

ALI 90 11384



ANSTO/E692

Ansto

**Acid Drainage from Waste Rock Dumps at Mine Sites
(Australia and Scandinavia)**

by

J. R. Harries

with Appendices by

**L. Å. Lindahl, Environmental Protection Board, Sweden,
and U. Qvarfort, Uppsala University, Sweden**

May 1990

ISBN 0 642 59903 3

ISSN 1030-7745



AUSTRALIAN NUCLEAR SCIENCE AND TECHNOLOGY ORGANISATION

A Review of Acid Drainage from Waste Rock Dumps at Mine Sites
(Australia and Scandinavia)

by

J. R. Harries

with Appendices by

L. Å. Lindahl, National Environmental Protection Board, Sweden,
and U. Qvarfort, Uppsala University, Sweden

ABSTRACT

This report reviews the literature from Australia and Scandinavia on acid drainage from pyritic waste rock dumps with an emphasis on measurements and theory of processes that control the rate of oxidation and the release of pollutants. Conditions within waste rock dumps have been measured at several mine sites and a range of rehabilitation treatments have been tried to reduce the release of pollutants. A number of models have been proposed to calculate air flow, water transport and geochemistry. The data and experience at the mine sites are compared with predictions of the models.

ISBN 0 642 59903 3

ISSN 1030-7745

The following descriptors have been selected from the INIS Thesaurus to describe the subject matter of this report for information retrieval purposes. For further details please refer to IAEA-INIS-12 (INIS: Manual for Indexing) and IAEA-INIS-13 (INIS: Thesaurus) published in Vienna by the International Atomic Agency.

AUSTRALIA; DRAINAGE; FLOW MODELS; HYDROGEOLOGY; LEACHING;
MINES; OXIDATION; PYRITE; REMEDIAL ACTION; SITE SURVEYS; SWEDEN;
TAILINGS; WATER POLLUTION.

EDITORIAL NOTE

The Australian Nuclear Science and Technology Organisation replaced the Australian Atomic Energy Commission on 27 April 1987. Reports issued after April 1987 have the prefix ANSTO with no change of the symbol (E, M, S or C) or numbering sequence.

Table of Contents

1. INTRODUCTION	1
2. WATER FLOW - HYDROGEOLOGY	1
2.1 THEORETICAL TREATMENTS OF WATER FLOW IN DUMPS	2
2.1.1 Rasmuson and Collin	2
2.1.2 ANSTO	6
2.2 CASE STUDIES - WATER FLOW	6
2.2.1 Woodlawn, NSW, Australia	6
2.2.2 Bersbo, Sweden	9
2.2.3 Captains Flat, NSW	10
2.2.4 Rum Jungle, NT	10
3. GAS TRANSPORT - OXYGEN	11
3.1 THEORETICAL TREATMENTS OF OXYGEN SUPPLY PROCESSES	11
3.1.1 ANSTO	11
3.1.2 Rasmuson and Collin	12
3.2 CASE STUDIES - OXYGEN SUPPLY	18
3.2.1 Woodlawn	18
3.2.2 Rum Jungle	18
4. ACID DRAINAGE	19
4.1 THEORETICAL TREATMENTS OF ACID GENERATION	19
4.2 CASE STUDIES - ACID DRAINAGE	22
5. DUMP CONSTRUCTION	22
5.1 THEORETICAL BASIS FOR ALTERNATE DUMP CONSTRUCTIONS ..	22
5.2 CASE STUDIES - DUMP CONSTRUCTION	23
5.2.1 Cessnock region, NSW	23
5.2.2 Woodlawn, NSW	24
6. RECOMMENDATIONS	24
6.1 FUTURE STUDIES	24
6.2 DRAFT CRITERIA FOR MANAGEMENT OF REACTIVE WASTE ROCK	25
7. REFERENCES	25
APPENDIX A. Description of Australian mine sites where waste rock is a source of acid drainage.	A1
APPENDIX B. Acid generation in waste rock at mine sites - the situation in Sweden, by Lars Åke Lindahl	B1
APPENDIX C. Acid Mine Drainage from mine sites in Bersbo and Falun, Sweden. by Ulf Qvarfort.	C1

1. INTRODUCTION

Acid drainage from waste rock at mine sites is a major problem that greatly complicates, and adds to the expense of, site closure. There are strong economic and environmental arguments for improving the accuracy of predicting if and when a particular waste rock dump will go acid. This requires an understanding of the geochemistry and the hydrology of dumps and the air flow and heat transport within dumps. Such an understanding would provide a basis for designing strategies that would prevent the oxidation of pyrite and the release of pollutants.

The release of acid drainage requires that the sulphides in the dump be oxidised and that the oxidation products be transported out of the dump by water flow.

The oxidation of pyrite occurs by a complex set of reactions, some of which are catalysed by bacteria. The following characteristics are important for understanding the rate controlling mechanisms in a waste rock dump.

1. A supply of both oxygen and water is required for oxidation to occur. In many dumps the oxidation rate is thought to be limited by the oxygen supply. There is nearly always enough water available.
2. Oxidation of pyrite produces sulphuric acid which can release metals from other sulphides.
3. Significant amounts of heat are released in oxidation.
4. The bacteria that catalyse the reaction require acid condition, $\text{pH} < 4$. This means that oxidation is slow until acid conditions are established. Acid consuming materials in dumps can prevent the buildup of acidity.

Water flow through the dump is the main pathway transporting acid and oxidation products from oxidation sites to the surrounding environment. Hence control of water flow through the dump is an important method of reducing the release of pollutants.

This report reviews the literature from Australia and Scandinavia on acid drainage from waste rock dumps and describes sites where data has been collected or different techniques been tried. A similar report is being written to discuss work from North America.

2. WATER FLOW - HYDROGEOLOGY

The release of polluted water is the obvious sign of acid drainage and it follows that a good understanding of the hydrogeology of dumps is essential if the problem of acid drainage is to be solved. The usual approach is to attempt to reduce the flow of water sufficiently to reduce pollution to acceptable levels. However there are numerous variations including:

1. covering the surface by a low permeability layer,
2. liming the surface so that percolating water is alkaline,
3. preventing infiltrating water making contact with reactive material,
4. making the dump saturated so that oxygen transport is severely restricted, and
5. growing vegetation on a dump to enhance evaporation and thereby reduce the amount of infiltrating water.

2.1 THEORETICAL TREATMENTS OF WATER FLOW IN DUMPS

2.1.1 Rasmuson and Collin

Rasmuson and Collin [1988] have modelled water and oxygen transport through covers consisting of, from below, a coarse grained capillary break layer, a fine grained layer, another coarse grained capillary break layer and a moraine layer on top, Figure 1. The lower capillary break layer would restrict loss of capillary water downwards from the fine grained material during rainfall and wet conditions and the upper capillary break layer would restrict drying of the fine grained layer by evaporation during dry summer conditions. The aim of the study was to see if the moisture content of the fine grained material would stay high enough to be an effective barrier to limit oxygen transport into the waste.

The model for evaluating cover effectiveness was divided into three parts, water transport, gas diffusivity and oxygen transport/reaction rate. Oxygen transport depended on air-filled porosity which in turn depends on the water content. Hence results from the water flow calculation became input to the gas transport calculations. The water flow calculations are discussed in this section; the other calculations which related to gas transport are discussed in section 3.1.2.

In an earlier report [Collin and Rasmuson 1986], the thicknesses and composition of the layers were given as, from below, a 0.3 m sand layer, a 0.2 m clay layer, another 0.3 m sand layer and a 1 m moraine layer on top. In the second paper they considered the use of fine sand in the fine grained layer.

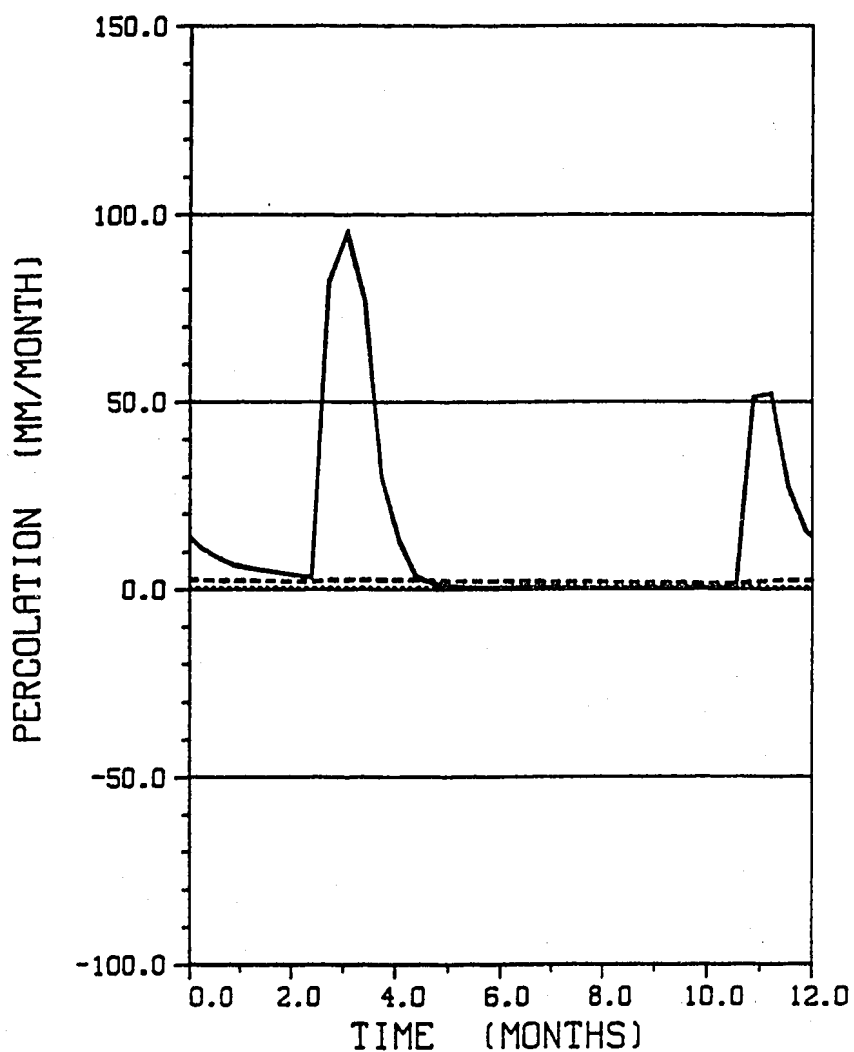
Water flow equations for saturated and unsaturated flow were solved using the TRUST computer program developed at Lawrence Berkeley Laboratories by T N Narasimhan. The TRUST results were verified by carrying out standard calculations and by comparing the results from TRUST with those from a one-dimensional finite element code GWH.

A basic requirement of any unsaturated water flow calculations is an estimate of the hydraulic conductivity of the various cover materials as a function of water content. Collin and Rasmuson [1986] used a method was proposed by Green and Corey [1971] in which the unsaturated hydraulic conductivity is derived from the more readily measured moisture characteristics, i.e. water potential versus water content. The method is based on the use of Poiseuille's law to determine the flow through particular size pores and a relationship between pore-size distribution and water potential.

Water flow through the cover layers were calculated for both one dimensional and a sloping moraine layer, using a slope of 1:20. Evaporation from the surface of the moraine during the summer was included in the calculation.

The results showed that the material in the fine grained layer remained almost saturated at all times, even during summer evaporation periods. Infiltrating water was partly accumulated in the moraine layer and percolation into the underlying coarse material was smooth compared to variations in precipitation rates [Collin and Rasmuson 1986].

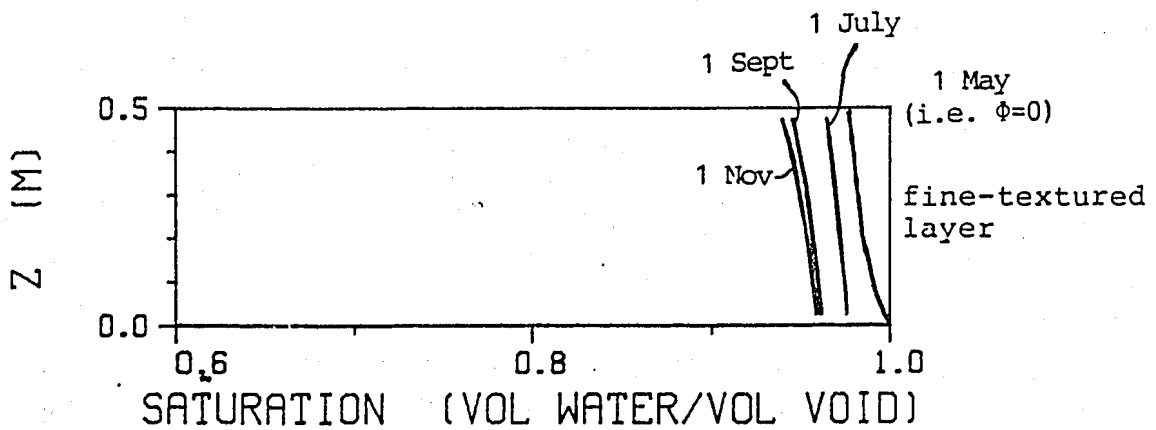
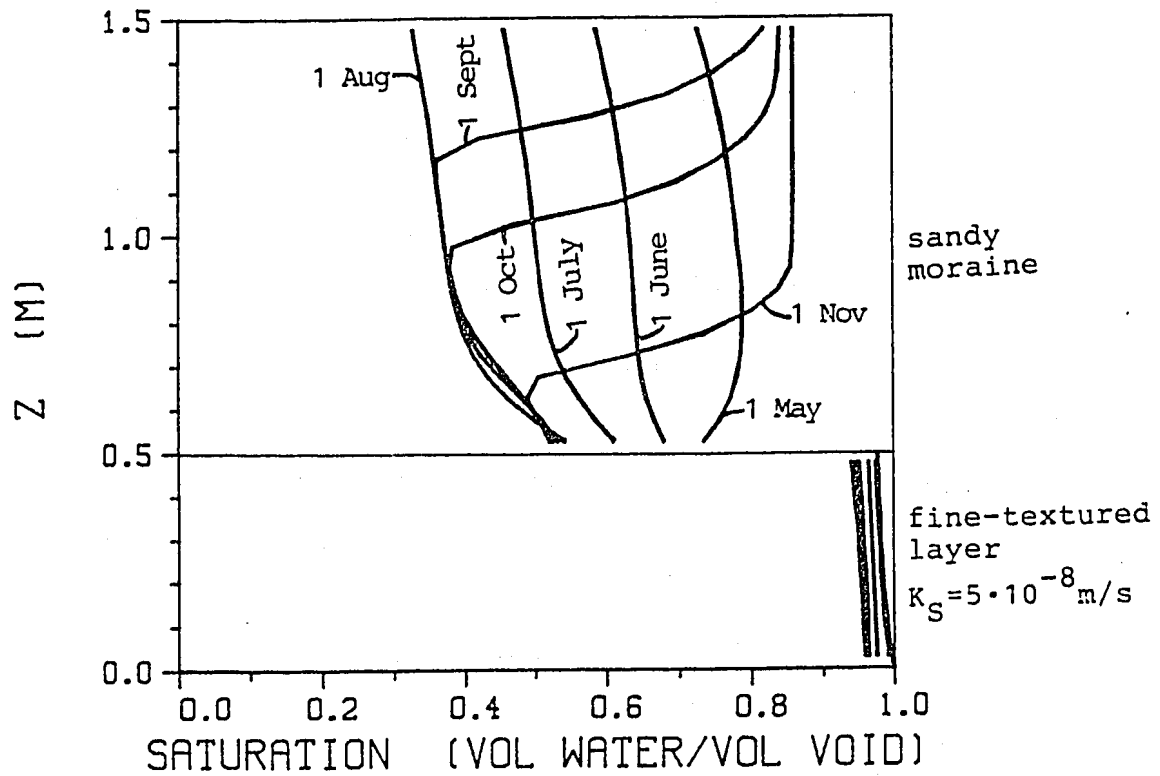
The water percolation was substantially reduced if the fine grained material had a hydraulic conductivity considerably below $5 \times 10^{-8} \text{ m s}^{-1}$, Figure 2. A low hydraulic conductivity meant that there could be considerable lateral flow in the coarse material overlying the fine layer. For materials with hydraulic conductivities less than $1 \times 10^{-9} \text{ m s}^{-1}$ it was found that the presence of capillary barriers did not significantly affect the moisture content in the fine grained material since the low conductivity itself prevented the water flow. Nevertheless a coarse grained layer above the fine grained layer might still be needed to act as a drainage layer and prevent ponding. The variation of the moisture content in the fine grained layer in the absence of a coarse grained layer is shown in Figure 3 for silt and in Figure 4 for clay.



Percolation through the fine-grained layer in a soil cover described in Figure 2. The water supply is equal to the percolation through 1 m sandy moraine during a normal year.

- $K_s = 5 \cdot 10^{-8}$ m/s for the fine-grained layer
- $K_s = 1 \cdot 10^{-9}$ m/s for the fine-grained layer
- $K_s = 1 \cdot 10^{-10}$ m/s for the fine-grained layer

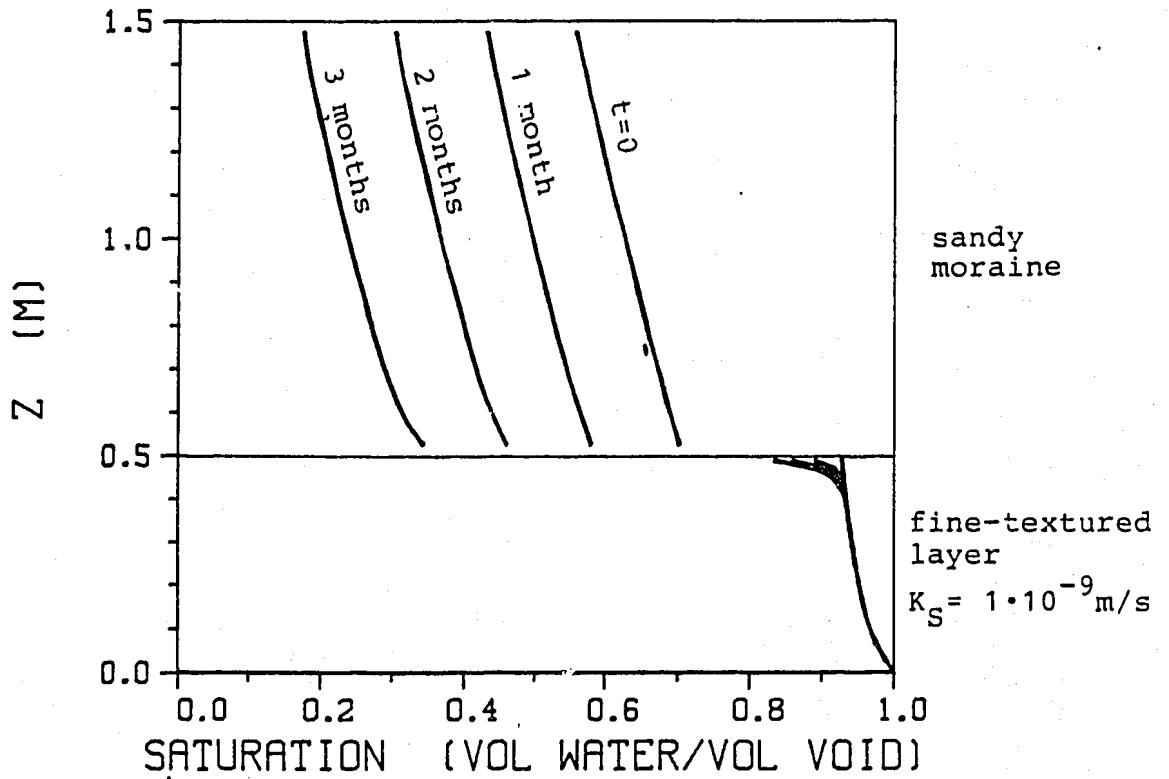
Figure 1. Source: Rasmuson and Collin [1988], Fig 9.



Drying of the fine-grained layer caused by capillary water flow upwards during the dry period. The fine-grained layer is represented by the silt ($K_S = 5 \cdot 10^{-8} \text{ m/s}$).

Figure 2.

Source: Rasmuson and Collin [1988], Fig 13.



Drying of the fine-grained layer caused by capillary water flow upwards to the moraine during drying of the moraine layer. The fine-grained layer is represented by the clay ($K_S=1 \cdot 10^{-9}$ m/s).

Figure 3. Source: Rasmuson and Collin [1988], Fig 14.

Rasmuson and Collin [1988] commented that plant roots could be very detrimental if they penetrated to the fine grained material and removed water.

2.1.2 ANSTO

Pantelis [1987] developed a saturated/unsaturated water flow model for a waste rock dump. The model was two-dimensional and assumed that the dump was constructed on gently sloping land with an underlying shallow unconfined aquifer. The model was used to investigate the effect of applying impermeable covers to the dumps at Rum Jungle.

The dump was taken to be 10 m high and 300 m long lying on an aquifer of uniform thickness 1 m and length 400m. The seasonal rainfall was introduced by assuming that all of the annual rainfall fell in the first half of the year and none in the second. Conservative contaminants were assumed to be produced uniformly through the heap prior to installation of the impermeable covers. The model was started by assuming there were zero contaminants and zero water contents at year 0. Graphical output showing specific discharge and contaminant profiles were produced for each quarter. Figure 5 shows the results at 19.24 years, halfway through the wet season.

At the beginning of the twentyfirst year both the infiltration of rainfall and the production of contaminants in the dumps were assumed to fall to zero, modelling the placement of impermeable covers. Recharge of the shallow aquifer continued to occur upstream of the dump. The model showed that there was still a significant unsaturated flow through the dump which slowly washed contaminants from the dump.

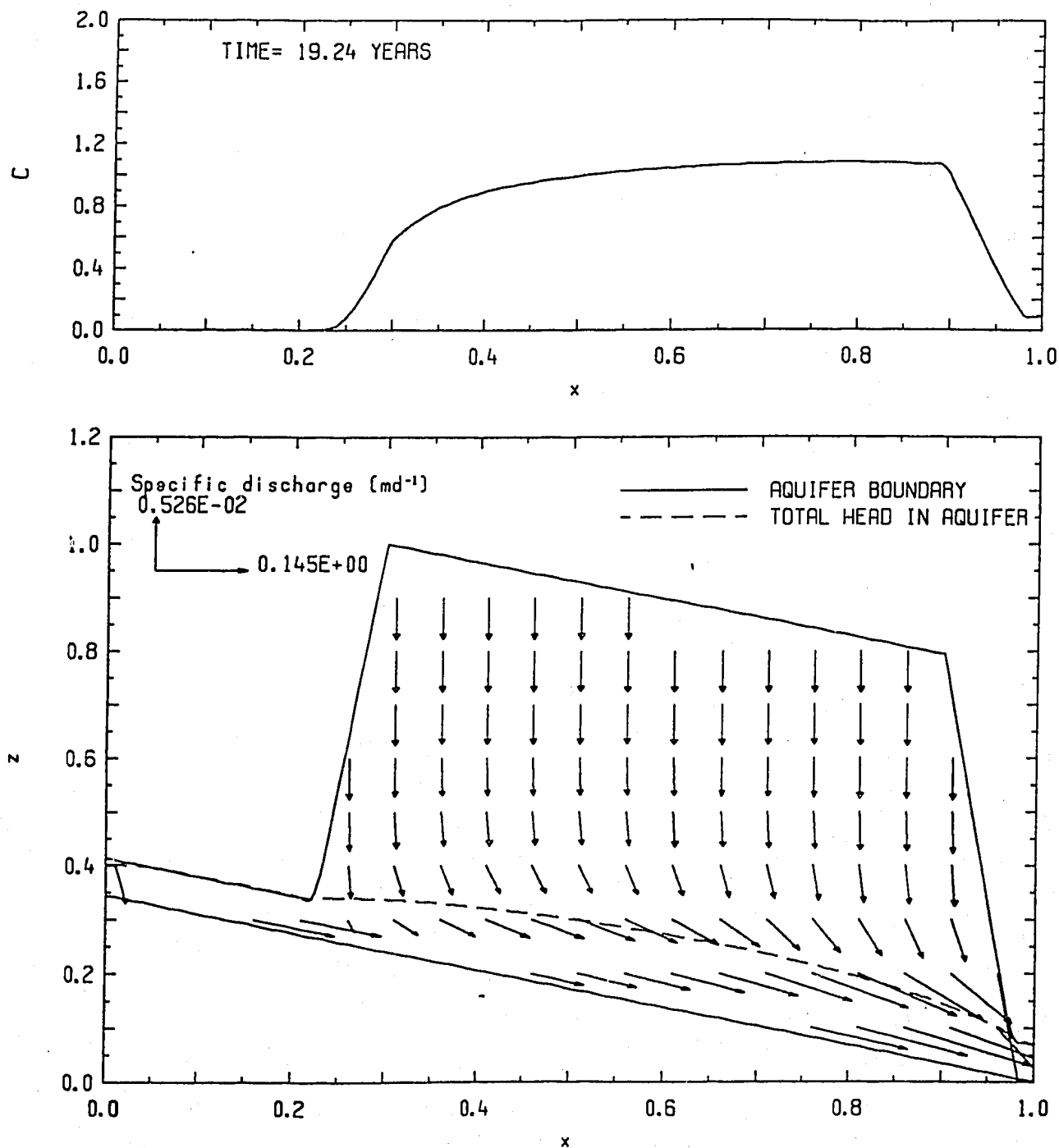
Figure 6 shows the horizontal water and contaminant discharge from the base of the dump. The concentration of contaminants decreased immediately the covers were installed, because the rain was no longer infiltrating the dump to the aquifer, but it then took 12 years until the pollutants in the unsaturated zone of the dump were effectively flushed. The slow build up from year 0 to year 20 is because the initial conditions assumed in the dump, zero contaminant and zero moisture content at year 0.

Gibson and Pantelis [1989] discuss the extension of the model to three dimensions and the inclusion of local site contours. Flow paths which would be followed by conservative contaminants were calculated for contaminants that started from the top of the dump. Infiltration after rehabilitation was assumed to be 5 % of the rainfall, down from the 50 % assumed to apply before rehabilitation. The longest flow path from the top of one of the Rum Jungle dumps to the nearest water way was about 5 years before rehabilitation and 15 years after rehabilitation. The 5 year shortening of the post rehabilitation period compared with the results given by Pantelis [1987] is due to the fact that there was still some, albeit small, infiltration after rehabilitation in this model.

2.2 CASE STUDIES - WATER FLOW

2.2.1 Woodlawn, NSW, Australia

The feasibility of reducing infiltration by compacting the top layer of a waste rock dump has been investigated at the Woodlawn mine. A series of compaction and infiltration tests were carried out by the NSW Public Works Department (PWD). Most of the dump material was not amenable to conventional compaction techniques but modified compaction methods produced acceptable compaction for all except the hardest dolerites. Maximum dry densities of 2.39 to 2.21 t m⁻³ and permeabilities of 0.32 to 3.31 m y⁻¹ were achieved in the laboratory. The PWD recommended that compaction to 2.19 t m⁻³ should provide the necessary barrier to infiltration [Southern 1937].



Top: C v. x curve, where

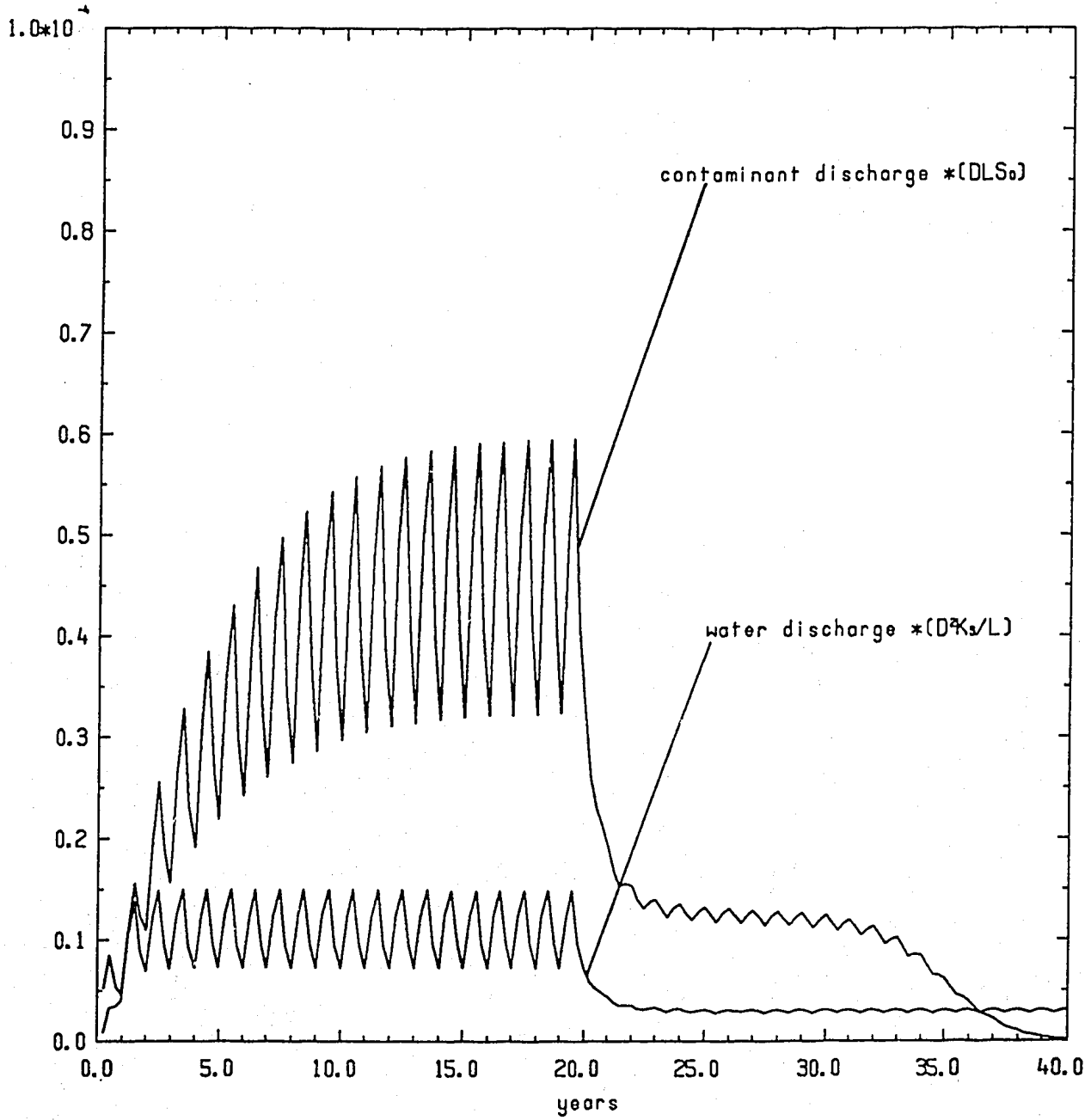
$$C = \int_g^h c' \theta dz / [DS_0 / (\epsilon K_s)] .$$

The total contaminant mass (per cross-sectional width) in the heap/shallow aquifer system is proportional to the area under the curve and is given by the expression

$$L^3 S_0 / K_s \int_0^1 C dx .$$

Bottom: The total head and specific discharge field in the heap/shallow aquifer system. The numerical values of the components of the specific discharge vectors are obtained by translating the arrows onto the origin of the axes labelled specific discharge (insert).

Figure 4. Source: Pantelis [1987], Fig 1.



Horizontal water and contaminant discharges (per unit cross-sectional area) being released into the stream at the base of the slope. Peaks are due to seasonal variation in rainfall.

Figure 5. Source: Pantelis [1987], Fig 3.

Field trials of the compaction technique were carried out on a 3 ha area and an average density of 2.40 t m^{-3} and a standard deviation of only 0.11. The compacted material was covered with 30 cm of "topsoil" and hydroseeded. Germination was excellent and establishment was still good at the time of the 1987 report. However acidic seepages continued to seep from the toe of the rehabilitated area and this could cause a problem at mine closure [Southern 1987]. The cost of rehabilitation was estimated to be \$(Aust)30 000 per ha (1985 dollars).

2.2.2 Bersbo, Sweden

Several alternate covers have been tested on test dumps at the Bersbo site in Sweden [Södermark and Lundgren 1988]. This site is discussed in detail in Appendix C. All covers consisted of a sealing layer of about 0.5 m covered with about 2 m of moraine as protection against freezing, drying and root penetration.

A capillary break (draining) layer between the sealing layer and the moraine was considered but it was found that such a layer did not significantly affect the drying of the sealing layer. The presence of the capillary break layer could reduce the amount of precipitation infiltrating into the waste rock dump but it was decided that it would be more cost-effective to spend resources on designing a properly functioning sealing layer than on adding a break layer.

Field trials were carried using three different sealing layers on plots of 200 to 600 m^2 with inclination of 1:10 to 1:5.

Cefill was tested on the first plot. Cefill is a product consisting of a cement stabilised flyash (cement 5-10%) and if necessary also activated with desulphurising producers (15-35%). With a water content of 30-40% the Cefill becomes a pumpable product which can be used to fill the voids in a bed of crushed rock. The Cefill solidifies in less than 24 hours to form a hard dense sealing layer which is also strong due to the rock aggregate stabilisation. The material is very resistant to acids and sulphate. The Cefill test plot was subdivided into nine areas for testing different cement proportions, water proportions, layer thickness and layer structures [Södermark and Lundgren 1988].

The Cefill was pumped onto a layer of crushed rock aggregates spread over the test plot, in some cases the material was vibrocompacted, and about 1 m of moraine was spread over the surface. Hydraulic conductivity was in the range $(2 \text{ to } 6) \times 10^{-10} \text{ m s}^{-1}$ which corresponds to an annual leach rate of 5 to 15 mm, i.e. a reduction of the normal gross precipitation by over 98%.

The second plot was used to test two types of clay, one dense and one semi-solid, in a 0.5 m layer. Both were very workable but the semi-solid had inferior bearing capacity and required a geotextile covering to facilitate the spreading of the moraine. Hydraulic conductivities of the clay were in the range $(1 \text{ to } 5) \times 10^{-10} \text{ m s}^{-1}$.

The third plot, which was somewhat steeper (1:5) than the others, was used to test a bentonite-mixed sand sealing layer. The surface was first covered with 0.3 m of sand and the plot subdivided into four areas to allow four different bentonite treatments to be applied. The bentonite-sand layer, which was 0.2 m thick, contained about 10% bentonite layer. Hydraulic conductivities ranges between 1×10^{-9} to $6 \times 10^{-10} \text{ m s}^{-1}$. For the highest bentonite proportion (11%) the values were in the range $(2 \text{ to } 6) \times 10^{-10} \text{ m s}^{-1}$.

The tests showed that all three methods could produce a satisfactory seal provided sufficient attention was given to quality control and on-site inspections. The density of the layers indicated that oxygen transport could be expected to decrease by more than 99%.

Details of the reclamation work at Bersbo are described in Appendix C.

2.2.3 Captains Flat, NSW

A major effort to rehabilitate the site of the abandoned Captains Flat site was undertaken in 1974. The rehabilitation consisted of two components,

1. stabilisation of the mine waste dumps to prevent ongoing erosion and leaching and to minimise the risk of major collapse;
2. covering the dumps to minimise infiltration of water into the wastes.
3. drainage modification to reduce the inflow of water into the mine.

The dumps were reshaped to a stable profile and covered with a three-layers of material.

Zone 1, on the bottom, was 225 mm thick and consisted of clay soil imported to the site. This layer was designed to provide an impermeable layer between the tailings and the surface water.

Zone 2 was a rockfill layer of minimum thickness 450 mm to inhibit capillary rise of polluted water and to provide a drainage layer. The material, which was obtained locally, consisted of slate and schist which broke up readily and was sufficiently resistant to acid and sulphur attack.

Zone 3, on the top, was a soil layer suitable for grasses and legumes.

Monitoring programs indicated reduced pollution levels in median flow conditions [Jacobson and Sparksman 1988]. There is a decrease in zinc concentrations and load and an increased biological diversity in the river close to Captains Flat. The remedial works are being evaluated and a report is expected soon.

2.2.4 Rum Jungle, NT

Runoff fractions and pollutant loads were measured on the waste rock dumps at Rum Jungle before they were rehabilitated [Harries and Ritchie]. The water balance indicated that about 50 to 60% of the incident rainfall percolated through the dump to the ground water [Daniel et al 1982]. It was estimated that most of the pollutant release was to ground water and less than 4% of the pollutants appeared in runoff.

The rehabilitation of the waste rock dumps at Rum Jungle was based on the premise that a reduction in infiltration of rain water into the dump material would reduce the pollution load to ground water and from there to the local river.

The dumps were covered with a three layer system:

- first, on the bottom, a compacted clay layer to act as a moisture barrier;
- second a layer of sandy loam as a moisture retention zone to support vegetation and prevent the clay layer drying out; and
- third, on the top, a gravelly sand to provide erosion protection and act as a pore breaking zone to restrict moisture loss by evaporation in the dry season.

Before the layers were put in place the dumps were reshaped so that the tops had a maximum slope of 5° and the sides a maximum slope of 1 in 3. On the top of the dumps the layers had minimum thicknesses of 225 mm, 250 mm and 150 mm while on the sides they had minimum thicknesses of 300 mm, 300 mm and 150 mm for the clay, loam and erosion layers respectively. On the sides crushed rock (nominal minimum size 75 mm) replaced gravelly sand as the erosion barrier [Verhoeven 1988].

Engineered runoff channels and erosion control banks were constructed on the tops and sides of the dumps. Vegetation was established to stabilise the dump surface against the long-term effects of erosion.

The amount of infiltration was measured using lysimeters installed in the reshaped White's and Intermediate dumps before emplacement of the clay layer. The amount of water collected by the lysimeters in White's dump in each of the three wet seasons between 1985 and 1989 was equivalent to less than 2.5 per cent of the incident rain. That collected in lysimeters in Intermediate dump corresponded to less than 5 per cent of the incident rain. These infiltrations were much less than the 50 per cent of the incident rain which percolated through the dumps before rehabilitation and indicated that the compacted clay cover achieved the desired reduction to 5 per cent or less of incident rain [Bennett et al 1988a, Bennett et al 1989b].

Intensive ground water monitoring at the site started in mid 1983. The monitoring data showed that the concentration of pollutants in the ground water close to the dumps four years after rehabilitation was substantially the same as before rehabilitation. The reduced infiltration of rainwater into the dump indicated that there was a reduction in the flow of ground water from the dump and hence a reduction in the pollutant load. Calculations showed that the large store of pollutants held in the pore space could take 10-20 years to be depleted [Pantelis 1987, Gibson and Pantelis 1988].

3. GAS TRANSPORT - OXYGEN

The oxidation of sulphides in the waste rock dumps requires oxygen. In many dumps it is thought that supply of oxygen limits the rate of oxidation.

Oxygen transport in dumps occurs by a combination of diffusion through the pore space, diffusion into the particles, advection driven by external pressure variations and thermal convection produced by the elevated temperatures resulting from the exothermic oxidation reaction. Once the oxidation reaction is underway there is a positive feedback, the release of heat raises the temperature which increases the thermal convection which brings more oxygen to the oxidation sites.

3.1 THEORETICAL TREATMENTS OF OXYGEN SUPPLY PROCESSES

3.1.1 ANSTO

Ritchie [1977] developed a gas diffusion model of an oxidising heap in which oxygen diffused from the top surface to react at oxidation sites within the heap. This model, also known as the simple homogeneous model (SHM), was based on the assumption that particle sizes were very small. The model predicted a reaction front between the oxidised and unoxidised zones which moved down the heap at a readily determined speed.

Davis and Ritchie [1986a] extended the oxidation model by including the effect of finite particle sizes, the single-sized particle model (SPM). The model assumed that the oxidation rate was limited by the rate at which oxygen was supplied to oxidation sites within the particles. Oxygen supply was by diffusion through the pore space of the waste followed by dissolution into the water film around the particles and diffusion into a moving reaction front within the particles. The reaction front starts at the surface of the particle and moves inwards until it reaches the centre, a 'shrinking core' model.

The model equations were solved by assuming pseudo-steady state diffusion within the particles [Davis et al. 1968b]. An approximate analytical solution was also derived. The model was used to predict the oxidation rate, sulphate production and heat source as a function of time within oxidising dump. It showed that a dump at Rum Jungle could take 250 years to oxidise.

The model was extended to include a range of particle sizes in the dump, the distributed particle size model (DPM) [Davis and Ritchie 1987]. Although this extension did not greatly change the predicted magnitude and longevity of the pollutant release, it did change the heat source distribution and the oxygen concentration profile, figure 2.7, 2.8 and 2.9 (Figure 2 3 and 4 of 1987). The DPM model is realistic for oxygen transport in a waste rock dump provided thermal convection or other advective processes are not significant.

A two-dimensional model which includes thermal convection has been developed although at this stage it is applicable to only a small range of geometries [Bennett et al 1989]. The model includes the effects of thermal convection caused by the elevated temperatures produced by heat released in the oxidation process. The results show that diffusion was the most important oxygen supply mechanism in all dumps for the first two years. At later times the relative importance of diffusion and thermal convection as oxygen transport mechanisms depended on the gas permeability of the dump. Thermal convection became a significant process in dumps containing reasonably porous material, i.e. gas permeabilities greater than about 10^{-10} m^2 . Figure 10 shows the gas flow and derived temperatures in a cylindrical dump (20 m high and 20 m radius) for permeabilities of 10^{-10} m^2 .

3.1.2 Rasmuson and Collin

Rasmuson and Collin [1988] have investigated the feasibility of controlling oxygen transport by using a fine grained layer in which the water content is maintained close to saturation. Their earlier work had shown that oxygen diffusion through a cover material was a highly controlling parameter for oxidation of mine wastes and that the effective gas diffusivity in a porous material decreases rapidly with increasing water content [Magnussen (Collin) and Rasmuson 1984]. In Collin and Rasmuson [1988], they reviewed experimental data on gas diffusivity and compared methods for estimating the gas diffusivity at high water contents.

A combined water flow, gas diffusivity and oxygen transport model was used to investigate a cover consisting of layers of coarse and fine materials. The water flow calculations have already been discussed in Section 2.1.1, where the materials and layer thicknesses were also described. Gas diffusivities were estimated using the semi-empirical methods of Millington and Shearer [1971], see also Collin and Rasmuson [1988].

It was assumed that oxygen flow through the covers was steady-state and that there was no oxygen consumption in the layers. This meant that the mass transfer coefficient, H_M , for a layered system could be calculated from

$$H_M = \frac{1}{\frac{l_1}{D_1} + \frac{l_2}{D_2} + \frac{l_3}{D_3} \dots}$$

where l_1, l_2, l_3, \dots are thicknesses of layers, and D_1, D_2, D_3, \dots are the gas diffusivities of the layers.

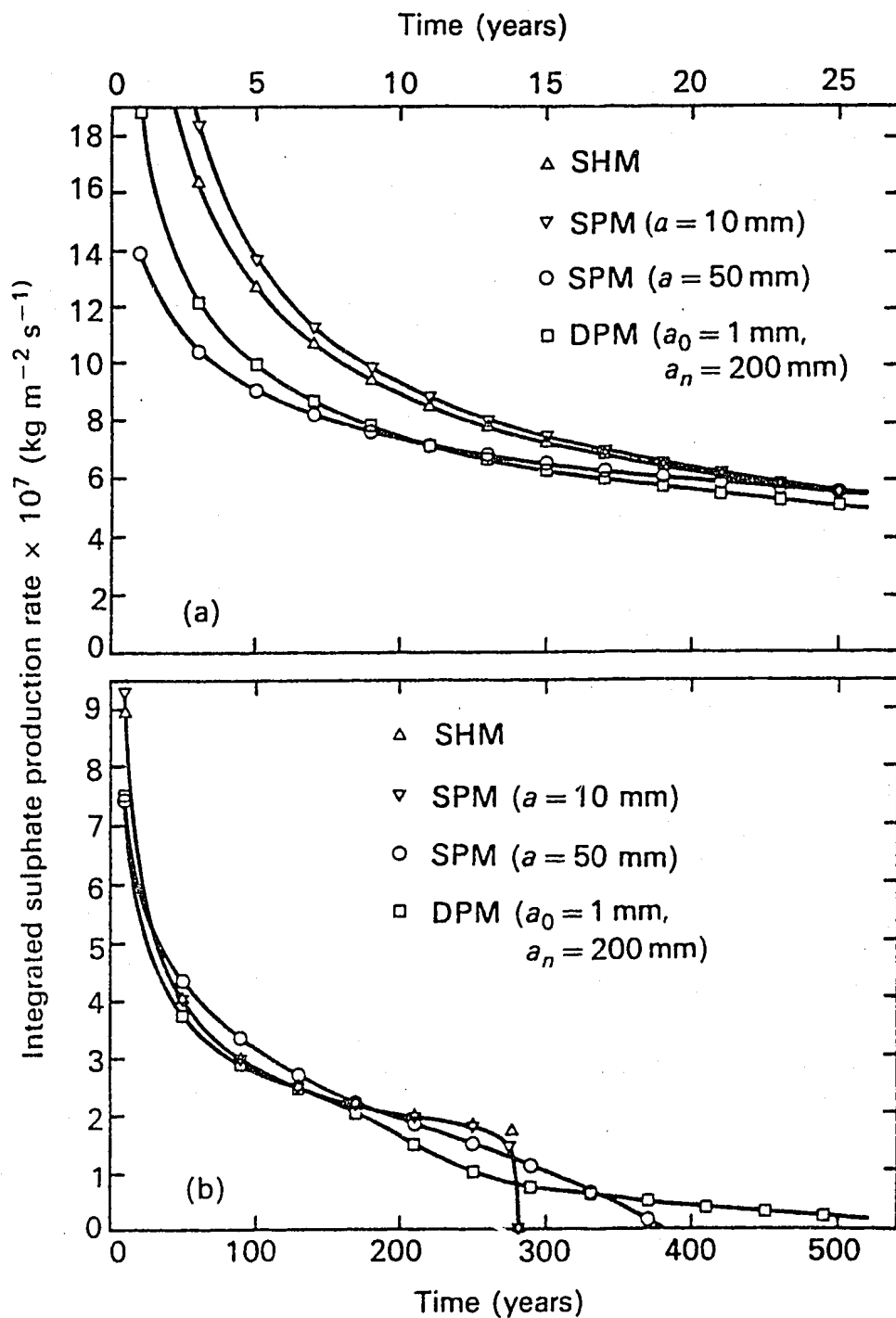
For most cases of interest the oxygen concentration at the top of the waste rock is nearly zero, hence the difference of oxygen concentration between the top and the bottom of the cover is effectively equal to the oxygen concentration in the atmosphere, c_{atm} . The flux of oxygen through the covers is then

$$J = -H_M c_{\text{atm}}$$

These equations were used to calculate the total oxygen flux through the covers over a whole year, using water contents obtained from the water flow calculations. This total oxygen flux is equivalent to the oxidation rate in the waste. Table 1 lists the derived oxidation rate for a range of cover designs.

Table 1. The efficiency of soil covers as a barrier against oxygen diffusion into the waste
 [Source: Rasmuson and Collin 1988]

Cover system	Climate data	Estimation of the oxygen diffusivity	Integrated mass transfer coefficient $\int H_M$	Oxidation rate in the waste calculated from $\int H_M$ [g pyrite/m ² ,year]
Without cover ¹⁾		non-agg	-	> 3000
Constant mass transfer coefficient $H_M = 1 \cdot 10^{-9}$	-	-	0.0311	10
Constant mass transfer coefficient $H_M = 1 \cdot 10^{-8}$	-	-	0.311	100
1.0 m sandy moraine 0.5 m silt capillary barrier	normal year	non-agg	0.0152	5
0.5 m silt, hydrostatic	-	non-agg	0.0270	9
0.5 m silt, hydrostatic	-	agg	0.393	126
Capillary barrier 0.5 m silt capillary barrier hydrostatic	-	non-agg	0.004	1
Capillary barrier 0.5 m clay capillary barrier hydrostatic	-	non-agg	0.0143	5
0.5 m clay transient	normal year	agg	-	< 20
1 m sandy moraine capillary barrier	normal year	non-agg	4.146	1330
1 m sandy moraine	normal year	non-agg	5.009	1610
1 m clayey moraine	normal year	non-agg	0.8096	260
2 m sandy moraine ²⁾	normal year	non-agg	1.176 (1.157)	376 (372)
2 m clayey moraine ²⁾	normal year	non-agg	0.1122 (0.0759)	36 (24)
2 m clayey moraine	normal year	agg	0.7489	241
2 m clayey moraine	normal year but dry summer	non-agg	0.4075	131



The total sulphate production rate as a function of time after creation of the wastes. Comparison of the SHM, SPM and DPM for (a) early time 0–26 years and (b) extended time 0–510 years

Figure 6.

Source: Davis and Ritchie [1987], Fig 2.

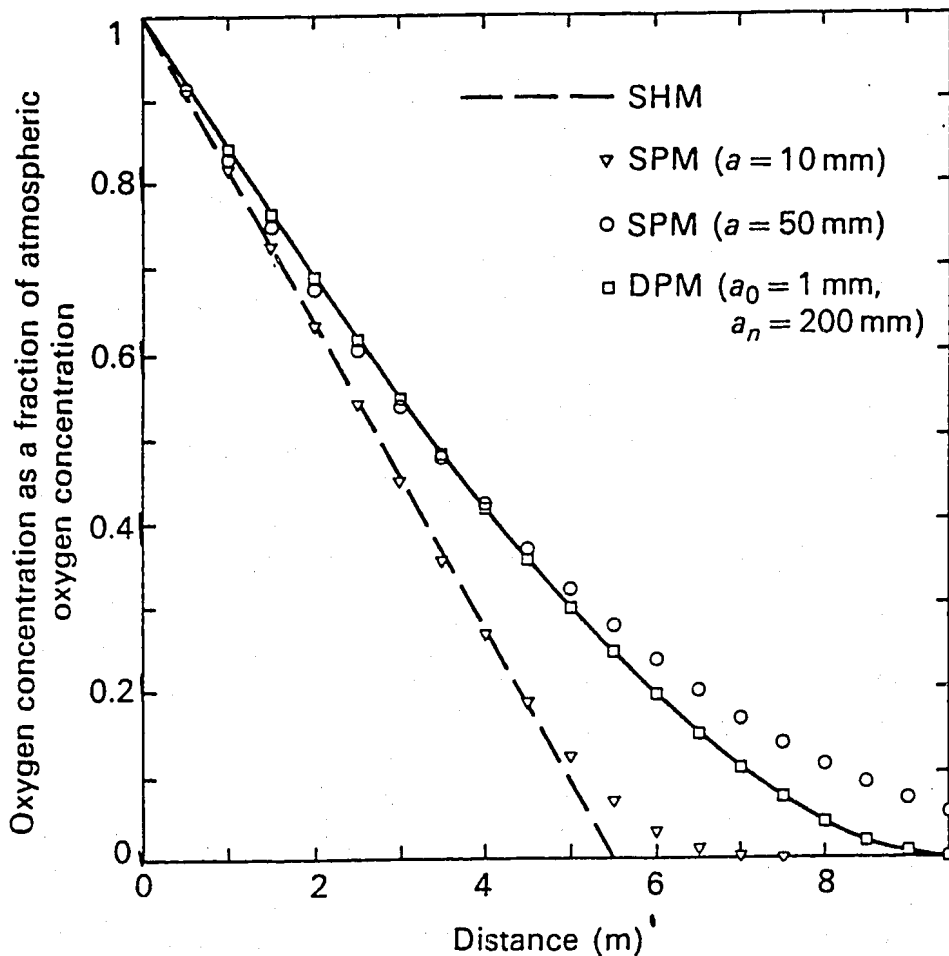
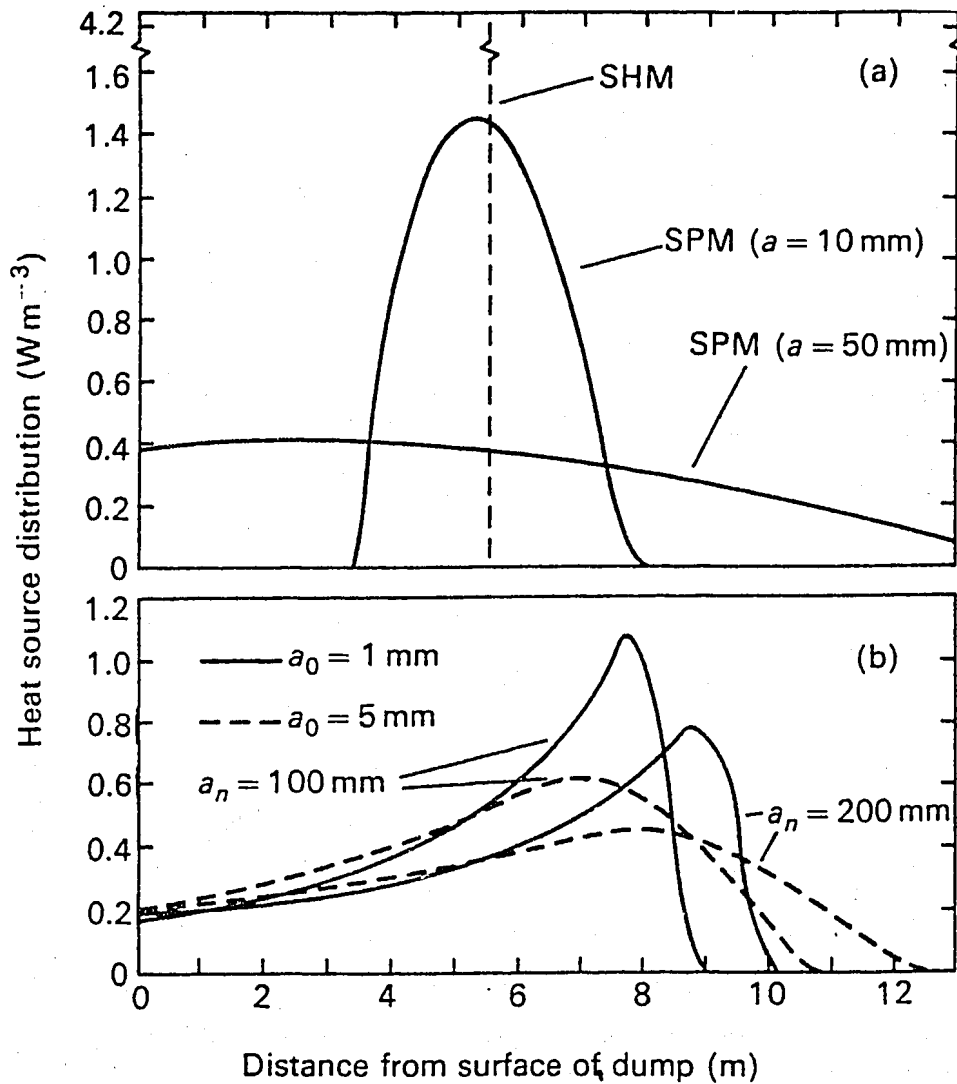


Figure 3 The dimensionless oxygen concentration as a function of distance from the surface of the wastes 26 years after creation of the wastes. Comparison of the SHM, SPM, and DPM

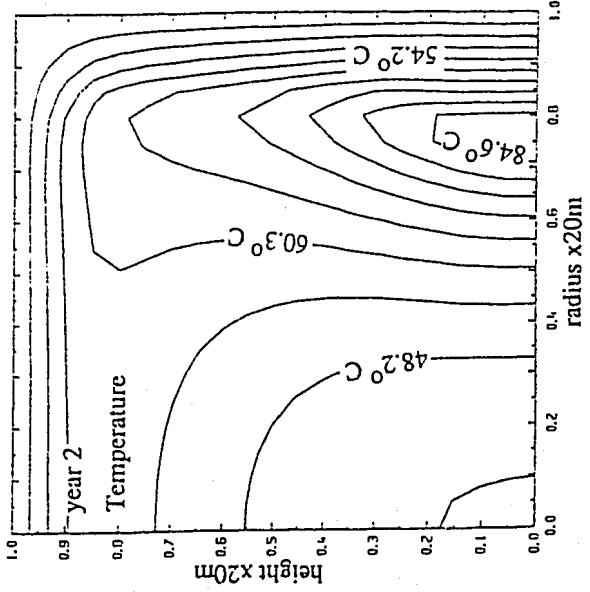
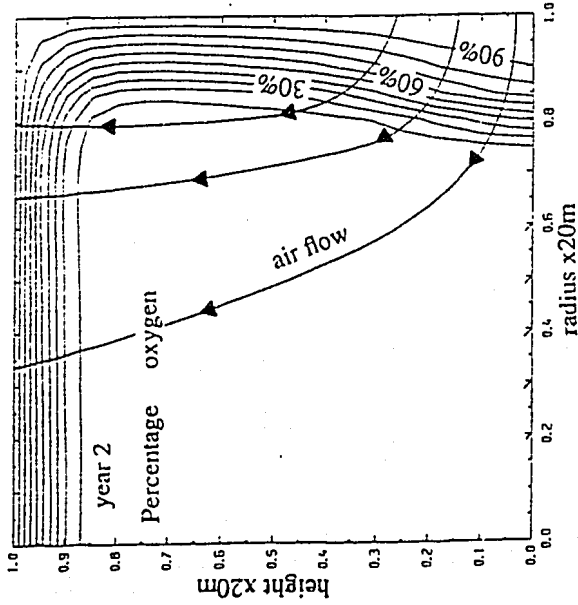
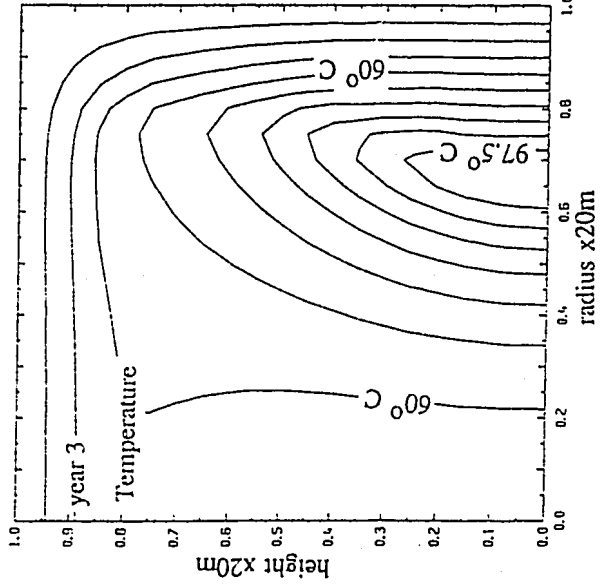
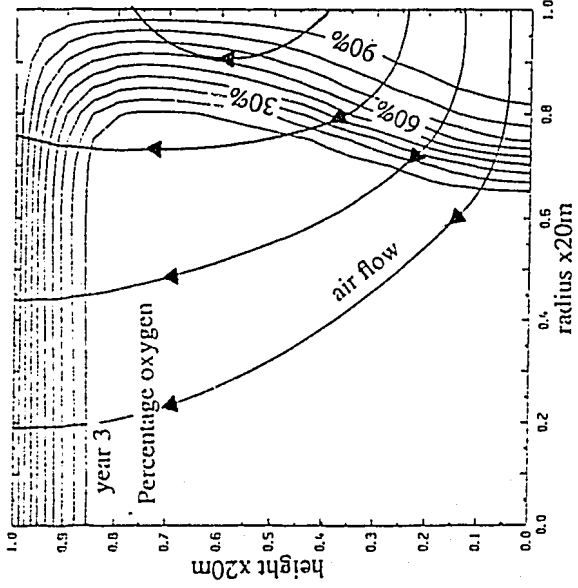
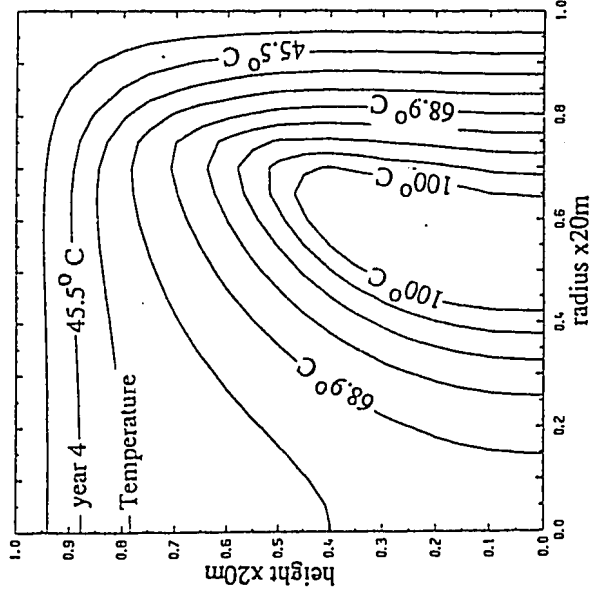
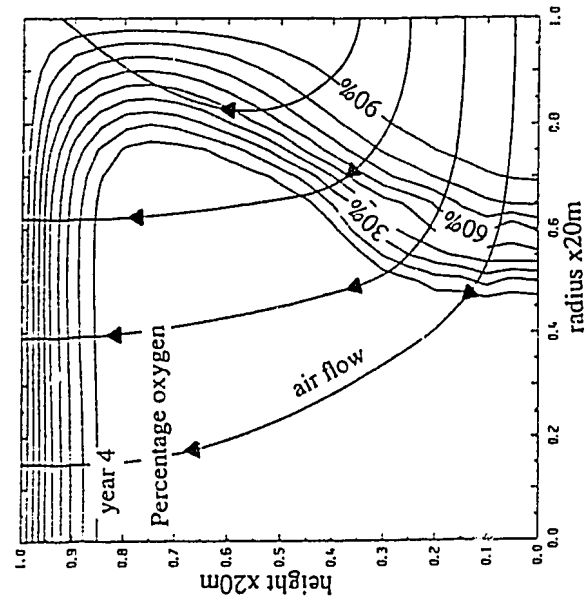
Figure 7. Source: Davis and Ditchie [1987], Fig 3.



The spatial heat source distribution at 26 years. Comparison of (a) the SHM and SPM and (b) the DPM for $a_0 = 1$ mm (solid lines), $a_0 = 5$ mm (dashed lines) and $a_n = 100$ mm and 200 mm (as indicated)

Figure 8.

Source: Davis and Ritchie [1987], Fig 4.



Oxygen content and temperature contours in a cylindrical heap at years 2,3 and 4. The permeability $K=10^{-10} m^2$ and the particle size $a=0.005 m$. Arrowed solid lines indicate air flowpaths.

Figure 9. Source: Bennett et al [1989a], Fig 2.

The result showed that very low oxygen transport rates could be obtained by a layer of fine-grained material (clay or fine sand) that was maintained at a high moisture content. It was important to maintain a very high moisture content in the fine grained material in order to keep the mass transfer coefficient sufficiently low. This was readily achieved for Swedish weather conditions for fine grained materials with very low hydraulic conductivities, less than about $1 \times 10^{-9} \text{ m s}^{-1}$, even if there were no capillary break layer. For fine grained layers of moderately low hydraulic conductivity, e.g. $5 \times 10^{-8} \text{ m s}^{-1}$, the use of capillary break layers were necessary to maintain the necessary moisture content.

3.2 CASE STUDIES - OXYGEN SUPPLY

3.2.1 Woodlawn

Jeffery et al [1988] carried out a long-term leaching study using waste rock from Woodlawn in large columns (2.3 m high and 0.3 m diameter). They found that the leach characteristics could not be properly assessed using short term bench-top test, that diffusion of oxygen was the major factor limiting the rate of oxidation, that particle size distribution was important, and that the rates of release of ions did not necessarily relate to oxidation rates in the column.

3.2.2 Rum Jungle

Probe holes in the Rum Jungle waste rock dumps were used for measuring temperature profiles and collecting gas samples for determining gas composition.

Before rehabilitation, the temperatures at some locations within the dumps exceeded 50°C . The elevated temperatures were caused by the release of heat in pyritic oxidation. A one-dimensional heat transfer model was used to derive the distribution of heat production, and hence locations where oxidation was occurring, from the measured vertical temperature profiles [Harries and Ritchie 1980, 1987]. In some locations heat was being produced at depths of 14 m in the dump.

After rehabilitation, the heat production was either very low or zero at all measuring locations. Comparison of heat production distributions before and after rehabilitation showed that the oxidation occurring before rehabilitation was effectively stopped by rehabilitation.

Measurement of oxygen concentrations in the pore gas of the Rum Jungle dumps before rehabilitation showed that oxygen was transported to the oxidation sites by a combination of diffusion, thermal convection, and advection driven by variations in atmospheric pressure [Harries and Ritchie 1985]. Each of the oxygen transport processes produces a characteristic oxygen concentration profile. Thermal convection caused the oxygen concentration to be higher near the base of the dump. Diffusion caused the oxygen concentration to decrease monotonically with depth. Advection driven by variations in atmospheric pressure lead to short-term changes in oxygen concentration over timescales of less than a day.

At Rum Jungle, which is near the equator, the dominant atmospheric pressure variations are atmospheric tides which have two maxima and two minima a day. Increasing pressure causes air to flow into the pore space and, because the incoming air has a higher oxygen content than air already in a dump, the oxygen concentration measured at a given point increases. This caused diurnal variations in the oxygen concentration at a given point in a dump with two maxima and minima a day.

A comparison of the oxygen concentrations with the oxidation rate profiles derived from the temperature profiles showed that the supply of oxygen was the main process limiting the rate of oxidation most regions of the waste rock dumps before rehabilitation.

The emplacement of the compacted clay covers greatly reduced the level of oxygen in most regions of the dumps. Before rehabilitation a tongue of oxygenated air at depth in both dumps indicated that thermal convection was transporting oxygen in from the sides of the dumps and up through the hot regions, Figure 11. After rehabilitation, the oxygen concentrations at depth decreased to low values indicating that the clay cover effectively stopped oxygen transport by thermal convection [Bennett et al 1989b].

The oxygen concentrations did show a seasonal variation. In the dry seasons, the oxygen concentrations were low and there was no diurnal variation. In the wet seasons, oxygen concentrations were still generally low but diurnal variations occurred at some locations near the surface early in the wet season. This was taken to indicate that there were some cracks in the clay layer in the dry season with the cracks providing paths over the whole dump surface for advection of air by atmospheric pressure variation. The reappearance of the diurnal variations in the wet season indicated that most of the cracks closed as the moisture content of the clay increased but the clay near the monitoring holes did not seal as well as that further away.

4. ACID DRAINAGE

4.1 THEORETICAL TREATMENTS OF ACID GENERATION

Harris [1969] developed a model for heap leaching of copper sulphide ores and compared the results with examples from Rum Jungle and Rio Tinto. He concluded that oxygen diffusion was the factor controlling copper leach rates at Rum Jungle and in other similar operations.

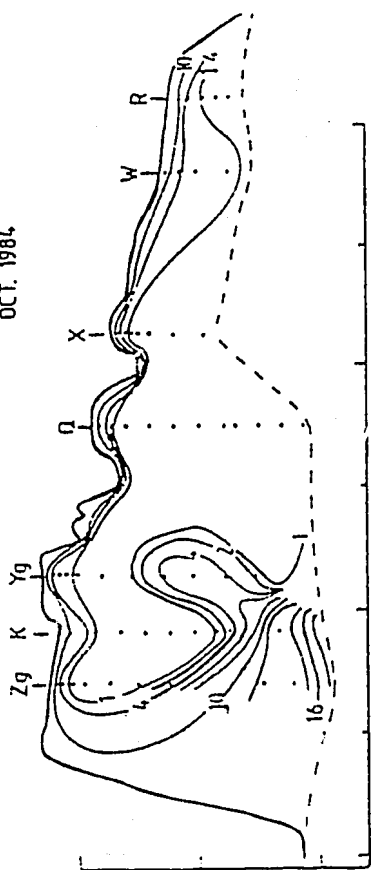
Waste rock and low grade ores often contain several minerals which may react with one or more reagents, e.g. minerals might be pyrite, chalcopyrite, malachite, calcite etc, and reagents might be ferric salts, oxygen, sulphuric acid, dissolved copper salts, dissolved ferrous salts etc. Prosser et al [1981, Prosser and Box 1983, Box and Prosser 1986] developed a generalised procedure for including the competing, complementary and sequential reactions in computer models for heap and dump leaching.

The procedure is based on the assumption that reactive mineral grains were located within lumps of inactive minerals, and that the reagents diffused in from the surface via cracks and pores in each lump. The reactive particles were assumed to be numerous and much smaller than the lumps. Although the pores in the lumps were filled with solution, the voids between the lumps were empty. For each mineral there was a distinct boundary between an outer zone of fully reacted zone and an inner unreacted zone. The complex chemical system of many minerals and many reagents was divided up into subsystems of reactions that were independent, and each subsystem was resolved into stable coherent groups of reactions that had a common boundary. Reaction rates were then calculated for each coherent group.

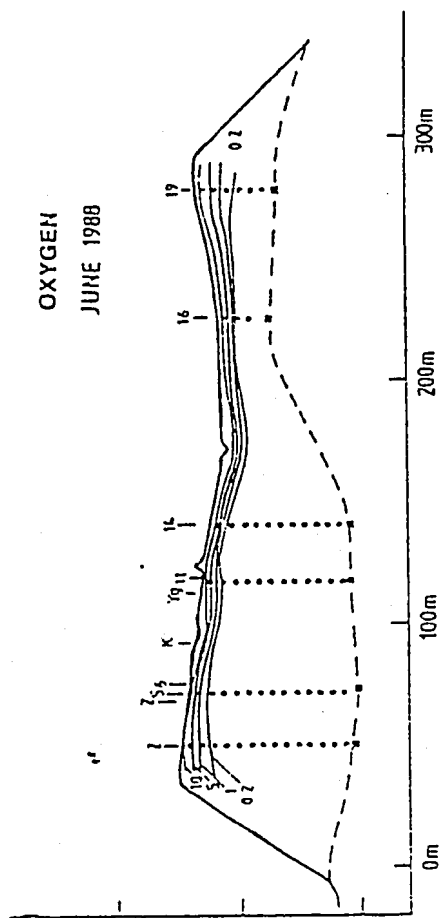
The procedure was tested against several column experiments and found to give reasonable results [Prosser and Box 1983]. Features in leaching predicted by the simulation included the delay before any copper appeared in the leachate, and the subsequent buildup of copper concentrations. This prediction was achieved without using any adjustable parameters [Box and Prosser 1986]. Prosser and Mansor [1989?] simulated the leaching of an ore containing both oxide and sulphide copper minerals.

Whittemore [1981] carried out an in-depth review of many models of vat, heap, dump and in-situ leaching systems, Table 2. The review concluded that a number of useful correlations had been produced, and that the most useful and widely applicable models were those of Madsen and Wadsworth, and of Cathles and Apps. Whittemore recommended that further work be directed to refining and developing the models and examining the validity of the various simplifying assumptions and approximations. There was also a need to obtain more data for testing the models.

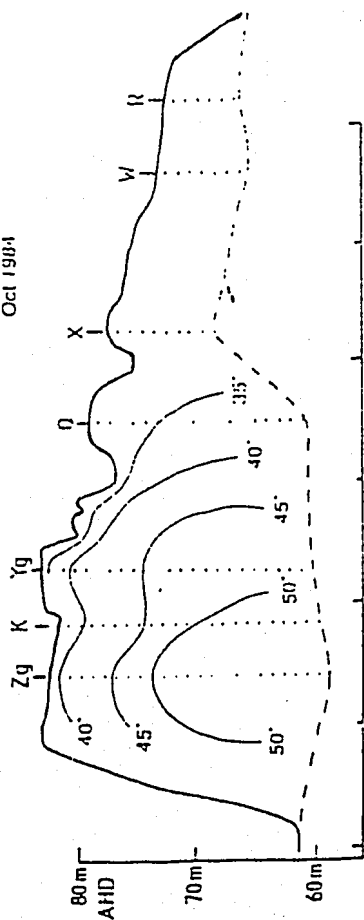
OXYGEN
OCT. 1984



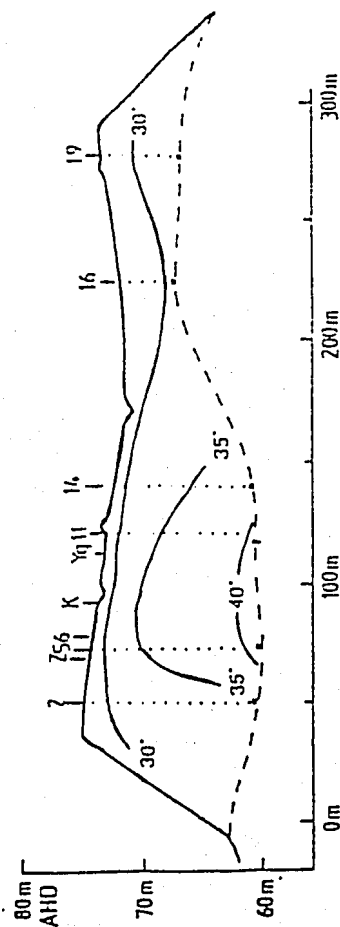
OXYGEN
JUNE 1988



TEMPERATURE
Oct 1984



TEMPERATURE
June 1988



Temperature and oxygen distributions in Intermediate dump before (Oct 1984) and after (June 1988) rehabilitation. Temperature contours are marked in °C and oxygen contours in percent oxygen (by volume)

Figure 10. Source: Bennett et al [1989b], Fig 5.

Table 2. Summary of models [Source Whittemore 1981]

Authors and Date	Type of Model	Application for which Model was Developed	Method of Application used by Authors	Notes
Harris et al 1969	Based on oxygen diffusion and surface reaction in heap.	Lab-scale tests and full size heaps of copper sulphide ore from Rum Jungle.	Mainly descriptive.	Made useful distinction between dump leaching processes controlled by properties of the dump itself and those controlled by properties of individual particles.
Braun, Lewis and Wadsworth (BLW) 1974	Particulate leaching model based on a shrinking core model with a narrow reaction zone, and controlled by a combination of diffusion and chemical reaction.	Leaching of chalcopyrite ores at high temperatures and pressures with oxygen as oxidant at scales from 200 g to 6 t - simulation of deep in-situ leaching.	By curve-fitting, with some predictive application.	Type of model which has attracted wide interest. The assumption of a narrow reaction zone may be unsound but reasonable correlations have been obtained, although modifying parameters were necessary to account for some discrepancies.
Madsen, Wadsworth and Groves 1975	BLW model with slight modifications.	Acid ferric leaching of monzonite and quartz monzonite copper sulphide ores at pilot scale.	Parameters from leaching data at one particle size used in model to predict leaching at another particle size.	Showed BLW model to be applicable to different systems. Predictions gave good agreement with experiment in one case, but not in the other because of solution flow problems.
Roman, Benner and Becker 1974	BLW model, with diffusion control only. Includes algorithm for obtaining lixiviant concentration profile in heap.	Acid leaching of malachite from sandstone ore, on 75-120 kg samples.	Same parameters found to enable model to be fitted to two tests under different conditions.	Assumption of narrow reaction zone probably more valid in case of diffusion control only. Results interpreted as showing model could be used predictively.
Shafer, White and Caenepeel 1979	BLW model as used by Roman et al with slight modifications.	Acid leaching of copper oxide ore on pilot-scale (40 kg-1 t).	Parameters defined by data from small-scale tests used in model to predict leaching from large-scale test.	Reasonable correlations obtained.
Averill 1976	BLW model with diffusion and chemical control, and with diffusion control only, with more sophisticated algorithm for lixiviant concentration profile than used by Roman et al.	Leaching of chalcocite and copper sulphide ores, on lab-scale.	As Braun, Lewis and Wadsworth.	Reasonable correlation obtained.
Bartlett 1972	Continuity equation model based on Fick's law diffusion equation and chemical reaction parameters.	As Braun, Lewis and Wadsworth, on 6 t scale.	Predictively, using measured physical parameters and known kinetic data.	Quite rigorous model, but requires very complex computation. Problems in measuring required parameters. Reasonable agreement obtained between predictions and experimental results.
Braithwaite 1976	Modification of the Bartlett model to give a quasi-steady-state form.	System similar to that studied by Braun, Lewis and Wadsworth, on lab-scale.	As Bartlett.	Computation simpler than with original Bartlett model.
Madsen and Wadsworth 1977	Effectively Braithwaite's version of the Bartlett model. Includes algorithm for obtaining ferric distribution profile.	As Madsen, Wadsworth and Groves, with addition of granitic chalcopyrite ore at pilot-scale.	A combination of curve-fitting and use of measured parameters to give some predictions.	Applies Bartlett-type model to a range of ores. Quite good correlations obtained in most cases.
Roach and Prosser 1975	Based mainly on diffusion control, with optional methods of making allowance for chemical reaction. Can be used to determine maximum possible recovery.	As Braun, Lewis and Wadsworth; also synthetic minerals.	As Bartlett.	Resembles the diffusion-controlled version of the BLW model. Rather empirical method of allowing for chemical reaction control. Fairly simple to use. Fair correlations obtained for leaching rates, but poor correlations for maximum possible recovery.
Cathles and Apps 1975	Combines BLW-type particle reaction model with heat and reagent balances in heap.	93,000 t dump of copper sulphide ore at Bingham Canyon.	As Bartlett.	Useful attempt to include properties of dump in leaching model. Some difficulties in measuring required parameters. Reasonable agreement between predictions and experimental results.
Letowski 1980	Specifically for leaching minerals exposed by dissolution of soluble gangue in ore.	Small-scale leaching of dolomitic copper sulphide ore.	Mainly descriptive.	Limited area of application.

Derry and Whitemore [1983] collected data from laboratory and pilot studies of acid leaching of copper- and zinc-bearing pyritic ore from the Avoca mine, Ireland. They used the data to test the validity of three models: the diffusion control model of Roach and Prosser [1978], the sharp-edged reaction zone model of Braun, Lewis and Wadsworth [1974], and the diffuse reaction zone of Bartlett [1973]. All three models described the progress of static leaching reasonably well for the larger sized fully wetted particles using parameters found by curve-fitting data from laboratory scale tests. At smaller particle size the chemical rate factor became more important and the mixed kinetics models were preferred, with the diffuse reaction model providing the best overall fit.

Kuzmanovska et al [1986] applied a moving reaction zone model to the leaching of a low grade chalcocite-covellite ore with a broad particle size distribution. There was good agreement between calculated extraction curves and results from column measurements. The model was used to predict likely recovery from dump leaching the ore over a period of five years.

Miller and Murray [1988] discussed the processes involved in predicting acid drainage. Data from a large gold project was used to show the importance of time dependent factors. They also stressed the need to carry out acid/base analyses on individual samples and not on composite samples.

4.2 CASE STUDIES - ACID DRAINAGE

There have been many studies on bacterial populations in sulphidic waste materials. There is a large and diverse microbial flora in the waste rock dumps at Rum Jungle [Goodman et al 1981a,b, and Babij et al 1981]. Hendy [1987] isolated a thermophilic iron oxidising bacteria from one of the Rum Jungle dumps and characterised its growth requirements.

Gottschlich et al [1986] measured the rate of oxidation of iron sulphide contained in small scale reactors designed to simulate conditions in coal storage heaps.

In 1981 the dumps at the Kjølvi mine in Norway were recontoured, consolidated and limed. About 30% of the piles were moved. During recontouring about 30 t of lime was mixed inside the dump and about 70 tonnes applied to the surface as a slurry. The amount of acid mine drainage increased following the reclamation work until 1985 but then decreased between 1985 and 1988 [Iversen 1988].

5. DUMP CONSTRUCTION

5.1 THEORETICAL BASIS FOR ALTERNATE DUMP CONSTRUCTIONS

It is clear from the earlier discussion that acid production could be controlled if the water and/or oxygen access is controlled or if the water is prevented from going acid. Various techniques have been proposed to achieve these ends.

Most dump management schemes rely on characterisation of the waste and selective placement of the reactive material. The aim is to encapsulate the reactive waste within non reactive material. It is important to establish that the encapsulation will indeed occur. It is not sufficient for selective placement to allow green vegetation to be established if the amount of acid drainage is not reduced.

The material used to encapsulate reactive material must have a low enough hydraulic conductivity and the dump be designed to divert water away from the reactive material. In addition the encapsulating material should reduce the supply of oxygen to the reactive material.

One method of reducing hydraulic conductivity is by compaction of silt or clay materials (e.g. see Section 2.2). At some sites suitable encapsulating material can be obtained by compacting selected waste rock (e.g. see Section 2.2.1). Joyce and Ryan [1989] suggested encapsulation using a capping of oxide waste material which is compacted by mine traffic and bulldozers.

discussed the plans for a proposed open-cut (250 000 tpa) polymetallic mine in semi-arid Queensland. The plans included the containment of sulphide waste within chemically inert and physically suitable material. The capping of the dump consists of oxide waste material traffic compacted by bulldozers and trucks prior to reshaping, topsoiling and revegetation. They did not discuss the criteria of the physically suitable material.

Liming and other neutralising and bactericide agents can be used to delay the build-up of acidity either by mixing into the waste or spreading into the top surface layer. However these techniques are often only a temporary solution.

5.2 CASE STUDIES - DUMP CONSTRUCTION

5.2.1 Cessnock region, NSW

The main source of pollution at the abandoned Caledon colliery was a small pit which had been backfilled with spoil. The spoil was highly acidic, devoid of vegetation and actively eroding. In view of the costs of obtaining sufficient top soil it was decided to attempt to establish vegetation directly on the acid spoil material. Sufficient lime was added to raise the pH to 6.5 down to a depth of 30 cm. Vegetation was established over most of the site with some difficulty. After eight years grasses and legumes were well established, and there has been some invasion by volunteer native and naturalised species. Some small areas of highly acidic spoil remained which steadfastly refused attempts to revegetate them. However vegetation was established to the point where erosion was under control [Hannan 1983].

Rehabilitation of the Maitland Main site consisted of redirecting "clean" water around the site, burying some 88 000 m³ of washery rejects in the open cuts, and covering the backfilled rejects with several metres of better quality spoil. The better quality spoil was also spread over all disturbed parts of the site. All areas were limed and revegetated [Hannan 1983].

Monitoring of the Maitland Main site showed that revegetation was encouraging but the ecosystem remained fragile. In 1983 maintenance was considered to consist principally of protecting the site from virtually any form of land use until such time as the native forest community had time to become properly established [Hannan 1983].

The Abermain No. 1 colliery was also a source of acid drainage mainly from carbonaceous material and washery reject emplaced or scattered over the surface. The area was reshaped, drainage works installed, sufficient lime added to neutralise the top 30 cm, and the area ripped. A 15 cm layer of clean topdressing was then applied. The vegetation was very successful and there was a continual improvement in density and vigour. The site was regarded as being completely rehabilitated [Hannan 1983].

The following lessons were learnt from experience in the Cessnock coalfield [Hannan 1983]:

1. Acid forming material should be buried. Even shallow burial to a depth of 30 cm helps.
2. Acid forming potential varies widely within a short distance.
3. There are clear benefits in topdressing acid sites with top soil or at least a better quality non-acid producing material.
4. The drainage system should separate clean from contaminated or acid water.
5. Hand planting of trees and complete exclusion of grazing animals.

6. Rehabilitation should leave the site in as tidy a condition as possible.
7. Maintenance is essential, at least in the first few years.
8. Old sites that have been operating for many years can present conditions which are extremely adverse to the growth of vegetation. The vegetation community will be very fragile for many years after rehabilitation.

Hannan [1984] recommended that selective handling should be used to handle acid materials at open cut coal mines in New South Wales. The acid overburden layers should be selectively placed at a depth greater than 4 to 5 m and above the ground water table.

These techniques allow vegetation of sites, but probably do not stop the release of acid pollutants.

5.2.2 Woodlawn, NSW

Trials have been carried out to investigate rehabilitation techniques at the Woodlawn mine. There were no readily available local sources of clay so other techniques were investigated. In 1983 an area of 3 ha was limed (4 and 8 tonnes per ha), covered with a nominal 10 cm of topsoil and planted with pasture. There was excellent germination but the grass began to die within 5 months because of acidification of the soil. The trial showed the importance of isolating the growing medium from the toxic waste [Southern 1987].

Trials were carried out to investigate whether compaction would form a suitable low hydraulic conductivity layer. These tests have been discussed in section 2.2.1.

6. RECOMMENDATIONS

6.1 FUTURE STUDIES

There are only a few well documented case studies on reactive waste rock dumps at mine sites, either of dumps that are known to have gone acid and of those which potentially could go acid. Such studies need to include a good characterisation of the dumps and the dump material. This can be achieved with difficulty by measurements at existing dumps, but potentially better and more definitive data can be obtained on dumps presently being constructed. Data should include information on geochemical and physical properties as well as construction techniques.

Selective placement techniques are being widely applied at mine sites, but there is a lack of basic understanding and experience to provide confidence that the all techniques will work in the long term. In most cases the selective placement techniques are a major advance on the old methods of dumping whatever came out of the pit on the closest available patch of ground. However even selective placement will not be effective if the dump design and material properties of the non-reactive materials allow significant quantities of water and air to reach the reactive material.

There is a need for further development of the various theoretical models for predicting water flow, gas transport and geochemistry in waste rock dumps. A set of models is required which together will predict the likely pollutant release for different dump designs. The models must address all of the important processes that occur in dumps, including unsaturated/saturated water flow, oxygen supply, geochemistry and perhaps microbiology.

Use of such models requires data on the properties of the dump materials, which must be obtained by measurements on real dumps. Finally the predictions of models must to be validated against data from real dumps.

6.2 DRAFT CRITERIA FOR MANAGEMENT OF REACTIVE WASTE ROCK

1. Characterise the different waste rock zones around the pit.
2. Determine effect of compaction and weathering on material properties of different rock types.
3. Design a dump structure which will limit access of oxygen and water to the reactive material and which is stable against erosion forces.
4. Predict long term behaviour of dump and pollutant load. Carry out an error analysis which considers the effect of uncertainties in design parameters and material properties on the predicted pollutant load.
5. Establish that predicted pollutant loads meet regulations.

7. REFERENCES

AGPS [1974] - Mine waste pollution of the Molonglo River, Final report on remedial measures. Joint Government Technical Committee, June 1974, Australian Government Publishing Service, Canberra.

Allen C G [1985] - The rehabilitation of Rum Jungle. Proceedings of the North Australian Mine Rehabilitation Workshop No 9, June 1985 Weipa, Queensland, Australia, Comalco, pp 197-215.

Andersen J E, Herwig G L and Moffitt R B [1966] - Heap leaching at Rum Jungle. Australian Mining, 15 April, 35-39.

Andersen J E [1980] - Mine dump vegetation: practise and problems. In: Biochemistry of Ancient and Modern Environments, (eds) P A Trudinger, M R Walter and B J Ralph, Australian Academy of Science & Springer-Verlag. pp 713-718

Andersen J E [1981] - Rehabilitation of residue dumps containing heavy metal sulphides. 1981 AMIC Environmental Workshop.

Andersen J E and Allman M B [1968] - Some operational aspects of heap leaching at Rum Jungle. Proc Australasian Institution Mining and Metallurgy, **225**, 27-31.

Ayre M W D and Hartley D G [1986] - The planned closure of Mt Lyell and its environmental and social implications. Proc AMIC Environmental Workshop, Launceston Oct 1986. Australian Mining Industry Council, Canberra, pp 306-323.

Babij T, Goodman A E, Khalid A M and Ralph B J [1981] - Microbial ecology of Rum Jungle, Part III: leaching behaviour of sulphidic waste material under controlled conditions. Australian Atomic Energy Commission (now Australian Nuclear Science and Technology Organisation) AAEC/E520.

Bell L C and Jones C J [1987] - Rehabilitation of base metal tailings - Australian case study. In: Mining Rehabilitation '87 (ed) T Farrell, Australian Mining Industry Council, Dickson, ACT, pp 88-102

Bell P R F, Greenfield P F, Kelly G J and Krol A A [1982] - Treatment of acid mine drainage. Environmental Workshop, Darwin, Sept 1982, Australian Mining Industry Council, Dickson, ACT pp 248-271.

Bennett J W, Gibson D K, Harries J R, Pantelis G, Ritchie A I M, Verhoeven T J, Alcock J, Henkel and Applegate R J [1989b] - Rehabilitation of the Rum Jungle Mine site. Proc. Conf. on Reclamation - a Global Perspective, Calgary, Alberta, August 27-31 1989, vol 1, pp 104-108.

Bennett J W, Harries J R, and Ritchie A I M [1988a] - Monitoring the rehabilitated waste rock dumps at the Rum Jungle mine site. Proc AMIC Environmental Workshop, Vol I, Darwin 1988. Australian Mining Industry Council, Dickson, ACT, pp 346-355.

Bennett J W, Harries J R, and Ritchie A I M [1988b] - Rehabilitation of waste rock dumps at the Rum Jungle site. Proceedings Conf Mine Drainage and Surface Mine Reclamation, Pittsburgh, April 19-21, US Bureau Mines Information Circular 9183.

Bennett J W, Harries J R, Pantelis G and Ritchie A I M [1989a] - Limitations on pyrite oxidation rates in dumps set by air transport mechanisms. International Symposium on Biohydrometallurgy, Jacksons Holes Wyoming, 13-18 Aug.

Blesing N V, Lackey J A and Spry A H [1975] - Rehabilitation of an abandoned mine site. In: Minerals and the Environment, (ed) Jones M J, Proc Intl Symp London, 4-7 June 1974, Institute of Mining and Metallurgy London, 341-368.

Box J C and Prosser A P [1986] - A general model for the reaction of several minerals and several reagents in heap and dump leaching. *Hydrometallurgy* **16**, 77-92.

Braun R L, Lewis A E and Wadsworth M E [1974]. In-place leaching of primary sulphide ores: laboratory leaching data and kinetics model. In: Solution Mining Symposium 1974, ed F F Aplan, New York AIME p 295-323.

Broman P G [1988] - Sulphidic mine tailings deposits in Sweden. Proceedings of International Conference of Environmental Problems from Metal Mines, 20-24 June 1988, Røros, Norway.

Brooks K, Holland K and Whitfield L [1989] - Mining rehabilitation and environmental management plans, a strategy for environmental management and review in New South Wales. Environmental Workshop, Ballarat, Vic, Australia, Oct 1989, Australian Mining Industry Council, Dickson, ACT, Australia pp 179-187.

Brooks K A and McIlveen G R [1988] - The impact of a derelict base metal mine on the aquatic environment. Third International Mine Water Congress, Melbourne, Australia, October 1988. Australasian Institute of Mining and Metallurgy, Parkville Victoria, pp 793-802.

Brooks K A and Summerhayes G J [1983] - Water Pollution Hazard Investigations at the Cowarra Mine, near Bredbo. Department of Mineral Resources, Sydney, Geological Survey Report GS 1983/228

Chesnut W S [1972] - Preliminary geological investigation of the nature and composition of mine tailings in the vicinity of Captains Flat. Department of Mines, Sydney, Geological Survey Report GS1972/402.

Collin M and Rasmuson A [1986] - Distribution and flow of water in unsaturated layered cover materials for waste rock. National Swedish Environmental Protection Board Report 3088

Collin M and Rasmuson A [1988] - A comparison of gas diffusivity models for unsaturated porous media. *Soil Science Society America Journal*, **52**, 1559-1565.

Corkery R W [1977] - Environmental monitoring at mining localities in New South Wales. Department of Mines, Sydney, Geological Survey Report GS1977/279.

Craze B [1977a] - Restoration of Captains Flat. *J Soil Conservation Service of NSW*, **33**, No. 2, April, 98-105.

Craze B [1977b] - Revegetation problems, Captains Flat mining area. *J Soil Conservation Service of NSW*, **33**, No. 3, July, 190-199.

Craze B [1977c] - Heavy Metal Toxicity in Captains Flat. J Soil Conservation Service of NSW, 33, No. 4, October, 265-271.

Craze B [1980] - Mine waste pollution control at Captains Flat, New South Wales. In: Biochemistry of Ancient and Modern Environments, (eds) P A Trudinger, M R Walter and B J Ralph, Australian Academy of Science & Springer-Verlag. pp 705-712

Davis G B, Doherty G and Ritchie A I M [1986] - A model of oxidation in pyritic mine wastes: part 2: comparison of numerical and approximate solutions. Appl Math Modelling 10 323-330.

Davis G B and Ritchie A I M [1986] - A model of oxidation in pyritic mine wastes: part 1: equations and approximate solution. Appl Math Modelling 10 314-322.

Davis G B and Ritchie A I M [1987] - A model of oxidation in pyritic mine wastes: part 3: import of particle size distribution. Appl Math Modelling 11 417-422.

Davy D R (ed) [1975] - Rum Jungle environmental studies. Australian Atomic Energy Commission, Sydney, AAEC/E365.

Derry R and Whittemore [1983] - Modelling the in-situ bioleaching of copper sulphide ore from Avoca, Ireland. Progress in Biohydrometallurgy, Proceedings of conference in Cagliari, May 1983, pp 243-263.

Fitzgerald B J and Haldane A D [1977] - Mine waste pollution study - Lake Burley Griffin catchment. Seventh Australian Waste Water Association conference September 1977, p 570-583.

Foskett W E [1971] - Environmental assessment of "Woodlawn" mining project near Tarago. Department of Mines, Sydney, Geological Survey Report GS1971/401.

Foskett W E [1976] - Rehabilitation of derelict mines sites. Department of Mines, Sydney, Geological Survey Report GS1976/103.

Gibson D K and Pantelis G [1988] - Forecasting the effect of mine site rehabilitation works on local groundwater quality. In: Proceedings 1988 Mine Drainage and Surface Mine Reclamation Conference, Amer. Soc. Surface Min. and Reclam. and U.S. Dept of Interior.

Gibson D K and Pantelis G [1989] - Ground water studies at Rum Jungle. Australian Nuclear Science and Technology Organisation ANSTO/C88.

Goodman A E, Khalid A M and Ralph B J [1981a] - Microbial ecology of Rum Jungle, Part I: environmental study of sulphidic overburden dumps, experimental heap-leach piles and tailings dam area. Australian Atomic Energy Commission (now Australian Nuclear Science and Technology Organisation) AAEC/E531.

Goodman A E, Khalid A M and Ralph B J [1981b] - Microbial ecology of Rum Jungle, Part II: environmental study of two flooded opencuts and smaller associated water bodies. Australian Atomic Energy Commission (now Australian Nuclear Science and Technology Organisation) AAEC/E527.

Gottschlich D E, Bell P R F and Greenfield [1986] - Estimating the rate of generation of acid drainage products in coal storage heaps. Environmental Technology Letters, 7, 1-12.

Green R E and Corey J C [1971] - Calculation of hydraulic conductivity: a further evaluation of some predictive methods. Soil Science Society of America Proc 35, 3

Greenwood E A N and Netchaef P [1988] - Quantitive dewatering of mine dumps by vegetation - a proposition. Third Intl Mine Water Congress, Melbourne, Australia, October 1988, p 785-791

- Hammond R [1982] - Tailings dam seepage at Woodlawn Mines - a case study. Environmental Workshop, Darwin, Sept 1982, Australian Mining Industry Council, Dickson, ACT pp 205-257
- Hannan J C [1983] - The prospects for long term revegetation of acid coal mine spoil and reject material. Northern Australian Mine Rehabilitation Workshop, Collinsville Queensland.
- Hannan J C [1984] - Mine rehabilitation: a handbook for the Coal Mining industry. New South Wales Coal Association, Sydney, Australia.
- Harries J R and Ritchie A I M [1980] - The use of temperature profiles to estimate the pyritic oxidation rate in a waste rock dump from an open cut mine. Water Air Soil Pollut. 15:405-423.
- Harries J R and Ritchie A I M [1985] - Pore gas composition in waste rock dumps undergoing pyritic oxidation. Soil Sci. 140:143-152.
- Harries J R and Ritchie A I M [1987] - The effect of rehabilitation on the rate of oxidation of pyrite in a mine waste rock dump. Environ. Geochem. Health 9:27-36.
- Harris J A [1969] - Development of a theoretical approach to the heap leaching of copper sulphide ores. Proc Aust Inst Min Met 230, 81-92.
- Hendy N A [1987] - Isolation of thermophilic iron-oxidising bacteria from sulphidic waste rock. J Industrial Microbiolgy 1 389-392.
- Hill P R H [1986] - Mine Closures in South Australia. Proc AMIC Environmental Workshop, Launceston Oct 1986. Australian Mining Industry Council, Canberra, pp 298-304.
- Hogg D [1990] - Evaluation of the remedial work at Captains Flat. in preparation.
- Iversen E [1988] - Lime treatment of waste dumps at Kjøli mines, Norway. Proceedings of International Conference of Environmental Problems from Metal Mines, 20-24 June 1988, Røros, Norway.
- Jacobson G and Sparksman G F [1988] - Acid drainage at Captains Flat, New South Wales. BMR J Australian Geology & Geophysics, 10, 391-393.
- Jeffery J J, Lawson R T, Sarbutt and Bowdler R J [1988] - Sulphide oxidation in pyrite overburden, a controlled leach column study. Third International Mine Water Congress, Melbourne Australia, pp 157-164.
- Jeffery J J, Marshman N A and Meynink W J C [1986] - A physical and chemical appreciation of the impact of discharge of mine wastes on a tropical river system. Proc AMIC Environmental Workshop, Launceston Oct 1986. Australian Mining Industry Council, Canberra, pp 485-503.
- John C D [1988] - Residue and cyanide management at the Boddington Gold Mine. Proc AMIC Environmental Workshop, Vol II, Darwin 1988. Australian Mining Industry Council, Dickson, ACT, pp 214-221.
- Jones C J [1986] - Strategies for Mount Isa lease rehabilitation. Proc AMIC Environmental Workshop, Launceston Oct 1986. Australian Mining Industry Council, Canberra, pp 181-192.
- Joyce C and Ryan P [1989] - The integration of the environment with engineering - case studies in mining. Environmental Workshop, Ballarat, Vic, Australia, Oct 1989, Australian Mining Industry Council, Dickson, ACT, Australia pp 146-164.
- Keane P A [1977] - Captains Flat mined area. J Soil Conservation Service of NSW, 33, No. 2, April, 92-97.

Kelly R [1987] - Rehabilitation of mined lands in the central Queensland coalfields. In: Mining Rehabilitation '87 (ed) T Farrell, Australian Mining Industry Council, Dickson, ACT, pp 120-126

Kuzmanovska C B, Poposka F A and Grizo A N [1986] - Leaching of low grade chalcocite-covellite in sulphuric acid solutions by percolation. Application of the moving reaction zone model. *J Serbian Chemical Society* **51** 55-62.

Lake P S, Coleman D, Mills B, and Norris R [1977] - A reconnaissance of pollution of the King River in the Comstock-Crotty Area, West Tasmania. In: Banks M R and Firkpatrick J B (eds), *Landscape and Man, Proceedings of Seminar, Royal Society Tasmania, June 1977*

Lowson R T, Evans J V, Sarbutt J V, Sinclair G and Folk E [1987] - the Rum Jungle tailings dam - chemical profile of the subsoil. Australian Nuclear Science and Technology Organisation ANSTO/E663.

Magnusson M and Rasmuson A [1984] - Gas diffusion in unsaturated porous media. National Swedish Environmental Protection Board report SNV PM 1843.

Miller S, Davies I, Murray G and Dale S [1985] - Predicting leachate quality from waste rock and tailings at a proposed gold mine, Waiki, NZ. Proc AMIC Environmental Workshop, Townsville Sept 1985. Australian Mining Industry Council, Canberra, p 395 (abstract only which gives no details about site)

Miller S and Murray G [1988] - Prediction of time dependent factors in acid mine drainage. Third International Mine Water Congress, Melbourne Australia, pp 165-172.

Millington R J and Shearer R C [1971] - Diffusion in aggregated porous media. *Soil Science* **111**, 372-378.

Murray G C, Grundy M J, Bell L C and Whiteman P C [1983] - Rehabilitation strategies at the Collinsville open-cut coal mine. Northern Australia Rehabilitation Workshop, Collinsville, Queensland, 13-18 May 1983.

Norris R H, Lake P S and Swain R [1980] Ecological effects of mine effluents on the South Esk river, North-eastern Tasmania - I. Study area and basic water characteristics. *Aust. J. Mar. Freshwater Res.* **31**, 817-827

Norris R H Swain R and Lake P S [1981] Ecological effects of mine effluents on the South Esk river, North-eastern Tasmania - II. Trace metals. *Aust. J. Mar. Freshwater Res.* **32**, 165-173

NTDME 1986. The Rum Jungle Rehabilitation Project, Final Project Report. 346 p. Northern Territory Department of Mines and Energy, Darwin, Australia, June 1986.

Pantelis G [1987] - Modelling water and contaminant transport in the Rum Jungle mine Overburden Heaps. Australian Atomic Energy Commission (now Australian Nuclear Science and Technology Organisation) AAEC/E653.

Pantelis G [1987] - Modelling water and contaminant transport in the Rum Jungle mine overburden heaps. Australian Atomic Energy Commission (now Australian Nuclear Science and Technology Organisation) AAEC/E653.

Prosser A P, Box J C and Beavis F C [1981] - Comprehensive models of in-situ and dump leaching. Australasian Institute of Mining and Metallurgy Conference Sydney NSW July, 317-327.

Prosser A P and Box J C [1983] - Simulation of the mineralogical and chemical aspects of heap and dump leaching as an aid to ore-process evaluation. *Australasian Institute of Mining and Metallurgy* **34** (Computers in Mining) 171-178.

- Prosser A P and Mansor A N [1989?] - Simulation of copper heap leaching as an aid to design and operation. University of New South Wales, preprint
- Rasmuson A and Collin M [1988] - Mathematical modelling of water and oxygen transport in layered soil covers for deposits of pyritic mine tailings. Proceedings of International Conference of Environmental Problems from Metal Mines, 20-24 June 1988, Røros, Norway.
- Ritchie A I M [1977] - Heap leaching: a gas diffusion rate-limited model. Australian Atomic Energy Commission (now Australian Nuclear Science and Technology Organisation) AAEC/E429.
- Roach G I D and Prosser A P [1978]. Prediction of rates of chemical processes for treatment of low-grade materials: theory and tests for mass-transfer controlled reactions. Trans IMM 1978, 87, C129-C138.
- Ruppin P [1987] - Tailings management at Bougainville Copper Limited. In: Mining Rehabilitation '87 (ed) T Farrell, Australian Mining Industry Council, Dickson, ACT, pp 71-87
- Ryan P [1987a] - Rehabilitation of the Rum Jungle Mining area, Northern Territory. In: Mining Rehabilitation '87 (ed) T Farrell, Australian Mining Industry Council, Dickson, ACT, pp 184-194
- Ryan P [1987b] - Rum Jungle mine rehabilitation, Northern Territory. J. Soil Conservation, New South Wales, 43(1), 18-27.
- Södermark B and Lundgren T [1988] - The Bersbo project - the first full scale attempt to control acid mine drainage in Sweden. Proceedings of International Conference of Environmental Problems from Metal Mines, 20-24 June 1988, Røros, Norway. pp
- Southern P [1987] - Rehabilitation of metalliferous mining wastes at Woodlawn Mines, NSW. In: Mining Rehabilitation '87 (ed) T Farrell, Australian Mining Industry Council, Dickson, ACT, pp 50-60.
- Southern P [1988] - Water management at Woodlawn Mine. private communication.
- Summerhayes G J [1986] - Water pollution hazard investigations at the Cowarra mine, near Bredbo, NSW. Proc AMIC Environmental Workshop, Launceston Oct 1986. Australian Mining Industry Council, Canberra, pp 518-532.
- Verhoeven T J [1988] - Rum Jungle Rehabilitation Project Site Management Plan. Report prepared for Rum Jungle Monitoring Committee, Power and Water Authority, June 1988.
- Webber J [1986] - The implications of the closure of the Hercules mine, Tasmania. Proc AMIC Environmental Workshop, Launceston Oct 1986. Australian Mining Industry Council, Canberra, pp 324-340.
- Whittemore R G [1981] - The modelling of vat, heap, dump and in-situ leaching systems. Warren Springs Laboratory Report No. LR 381 ME.

APPENDIX A.

DESCRIPTION OF MINE SITES WHERE WASTE ROCK IS A SOURCE OF ACID DRAINAGE

A1. AUSTRALIA

A1.1 Ardelthan, NSW

Mining for tin at Ardelthan began in early 1918 and in 1977 the mine was producing about 2200 tonnes of tin concentrate from an open cut. The waste rock/mullock dumps releases contaminated water containing copper, zinc and cadmium [Corkery 1977]. The mine closed in 1986 although there was a brief period of operation when tailings were processed in 1988 [Brooks et al 1989]

Earthworks have been constructed to separate clean and dirty water. Dirty water, mainly seepage water from the dumps, is fed to a series of evaporation ponds. The ponds are designed to release water off site only at times of very heavy rainfall when the receiving creek will have a high flow.

A1.2 Broken Hill, NSW

There are several metalliferous waste rock dumps at Broken Hill but the low annual rainfall (230 mm) means that seepage from dumps and acid generation are usually not problems. Most mining at Broken Hill is underground.

The lead-zinc mine at Zinc Corporation has three major residue dumps containing a sandy loam material with average sulphur content of 1.08%. Material in the oldest dump was being retreated in 1981. The other two had been or were being rehabilitated by covering the top with calcareous soil and trickle irrigating with effluent water from the local city sewerage treatment plant. Trees and shrubs were planted on the dump. Grasses and herbs were found to rapidly pioneer the area once the ground was wet [Andersen 1980, 1981].

The material in the dumps at the North Broken Hill Limited mine has a sulphur content of 0.6%. The site is remote from residential areas and is downwind of them. The main aim of rehabilitation was dust suppression and stability of the dumps against high winds and rainfall events. The dumps are covered with a layer of limestone or highly calcareous soil and treated with sewerage sludge and some fertiliser [Andersen 1981].

A1.3 Brukunga Pyrite, SA

Nairne Pyrite operated the Brukunga pyrite mine between 1955 and 1972 to produce sulphuric acid. The ore consisted mainly of pyrite and pyrrhotite (totalling 15%) in a conformable pyritic metasediment. Coarse waste rock produced during the excavation of the mine was dumped in two large waste rock dumps (700-800 m long and 30-40 m high and containing about 1×10^7 m³ of material). About 2000 m³ per year of highly polluted water seeped from the base of the dumps. Other important and larger sources of polluted water were the tailings dam and seepages in the open cut quarry. The annual load of pollutants in the local Dawesley Creek were 150 t iron, 200 t aluminium, 30 t zinc, 0.7 t copper and 0.15 t cadmium [Blesing et al 1975].

In 1975 an acid water collection system was installed and a lime neutralising plant was commissioned in 1979. In the treatment plant the pH of the seepage water is raised from 2.4 to 6 or 9 using waste lime. Running costs were \$(Aust)240 000 per year. Brukunga was considered the most difficult problem facing rehabilitation experts in South Australia. Further containment options were being implemented but the long term solutions were out of the financial reach of the State Government at that time [Hill 1986].

The following options have been considered for the waste rock dump:

- continue treatment of seepage water,
- cover dumps with soil (there is no available clay within 30 km of the site), and
- cover dumps with soil and grow pine trees (test have shown that pine trees can grow and it is hoped that a ground covering of pine needles will stop oxygen getting into the dump).

Consideration has been given to using organic rubbish to produce a reducing atmosphere and thereby control ingress of oxygen. Septic tank rubbish has been used with some success in sections on the pit and overburden dump. However local town of Mt Barker does not generate enough septic tank rubbish to completely treat the site.

A wetland has also been considered but suitable areas downstream of the mine are prime farm land.

A1.4 Captains Flat, NSW

Mining commenced at Captains Flat in 1874 and continued periodical until 1962. Over 4 million tonne of ore was milled to produce zinc, pyrite, lead, copper and gold. About 2.5 million tonne of mine waste was piled in dumps covering an area of 15 hectares. Underground workings which extended to a depth of 600 m were incompletely backfilled with quarried rock.

The wastes were stored in above ground dumps at two locations, the northern dumps and the southern dumps. The fine fraction of the wastes, or slimes, were pumped to earthen dams which were continually built up with fresh material. The coarser fraction was dumped in the northern solids dump. The wastes in the northern area, about 90% of the waste, contain about 1.1 % zinc, 0.5 % lead, 0.05 % copper and 17.4 % sulphur (as S, equivalent to 32.5 % FeS₂) [AGPS 1974, Craze 1977abc, Corkery 1977, Fitzgerald and Haldane 1977, Keane 1977, Craze 1980, Jacobson and Sparksman 1988, Hogg 1990].

The operation of the mine and the dispersion of tailings resulted in high levels of zinc and other heavy metals in the Molonglo River between Captains Flat and Canberra. The source of this pollution was erosion and leaching from the dumps, flow of water through the mine, and accumulated bed sediments in the creeks and Molonglo river. In 1939 one of the dams burst and released polluted water and fine tailings into the river and in 1943 one of the dumps collapsed and 30 000 m³ of fine tailings into the town water supply reservoir. Then in 1945 a flood in the Molonglo river spread the polluted sediments downstream [Hogg 1989]

Monitoring of zinc loads found that 42 per cent of the dissolved and suspended zinc in the river arose from erosion and leaching of the dumps, 20 per cent from mine water outflow from a spring, 30 to 35 per cent from bed load already in the creek and river and 3 to 8 per cent from other minor sources.

The rehabilitation program undertaken at Captains Flat and the results of monitoring are discussed in Section 2.2.3.

A1.5 Cessnock region, NSW

Mining in the Cessnock area commenced in the late 1800s primarily along the Greta seam. The area has been extensively mined by underground and open cut methods (mines include Aberdare East, Aberdare North, Pelton, Bellbird, Neath and Caledon). The Greta coal seam has a high pyrite content (0.1 to 0.54% pyrite S) and produces acid drainage problems at a number of abandoned and operating sites. Pelton colliery commenced operations in 1916 and, in 1977, water leaving the site had pH 3.1 [Corkery 1977].

In the early 1970s there were only a few active mines but a large number of disused sites. At that time the various companies involved initiated a program of progressively rehabilitating the disused sites. By 1983 some 14 sites were in various stages of rehabilitation, and there were another 5 or 6 sites on which work had not started [Hannan 1983].

The abandoned Caledon mine was a small site about 30 ha which had been mined by truck and shovel methods to a maximum depth of about 20 m during the 1950s. The area had been progressively backfilled leaving a void of up to 15 m when the site was abandoned. The backfilled spoil was highly acidic, devoid of vegetation and actively eroding [Hannan 1983].

The abandoned Maitland Main and Washery in the south Maitland district produced extremely poor water, pH 3.0 1 km downstream of the site [Corkery 1977]. It was primarily an underground mine but in the later stages of operation several small open cuts were excavated. The site was worked continuously from the early 1900s to 1972. The pH of the runoff water leaving the site fell to 1.7 during dry periods and rarely rose above 3.0. The principal sources of acid water were runoff and leachate from the stockpiling areas and washery reject emplacements and ponds [Hannan 1983].

The Abermain No. 1 colliery was an underground mine and there were no associated open cut operations. The sole source of acid drainage was carbonaceous material and washery reject emplaced or scattered over the surface.

Rehabilitation works carried out to rehabilitate the Cessnock mines are discussed in Section 5.2.1.

A1.6 Collinsville, Qld

Coal mining has been carried out by opencut and underground techniques at the Collinsville site since the 1920s. Collinsville receives an annual rainfall of 0.74 m and evaporation exceeds precipitation in all months except February. Pyrite occurs above the coal seams at all levels except for the weathered zone 3 to 9 m beneath the surface. The major problems include acid runoff, erosion and lack of vegetation. Initial studies were directed at revegetation options. Murray et al [1983] carried out field trials to investigate the effect of different soil depths and lime rates on the growth of pasture, shrub and tree species.

A1.7 Conrad, NSW

The Conrad mine is an abandoned mine located about 21 km SSW of Inverell which produced zinc, copper and lead-silver concentrates. There are three main waste rock dumps at the site with sulphur contents of 3.1, 1.4 and 0.9 %. The dump with the highest sulphur content, which contains 22 000 tonnes of a fine grained tailings, is undergoing extensive active leaching of toxic metals. Under low flow conditions mine water from the flooded workings was the dominant pollutant source however leachate and runoff from the dumps were significant under and after wet conditions [Brooks and McIlveen 1988]

A1.8 Mt Lyell, Tas

The Mt Lyell is a copper mine which commenced operations in 1896 using open cut methods. Both open cut and underground techniques have been used over the years. A total of 100 million tonnes of ore have been treated. It is planned to close the mine in 1989 [Ayre and Hartley 1986].

West Lyell was the largest open cut and at completion in 1972 had supplied 57 million tonnes of ore and produced 47 million tonnes of waste. Mining the ore body continued underground after this date.

Surface water percolating through mine workings and waste dumps flows into the King and Queen rivers. Chief contaminants are iron and copper, together with minor zinc and low lead values. Cadmium, arsenic and mercury have been traced but are usually below detection levels. Closure of the mine will stop the release of tailings into the Queen river and result in a major improvement in the visual appearance of the water. However there is not expected to be much improvement in levels of heavy metals or pH [Ayre and Hartley 1986].

The small Cape Horn orebody produces water relatively high in copper and low in iron. It is planned to extract copper by cementation and reduce the pollutant potential of this stream [Ayre and Hartley 1986].

Two mines operated in the upper tributaries of the King river; the Comstock mine between 1901 to 1942 and the Iron Blow mine from 1889 to 1929. Pollution in the King River from these sites has been investigated by Lake et al [1977].

A1.9 Ottery, near Tent Hill NSW

Underground operations at the Ottery mine were carried out between 1881 and 1958. The mine produced about 2700 tonnes of tin oxide concentrate and 1900 tonnes of arsenic trioxide. Water from an adit had more than 5 ppm arsenic. The water pollution problem was aggravated by runoff during wet weather across mill tailing dams, mineralised mullock heaps and mine workings spread over about 8 ha. Rehabilitation consisted of sealing adits, and building diversion drainage channels around the site [Foskett 1976].

The quality of water close to the mine site was very poor with pH 2.5 and high cadmium, copper and zinc [Corkery 1977].

A1.10 Rum Jungle, NT

Open cut mining to extract uranium/copper ore was carried out at Rum Jungle in the Northern Territory, Australia, between 1954 and 1964. The East Branch of the Finnis river flows through the site. Three main ore bodies were mined using open cut techniques. When the site was abandoned in 1971 there were three water-filled open cuts, four waste rock heaps (dumps) containing pyritic material, a tailings disposal area and a pile of low grade ore where an attempt had been made to extract copper by heap leaching.

The largest of the open cuts was White's which was completed in 1958 at a depth of just over 100 m. The waste rock from this open cut was formed into a dump the top of which was some 13-18 m above the original ground surface and was generally smooth and well graded. The top surface sloped gently down towards the centre of the dump where a main drainage channel collected run-off. The sides were steep (about 30°) and made up about 25 percent of the 26.4 ha area of the whole dump.

White's overburden dump contains about seven million tonnes of material consisting of carbonaceous shales and graphitic schists with an average sulphur content of 3 per cent, mainly in the form of pyrite. Some dolomite is interspersed through the dump.

Dyson's ore body was mined between 1957 and 1958. The overburden dump was similar in shape and composition to White's but smaller with a content of about two million tonnes and an area of 9 ha.

In 1963, the Intermediate ore body, which was only about 500 m west of White's ore body, was also extracted by an open cut operation. This ore body was mined for its copper content not its uranium content. The waste rock dump had an area of about 7 ha and contained about two million tonnes of material with a similar composition to that in White's dump. The oxide and low grade sulphide ore from the ore body were placed in a heap leach pile where an attempt was made to extract the copper by heap-leaching techniques.

Significant pollution of ground and surface waters with acid and heavy metals was apparent when mining operations ceased. An extensive survey in 1973-74 [Davy, 1975] showed that the major sources of pollution were the waste rock dumps and the heap leach pile, copper being the main heavy metal pollutant.

The Rum Jungle mine site was rehabilitated between 1983 and 1986 by the NT Department of Mines and Energy [NTDME] using money provided by the Commonwealth Government [Allen 1985, NTDME 1986, Bennett et al 1988, 1989]. The total cost was \$(Aust)18.6 million.

The objectives stated in the 1982 agreement between the Northern Territory and the Commonwealth Governments were: a major reduction in pollution in water courses feeding the East Branch of the Finnis River and in particular the reduction of the annual average releases of copper, zinc and manganese into the river by 70 per cent, 70 per cent and 56 per cent respectively; a reduction in public health hazards and in radiation levels at the site; a reduction of pollution in the water contained in White's and Intermediate open cuts; and aesthetic improvement including revegetation.

The rehabilitation strategy for the waste rock dumps is discussed in Section 2.2.4.

There has been an extensive monitoring program at Rum Jungle. Results of monitoring the waste rock dumps is discussed in Section 2.2.4 and 3.2.2.

The revegetation program on the waste rock dumps is describes by Ryan [1987a,b] and NTDME [1988].

A1.11 Woodlawn, NSW

Woodlawn is a lead-zinc-copper-silver mine west of Tarago which commenced operations in 1978. Initially an open cut was used to recover the ore and in 1987 most of the operations were underground. The main ore body is more than 75% sulphides, with pyrite predominating [Register Aust Mining 1988/89].

The mine is located on the boundary of two environmentally sensitive water catchments. To the east the catchment flows into the Warragamba dam which supplied most of the water for the city of Sydney, and to the west water leaving the site flows into the land locked Lake George. A total containment water policy has been implemented to prevent the release of contaminated water [Southern 1987, 1988]. Hammond [1982] discusses seepage problems from the tailings dam.

The dump containing the waste rock from the open cut is located on the western side of the site. The original EIS did not foresee the problem of acid leachate from the waste rock dump [quoted by Southern], although Foskett [1971] did note that pollution of creeks could occur and that this effect would 'be largely influenced by the pyrite and other sulphide content of the dumped material, and will need to be evaluated as the project develops'. Corkery [1977] also provides water quality data at an early stage in the mine development. Seepage from the dumps turned acid nearly four years after the initial placement of waste rock.

At the completion of open cut operation the waste rock dump covering about 92 ha and containing about 80 000 000 tonnes of overburden with sulphide levels of approximately 6 to 7 % mainly as pyrite. The dump was constructed in 15 m lifts by back dumping, resulting in gently sloping benches of 1 to 3% and steep batters of 36°. Some of the overburden has <1% sulphides but the nature of the mining operation precluded preferential placement of the low sulphide material. There has been little voluntary establishment of plants. Typical leachate from the dump has a pH of 2.8 and contains 290 mg L⁻¹ copper, 6000 mg L⁻¹ zinc and 5000 mg l⁻¹ magnesium .

Trials to investigate rehabilitation techniques are discussed in Sections 2.2.1, 3.2.1 and 5.2.2.

By 1989 all mining was underground and it was planned to rehabilitate the dump by compacting the top layer and covering with soil.

Jeffery et al [1988] carried out a long-term leaching study using waste rock from Woodlawn in large columns (2.3 m high and 0.3 m diameter). They found that the leach characteristics could not be properly assessed using short term bench-top test, that diffusion of oxygen was the major factor limiting the rate of oxidation, that particle size distribution was important, and that the rates of release of ions did not necessarily relate to oxidation rates in the column.

A2. PAPUA NEW GUINEA

A2.1 Bougainville Copper

Bougainville Copper Limited mines a large low grade, porphyry copper deposit on Bougainville Island in Papua New Guinea. The latitude of the site is 6°S, and it receives a 4.5 m annual rainfall [Jeffery et al 1986]

In 1989, the waste rock dumps covered an area of about 4 km² and contained about 430 million tonnes of rock averaging 0.2% copper and 3 to 5 % sulphur/sulphide. This means the dumps contain about 800 000 tonnes of copper. The feasibility of recovering some of the contained 800 000 tonnes of copper from the dumps by leaching with sulphuric acid is being investigated.

The dumps drain into the Jaba/Kawerong river system which flows 35 km to the sea. Mean discharge from the dumps is about 0.5 cumecs and averages about 70 mg/L copper at a pH of 4.5 to 5.5. The concentration of other metals is not environmentally significant [Jeffery et al 1986].

Tailings are the other major waste stream. The tailings are highly alkaline and while the tailings were discharged into the river they neutralised the seepage from the dumps and absorb the metal ions. About 40% of the tailings have been retained in the river valley while the rest has formed a delta in Empress Augusta Bay. Plans have been developed for rehabilitating the river valley [Ruppin 1987].



APPENDIX B.

Acid generation in waste rock at mine sites - the situation in Sweden

Background

The history of mining in Sweden goes back at least 1000 years. The number of abandoned mines is not known but is probably well over one thousand. Many of these mines are very small. At least three old mines are known to be major sources of heavy-metal emissions to water: the Falun mine, the Garpenberg mine and the Bersbo mine. The Falun and Garpenberg mines are still in operation.

At a number of old mine sites waste-rock dumps have been identified as a significant local problem.

There are about 20 sulphide mines in operation in Sweden today, of which four are open pits. The Aitik open pit mine is Swedens biggest sulphide mine, with a yearly ore-production of 12 Mtonnes. This is now under expansion to 14 Mtonnes.

A summary of the investigations and actions that have been performed in the Falun and Bersbo mining areas has been presented in Ulf Qvarfort's paper.

Legal Aspects

The Environment Protection Act states that anyone who performs any kind of polluting activity is obliged to take necessary remedial actions to avoid future negative effects on the environment. This pertains to all activities that have taken place since the Act was passed in 1969. The Act cannot be applied to activities that were terminated before 1969.

The Environmental Protection Board has estimated the cost of remedial actions at sites where no company or other party can be made responsible to be approximately 140-150 M SEK per year (22-24 M US \$) during a period of 15-20 years. These figures include all kinds of contaminated areas, on land and in water. The Board proposed that the state should establish a fund for the purpose of financing these kinds of actions. It was suggested that the fund should be financed by an "environmental fee" on emissions of certain pollutants. The Government decided not to establish any fund for the time being, but provided the Board with an allowance of 50 M SEK (8 M US \$) to be used for investigations and remedial actions in contaminated areas.

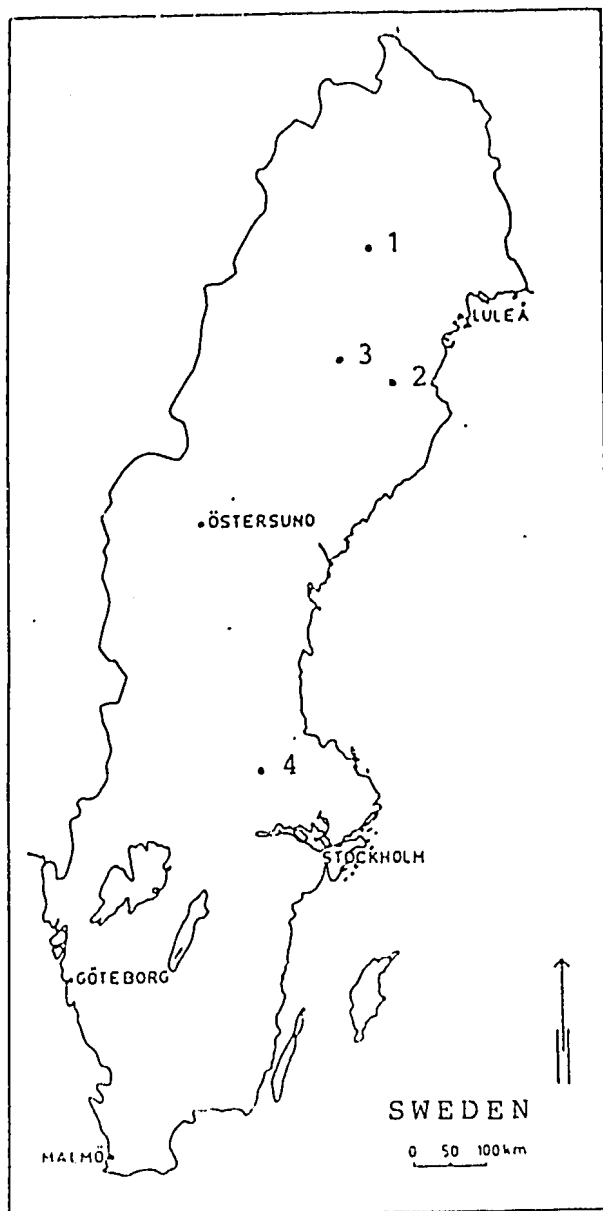
This work will be evaluated after three years (1991). After that, the Government will decide how the work with old contaminated areas shall be organized and financed in the future.

Research program

The National Swedish Environmental Protection Board decided in 1983 to allocate funds for a research program on mine waste. The program was mainly focused on the problems and possible solutions at tailing ponds. Many of the results, concerning for example soil-covers, are not dependent on whether the wastes are tailings or waste-rock. Models for infiltration, unsaturated and saturated flow, and the effective oxygen diffusivity have been developed and used to evaluate different cover designs. Reports produced within the program are listed in Appendix B:1.

Site descriptions

The following sections contain a short summary of the information available about sites with acid mine drainage (AMD) generating waste-rock. The locations of the mines are shown on the map below.



1. The Aitik Mine
2. The Boliden area
3. The Kristineberg area
4. The Garpenberg area

1. The Aitik Mine

- In operation: Since June 1968
- Ore: Cu 0.38%, S 1.5%
- Production:

Ore	1968-1987	140 Mtonnes
	1968-2007	450 Mtonnes
Waste-rock	1968-1987	87 Mtonnes
	1968-2007	250 Mtonnes
- Water quality: Water in the ditches collecting drainage from the waste-rock dumps contains up to 20 mg/l of copper and has a pH below 4. This water is today used as process water in the ore dressing plant. Through recycling of this and other contaminated waters, copper emissions are held at a very low level, less than 100 kg of copper per year. These emissions will increase dramatically when operations come to an end. A moderate estimate is 10 tonnes of copper per year if no remedial actions are taken.
- Rehabilitation: The existing rehabilitation plan (from the early 1970's) doesn't consider the AMD-problem. The operating company, Boliden Mineral AB, is now obliged to investigate and to propose solutions for solving the AMD-problems in Aitik. The results shall be reported in the end of 1993. The investigations will include methods for prediction of the weathering potential of waste-rock, segregation of reactive from "harmless" waste-rock, alternative deposition strategies, possible reclamation methods, etc.

Coordination with the Canadian Mine Environment Neutral Drainage program (MEND) is desirable.

2. The Boliden area

In the Boliden area a dozen sites have been mined during this century. Five of these are still in operation. Two new mines will be opened in 1990.

The Boliden mine

- In operation: 1926-1967
- Ore: Cu/Pb/Zn
- Production: 1 M(m³) waste-rock mixed with over-burden (glacial till) has been deposited close to the mine.

- Water quality: The drainage water from the deposit contains: Cu 5-8 mg/l, Zn 50-75 mg/l and a pH of 3-4.

The last few winters the local community has been using the area as a snow-dump. This has caused an increase of drainage flow through the dump and probably an out-wash of metals and other contaminants. In 1988, grazing cattle downstream from the dump showed signs of intoxication. The investigation of the event is not yet completed, but most likely it has been caused by the emission of metals from the dump.

- Rehabilitation: No permanent remedial actions have been planned. A short-term solution that has been discussed is to collect the drainage water and to pump it either to the concentrator or to the tailings pond.

The Kankberg mine

- In operation: 1966-1969
1989-

- Production: The waste-rock has been dumped on a slope close to the open pit. The amount of waste-rock is estimated to be 0.65 Mtonnes. The sulphurcontent is high, approx. 25%.

- Drainage water quality:

Cu	0.3 - 1.5	mg/l
Zn	30 - 250	mg/l
Cd	0.1 - 0.6	mg/l
pH	2.5 - 3.5	

- Rehabilitation: One of the conditions for Boliden's license to re-open the mine was to back-fill the old waste-rock under water in the open pit.

3. The Kristineberg area

The Kimheden mine

- In operation: 1968-1975 (in shorter periods)
- Drainage water quality (1987):

Cu	3 - 19	mg/l
pH	2.8 - 4.0	

- Rehabilitation: Waste-rock (approx. 0.2 Mtonnes) and contaminated soil have been back-filled in the two open pits. To raise the ground-water table in the pits an earth-dam has been constructed in the lower end of the pit. The waste-rock is covered with 1.0 meter moraine (glacial till).

The total cost is estimated to be 850 000 SEK (135 000 US \$). The measures were completed in september 1989. It's too early to tell if the actions have been successful or not.

The Rakkejaure mine

- In operation: A short period in the late 1960's.
- Production: The ore is difficult to beneficiate due to a fine grained and complex structure.
- Drainage water quality:

Cu	1 - 3.5	mg/l
Zn	100 - 325	mg/l
Cd	0.1 - 0.3	mg/l
pH	3 - 4	

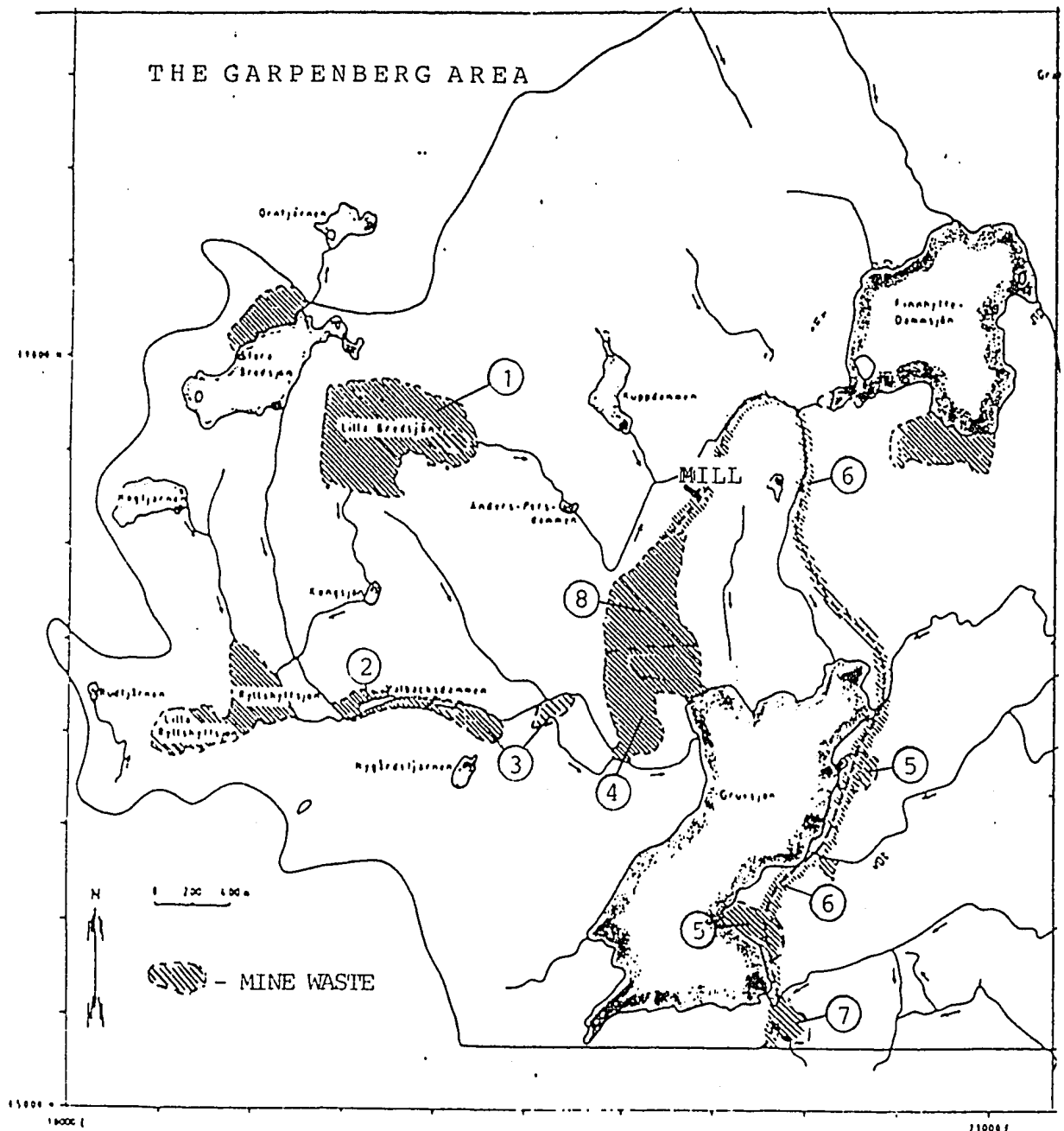
- Rehabilitation: One out of two waste rock dumps has been covered with a thin layer of moraine (glacial till). This has not had any significant effect on the drainage water quality. Since Boliden is holding a permit to perform testmining the matter of final reclamation is pending.

Meanwhile, the drainage-water from the waste-rock dumps and from the open-pit is led to a lime-treatment plant. The zinc-content of the outgoing water is less than 2 mg/l.

4. The Garpenberg area

The mining activities in this area began 1 000-1 200 years ago. Garpenberg lies within the drainage basin of the river Dalälven. The Swedish government has initiated a project aiming to reduce heavy metal emissions to the river Dalälven. The state has funded 100 M SEK (~ 16 M US \$) to finance investigations and measurements.

The mine waste deposits in the area have been inventoried. Eight possible major sources of metal emissions can be identified (see map below): five tailing impoundments ("1" - "5"), an old railway embankment constructed of slag and waste-rock ("6"), one area with slag ("7") and finally the "Odal" field with a number of old waste-rock heaps ("8").



Drainage from all the areas is to the lake "Gruvsjön". The in- and outflow of metals to the lake are given in the table below.

In- and outflow of metals to "Gruvsjön" ("the Mine Lake"), 1983, kg/year

	In	Out
Cu	1 440	400
Pb	2 040	170
Zn	47 700	36 000
Cd	75	70

Since 1988 most of the drainage water from areas "1" - "4" and "8" is collected and pumped to the new tailing pond. This has led to a significant decrease in metal emissions, as shown in the following table.

Inflow of metals to "Gruvsjön", 1988, kg/year

Cu	~ 300
Pb	~ 650
Zn	~ 10 000
Cd	~ 40

A preliminary "plan of action" has been presented by Boliden Mineral. Two principle activities are suggested: deposition under water and/or covering with 1.0 m moraine (glacial till, hydraulic conductivity $< 1 \times 10^{-7}$ m/s).

The cost to cover areas "1" - "5", the tailing ponds, with 1.0 m moraine has been estimated to be 30 M SEK (5 M US \$). The total area is 640 000 m².

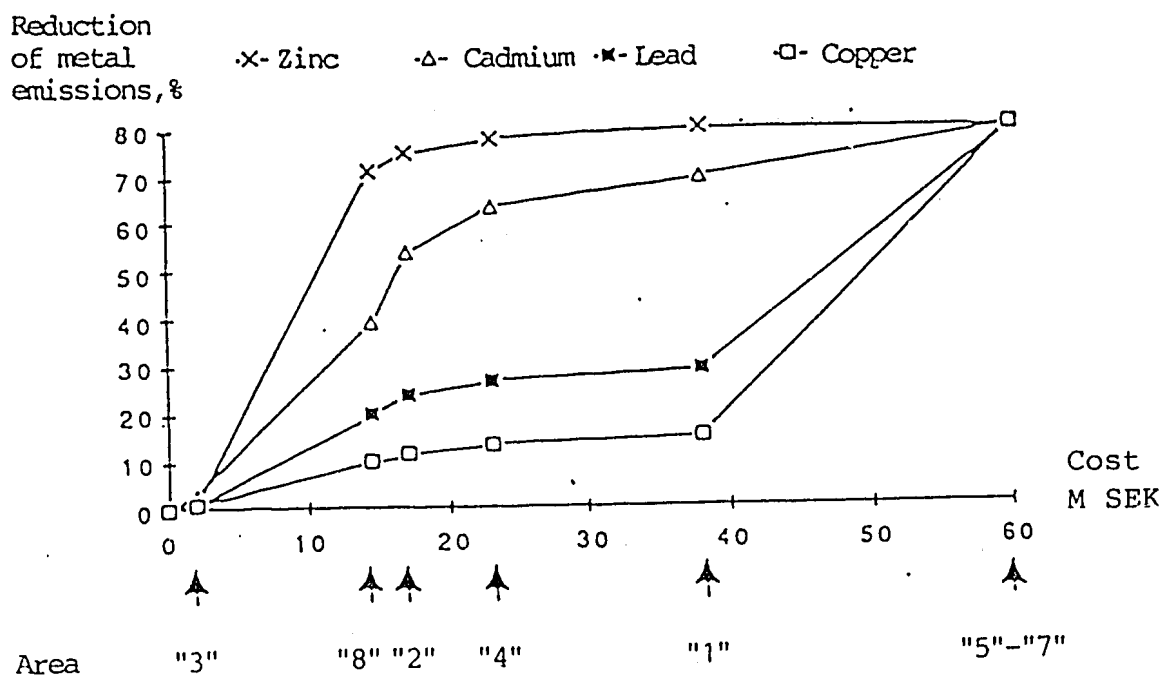
The slag and waste-rock in the old railway embankment (area "6"), with a volume of approx. 60 000 m³, should be removed and deposited under water in the lake. The estimated cost is 2 M SEK (0.3 M US \$).

The actions proposed for area "7" are to remove approx. 1 m slag from an area of approx. 60 000 m² and to deposit the slag under water in the lake. The estimated cost is 5 M SEK (0.8 M US \$). Some old buildings will have to be pulled down and eventually rebuilt which will cost at a minimum 10 M SEK (1.6 M US \$).

In area "8" 150 000 m³ of waste-rock can be back-filled in old shafts. The plan suggests that the remaining waste-rock, approx. 70 000 m³, should be deposited under water in the lake. The uncovered underlying soil would be partly removed, partly covered with moraine.

An old mine-worker's chapel will have to be moved and some old buildings pulled down. The total costs for area "8" are estimated to be 12.5 M SEK (2 M US \$).

Altogether, the costs for rehabilitation of the Garpenberg area are estimated to be approx. 60 M SEK (9.5 M US \$). The effect on the metal emissions will presumably be an 80 % decrease for all metals. A rough cost-benefit analysis has been made and is shown in the figure below.



An evaluation of the plan described above will be made in 1990. Since the economic resources of the Dalälven project are limited, the need for action in Garpenberg must also be balanced against the needs in other areas, for instance in Falun. Detailed planning will probably start in 1991.

NATIONAL SWEDISH
ENVIRONMENT PROTECTION BOARD

Report List
1988-08-31

Project group for mine waste
deposits from the mining industry

PROJECT FIELD: WASTE DEPOSITS FROM THE MINING INDUSTRY

PROJECT REPORTS AUGUST 1988

Published reports

1. S Hydrogeological prerequisites for the reduction of leachate production at waste deposits. a)
2. S Report from a mining symposium and visits at some mines in U.S.A., December 1981. a)
3. S Transport calculations on the weathering course in mine waste. a)
4. S A survey of sand impoundments from the processing of sulphidic ores in Sweden. a)
5. S A study of base mineral index in sand impoundments from the processing of sulphidic ores. b)
6. S Gas diffusion in unsaturated porous media. a)
7. S Pyrite weathering in sulphidic sand deposits, laboratory tests. b)
8. S Bacterial weathering in sulphidic sand deposits. b)
9. S HBV-model study for tailings in Kristineberg, Sweden. b)
10. E Distribution and Flow of Water in Unsaturated Layer Cover Materials for Waste Rock. a)
11. S Solid Mine Waste. Information from the National Swedish Environment Protection Board. SNV. Broschure 9019, 1986.
12. E Mathematical Modelling of Water and Oxygen Transport in Layered Soil. Covers for deposits of Pyritic Mine Tailings. f)

Working reports

20. S PM concerning a proposed research work on sand impoundments. c)
21. S PM concerning reclamation of sand impoundments at sulphidic ore mines. c)
22. S Waste deposits from the mining industry. Research plan for SNV. d)
23. S Supplementary survey of sand impoundments from the processing of sulphidic ores in Sweden. b)
24. S The distribution of soil types under and around present sand impoundments from the processing of sulphidic ores. b)
25. S Methods for the evaluation of future potential acid formation in sulphidic residues. b)
26. S Waste deposits from the mining industry - Field surveys with water balance assessment in Kristineberg. e)
27. S Infiltration of precipitation through sealing layers Field tests in a sand impoundment at the Kristineberg mine field. b)
28. S 45 mine tailings in Sweden - a botanic survey. Report, December 1984.
29. S Laboratory study on the water balance in a sealing layer on a sand impoundment. d)
30. S Survey of sand impoundments from the mine fields of Zinkgruvan, Vassbo, Laisvall, Stekenjokk and Viscaria, Sweden. b)
31. S Metal balance - Kristineberg. Status Report, May 1985. b)
32. S Model calculations for sealing layers on the tailings of Bersbo. f)
33. S Environment Protection Board project field: "Waste deposits from the mining industry" - Status Report, January 1986. d)
34. S Course at weathering of sulphides - laboratory tests. b)
35. E Capillary Barriers in Covers for Mine Tailing Dumps. a)
36. S Waste deposits from the mining industry - Field surveys with a water balance study in Kristineberg. g)

37. S Metal balance - Kristineberg - Final Report. b)
38. S Oxygene and water barrier prooporties of some material of interest covering sandy mine tailings. h)
39. S Water barrier prooporties of different covering materials for waste disposal. i)
40. E Biomass, root penetration and heavy metal uptake in birch in a moraine layer on copper tailings. d)
41. E Utilization of waste products in restoration of iron mine tailings. d)
42. E Efficiency and design of layered soil covers for mine tailings dumps containing pyrite. f)
43. E Oxygen transport and oxidation reactions in waste rock dumps and soil covers. f)
44. S Sand impoundments from the processing of sulphidic ores in Sweden - surveys and inventories.
45. S Travel Report. Participation at the conference on "Mechanisms of Ion Transport in Soils" in Zürich, May, 20-23, 1985. f)
46. S Collection of basic data within the frame of current studies at the Kristineberg Mining Area. Water balance and Metal balance, september 1987. g)
47. E Gas Diffusion in Unsaturated Porous Media II. f)
48. S Assessment of measures to counteract environmental impacts from tailings at the siting operation. g).
49. S A test area for the investigation of the efficiency of sealing and covering layers at waste disposal. i)

Key notes:

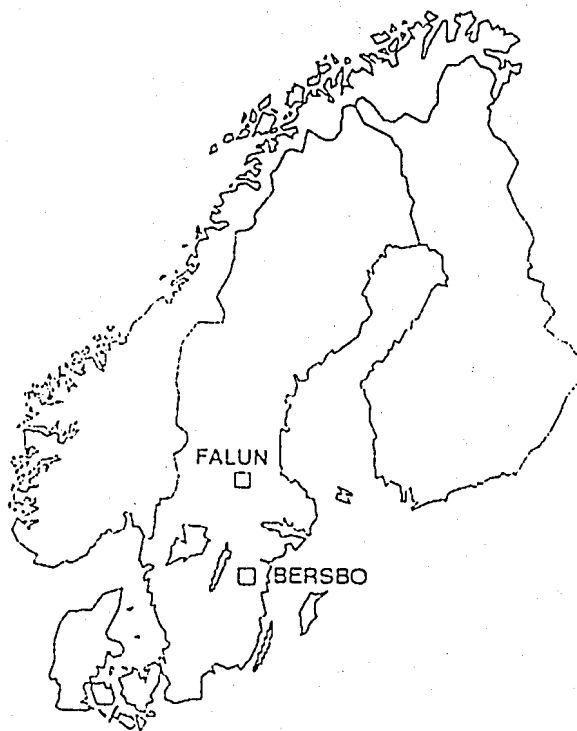
- a) National Swedish Environment Protection Board
- b) Dept. of Quaternary Geology. University of Uppsala
- c) VBB AB (consulting company). Stockholm
- d) Swedish Geotechnical Institute
- e) Swedish Geological Survey, Dept. of Chemical Engineering
- g) Uppsala Geosystem AB (consulting company)
- h) Swedish University of Agricultural Sciences, Res. Dept. of Hydrotechnology
- i) Swedish Geological AB (consulting company)

Note: S = In Swedish only
E = Translated into English

**APPENDIX C. Acid Mine Drainage from mine sites in Bersbo and Falun, Sweden.
by Ulf Qvarfort.**

ACID MINE DRAINAGE IN SWEDEN.

Description of the mine sites in Bersbo and Falun.



Ulf Qvarfort
November 1989
Terratema AB.

BACKGROUND.

The presence of sulphides in and around many mining areas leads to environmental problems. The processes in the residual products are normally composed by chemical decomposition reactions of which oxidation is the most important. The sulphides in the waste rock heaps and in the tailings dams are easily oxidized and decomposed under humid aerated conditions, leading to the production of sulphuric acid. The percolating water is contaminated by the sulphuric acid which increases the decomposition of other minerals in the heaps and tailings. This weathering process causes considerable amounts of metals and acid to leach out to near-by lakes and streams.

Ever since the environmental problems began to attract attention in the mid 1970's the Environment Protection Board has given priority to the work of developing techniques suitable for solving the problems connected with mine waste. The conclusion was generally speaking the only reasonable measure to counteract environmental problems at old waste-rock heaps is to cover the waste with an effectively designed coverage. A fundamental element has been that the transport of oxygen and water to the waste is effectively limited for a very long time.

The state of the knowledge at this time regarding oxidation and metal leaching from sulphide mine waste shows that it would be possible in the long run to reduce metal leaching by roughly 90 % if the heaps are sealed and screened off to prevent oxygen and water from penetrating.

This paper reports a summary of the investigations made so far in the Bersbo and Falun mining areas.

According to the Bersbo area the summary includes a description of the research material as well as a presentation of the rehabilitation work. However, in the Falun area the summary only includes the first part of the research work as no final design have been made yet.

THE BERSBO MINING AREA.

The Bersbo mining area is situated 8 km north-north west of Ätvidaberg. The ore mined have been a complex ore containing chalcopyrite, galena, pyrite, sphalerite and pyrrhotite. The average contents for mined ore were 25 % iron, 0,5-3 % copper, 1-3 % zinc, 1 % lead and 25 % sulphur.

The mining in the area, from the 13th century up to the beginning of the 20th century, has given rise to large quantities of waste material, mainly in form of waste rock heaps. A map of the mining area is shown in fig 1 and in table 1 is a summary of the waste material found in the area.

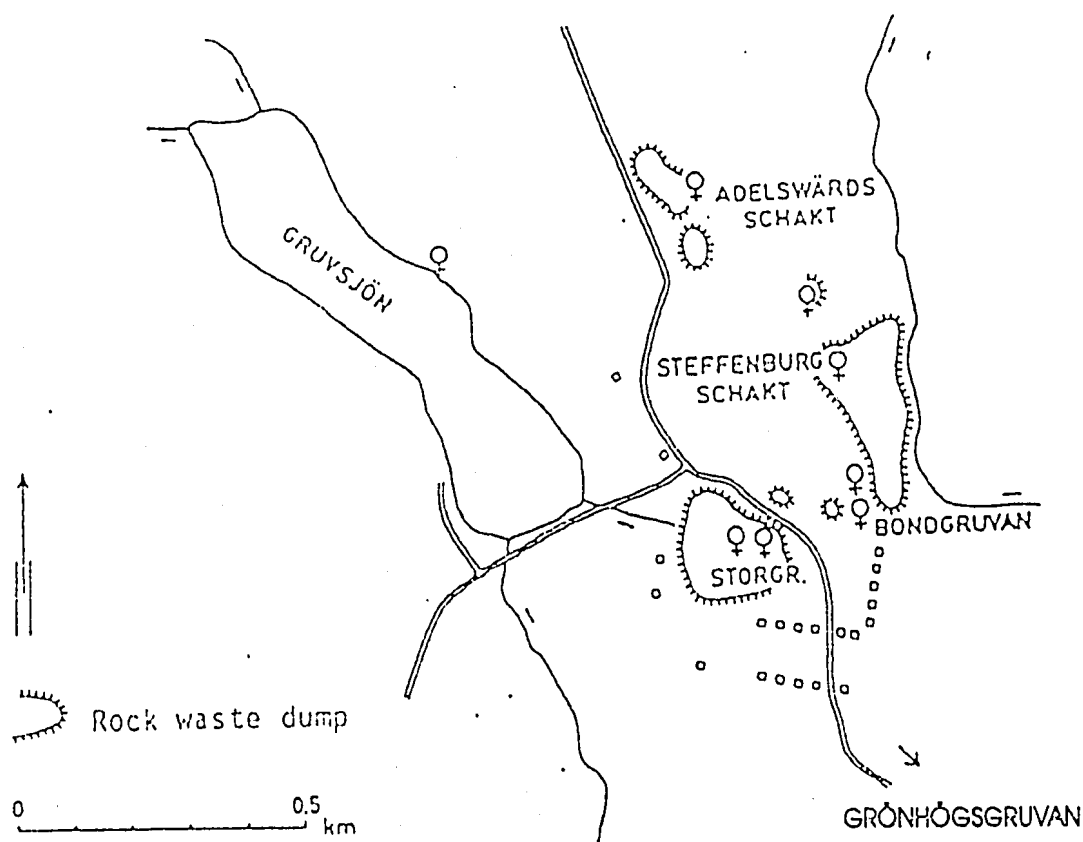


Fig 1. Bersbo mining area.

TABLE 1. Waste materials and their metal contents for the Bersbo mining area. The different sites can be seen in fig 1.

Waste dump	Total kton	Content kg/ton		Total amount	
		Cu	Zn	Cu	Zn
Steffenburg	800	2,9	7,0	2320	5600
Storgruvan.	500	2,9	7,0	1450	3500
In roads	20	2,9	7,0	58	140
Grönhögsgruvan	30	2,5	19,6	75	588
Tailings	20	2,0	8,6	40	249
Adelswård	10	2,0	19,6	25	196
Kuntebo	10	2,0	6,0	20	60
Small waste heaps in the area	10	1,5	5,0	15	50
Summary	1.400.000			4003	10306

Measurements made in recent years show that the feeder stream to the lake nearby the mining area, called Gruvsjön, has pH-values below 4, zinc contents of 50-150 mg/l, copper contents of 15-15 mg/l and cadmium contents of 0,2-0,6 mg/l.

One can estimate that 15-50 tons of zinc, 2-6 tons of copper, 40-70 kg of lead and 30-50 kg of cadmium leach out from the area ever year.

An example of the leaching of copper for the years 1986 and 1987 are also given in fig 2. The figure showing that the main leaching and metal-transport take place during spring and autumn.

Despite the heavy metal leaching that already has taken place, there is enough waste in the area for several hundreds of years of continued leaching at the same proportions as today. The production of sulphate from the pyrite oxidation, shown by the water analysis indicate a mean oxidation rate of $500 \text{ g FeS}_2/\text{m}^2, \text{år}$. This means a weathering of totally 7500 tons of FeS_2 or 8-16 % of the total amounts of the sulphides in the wastes through the years.

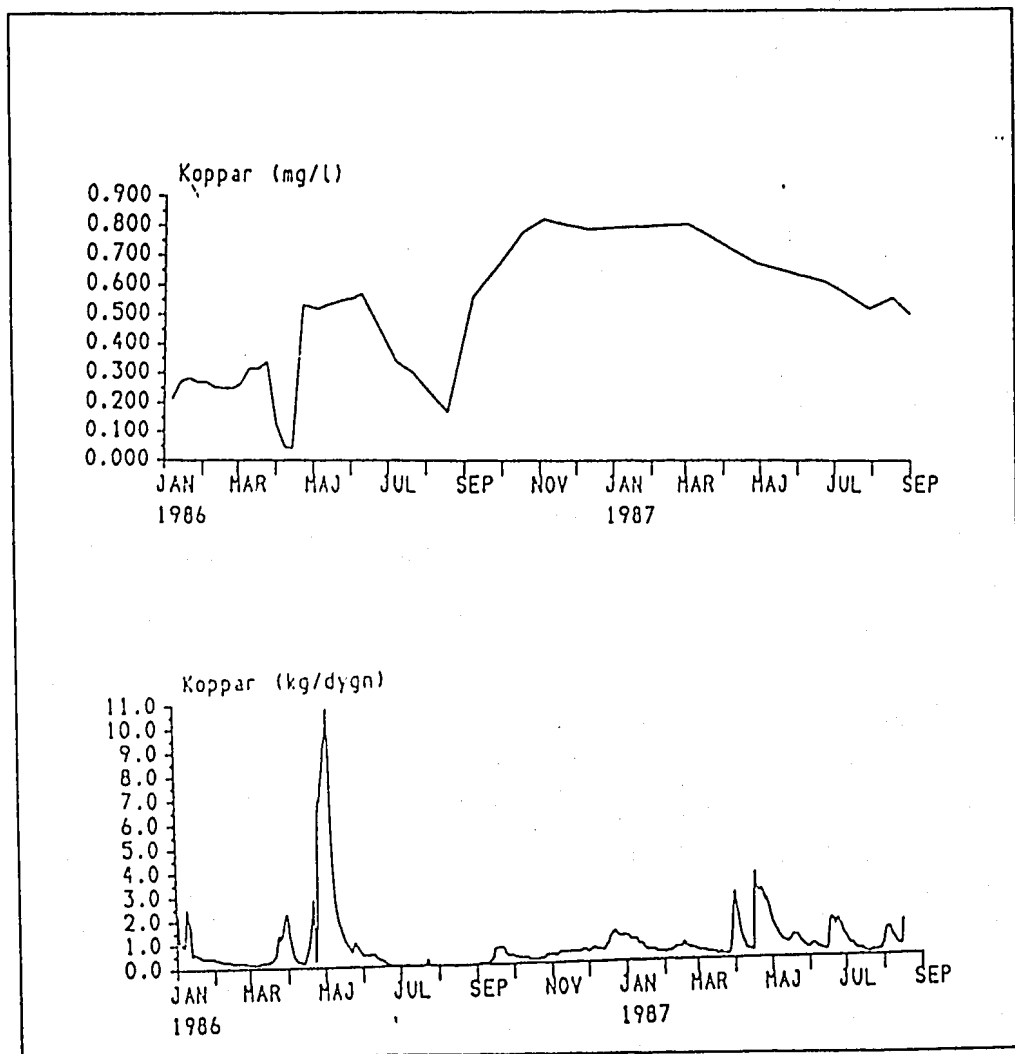


Figure 2. The amount of copper in the river Kuntebäcken. The sampling point is located 400 meters north of Steffenburg, cf. fig 1. The data are from Theme Water Research, University of Linköping.

Based on water analyses and field data an approximate metal balance for the area have been made. The results are shown in figs. 3 - 5. The data are mainly from the investigations made by Tema Vatten, University of Linköping and the Swedish Geotechnical Institute.

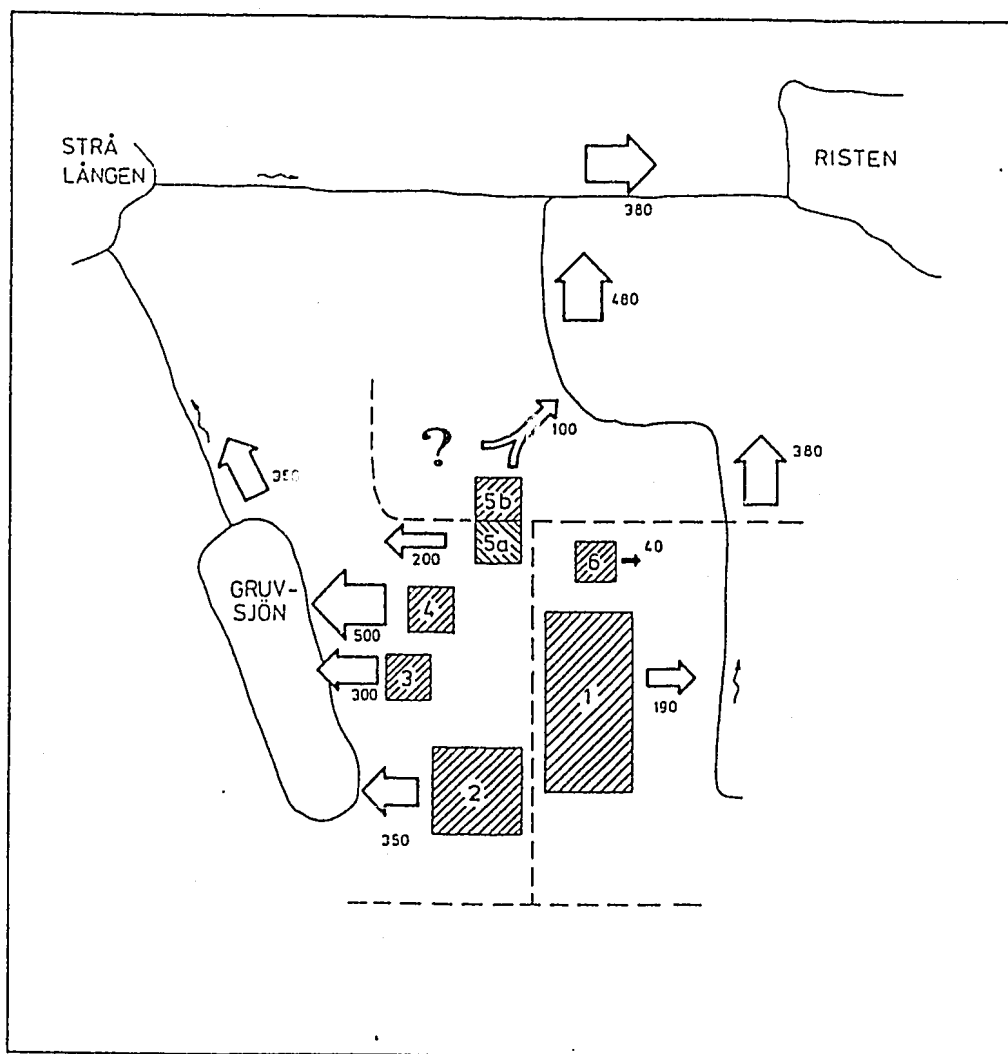


Fig 3. Estimated transport of Copper in kg/year from different sources in the Bersbo mining area.

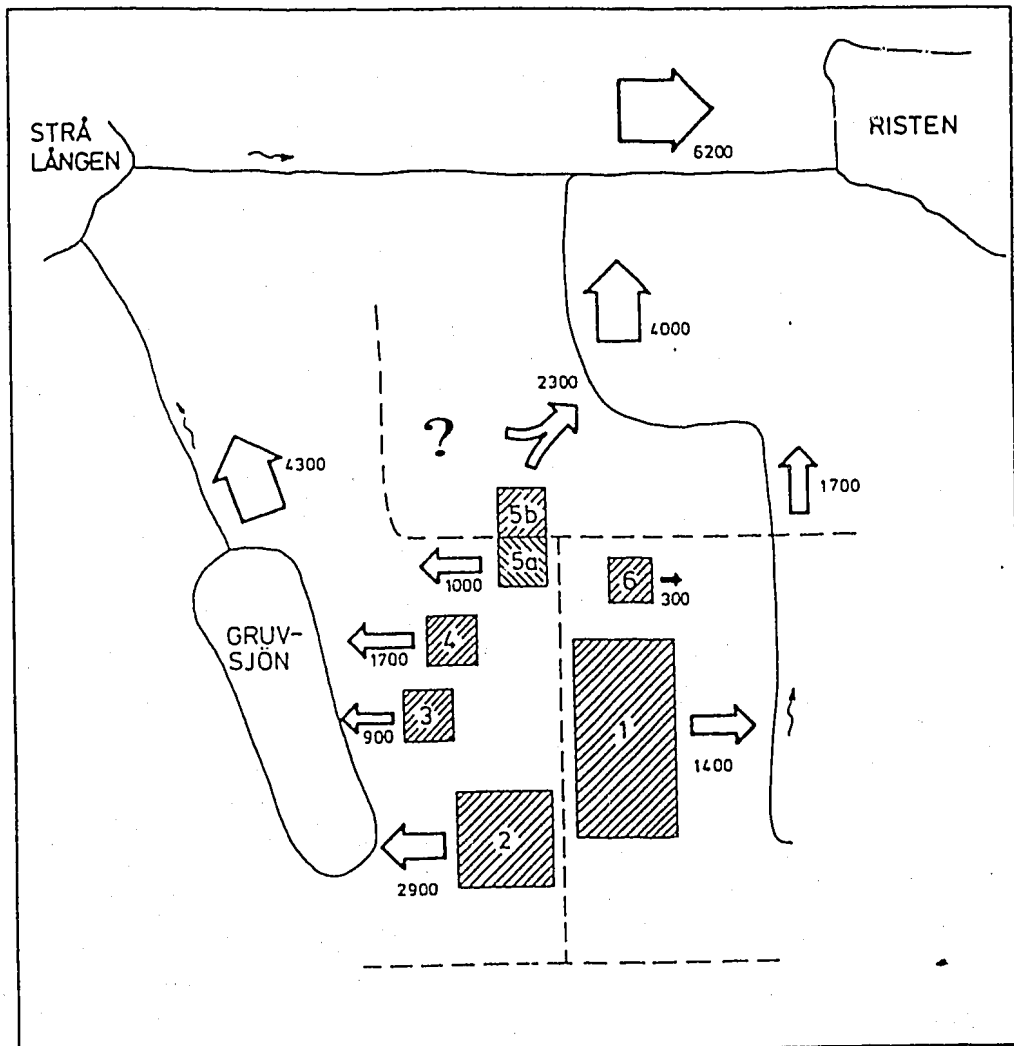


Fig 4. Estimated transport of zinc in kg/year from different sources in the Bersbo mining area.

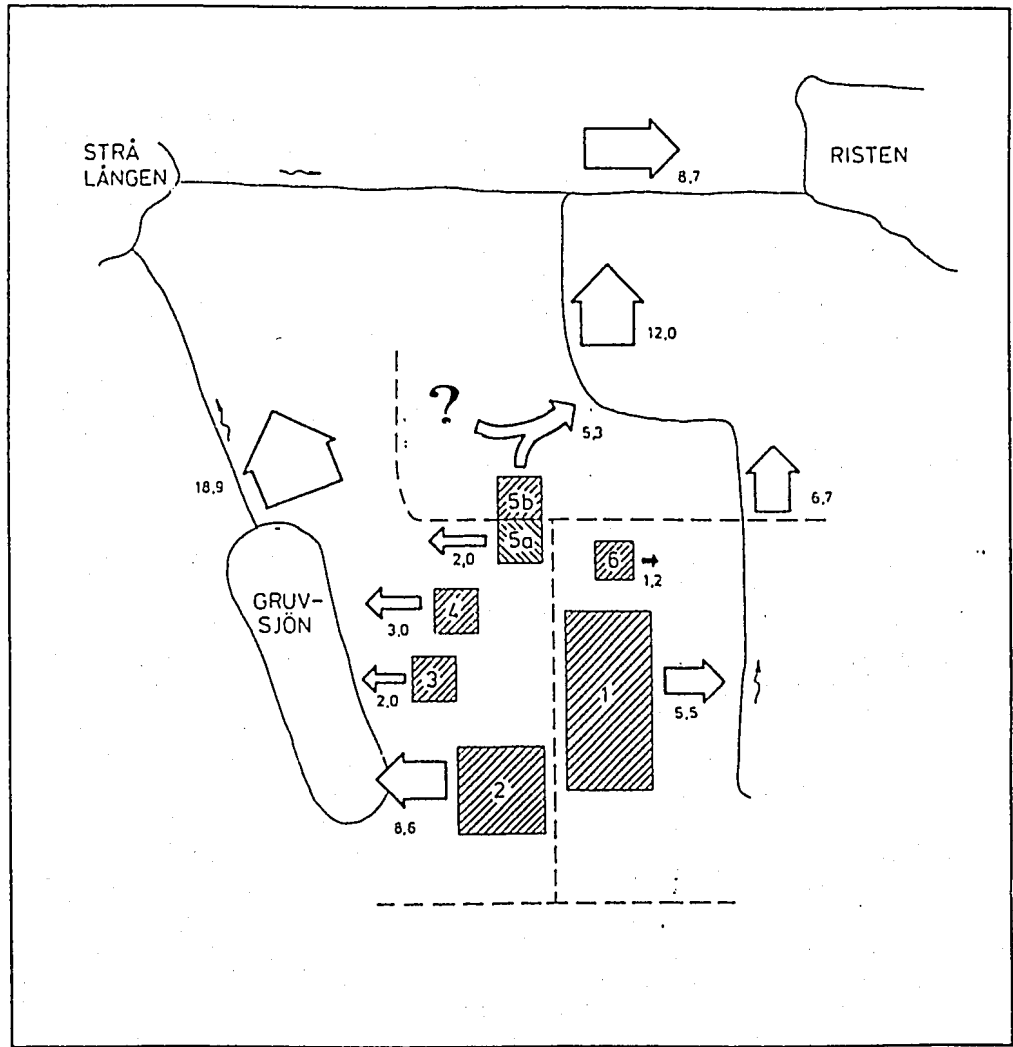


Fig 5. Estimated transport of cadmium in kg/year from different sources in the Bersbo mining area.

DESCRIPTION OF THE RECLAMATION WORK.

A fundamental conditions of the investigation within the Bersbo project has been that the mine heaps should be covered in such a way that the transport of oxygen and water to the waste is effectively limited for a very long time. According to that the work on the rehabilitation of the area can be summarized as followed.

Reducing the leaching area as much as possible.

First all shafts, and the open cast mine was filled with waste dumps. Then a number of small waste heaps, in all 50.000 m³, plus some waste sand and polluted surface soil about 15.000 m³ was brought together and concentrated in two main heaps, the Storgruve dump and the Steffenburg deposit. (se fig.1).

The two dumps reshaped with suitable slopings. To this end, about 30.000 m³ of waste rock was moved within and between these dumps. The ground surface from which the waste rock was moved was treated with a lime slurry and covered with about 0,3 m moraine. In this way, a total area of about 80.000 m³ was treated.

Reducing the oxygen supply and minimizing the weathering.

The objectives was achieved by sealing off the surface of the two dumps. Two different sealing material were used. The bigger and rather flat Steffenburg dump was covered with clay. The clay was laid in three layers to obtain a final thickness of about 0,5 m. Each layer was worked to get a homogenous and impervious layer.

The smaller Storgruve dump, which has steeper slopes and more difficult boundary zone connections, was covered with CeFill. CeFill is a trade name of a cement stabilized flyash (cement 5-10%) which if necessary also is activated with desulphur-izing products (15-35%). At a water ratio of 30-40% a pumpable products is obtained which can be used for instance to fill the voids of a bed of crushed rock. CeFill solidifies in less than 24 hours into a hard, dense sealing layer which also is strong thanks to the rock aggregate stabilization. This product is pumped out as a slurry to fill the voids of a crushed rock layer previously spread on the heap. That layer is 0,25 m thick constructed of sorted out, crushed rock aggregats, 60-90mm, originating from blocks of the remaining moraine pit.

In order to protect the sealing layers against freezing, drying and root penetration, the dumps was covered with a moraine layer of approx. 2 m thickness.

Around the dumps, impervious connections to rock, moraine and clay was constructed by means of bentonite and CeFill grouted crushed rock aggregates.

The heaps will finely be revegetated with trees and grass.

To obtain the necessary quantities of clay, crushed material and moraine, a clay-pit as well as a moraine and stone pit was opened north of Bersbo. The moraine contains plenty of boulders, so that the moraine which will be laid next to the sealing layers must be cleared of boulders exceeding 300 mm. A crushing plant was used for the crushing of rock aggregates and other aggregates needed for CeFill sealing, road construction and road stabilization. The pits will be landscaped upon completion of the operation.

Environmental control program.

An environmental program have recently been designed for monitoring the area. The program have not yet started but it will include monitoring of the streams and the lakes in the area. Also the groundwater quality will be included in the programme.

To this date, nov. 89, same checking have taken place of the function of the sealing layer and the effects of the sealing conditions of the dumps. According to them the oxygen contents in some probe holes have been measured. The results showed that rehabilitation greatly reduced the oxygen inflow to the dumps and also reduced or stopped the oxidation of pyrite.

THE FALUN MINING AREA.

No one can tell exactly when and how the Falun Copper Mine began to be worked. The mine is, however, one of the oldest in the world, where work has been in progress at various times from about 700 A.D. (late Iron Age) until the present day.

The bedrock of the area is dominated by granites and leptites of Precambrian age. The chief ore mineral nowadays is pyrite, together with some chalcopryite, sphalerite, and galena. The pyrite ore forms compact, lens-shaped masses and has to a large extent replaced limestone or dolomite which are now found as scattered remnants in the ore body. Some sulphide minerals also occur as impregnations in quartzite, occasionally with some gold and selenium. The rich copper ore which previously made Falun renowned is now of little importance.

At the present, the total production of the mine is about 200.000 tons of ore per year. The average composition of the ore mined is: 21 % iron, 3 % zinc, 1 % Lead and 0,5 % copper.

The nature and the amount of contamination due to the mining in the area depends on many factors as;

First the long mining activities in the area have give enormous amounts of waste dumps and slag heaps.

Second when the "Great Pit" was formed in Falun the amount of leaching must have increased enormous. As the whole cave-in is composed of large blocks of rock, water and oxygen can readily penetrate it. Maximum weathering and oxidizing conditions and maximum removal of metals are therefore to be expected.

Third the industrial production of sulphuric acid has generated a relatively large amount of waste material containing leacheable amount of zinc.

For the Falun area a comprehensive and detailed research material will be available during 1990 in connection with the project "Dalälven". A short presentation of the most important results is therefore given below, based on former investigations.

Types of waste material.

The mining at Falun has given rise to large quantities of waste, mainly in form of;

- * Waste-rock heaps
- * Tailings
- * Slag heaps
- * Waste from the sulphuric acid production.
(Roast Residues).

A map of the waste material in the Falun area (the town) can be seen in fig 6. In table 2 gives a summary of the amounts of the different waste materials.

TABLE 2. The amounts of mining waste materials in Falun.

Types	m ³
Waste-rock heaps	710.000
Talings	2.200.000
Slag heaps	5.300.000
Roast Residues	500.000

The metal discharged from the Falun area.

Based on analysis of water from the Faluån River the metals discharged from the area are given in table 3. The Faluån River can be seen in fig 6.

TABLE 3. Metal outflow to the River Faluån.

Fe	1100 tons/year
Cu	29 "
Zn	700 "
Cd	0,6 "

1987 started a purification of the mine-water in the Falu town waste water plant. According to that the total amount of zinc to the river was recently (1987) decreased to about 300 tons/year.

In 1989 VIAK AB, a Swedish firm of consultants and STORA AB, Falun made a metal balance study for the area. The purpose of that study was among other things to investigate the metal contribution from different subareas to the river Faluån. A summary of the investigation is given in table 4 and in fig 7.

TABLE 4. The contribution of iron, copper and zink from different subareas. The location of this areas can be seen in fig 7. The amounts are in tons/year.

Location	Fe	Cu	Zn
1. Skålpussen(lake)	22	0,18	44
2. *Rost residues	78	1,75	124
3. Tailings dam	138	10,5	88
4. Mining area	98	1,1	30
5. Old tailings dam	18	0,44	4
Old slag heaps in the area	0,75	0,003	0,225

* Waste from the sulphuric acid production.

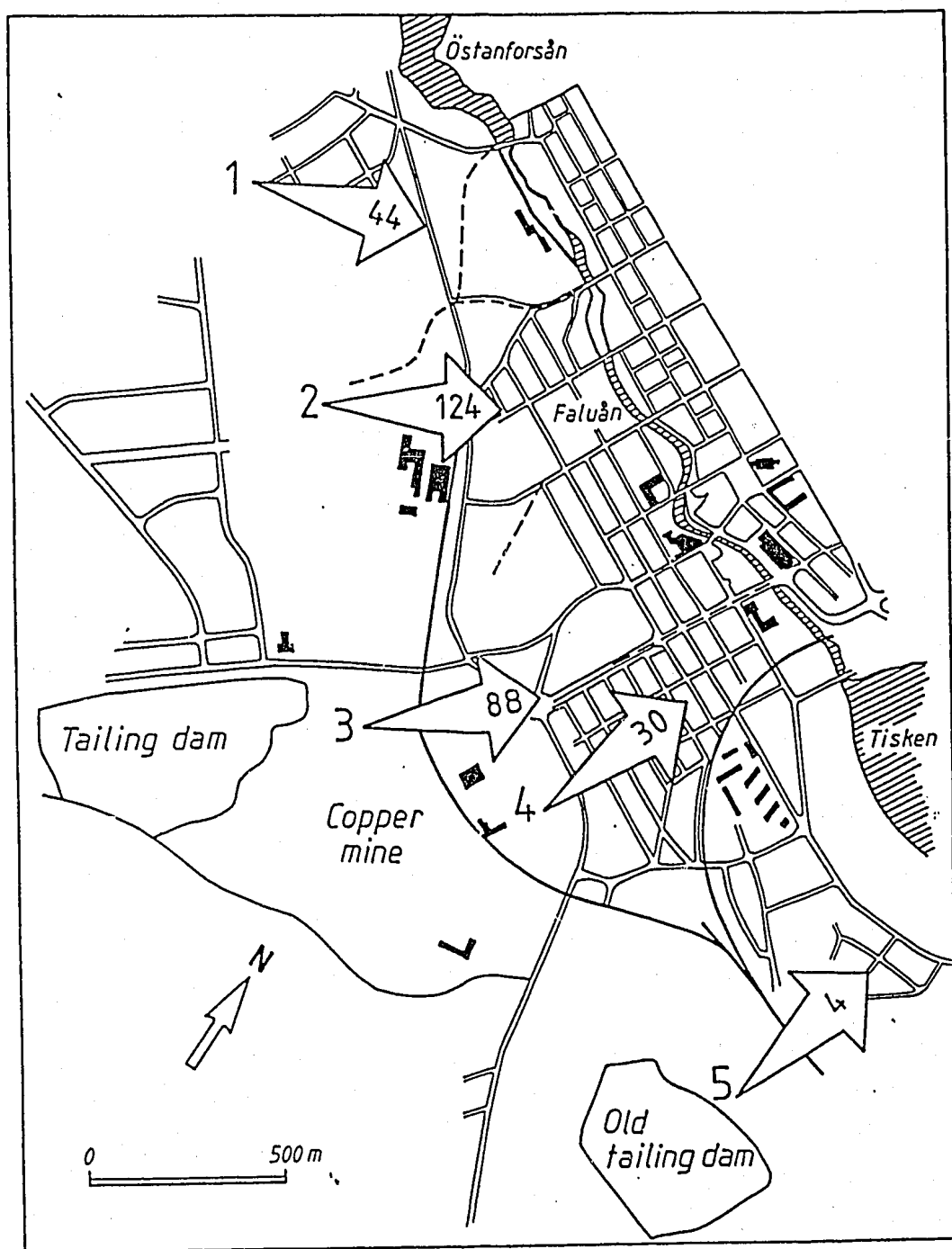


Fig 7. Simplified map of the contribution of zinc to River Faluån from different subareas. The arrows show the amounts in tons/year.

RECLAMATION WORK.

In view of the former result obtained in the Falu area and the complexity of the area a project started 1988 with a detailed investigation of the different waste material in the area. After that a detailed planning and design work should be started during 1990.

REFERENCES (mostly in Swedish).

SOU 1988:34: River Dalälven - Environmental Reuodial Actions.

Depart.of Theme Resarch (1987): Metal transport from the Bersbo area in connection with covering. Hydrochemical monitoring program. Linköping University.

Jacks, G. (1976): Heavy metals from Sulfide Tailings. Report 3:20. Royal Technical Institute, Stockholm.

Karlqvist, L & Qvarfort, U (1979): The Bersbo project. Municipality of Åtvidaberg.

Södermark, B & Lundgren, T (1987): The Bersbo project - The first full scale attempt to control acid mine drainage in Sweden. The National Swedish Protection Board and The Swedish Geotechnical Institute. In English.