken which flows into the Gaula river 2 km below the mine site.

4.3 Remediation

The remediation action taken at the old mine site in 1990 consisted of consolidating the discreet mine rock piles. The consolidated rock pile (Plate 10) was then recontoured and covered with till to a depth of 1 m and a course rip-rap layer.

The consolidated rock pile was located up-slope from the upper mine adit to allow collection of the seepage from the rock pile and the routing of it to enter the mine at the adit. Prior calculations had shown that the natural convection through the mine provides evaporation in excess of the rock pile water drainage. It was therefore anticipated that water draining into the mine would evaporate as it flowed down through the workings and would result in the accumulation of the oxidation products in the mine.

To achieve this, a collection ditch was designed at the toe of the mine rock pile, equipped with a drainage collection pipe. The location and nature of the collection ditch is illustrated in Plates 11 and 12.

A total of 5 000 000 NOK ($850 000 CAD) was spent on the remediation work.

Plate 9: Killingdal - Rock piles and minor dumps at upper mine shaft prior to remediation

Plate 10: Killingdal - Rock pile after consolidation and covering

Plate 11: Killingdal - Covered rock pile showing location of drainage ditch

Plate 12: Killingdal - Portion of drainage ditch with drain pipes still to be completed

4.4 Monitoring and Performance

At the time of the site visit, the collection ditch had, as yet, not been completed, i.e. the perforated collection pipe and access manholes had been constructed, but joint repairs and sealing had not been completed and the ditch had not been refilled. The collection pipe was broken at several locations, resulting in no or very little ARD from the rock pile actually entering the mine adit. It was also noted that, although the diversion ditch had, for at least part of its length, been blasted into bedrock, the rock was highly fractured and has a poor water retention ability. Thus the seepage entering the ditch did not follow the course of the ditch but passed through it and appeared below it as seepage. It was also noted that the ditch appeared to be located within soils or mine rock for at least part of its length and can therefore only be partly effective. The elevation of the ditch is restricted by the elevation of the adit and deepening the ditch to locate it in more competent bedrock is not feasible.

Seepage from the soils and mine rock exposed below the diversion ditch was observed to be richly coloured by the high metal content, and a pH of 2.9 was measured for a number of seeps. Very little surface runoff was observed on the cover, suggesting a significant infiltration rate to the pile. Some cracking of the till cover was also noticed.

Runoff in the fresh water diversion ditch also showed some contamination with the pH ranging from 4.9 to 5.0.

Mr. Ese of Bergvesenet indicated that copper load measurements in the run-off from the site had indicated large improvements, despite the incomplete drain. This he attributed to the effectiveness of consolidation of the mine rock pile and the till cover. No copper load measurement data was available at the time of the visit to determine the nature of the improvement.

Highly contaminated runoff with low pH values were observed from the mine rock placed at the new (Sondre) mine entry. Although it is recognised that the relative scale of this deposition is small in comparison with the main mine rock pile, the environmental impact from this site cannot be neglected.

4.5 Recommendations and Conclusions

The concept of utilising the natural convection through the mine as an evaporative treatment process presents an ingenious remediation measure, provided the evaporation rate can cope with the total hydraulic load.

However, to utilise the potential of this idea it is necessary to introduce the seepage to the mine. Since the effectiveness of the collection ditch is impaired by its elevation above sound low permeability bedrock, it may be necessary to investigate alternative collection strategies.

Monitoring of the copper load in Gruvebekken will define the effectiveness of the current system, once implementation is completed. If copper reductions are inadequate then it may be appropriate to construct a collection ditch at a lower elevation to drain to a sump from which drainage can be pumped to the upper adit. As an alternative to pumping, a borehole drilled to intersect the old stopes may be considered to drain directly into the mine. A third alternative may be the placement of a geomembrane over the till cover to improve cover performance.

It is expected that, with ever increasing acid and metal concentrations in the accumulated seepage in the mine, the efficiency of the evaporation process will be reduced. This effect should be evaluated thermodynamically, and the concentrated solution accumulation rate should be evaluated and be compared with the mine storage capacity.

The metal load in the seepage from the mine rock at the new (Sondre) mine adit should be surveyed to determine if any remediation may be necessary.

The bars placed on the ventilation shaft were secured by means of wire bindings only. The bars were found to be relatively loose and could be moved by hand. From a safety point of view, the ventilation shaft should be equipped with a more permanent structure to prevent accidental access to the shaft.

5.0 SKOROVAS

5.1 History and Setting

The Skorovas mine is located approximately 260 km north of Trondheim. A highly mineralised pyritic vein outcrops on the side of Rauberget (the Red Mountain), and significant natural ARD has been generated from this site for the centuries since the last glacial scouring of the area. The natural ARD drained from the outcrop into a small lake, Dausjøen (Dead Sea), located at the foot of the mountain. Based on sludge deposition depths measured in the lake, it has been estimated that between 8 and 15 million tonnes of metal hydroxide precipitate has been deposited in the bottom of lake Dausjøen providing an excellent opportunity to study the long term stability and fate of such sludge on aging.

The water from the Dausjøen drains into lake Store Skorovatn, from where the Skorovas river flows to the Grøndals river and finally to the Namsen river as indicated on Figure 37. The Namsen river is considered one of the most valuable salmon rivers in Norway.

During the late 1800s' and up to 1940 a number of surveys and test mining operations had been conducted at the site. The disturbances created by these test operations further contributed to the ARD originating from the site. Mine production at a rate of 500 tonnes per day from an underground operation started in 1952. The ore contained an average of 37% sulphur, 1.1 % copper and 2.2% zinc. The mine produced a pyrite concentrate for acid production until 1976. Subsequently copper and zinc concentrates were produced until the ore deposit was depleted in 1984. Approximately 1.5 million tonnes of tailings were deposited directly into the Dausjøen during this period. A mine rock pile of approximately 160 000 m^3 was developed on the side of the mine adjacent to the mine entrance at this time (Plate 13).

5.2 ARD

ARD from the site impacts on two different drainage areas, as shown in Figure 37. Mine water drained at a rate of 2.1 L/s from an adit (originally an exploration adit) to the Stallvik river which flows into Lake Tunnsjøen (90 km^2), 10 km from the site. Annually, approximately 8-10 tonnes of copper and 17-21 tonnes of zinc was transported from this source. While the Stallvik river is highly contaminated, the effect of the ARD is limited to the bay of this river in Lake Tunnsjøen.

Three sources of ARD from the surface of the site were identified, which drain via Skorovaselva to the Namsen:

- Natural runoff from Rauberget, most of which flows along Gruvebekken (Mine Creek) to Dausjøen;
- seepage from the mine rock pile adjacent to the mine entry; and
- several smaller rock piles scattered in the area.

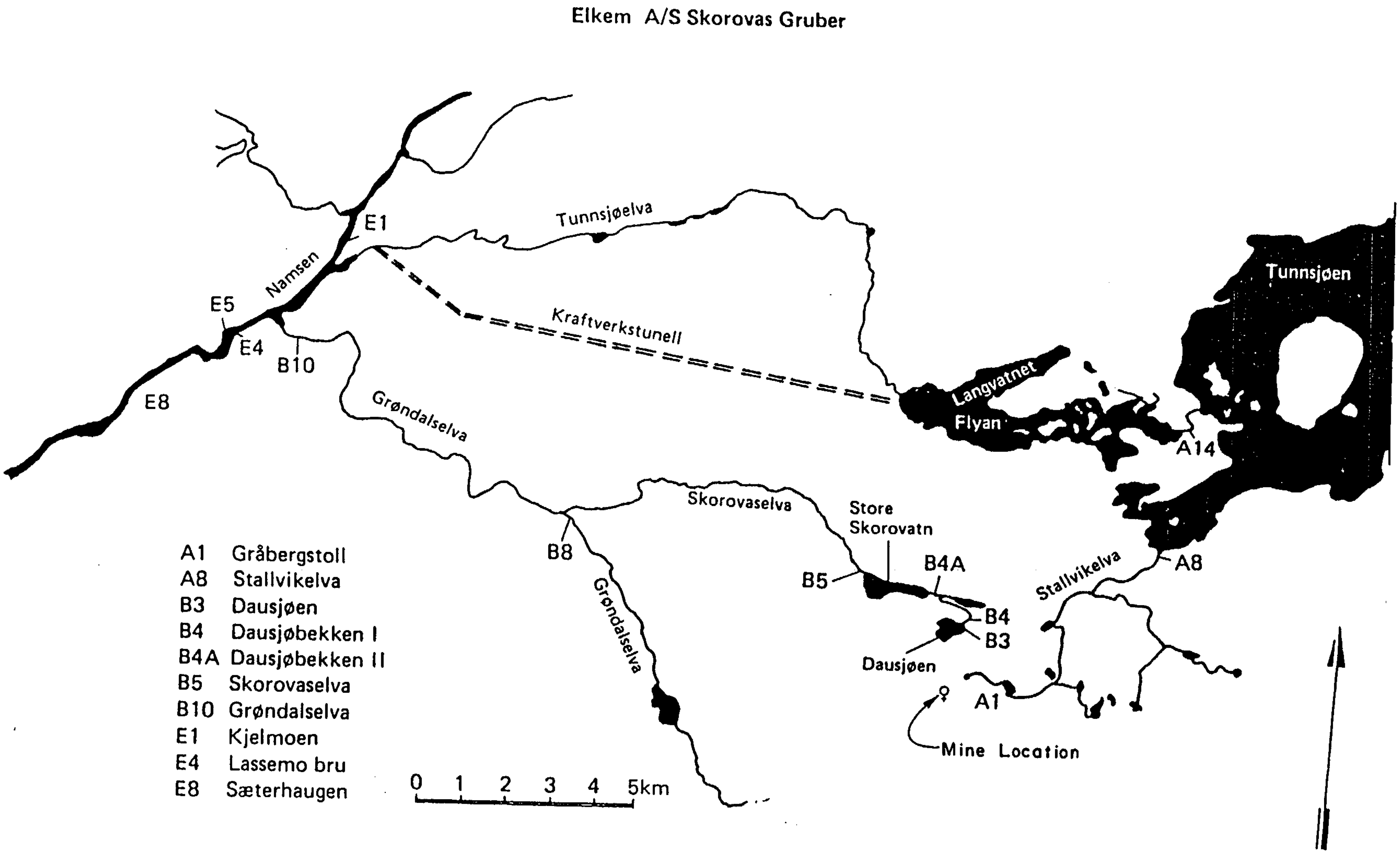


FIGURE 37 LOCATION OF SKOROVAS MINE AND SAMPLING STATIONS

Welcome Screen Search Report List

Plate 13: Skorovas - General view of mine rock prior to removal

Plate 14: Skorovas - Metal hydroxide accumulation in grassy flood plain of Dausjøen Creek upstream of Store Skorovatn

As a consequence of mine development and operation a significant drop in the pH of the drainage from Dausjøen was observed, resulting in an increased metal transport from Dausjøen. The following abatement measures were applied in an attempt to reduce the metal transport:

- The acidic seepage from the mine rock pile and water from Gruvebekken was diverted downstream of Dausjøen, and
- a liming station was installed in Dausjøen creek. The pH of the water flowing from Dausjøen was monitored on a continuous basis and milk of lime added directly to the stream flow on a proportional basis. Mixing and flocculation occurred in the stream flow with some resulting metal hydroxide precipitation accumulation in grassy shallows in the stream floodplain (Plate 14) or in lake Store Skorevatn. This lower lake is shallow and wind and wave action stirring of the sludge appears to have resulted in most of the hydroxides having been flushed through the lake. Approximately 400 tonnes of lime was consumed annually.

The annual transport of copper and zinc from the mine site post closure of mining operation is provided in Table 6.

TABLE 6
Surface Runoff Transport of Copper and Zinc from the Mine Site to Dausjøen Creek

YEAR	COPPER (t/a)	ZINC (t/a)
1985	4.2	10.1
1986	4.5	10.6
1987	5.0	12.4
1988	9.6	21.9
1989	11.6	23.9

The approximate contribution of each of the surface runoff ARD sources, as previously identified, to the overall ARD production is summarised in Table 7.

TABLE 7
Source Contribution to Total Surface Runoff Metal Transport

SOURCE	COPPER (%)	ZINC (%)
Gruvebekken (natural)	8	5
Main Rock Pile	82	84
Minor rock piles (seepage directly to Dausjøen)	10	11

In 1988 the State Pollution Control Authority informed the mine owners to reduced the metal load to the Stallvikelva by 80%. In addition, the increasing surface runoff metal load to lake Dausjøen should be curbed and should not exceed levels observed following the initial years post closure of the mine. The owners were also requested to restore the terrain to its original condition.

5.3 Remediation

Alternatives

To avoid the necessity of long term active intervention and operation the option of collection and treatment was rejected.

Plugging and allowing the mine to fill was considered the most feasible option to reduce the production of ARD to the Stallvik river. The ability to flood the mine is limited by geometry, openings and subsidence cracking. The water in the mine, when flooded to the highest achievable level, will occupy approximately 70% of the mine volume. At this level almost all the rock surface area which exposes pyrite will be covered and it is expected that a substantial reduction in the ARD production will be observed.

The evaluation of sources contributing to the surface runoff to Dausjøen indicated that remediation measures should be concentrated on the rock piles. Two alternatives were evaluated.

- The first alternative required diversion of the surface runoff around the mine rock piles and placement of a geomembrane cover.

- The second required the placement of the mine rock underwater in lake Dausjøen.

Although the remediation cost was similar for both options, there were considerable practical difficulties anticipated with the installation and maintenance of a geomembrane cover on the steep rock pile slopes, and therefore the second alternative was considered to be the better long term solution. Due to the steep topography slope flattening of the rock pile would be impractical and with the extreme climatic conditions of this area, it was unlikely that a stable cover layer over the geomembrane could be maintained.

A concern for the risk of releasing large quantities of stored ARD products from the mine rock once placed in the lake was identified. In 1988 a program of mine rock pile sampling and material characterization was initiated. This involved the determination of the quantity of stored acid products in the aged mine rock and the evaluation of alternative lime stabilization methods. To estimate the lime requirements to control acidity, solution extraction tests were conducted on aged mine rock samples taken from various locations and depths in the rock pile. A large variability in the stored acid and metal content was observed. Lime requirements for neutralization, as indicated by individual samples, ranged from 25 to 2500 tonnes for the entire rock pile.

Due to the uncertainty of the lime requirements, a lime contribution capacity of 1 500 tonnes over the 3 month construction period was decided on. It was elected that lime addition would not be directly to the mine rock, but would be added to the lake water. This would avoid the risk of excessive liming and high pH values in the lake.

As part of the evaluation of methods of controlling stored oxidation products during the placement of the mine rock, a test location was isolated from the lake. Mine rock, from an area indicating a high level of stored products, was placed in this isolated area in the same ratio of the total mine rock to water volume in lake Dausjøen (100 m^3 of rock in 3500 m^3 of water). The pH in the isolated water mass decreased from 3.8 to approximately 2.4, while the copper concentration increased from 0.58 mg/L to approximately 4.5 mg/L and the zinc concentration from 1.3 to approximately 5.5 mg/L.

An additional concern with the in-lake placement was the potential placement problems that could result due to instability of the rock pile being placed over the very soft metal hydroxide sediments in the lake. Investigations were conducted to determine the lake bottom geotechnical conditions and to develop an appropriate placement method. Based on the geotechnical evaluation it was concluded that the mine rock placement would have to be done behind a stabilizing berm.

Implementation

The remediation work was concluded mainly in the summer of 1990. A total of 4 adits were plugged with concrete and an overflow through a ventilation shaft in the upper part of the mine was prepared. Additional surface water was diverted to the mine to accelerate the filling, and it is expected to overflow in approximately 3 to 5 years. An access shaft to the main chamber of the mine has been prepared to allow monitoring of the water level and quality.

A control strategy was developed to control release of oxidation products to the environment during mine rock relocation and to maintain the pH within the limits set by SFT ($5<pH<8$). The strategy included the following:

i) The lake water level was drawn down by 5 m by pumping to allow mine rock to be placed in the dry.

ii) It was decided to not add an alkali (lime or limestone) directly to the mine rock but to control the pH in the lake water during mine rock placement and subsequent flooding.

iii) Three lime addition facilities were provided as follows:

1. A lime addition facility discharging dilute milk of lime through a submerged high velocity jet (6 m/s at a flow rate of 20 L/s) directed from the bottom of the lake upwards, to achieve mixing while controlling the pH of the water in the lake. In practice this method proved to highly efficient in controlling the lake pH.

2. A second liming station was provided downstream of the lake to control the total runoff from the lake and overall mine area.

3. The third liming station was provided for the control of seepage from the mine rock pile and consisted of a single air stirred tank overflowing into a thickener. The sludge from the thickener was piped into lake Dausjøen at a depth of 10 m.

For the relocation of the mine rock, extensive construction of access roads to the mine rock pile and to the deposition site in Dausjøen, were required. The mine rock was excavated using of a backhoe and 6 to 8 dump trucks. Working two shifts per day, 3 months were required for the relocation of the mine rock. The mine rock was removed in lifts (Plate 15), starting at one end of the pile and gradually proceeding to the other side, working from the top down, to minimise the excavation area (see Figure 38), thus minimising the area exposed to rain. As foundation bedrock was exposed it was carefully and rapidly cleaned by hand (2 labourers) to remove residual mine rock. The resulting bedrock surface was remarkably clean (Plate 16).

A retaining berm was constructed along the planned perimeter of the mine rock deposition area in the Dausjøen to stabilize the toe of the final rock pile. This berm was constructed with 2 m diameter quarried rock boulders and explosives were used to consolidate and firmly imbed the rocks in the sediments. The relocated mine rock was placed in layers of approximately 2-3 m thick, and gradually expanded from the shore to the retention berm (Plate 17). The construction method is illustrated in Figure 39. After relocation, the mine rock was covered with 0.3 m of compacted till.

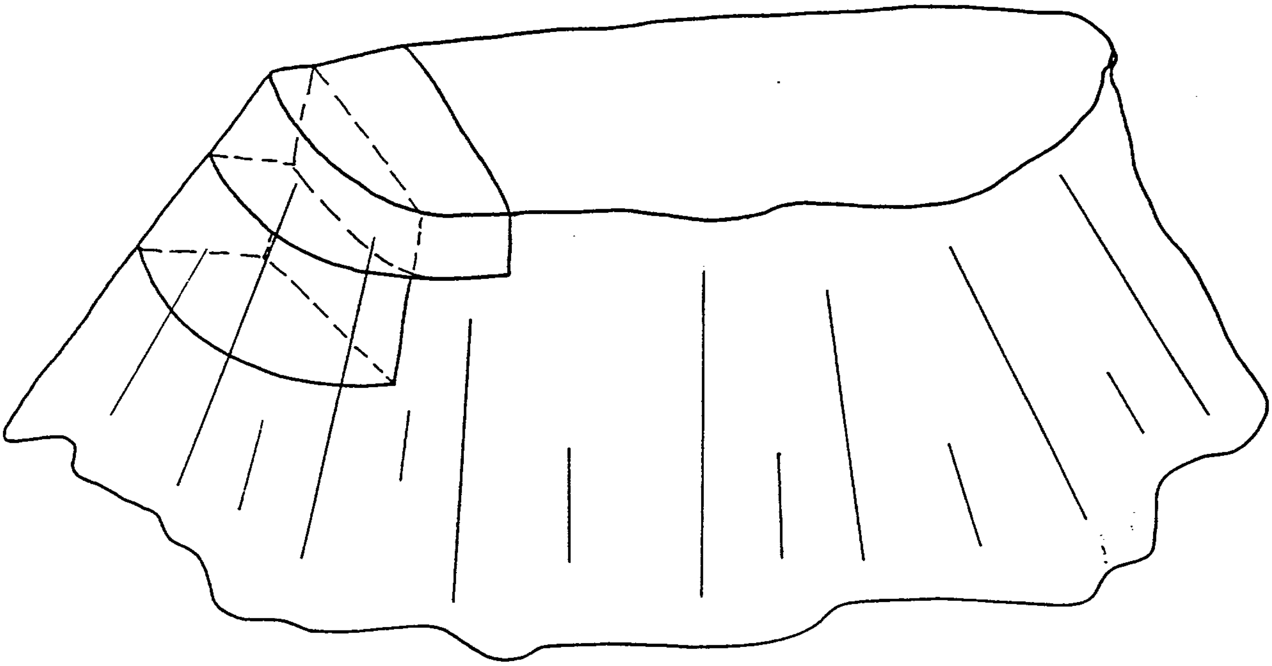


FIGURE 38 SKOROVAS– MINE ROCK PILE EXCAVATION

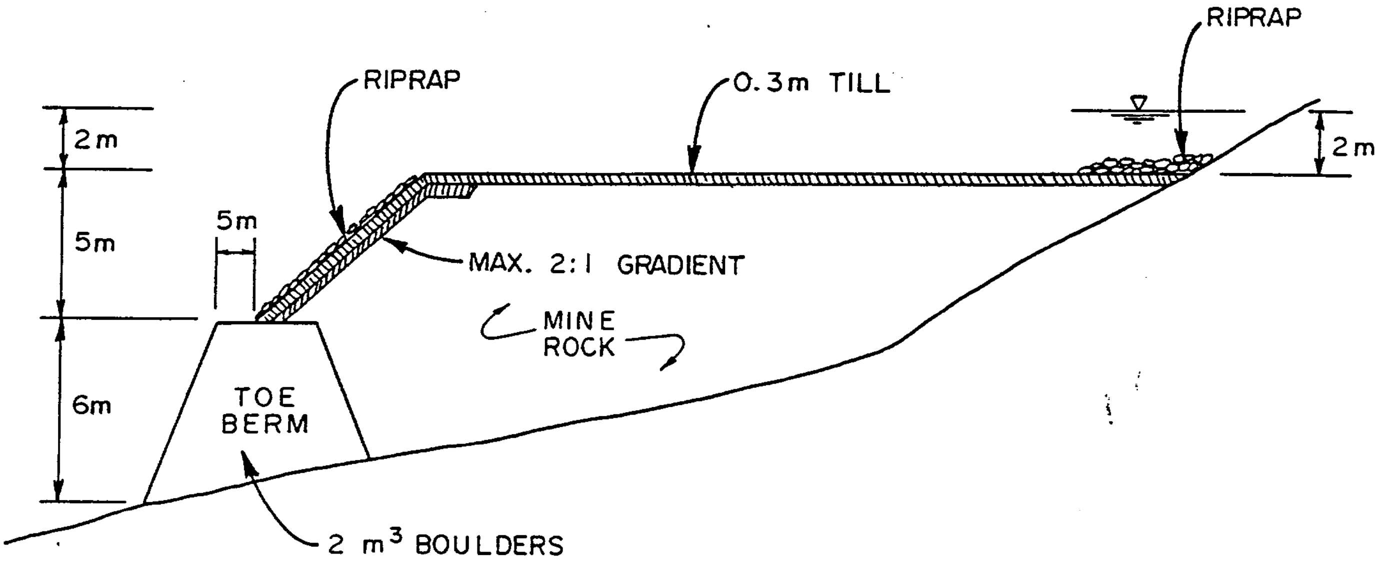


FIGURE 39 SKOROVAS- CROSS SECTION THROUGH CONSTRUCTED MINE ROCK PILE

Plate 15: Skorovas - Mine rock excavation by backhoe

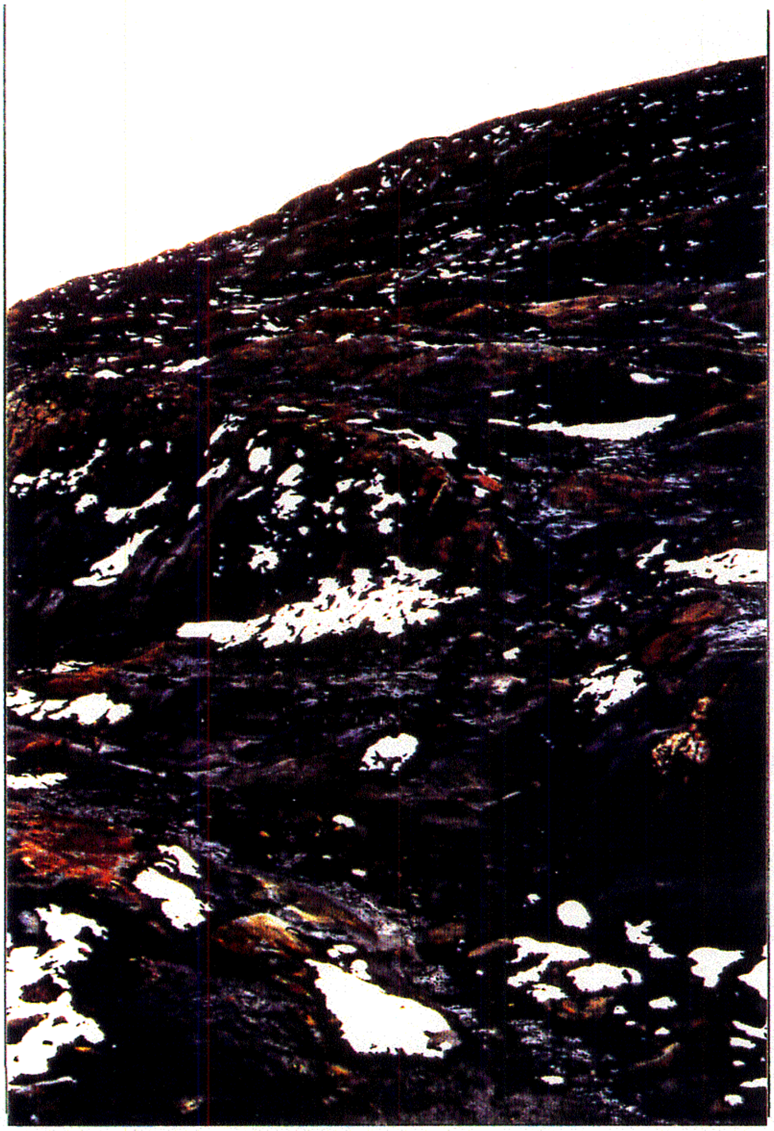


Plate 16: Skorovas - Bedrock after cleaning mine waste pile

Plate 17: Skorovas - Placement of mine rock in Dausjøen Lake

Plate 18: Skorovas - New dam at Dausjøen Lake outlet

The till layer on the front face of the rock pile could not be compacted and is not continuous over either the mine rock or the berm. Rip rap was placed on the face of the mine rock slope and at the shore edge to minimise erosion due to wave action. The relocated mine rock pile was instrumented with sampling points within the mine rock at two locations for future monitoring.

The liming system proved very effective for the duration of the construction period. On completion of the project, approximately 400 tonnes of lime had been consumed. At a cost of 1000 NOK per tonne (CAD $170 per tonne), the lime cost was only 400 000 NOK (CAD $68 000).

A new dam wall was constructed at the overflow of the lake (Plate 18), equipped with a V-notch for flow measurement, raising the lake level by 2 m above the original level.

Cost

The cost for the completion of the project is summarised in Table 8.

TABLE 8
Cost Breakdown for the Remediation at the Skorovas Site

ITEM	COST	
	NOK	CAD $
Planning; designs; call for tenders; pollution monitoring	3 500 000	595 000
Safety; liming systems; water displacement; dam construction	2 500 000	425 000
Construction; relocation of mine rock	15 000 000	2 550 000
Cleaning up; removal of buildings, roads and railroads	5 000 000	850 000
TOTAL	26 000 000	4 420 000

(Note : Costs for plugging excluded)

This equates to approximately CAD $8.90 per tonne (exclusive of removing buildings etc.), which compares well with the cost of consolidation and a geomembrane cover as installed at Kjøli mine.

5.4 Monitoring and Performance

During the construction phase the water quality standards targeted at the onset were met at all times. Early indications are that a reduction in the copper and zinc transport from the mine site has been achieved in 1991 over the preceding years, despite the fact that all liming has been discontinued. During the first half of 1991 the copper transport from surface runoff has been reduced by 80 % and the transport of zinc by 70%, compared to 1990. However, it should be noted that the water course has been changed significantly, and it is considered that the results are as yet atypical.

Following the construction phase, the pH in Dausjøen has slowly decreased and appears to have stabilized marginally above 4.0. The liming system at the overflow of Dausjøen has been discontinued since the completion of the remediation program. Water samples taken from various locations within the submerged rock pile indicate that oxygen has been successfully excluded from the rock pile. However, the pH of the water in the interstices of the submerged mine rock is at 2.4. Thus, there is a large storage of highly acidic and metals rich water in the submerged mine rock pile. Migration of the metals in this water to the lake water can be expected due to convective and diffusive mechanisms. Convection may occur quite readily as the till cover is not continuous over the mine rock face and perimeter berm. This migration will result in reduced pH and elevated metals concentrations in the Dausjøen until there is an approximate equilibration between the rock pile water and lake water quality. This period of depressed water quality could have been avoided had finely crushed limestone been added to the aged mine rock as it was excavated and relocated.

During the site visit it was noted that significant seepage flows of low pH water was seeping from the remaining mine rock piles located directly behind the remaining mine buildings. The pH of the seepage from this source was approximately 2.4 to 2.9, while the coloration suggested a high metal content. Similarly, highly acidic water (pH 2.4 to 2.9) was flowing from the sealed underground silo located behind the remaining mine buildings (pers. comm. Per Åge Beck).

A significant deposition of metal sludge was observed in the Dausjøen stream floodplain near the inflow end of lake Store Skorovatn (Plate 19). The sludge has settled to a firm compacted medium. Vegetation has established naturally in the sludge and appears to be thriving. The long term dissolution of this sludge is an issue which could be investigated to evaluate the long term stability of the sludge, and may provide guidelines for sludge disposal.

During inspection of the upper regions of Gruvebekken, from where the natural ARD is occurring, an extensive pyritic mineralised outcrop was noted (Plate 20). In addition, several small, man excavated, rock piles of a very high grade sulphide (likely pyrite) mineral were observed. Although the sulphide surfaces do not appear to be oxidising (shiny crystals), the general appearance of the massive sulphides is crumbly. This suggests that continued oxidation of the sulphides is occurring with complete removal of the oxidation products. It was concluded that the shiny appearance of the crystals are not indicative of no oxidation but rather that the oxidation is producing a pH condition in which all oxidation products are leached away. Staining of the sulphidic material is absent since no neutralising agents are available in the massive sulphide structure to produce the coloured precipitations usually associated with ARD.

5.5 Conclusions and Recommendations

The relocation of the mine rock has been achieved effectively and efficiently. A high standard was maintained during the clean up process. Very little residual mine rock was found on the cleaned up portions of the site. Results observed at the site reflect the careful planning and thorough execution of the project.

The team involved in the planning and completion of the project should be commended. The site drainage water quality monitoring results may yet demonstrate that the water quality objectives for the site remediation will be met.

However, the authors feel that the following activities would have resulted in significant improvements in the metal contaminant discharge rates from the site:

- The mine rock that is located near the remaining mine offices could have been removed along with the main mine rock pile. This would have required very little additional effort and the cost incurred would have been small.

- Limestone could have been blended in with the mine rock during relocation to neutralize the acidity in the mine rock and minimize the potential for soluble metal migration to the lake water.

- The seepage from the underground ore silo is significant. Remediation of this source could also have been considered.

Plate 19: Metal hydroxide sludge deposits in Dausjøen stream near Lake Store Skorovatn

Plate 20: Skorovas - Pyrite outcrop contributing to natural acidic drainage

The cost of placing the mine rock under water at Skorovas was found to be comparable with that of placing a geomembrane cover at Kjøli. However, the advantages of a water cover, when comparing the long term stability and efficiency of a water cover to the life expectancy of the synthetic membrane, outweighs that of a geomembrane cover.

This report 66006, ARD Remediation Measures at Four Norwegian Mines is respectfully submitted by:

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

A.C. Clarke
for
Dr. A. MacG Robertson., P.Eng.
Principal

J. T. Chapman, M.Sc(Eng.), P.Eng.
Chemical Engineer

AMR/JTC/R-2

Welcome Screen Search Report List

APPENDIX A

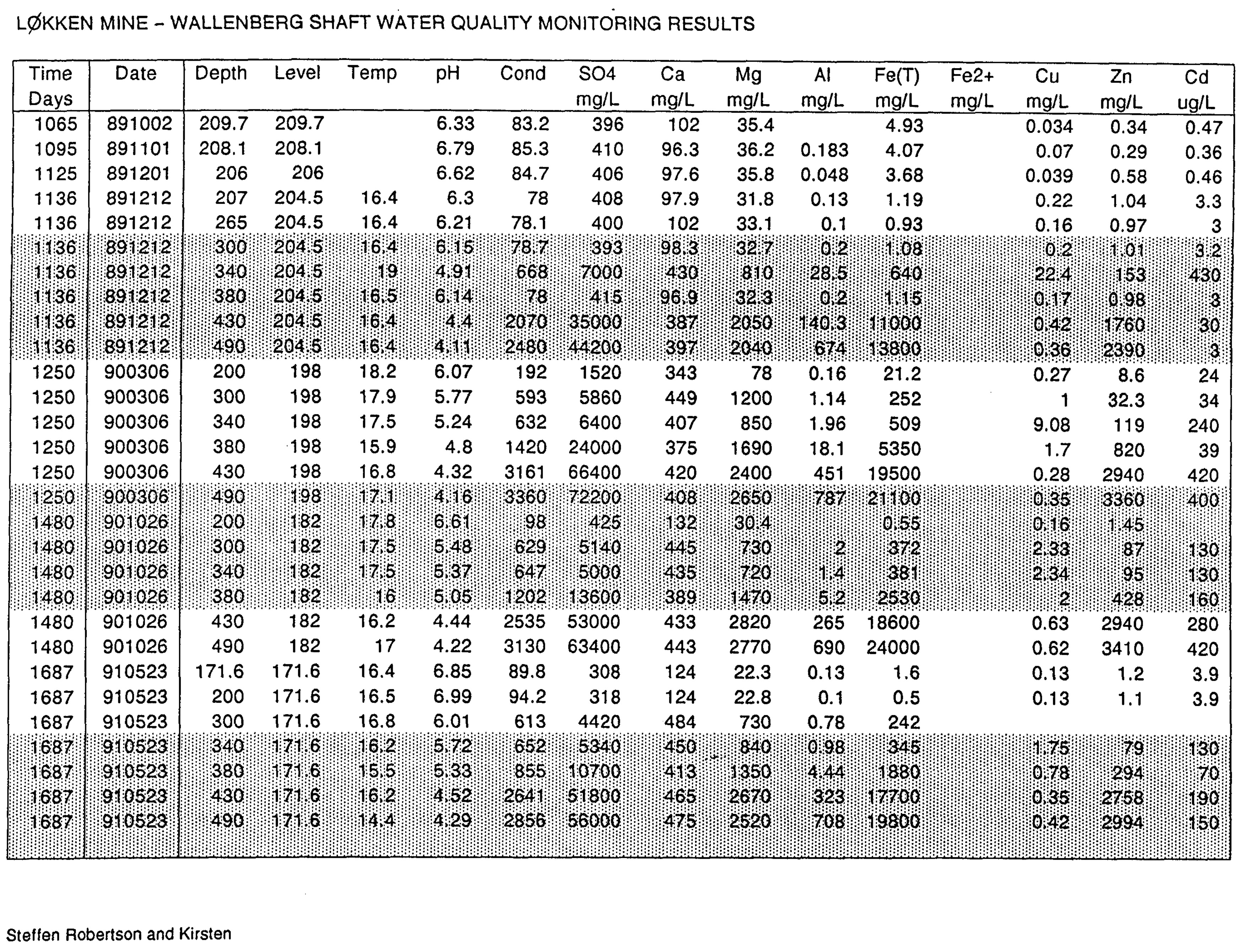
Løkken Mine - Wallenberg Shaft
Water Quality Monitoring Results

LØKKEN MINE – WALLENBERG SHAFT WATER QUALITY MONITORING RESULTS

Time Days	Date	Depth	Level	Temp	pH	Cond	SO4 mg/L	Ca mg/L	Mg mg/L	Al mg/L	Fe(T) mg/L	Fe2+ mg/L	Cu mg/L	Zn mg/L	Cd ug/L
0	860917	390	336	16.2	2.3	1790	23400	476	1610	426	5750	700	120	1240	1980
0	860917	490	336	16.2	3.96	3490	74200	600	3100	1362	25700	1000	0.7	4130	105
47	861104	333	331	16.2	5.35	69.1	340	88	17.5		1.63	1.15	0.31	1.54	4.4
47	861104	381	331	16.5	2.54	1376	15000	470	1035	323	2580	3000	157	504	1540
47	861104	440	331	14.5	3.32	3180	52500	520	2000	988	16900	18750	47.2	2900	7200
74	861201	328	328							0.44	2.14		0.37	1.68	5.1
109	870106	324	324		6.6	60.1	280	82.3			1.32		0.13	4.54	11.2
165	870302	321	321		5.88	62.6	311	76.6	19		0.02		0.03	3.71	4.1
195	870402	320	320		5.67	63.5	301	81.2	20.2		0.7		0.17	0.94	2.4
227	870504	314	314		5.7	61.6	320	78.8	17.7	1.17	5.64		1.37	4.03	20
254	870601	310	310		5.94	63.9	305	81	18.8	0.005	0.36		0.29	2.27	10
270	870617	311	310	16	5.89	64.3	318	84.1	19.4	0.12	0.86	0.23	0.64	2.87	10
270	870617	320	310	16	5.77	65.7	304	85.3	19.4	0.13	0.98	0.23	0.49	2.6	10
291	870708	328	308	16	5.1	70.6	336	85.7	20.9	0.12	1.07	0.19	0.28	1.71	4.1
291	870708	358	308	17	3.78	915	8850	439	855	144	1280	1400	36.5	303	880
291	870708	390	308	16.2	3.72	2800	45200	507	2580	570	14000	16600	18.8	2370	3050
291	870708	440	308	17	3.41	3040	52100	527	2750	1035	20000	17000	30.1	2650	6290
291	870708	490	308	15	3.92	1640	71700	519	2920	1165	26200	26000	1.49	3730	45
317	870804	307	307		5.09	68	356	110	21.7	0.35	8.2		0.31	1.32	4.2
344	870901	306.5	306.3		6.03	69.3	328	83	22	0.11	0.38		0.16	1.19	4.3
378	871005	302	302		6.12	72.9	350	94.7	22.4	0.14	0.95		0.22	1.27	3.4
412	871109	300.5	300.5		4.07	373	2640	327	305	16.5	157		19	48.1	160
414	871111	302	299.8	17.3	5.45	478	3080	416	501	3.27	121	0.5	15.2	41.2	230
414	871111	320	299.8	16.3	4.99	602	4920	415	640	3.19	315	<0.01	9.1	95	230
414	871111	360	299.8	16.7	4.19	830	8000	420	820	48.7	950	0.65	25.9	287	760
435	871202	294	294		5.4	598	4800	485	780	2.72	241		16.8	65	150
468	880105	294	294		7.1	74.7	290	94.8	29	0.4	1.34		0.28	1.22	3.9
495	880202	294	294		6.71	79.7	344	103	38.4	0.47	2.74		0.22	1.18	3.7
530	880307	291.3	291.8		6.52	82.7	400	98	36.4	0.25	4.71		0.17	0.84	3.2
559	880406	291.6	291.6		6.76	87.9	414	103	40.2	0.26	5.94		0.17	0.63	1.9

LØKKEN MINE – WALLENBERG SHAFT WATER QUALITY MONITORING RESULTS

Time Days	Date	Depth	Level	Temp	pH	Cond	SO4 mg/L	Ca mg/L	Mg mg/L	Al mg/L	Fe(T) mg/L	Fe2+ mg/L	Cu mg/L	Zn mg/L	Cd ug/L
593	880510	288.6	286.6	17	6.57	69.9	302	86.9	26.2	0.46	3.4	0.84	0.21	0.73	2.9
593	880510	292	286.6	17.2	6.59	69.9	290	86	26	0.42	3.34	0.85	0.22	0.74	2.9
593	880510	300	286.6	18.9	5.95	681	4620	497	870	0.62	114	110	0.62	35.1	50
593	880510	315	286.6	17.6	5.36	681	5000	494	860	0.31	148	145	0.31	28.8	42
619	880606	278.8	278.8		6.68	69.5	300	88.5	27.8	0.04	3.88		0.28	0.94	3.4
647	880704	278.8	278.8		6.53	74.5	308	87.2	33		6.29		0.23	0.78	2.2
675	880802	277	277		6.59	78.3	340	92.4	32.2		6.46		0.24	0.93	2.9
708	880905	273.9	273.9		6.51	79	365	100	35.5	0.33	7.1		0.17	0.63	6
732	880929	273	271	17	6.12	61.5	400	100	34.9	0.19	1.57		0.56	2.41	6
732	880929	285	271	17	6.19	62	388	102	35.5	0.2	1.67		0.55	2.46	6
732	880929	300	271	17.2	6.08	80	494	105	57	0.24	8.35		0.78	3.79	8
732	880929	340	271	17.7	5.7	455	5325	485	870	0.29	240		3.5	91	160
732	880929	380	271	16.9	4.93	490	6560	437	870	4.19	608		5.98	188	380
732	880929	430	271	15.3	4.39	1140	20600	392	1700	122	6650		0.83	1240	160
764	881101	265.8	265.8		6.92	83.8	384	137	33.7	0.1	1.75		0.14	2.05	3.5
798	881205	259.3	259.3		6.76	81.7	378	109	30.8	0.15	0.55		0.22	1.99	5
825	890102	255.6	255.6		6.32	75.9	366	98.3	28.1		4.73		0.73	2.1	6.9
855	890201	245	245		5.5	67	342	87	20		14.1		2.69	5.9	20
892	890308	238.4	236.4	15.4	6.09	71	336	91.3	24	0.1	1.15		0.16	2.22	6
892	890308	300	236.4	17.9	5.59	696	5550	469	810	9.64	456		5.79	112	230
892	890308	340	236.4	17.5	5.49	736	6150	478	880	0.71	437		6.62	152	290
892	890308	380	236.4	17.5	5.39	743	7000	473	900	2.01	473		15.7	184	430
892	890308	430	236.4	16	4.03	2803	47600	454	2400	681	15900		0.21	2540	110
892	890308	480	236.4	16.2	4.2	3527	66000	461	2600	800	21800		0.38	3400	220
917	890403	232.4	232.4		6.14	74	308	96	23		5.09		1.22	3.57	10.9
946	890502	223.8	223.8		6.38	64.3	248	85.2	21.7	0.07	1.52		0.17	1.81	4.9
955	890511	430	221.6	15.9	4.45	855	7300	365	880	49.3	1070		23.2	323	580
976	890602	218.4	218.4		6.31	66.6	272	81	24	0.09	4.52		0.13	1.55	3.8
1009	890705	430	215.8	15.9	4.3	2470	34400	426	2070	214	12500		0.41	1960	12.1
1035	890901	211.7	211.7		6.56	81.6	376	93.8	32		2.13		0.05	0.54	0.85



LØKKEN MINE – WALLENBERG SHAFT WATER QUALITY MONITORING RESULTS

Time Days	Date	Depth	Level	Temp	pH	Cond	SO4 mg/L	Ca mg/L	Mg mg/L	Al mg/L	Fe(T) mg/L	Fe2+ mg/L	Cu mg/L	Zn mg/L	Cd ug/L
1065	891002	209.7	209.7		6.33	83.2	396	102	35.4		4.93		0.034	0.34	0.47
1095	891101	208.1	208.1		6.79	85.3	410	96.3	36.2	0.183	4.07		0.07	0.29	0.36
1125	891201	206	206		6.62	84.7	406	97.6	35.8	0.048	3.68		0.039	0.58	0.46
1136	891212	207	204.5	16.4	6.3	78	408	97.9	31.8	0.13	1.19		0.22	1.04	3.3
1136	891212	265	204.5	16.4	6.21	78.1	400	102	33.1	0.1	0.93		0.16	0.97	3
1136	891212	300	204.5	16.4	6.15	78.7	393	98.3	32.7	0.2	1.08		0.2	1.01	3.2
1136	891212	340	204.5	19	4.91	668	7000	430	810	28.5	640		22.4	153	430
1136	891212	380	204.5	16.5	6.14	78	415	96.9	32.3	0.2	1.15		0.17	0.98	3
1136	891212	430	204.5	16.4	4.4	2070	35000	387	2050	140.3	11000		0.42	1760	30
1136	891212	490	204.5	16.4	4.11	2480	44200	397	2040	674	13800		0.36	2390	3
1250	900306	200	198	18.2	6.07	192	1520	343	78	0.16	21.2		0.27	8.6	24
1250	900306	300	198	17.9	5.77	593	5860	449	1200	1.14	252		1	32.3	34
1250	900306	340	198	17.5	5.24	632	6400	407	850	1.96	509		9.08	119	240
1250	900306	380	198	15.9	4.8	1420	24000	375	1690	18.1	5350		1.7	820	39
1250	900306	430	198	16.8	4.32	3161	66400	420	2400	451	19500		0.28	2940	420
1250	900306	490	198	17.1	4.16	3360	72200	408	2650	787	21100		0.35	3360	400
1480	901026	200	182	17.8	6.61	98	425	132	30.4		0.55		0.16	1.45	
1480	901026	300	182	17.5	5.48	629	5140	445	730	2	372		2.33	87	130
1480	901026	340	182	17.5	5.37	647	5000	435	720	1.4	381		2.34	95	130
1480	901026	380	182	16	5.05	1202	13600	389	1470	5.2	2530		2	428	160
1480	901026	430	182	16.2	4.44	2535	53000	433	2820	265	18600		0.63	2940	280
1480	901026	490	182	17	4.22	3130	63400	443	2770	690	24000		0.62	3410	420
1687	910523	171.6	171.6	16.4	6.85	89.8	308	124	22.3	0.13	1.6		0.13	1.2	3.9
1687	910523	200	171.6	16.5	6.99	94.2	318	124	22.8	0.1	0.5		0.13	1.1	3.9
1687	910523	300	171.6	16.8	6.01	613	4420	484	730	0.78	242				
1687	910523	340	171.6	16.2	5.72	652	5340	450	840	0.98	345		1.75	79	130
1687	910523	380	171.6	15.5	5.33	855	10700	413	1350	4.44	1880		0.78	294	70
1687	910523	430	171.6	16.2	4.52	2641	51800	465	2670	323	17700		0.35	2758	190
1687	910523	490	171.6	14.4	4.29	2856	56000	475	2520	708	19800		0.42	2994	150

Steffen Robertson and Kirsten

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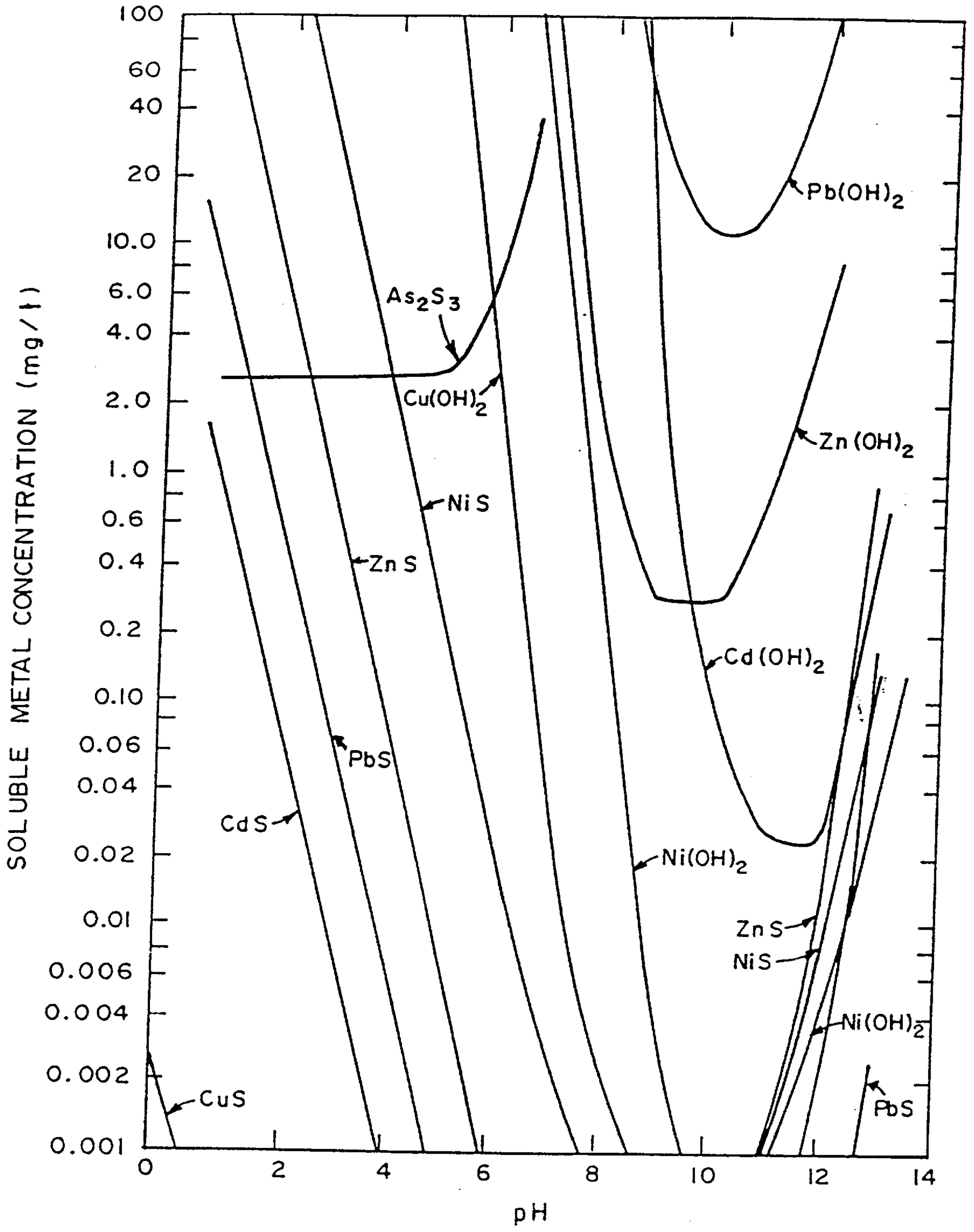

FIGURE A-1

Calculated Metal Sulphide and Hydroxide Solubilities
(Bhattacharrya Et Al 1981)

STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers

Welcome Screen Search Report List

APPENDIX B

Kjøli Mine - Mine Rock Pile
Oxygen and Temperature Monitoring Results

Welcome Screen Search Report List

KJØLI MINE – ROCK PILE MONITORING RESULTS

DATE		14 Oct 88	16 Mar 89	7 Jul 89	22 Aug 89	3 Apr 90	2 May 90	15 Aug 90	19 Sep 90	8 Feb 91	13 Jul 91
TIME (months)		0	5	9	10	17	18	21	22	27	32
Borehole #	Depth (m)										
		TEMPERATURE (C)									
DH1	0.50	1.5	-1.1	9.8	8.8	-0.3	0.1	8.4	8.7	0.2	7.2
	0.96	4.3	0.9	9.5	9.5	1.5	1.8	8.6	10.0	2.5	6.1
	1.44	6.6	3.0	9.5	10.0	3.2	3.4	8.7	10.8	4.6	5.6
	2.44	9.6	6.6	9.3	10.4	6.1	6.1	8.7	11.3	7.7	6.1
	4.48	9.9	8.8	8.7	9.1	7.5	7.5	7.6	9.9	8.4	6.7
	6.50	6.6	6.5	5.0	6.0	5.2	5.2	5.2	7.3	6.1	5.3
	9.50	4.8	5.0	4.4	4.6	4.4	4.6	4.5	6.0	5.0	5.1
		OXYGEN (%)									
	0.52	20.0	19.8	19.7	15.0	5.4	5.6	4.3	5.8	10.3	7.7
	1.02	20.0	21.0	19.0	15.0	4.5	5.6	4.0	5.4	10.3	7.5
	1.53	19.9	20.0	19.7	15.0	5.5	6.8	5.7	7.5	10.5	7.5
	3.00	20.4	20.8	20.7	15.1			6.0	7.5	10.3	7.5
	3.99	19.8	19.9	19.6	15.1	5.7	6.6	5.5	7.7	10.2	7.5
	5.04	20.0	19.8	19.8	15.1	5.4	6.6	5.4	7.5	10.1	7.5
	6.04	20.0	19.9	19.8	15.9			5.5	7.5	10.2	7.5

KJØLI MINE – ROCK PILE MONITORING RESULTS

DATE		14 Oct 88	16 Mar 89	7 Jul 89	22 Aug 89	3 Apr 90	2 May 90	15 Aug 90	19 Sep 90	8 Feb 91	13 Jul 91
TIME (months)		0	5	9	10	17	18	21	22	27	32
Borehole #	Depth (m)										
DH3		TEMPERATURE (C)									
	0.45	1.0	0.1	12.2	8.5	1.3	1.4	8.8	8.2	2.8	9.0
	0.95	4.0	1.8	12.0	9.5	2.4	2.7	9.3	9.6	4.2	8.6
	1.45	7.1	3.6	12.1	10.8	3.9	4.3	10.0	11.1	6.0	8.5
	2.45	12.4	7.6	12.5	13.3	7.4	7.7	11.1	12.9	9.5	9.5
	3.45	17.1	12.9	14.2	14.9	11.6	11.8	12.1	13.7	12.8	12.2
	6.45	15.8	14.5	13.9	13.7	11.8	12.2	11.2	12.4	11.8	12.1
	11.75	7.6	8.2	8.6	7.8	5.6	6.1	5.2	6.0	4.6	5.8
		OXYGEN (%)									
	0.45	6.2	7.1	2.0	0.4	0.3	6.0	0.6	2.0		0.5
	0.95	10.5	12.5	2.7	0.2	0.0	7.0	0.5	1.4	2.6	7.0
	1.95	15.5	16.2	9.7	1.3	1.5	6.6	2.0	3.7	8.0	5.5
	2.95	17.5	16.8	15.5	8.4	2.5	6.3	1.3	3.0	8.0	5.5
	3.45	20.0	19.5	19.0	14.5	6.0	7.7	3.1	5.0	10.5	6.0
	4.95	20.5	20.0	19.2	15.5						
	5.95	20.0	20.2	18.9	13.8	5.5	6.7	3.5	6.0	9.0	5.0
	6.95	20.9	20.2	19.8	17.5	6.0	9.0	4.2	6.8	10.4	5.2

KJØLI MINE – ROCK PILE MONITORING RESULTS

DATE		14 Oct 88	16 Mar 89	7 Jul 89	22 Aug 89	3 Apr 90	2 May 90	15 Aug 90	19 Sep 90	8 Feb 91	13 Jul 91
TIME (months)		0	5	9	10	17	18	21	22	27	32
Borehole #	Depth (m)										
DH4		TEMPERATURE (C)									
	0.50	1.1	0.4	11.2	9.8	1.8	2.1	9.0	8.0	0.4	8.5
	1.02	4.6	2.6	10.6	10.6	4.0	4.6	9.5	9.5	2.5	7.8
	1.48	7.8	4.9	10.9	11.7	6.5	7.2	10.2	10.6	4.7	7.6
	2.43	11.9	8.4	10.9	11.7	10.6	11.4	11.1	12.0	7.9	8.4
	4.48	15.5	19.8	11.6	13.1	14.7	15.6	12.1	12.6	10.3	10.8
	6.47	15.7	12.7	12.5	14.4	15.5	16.4	10.9	11.5	10.4	11.0
	11.70	10.7	10.4	11.0	11.6	9.6	10.4	5.0	6.1	5.4	5.3
		OXYGEN (%)									
	0.50	19.9	20.1	19.2	19.2	6.0	2.0	0.4	1.5		0.7
	1.02	19.5	19.7	19.7	18.8	4.0	2.0	0.5	1.0	5.0	0.8
	2.03	19.8	20.2	20.3	19.0	4.8	2.5	0.4	0.4	4.4	0.6
	3.02	20.2	20.0	20.3	19.6	5.3	2.8	0.4	0.5	4.5	0.5
	4.02	20.4	20.1	20.5	19.8	5.8	2.8	0.5	0.8		0.7
	5.01	20.5	19.2	19.8	19.5	6.0	2.9	0.2	0.8		0.9
	6.01	19.7	–	–	20.2	4.1	2.0	0.3	0.5	4.5	0.3