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Metals Exploration Limited.

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# MOUNT BISCHOFF JOINT VENTURE

## TIN PROJECT

## MINING OPTIMISATION STUDY

G. G. JONES

JUNE 1984



AMG REFERENCE POINTS ADDED

## SUMMARY AND RECOMMENDATIONS

The enclosed study examines three individual cases for the mining of the Mount Bischoff orebody. These are:

(i) Main pit with one pre-production year,

(ii) Main pit with the Stanhope extension pit without a pre-production year, and

(iii) Main pit by itself without a pre-production year.

Ore reserves calculated by Douglas McKenna and Partners were used as the basis for the study. Using the geological cross sections showing these reserves, ideal open pit mining parameters were designed and Mineable Ore Reserves (MO/R) for Dolomite Sulphide Lode (D.S.L.) and the Quartz Porphyry (QP) ore were established. These were:

| | | | |
|---|---|---|---|
| DSL | 963282 tonnes | @ | 0.90% Sn |
| QP | 719771 tonnes | @ | 0.46% Sn |

Due to the configuration of the ore blocks an exercise to establish the amount of dilution and the percentage of MO/R recovered is included. The results of the exercise indicate that the following tonnage of ore will be recovered from each case.

| | | | | |
|---|---|---|---|---|
| (i) | DSL | 1024036 tonnes | @ | 0.78% Sn |
| | QP | 711192 tonnes | @ | 0.43% Sn |
| (ii) | DSL | 1023342 tonnes | @ | 0.78% Sn |
| | QP | 1223081 tonnes | @ | 0.41% Sn |
| (iii) | DSL | 1022918 tonnes | @ | 0.78% Sn |
| | QP | 777688 tonnes | @ | 0.43% Sn |

Production rates from each case are determined by the location of the ore relative to the haul roads and by the restricted space caused from the desire to keep the stripping ratio to a minimum. The production rate from cases (ii) and (iii) is kept to a minimum in the initial two years to allow the necessary development work to be completed.

An ore reserve which can only be mined by underground mining or by scavenging at the completion of the pit is also established. This reserve is:

| | | | |
|---|---|---|---|
| DSL | 118500 tonnes | @ | 1.39% Sn |
| QP | 72460 tonnes | @ | 0.85% Sn |
| QP | 166060 tonnes | @ | 0.80% Sn |

It is estimated that 50% of the DSL may be recovered using a backhoe from the bottom of the Main pit (Fig. 2.6). The 238520 tonnes of Porphyry ore can only be recovered by underground stoping methods which will require extensive development (both decline and horizontal) and ground support (filling). These underground ore blocks by themselves would appear to be very marginal at this stage but future exploratory drilling may discover new ore reserves in the same zones which may make underground mining more attractive. By that time greater confidence will be established with regard to overhead costs, rock structure, tin recovery, etc.

Examination of the following indicative net revenue statements, result in Case (iii) being preferred in view of the level of revenue achieved over project life and the minimisation of initial funds exposure.

It is therefore strongly recommended that Case (iii) be accepted as the optimum and that a test mining programme in the Main pit area take place immediately to establish confidence in the ability to be able to mine to the forecast parameters.

G. G. JONES
JUNE 1984

S00910:CHR/40

TABLE 1(i)

## MAIN PIT AND PRE-PRODUCTION
## INDICATIVE NET REVENUES

| DETAILS | | YEAR -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Tonnes mined | (000t) | | 137.6 | 195.1 | 203.4 | 189.5 | 182.5 | 179.6 | 174.7 | 181.5 | 144.0 | 147.3 | 1735 |
| Ore grade mined | % Sn | | 0.68 | 0.67 | 0.71 | 0.64 | 0.64 | 0.71 | 0.53 | 0.62 | 0.58 | 0.60 | 0.64 |
| Sn tonnes @ 61.5% overall recovery | | | 575.44 | 803.91 | 888.15 | 745.87 | 718.32 | 784.22 | 569.43 | 692.06 | 514.65 | 543.54 | 6835.59 |
| Sn price A$13880/tonne | | | | | | | | | | | | | |
| Gross Revenue | ($'000) | | 7,987 | 11,158 | 12,328 | 10,353 | 9,970 | 10,885 | 7,904 | 9,606 | 7,143 | 7,544 | 94,878 |
| Total Operating Cost | ($'000) | (1422) | (6371) | (8346) | (8907) | (8433) | (8120) | (8016) | (7654) | (7353) | (5848) | (5833) | (77281) |
| Royalties | ($'000) | | ( 200) | ( 279) | ( 308) | ( 259) | ( 249) | ( 272) | ( 198) | ( 240) | ( 179) | ( 189) | |
| Net Revenue | ($'000) | (1422) | 1416 | 2533 | 3113 | 1661 | 1601 | 2597 | 52 | 2013 | 1116 | 1522 | 16202 |

S00932:CHR/1

TABLE 1(ii)

## MAIN AND STANHOPE EXTENSION PITS
## INDICATIVE NET REVENUES

| DETAILS | | YEAR 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Tonnes mined | (000t) | 140.9 | 180.4 | 275.5 | 352.3 | 290.2 | 291.4 | 204.8 | 181.8 | 152.4 | 176.8 | 2246.5 |
| Ore grade mined | % Sn | 0.69 | 0.46 | 0.54 | 0.56 | 0.54 | 0.64 | 0.62 | 0.63 | 0.55 | 0.59 | 0.58 |
| Sn tonnes @ 61.5% overall recovery | | 597.9 | 510.4 | 914.9 | 1213 | 964 | 1147 | 781 | 704 | 515.5 | 641.5 | |
| Sn price A$13880/tonne | | | | | | | | | | | | |
| Gross Revenue | ($'000) | 8299 | 7084 | 12699 | 16841 | 13377 | 15920 | 10839 | 9777 | 7155 | 8904 | 110895 |
| Total Operating Cost | ($'000) | 7241 | 8096 | 12617 | 15046 | 12799 | 12605 | 9007 | 7465 | 6307 | 70546 | 98239 |
| Royalty | ($'000) | 207 | 177 | 317 | 421 | 334 | 398 | 270 | 244 | 179 | 223 | 2770 |
| Net Revenue | ($'000) | 851 | (1189) | ( 235) | 1374 | 244 | 2917 | 1562 | 2068 | 669 | 1625 | 9886 |

S00932:CHR/4

TABLE 1(iii)

MAIN PIT (NO PRE-PRODUCTION)
INDICATIVE NET REVENUES

| DETAILS | | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Tonnes mined | (000t) | 140.9 | 137.6 | 216.7 | 205.8 | 208.4 | 205.3 | 204.8 | 193.2 | 152.1 | 135.9 | 1801 |
| Ore grade mined | (% Sn) | 0.69 | 0.65 | 0.66 | 0.65 | 0.58 | 0.68 | 0.62 | 0.59 | 0.58 | 0.59 | 0.63 |
| Sn tonnes @ 61.5% Recovery | | 597.9 | 550 | 879.6 | 822.7 | 743.4 | 858.6 | 780.9 | 701 | 542.5 | 493.1 | |
| Sn price A$13880/t | | | | | | | | | | | | |
| Gross Revenue $000's | ($'000) | 8299 | 7635 | 12209 | 11419 | 10318 | 11917 | 10839 | 9730 | 7530 | 6844 | 96740 |
| Total Operating Cost | ($'000) | 6505 | 6314 | 9662 | 9377 | 9389 | 9122 | 8634 | 7966 | 6114 | 5492 | 78575 |
| Royalty | ($'000) | 207 | 191 | 305 | 285 | 258 | 298 | 271 | 243 | 188 | 171 | 2417 |
| Net Revenue | ($'000) | 1587 | 1130 | 2242 | 1757 | 671 | 2497 | 1934 | 1521 | 1228 | 1181 | 15748 |

S00932:CHR/6

# INDEX

# APPENDICES

1. ORE RESERVES

2. CONTRACTORS' PRICES

3. MINING OPTIMISATION STUDY BRIEF

4. GUIDELINES FOR 1982 GEOLOGICAL ORE RESERVES

5. RIPPABILITY REPORT

# TABLES

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# DRAWINGS

1. PLANS

1.1 General Locality

1.2 Main Pit (in pockets @ back)

1.3 Main and Stanhope Extension Pits (in pockets @ back)

1.4 Main and Stanhope Extension Pits Showing (in pockets @ back)
Possible Underground Stoping Blocks

2. SECTIONS

2.1 1000mE Main Pit

2.2 1060mE Main Pit

2.3 1140mE Main Pit

2.4 1240mE Stanhope Extension Pit

2.5 1380mE Stanhope Extension Pit

2.6 Idealised Long Section

3.1 Recovery and Dilution

S00910:CHR/4

FIG. 1-1.

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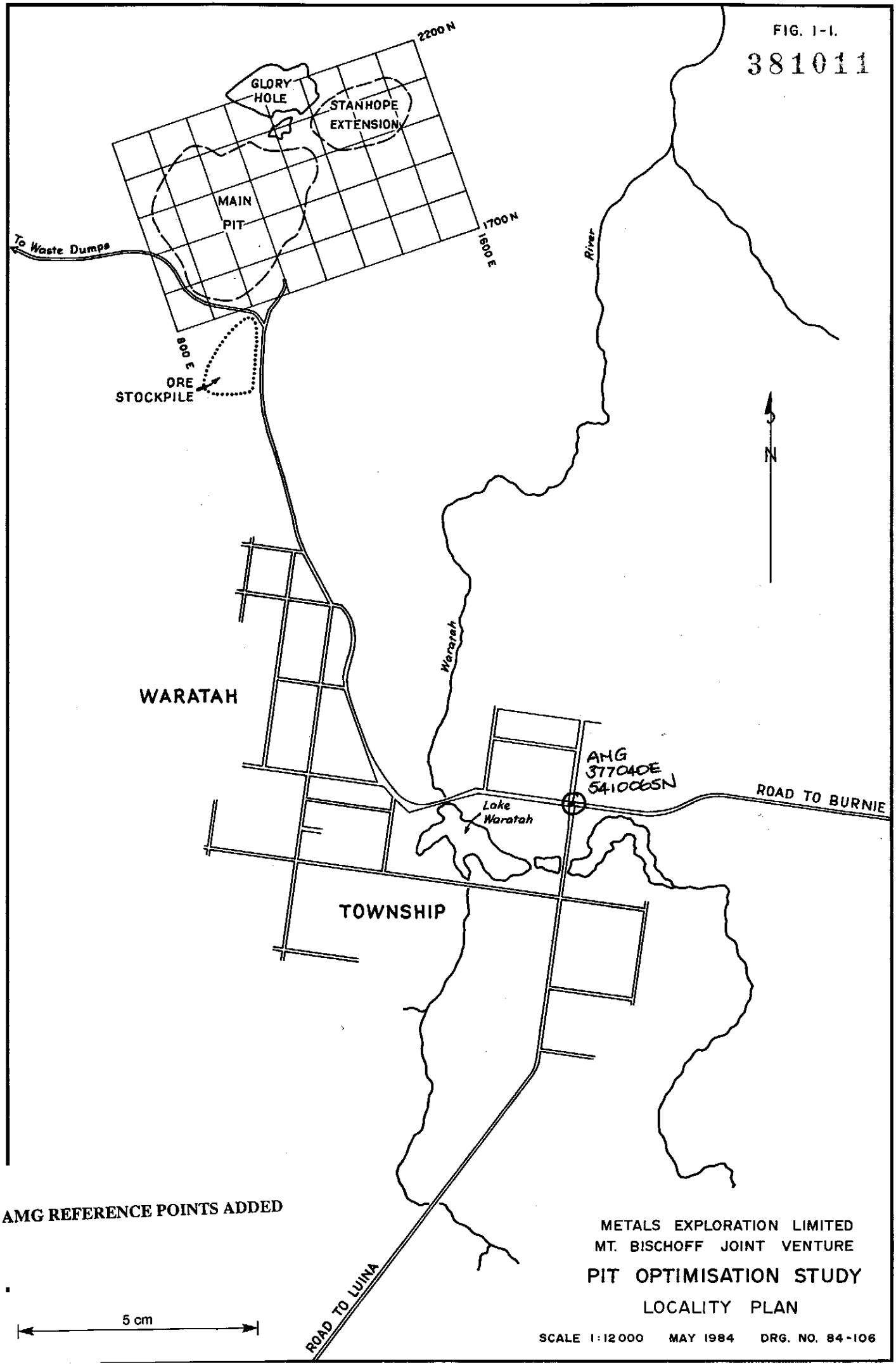


AMG REFERENCE POINTS ADDED

METALS EXPLORATION LIMITED
MT. BISCHOFF JOINT VENTURE

# PIT OPTIMISATION STUDY

## LOCALITY PLAN

SCALE 1:12000 MAY 1984 DRG. NO. 84-106

# 1. INTRODUCTION AND BACKGROUND

## 1.1 Introduction

The Mount Bischoff tin deposit is situated close to the town of Waratah in north-west Tasmania. The deposit was originally discovered in 1871 and was continually mined by one company or another until 1947.

The deposit comprises a patchy and variable cassiterite mineralisation within a more widespread but variable (massive to disseminated) sulphide mineralisation (pyrrhotite and pyrite), which replaces a Pre-Cambrian dolomite in the zone of its contacts with an underlying Pre-Cambrian siltstone/sandstone and a transgressive Devonian porphyry dyke set. Cassiterite mineralisation also occurs in association with disseminated sulphides within the dykes, which are themselves slightly griesenised. The material replacing dolomite has been named the Dolomite Sulphide Lode (D.S.L.) and the sulphide - bearing porphyry simply Porphyry.

The major tin mineral in the Porphyry ore is cassiterite, associated with quartz, topaz, fluorite and with pyrrhotite in the sulphide orebodies.

## 1.2 Background

Numerous engineering studies and mine planning exercises have been carried out with the intention of establishing the most cost effective means of mining and processing the ore from the Mount Bischoff tin deposit. A brief history of the more recent studies is as follows:

1.2.1 1981 - Austin Anderson were commissioned by the Joint Venture managers (Metals Exploration Ltd) to carry out a preliminary engineering study. This study dealt with three individual mining production rates, namely

1.2.1.1 Case 1

Mining and concentrating at Waratah with one 'pre-production' year followed by an initial annual production of 200 000 tonnes of D.S.L. ore increasing to an ultimate annual production rate of 550 000 tonnes combined D.S.L. and Porphyry ore.

| | |
|---|---|
| Total Ore | $3.82 \times 10^6$ tonnes |
| Combined Head Grade | 0.48% Sn |
| Life of Mine | 8 years |
| Overall Stripping Ratio | 1 : 5.9 (ore t : waste t) |

1.2.1.2 Case 2

Mining and concentrating at Waratah with one 'pre-production' year followed by an initial annual production of 150 000 tonnes of D.S.L. ore

increasing to an ultimate annual production rate of 400 000 tonnes combined D.S.L. and Porphyry ore.

| | |
|---|---|
| Total Ore | $3.82 \times 10^6$ tonnes |
| Combined Head Grade | 0.48% Sn |
| Life of Mine | 8 years |
| Overall Stripping Ratio | 1 : 5.9 (ore t : waste t) |

1.2.1.3 Case 3

Mining and stockpiling operations at Waratah with transport to custom milling at Aberfoyle's 'Cleveland' concentrator. One 'pre-production' year would be followed by an initial annual production of 150 000 tonnes of D.S.L. ore increasing to an ultimate annual production rate of 260 000 tonnes of D.S.L. ore only.

| | |
|---|---|
| Total Ore | $0.89 \times 10^6$ tonnes |
| Head Grade | 0.80% Sn |
| Life of Mine | 4 years |
| Overall Stripping Ratio | 1 : 3.8 (ore t : waste t) |

In all the cases the mining operations were to be carried out by J.V. personnel.

1.2.2 1982

1.2.2.1 As a result of the Austin Anderson study the Joint Venture decided that a modified Case 3 operations plan would be the most acceptable. The modifications to the plan were broadly set as follows:

(a) D.S.L. ore to be mined in Years 1 and 2 and from then on blended together with Porphyry ore, the ultimate tonnage to be made up of -
  (i) Approximately 270 000 tonnes of Porphyry ore to be mined per annum, this to be stockpiled, transported to Cleveland then fed into a pre-concentrator which would reject 50% as tailings, the remainder delivered to the concentrator as an upgraded flotation feed.
  (ii) 130 000 tonnes of D.S.L. ore to be mined, stockpiled, transported to Cleveland and fed directly into the concentrator.

(b) A complete reassessment of the geological information and updated drilling results would be carried out.

When the new ore reserves were examined and a new mining design completed the following ultimate production rates were set.

| | |
|---|---|
| D.S.L. ore | 155 000 tonnes per annum |
| Porphyry ore | 245 000 tonnes per annum |

These production rates were to be preceded by one 'pre-production' year. Year 1 would only yield 67% of ultimate production to allow for a 'settling-in' period.

Mining operations were to be carried out by J.V. personnel.

| | |
|---|---|
| Total Ore Mined | $3.563 \times 10^6$ tonnes |
| Combined Head Grade | 0.73% Sn |
| Life of Mine | 10 years |
| Overall Stripping Ratio | 1 : 5.3 |

Contractors were to be used in the 'pre-production' year and to assist in waste removal thereafter.

1.2.2.2 The 1978 Comstaff Joint Venture - Metals Exploration Ltd. Agreement called for a Decision to Mine to be made at a minimum rate of 100 tonnes per annum contained tin. In order to be assured that this would be possible, the Mt. Bischoff Joint Venture performed several studies incorporating the use of a small modular plant at Waratah and contractor mining of high grade DSL ore at low stripping ratios. The first such plan called for mining in the Greisen face area. The second plan called for mining in the Pit Flat area, as follows:

| | |
|---|---|
| Total Ore | $0.238 \times 10^6$ |
| Head Grade | 1.02% Sn |
| Life of Mine | 4 years |

## 1.3 Scope of Work

The brief requires mine planning to optimise cash flow without restrictions imposed by:

(i) Joint Venture agreement requirements, and
(ii) specific maximum tonnage to an offsite plant.

The optimum ore production rate should fall within the range 150 000 - 250 000 tpa and utilise the most recent geological estimates, as conducted by Douglas McKenna and Partners. The following specific items are to be included.

1.3.1 Annual tonnes and grade

i) D.S.L.
ii) Porphyry

1.3.2 Optimisation of grade

i) low grade stockpiling
ii) effective sampling and grade control
iii) minimise dilution

1.3.3 Rippability of surface oxidised zone

i) Waste
ii) Ore

1.3.4 Pre-stripping

i) Time
ii) Tonnage
iii) Cost

1.3.5 Waste Disposal

i) Location
ii) Distance

1.3.6 Mining Operations

i) Contractors
ii) Supervision

1.3.7 Underground Operations

i) Whiteface area
ii) Stanhope area

1.3.8 Costs

i) Capital
ii) Operating
iii) Annual
iv) $/tonne

1.3.9 Drafting

i) Pit outlines and sections
ii) Underground ore zones

1.3.10 Pit and Stockpile Parameters

1.3.11 Contractor enquiries and quotes

M.E.L. to be used as the vehicle

## 2. DESIGN PARAMETERS

### 2.1 Ore Reserves

#### 2.1.1 1982 Geological Reserves

Douglas McKenna and Partners produced a set of geological ore reserves in February 1982. These were calculated with the parameters set by the then Joint Venture managers - CRA Services Ltd. (refer Appendix 4).

DSL Ore

| Cut off Grade %Sn | Proven Tonnes ('000s) | %Sn | Probable Tonnes ('000s) | %Sn | Possible Tonnes ('000s) | %Sn | Total Tonnes ('000s) | %Sn |
|---|---|---|---|---|---|---|---|---|
| 0.4 | 40.0 | 1.5 | 910.0 | 1.1 | 150.0 | 0.89 | 1100.0 | 1.1 |
| 0.3 | 55.0 | 1.1 | 1000.0 | 1.0 | 200.0 | 0.80 | 1300.0 | 1.0 |
| 0.2 | 58.0 | 1.1 | 1100.0 | 0.98 | 290.0 | 0.69 | 1400.0 | 0.93 |

Porphyry Ore

| Cut off Grade %Sn | Probable Tonnes ('000s) | % Sn | Possible Tonnes ('000s) | % Sn | Total Tonnes ('000s) | %Sn |
|---|---|---|---|---|---|---|
| 0.3 | 1000.0 | 0.54 | 29.0 | 0.63 | 1029.0 | 0.54 |
| 0.2 | 1500.0 | 0.46 | 50.0 | 0.51 | 1550.0 | 0.46 |

#### 2.1.2 1983 Geological Reserves

The Geological Reserves for Bischoff were added to in June 1983 reflecting all surface information available and neglecting the spatial restriction placed on the February 1982 Estimates. A new category of pp ore was introduced to reflect surface mineralisation with lower confidence than the possible category.

Bischoff Geological Reserves are therefore:

DSL Ore (0.3% Sn Cut Off)

| Proven Tonnes (000s) | %Sn | Probable Tonnes (000s) | %Sn | Possible Tonnes (000s) | %Sn | PP Tonnes (000s) | %Sn | Total Tonnes (000s) | %Sn |
|---|---|---|---|---|---|---|---|---|---|
| 55.0 | 1.1 | 1054.0 | 1.05 | 221.0 | 0.76 | 99.0 | 0.53 | 1428.0 | 0.90 |

Porphyry Ore (0.2% Cut Off)

| Probable Tonnes (000s) | %Sn | Possible Tonnes (000s) | %Sn | PP Tonnes (000s) | %Sn | Total Tonnes (000s) | %Sn |
|---|---|---|---|---|---|---|---|
| 3326.0 | 0.47 | 76.0 | 0.50 | 120.0 | 0.33 | 3522.0 | 0.47 |

### 2.2 Ore Distribution

All ore reserve blocks were outlined on the appropriate cross section or level plan by Douglas McKenna and Partners. The volumes of ore and waste contained within the designed bench parameters have been calculated and are detailed in Tables 3(i) - (iii).

2.3 Rock Density

The densities used in previous studies were as follows:

| | | |
|---|---|---|
| D.S.L. ore | - | 3.5 $t/m^3$ |
| Porphyry ore | - | 3.0 $t/m^3$ |
| Waste | - | 2.85 $t/m^3$ |

These figures have been substantiated by Douglas McKenna and Partners.

2.4 Final Wall Angle

An overall 'Final Wall Angle' of 45° is considered optimum for the purpose of this exercise. Individual benches will be mined at 70° and 5 m safety berms will be left behind. Further analysis of joint sets and rock structure needs to be carried out before any steeper angle may be considered.

2.5 In-Pit Haul Roads

The width of haul roads is set at 20m being sufficiently wide to allow two Cat 769 haul trucks to safely pass. Allowance is also made for a large drain and an adequate safety barrier. The gradient of the haul roads is set at 9% on the 'straights' and 5% on all curves.

2.6 Bench Configuration

Bench height is set at 10m in all the Porphyry ore. In the DSL lodes bench height may vary dependent on the configuration of the lode. In general a 5m bench will be used.

2.7 Rippability

Golder Associates were commissioned to investigate the practicalities of being able to 'rip and doze' a percentage of the overburden. A preliminary exercise was undertaken, using short seismic velocity traverses on selected sections in conjunction with examination of the diamond drill cores. The results of this (Appendix 5) indicated that significant volumes of material were conducive to ripping and that additional work should be undertaken to establish defined areas and volumes.

To complete the second exercise, Golders carried out seismic velocity traverses along the surface to correspond with the 20m section spacing used for ore reserve computation. The drill cores were again examined and the results are plotted accordingly on the respective sections. A seismic velocity limit of 2500 m/s was set as the maximum to be designated rippable material (which a Cat D9L bulldozer filled with a single shank ripper could handle). The limits of rippability marked on Golders' sections were extended to the pit limits for ease of computation.

Total quantity of rippable material is 1,311,703 tonnes
Say 1,500,000 t

## 3. DESCRIPTION OF PROJECT

### 3.1 General

It is intended to utilise contractors for all drilling, blasting, loading and hauling operations. To enable the required volumes of waste and ore to be moved and the required development work to be completed on schedule it will be necessary to operate the mine for eleven shifts per week i.e. 2 shifts/day - weekdays
1 shift on Saturday

The D.S.L. and Porphyry ore will be stockpiled separately, adjacent to the open pit at Waratah.
Stockpiles will be limited to two weeks plant requirement.

Close attention to supervision of the mining operation will be required as strict grade control will be necessary.

### 3.2 Mining

The Austin Anderson exercise and the Stage 3B Mining Study (March 1982) were both studied and used to develop the rationale for the current mining plan. In order to optimise the mining to maximise cash flow it is necessary to consider three different sets of mine plans for open pit methods.

#### 3.2.1 Main Pit (Pre-Production) (Figs. 1.2 and 2.1-2.3)

Production from the Main Pit area (bounded by section lines 940mE - 1240mE) of an annual rate of approximately 175 000 tonnes of combined D.S.L. and Porphyry Ore. Ore production will be preceded by one pre-production year.

#### 3.2.2 Main and Stanhope Extension Pits (Figs 1.3 and 2.4-2.5)

The Stanhope dyke is situated to the east of the Main Pit area and outcrops at +700m RL. To mine the total ore reserve in this area is not economic due to the high stripping ratio required and the relatively low overall grade. It is however possible to mine a reasonable grade of ore from the upper horizons + 650m RL and still retain a stripping ratio consistent with that existing in the Main Pit area. The Stanhope ore will be depleted over the first five years of the 'life of mine' in an effort to improve the cash flow during that period.

Instead of allowing for one pre-production year this plan allows a decreased mining rate of DSL and Porphyry in years 1 and 2 with the ultimate mining rate of approximately 250 000 - 300 000 tpa being established in year 3.

3.2.3 Main Pit (no Pre-Production) (Figs. 1.2 and 2.1-2.3)

This alternative is the result of the Stanhope Extension Pit exercise which indicated that the addition of the Stanhope ore reduces the overall head grade from 0.64 to 0.58% Sn. This latter grade is very marginal and may not result in the required returns being achieved.

The concept of the pre-production year being replaced by two initial years with reduced production is considered very attractive as there will be a cash return from year one. Production from the Main Pit area has therefore been re-scheduled to adjust for no pre-production period.

3.2.4 Underground (Fig. 2.6)

Assuming that one of the above production plans is adopted, there will remain three zones containing what appears to be payable ore. These zones are:

| | | | |
|---|---|---|---|
| i) | D.S.L. | (1040mE - 1120mE) | 118500 t @ 1.39% Sn |
| ii) | Porphyry | (1100mE - 1200mE) | 72460 t @ 0.85% Sn |
| iii) | Porphyry | (1380mE - 1480mE) | 166060 t @ 0.80% Sn |

3.3 Mineable Ore Reserves

3.3.1 General

In order to establish the optimum mining depth, all available cross sections, indicating the ore reserve block outlines, were examined in detail. The result of this exercise was that the +570m RL was established as the optimum depth. The open pit final limits were constructed on all sections and plans and the ore reserve blocks contained within these parameters were calculated.

For the purpose of optimisation the 0.2% Sn Porphyry ore reserve and the 0.3% Sn D.S.L. ore reserve were accepted as cut-off grades.

Details of all blocks included in the mineable ore reserves (M O/R) are listed in Appendix 1.

3.3.2 Main Pit Area

Section lines 940mE - 1240mE establish the east and west boundaries of the mineable ore reserves in this area.

The following tabulations summarise the D.S.L. M O/R (by elevation), the Porphyry M O/R (by section line) and the pp reserve (by section line) (Tables 2(i) - (iv)).

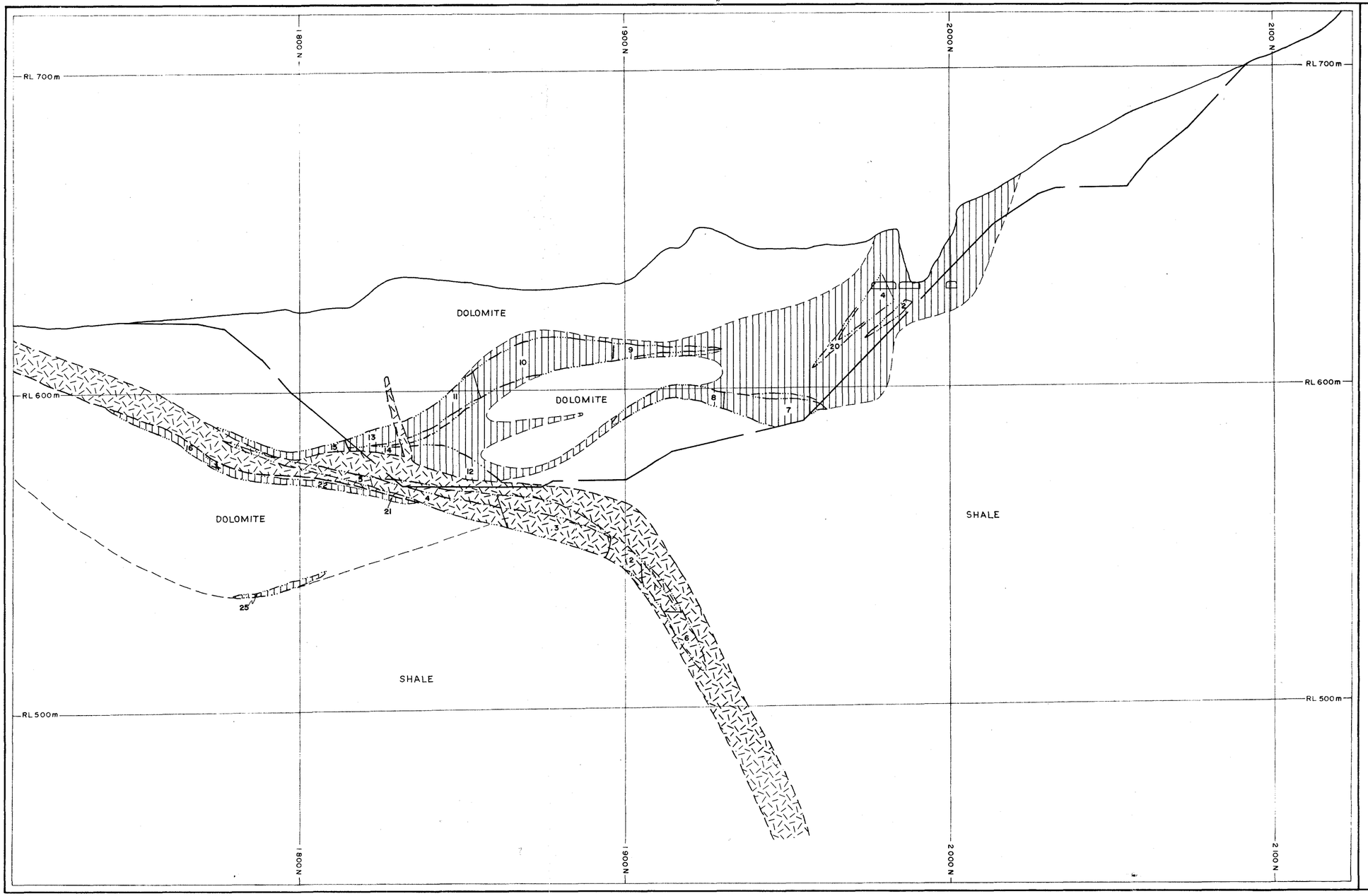


FIG. 2-1.

## LEGEND

### GEOLOGY

- GEOLOGICAL BOUNDARY
- PORPHYRY
- DOLOMITE SULPHIDE
- 3 ORE BLOCK

### PIT DESIGN

- ORIGINAL SURFACE
- SURFACE AFTER MINING TO ULTIMATE PIT LIMIT

5 cm

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# PIT OPTIMISATION STUDY

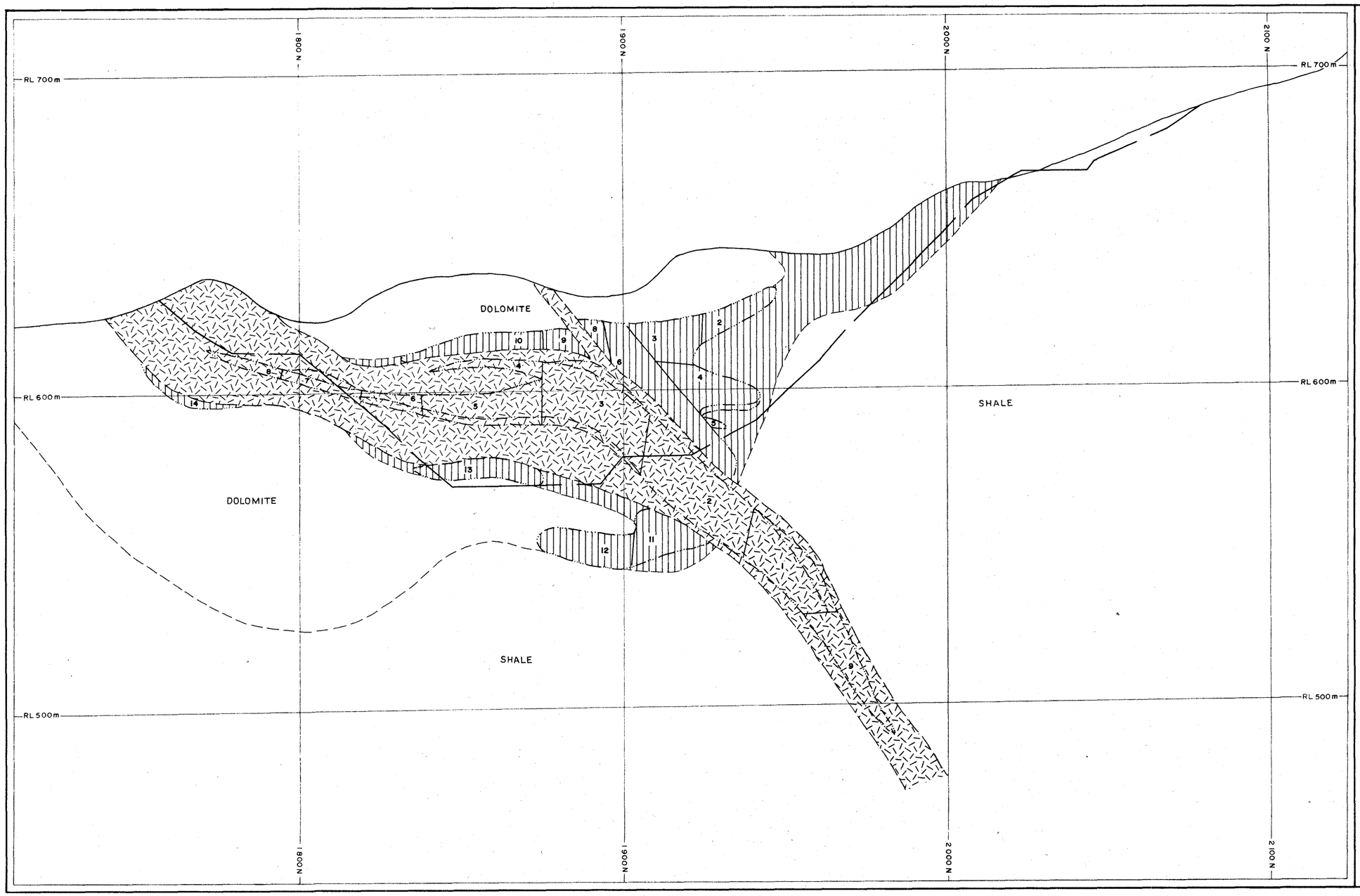
## CROSS SECTION 1000 mE

SCALE 1:1000 LOOKING WEST

APRIL 1984 DRG. NO. 84-079.

0208

0208 381021 FIG. 2-2.




## LEGEND

### GEOLOGY

- GEOLOGICAL BOUNDARY
- PORPHYRY
- DOLOMITE SULPHIDE
- 3 ORE BLOCK

### PIT DESIGN

- ORIGINAL SURFACE
- SURFACE AFTER MINING TO ULTIMATE PIT LIMIT

5 cm

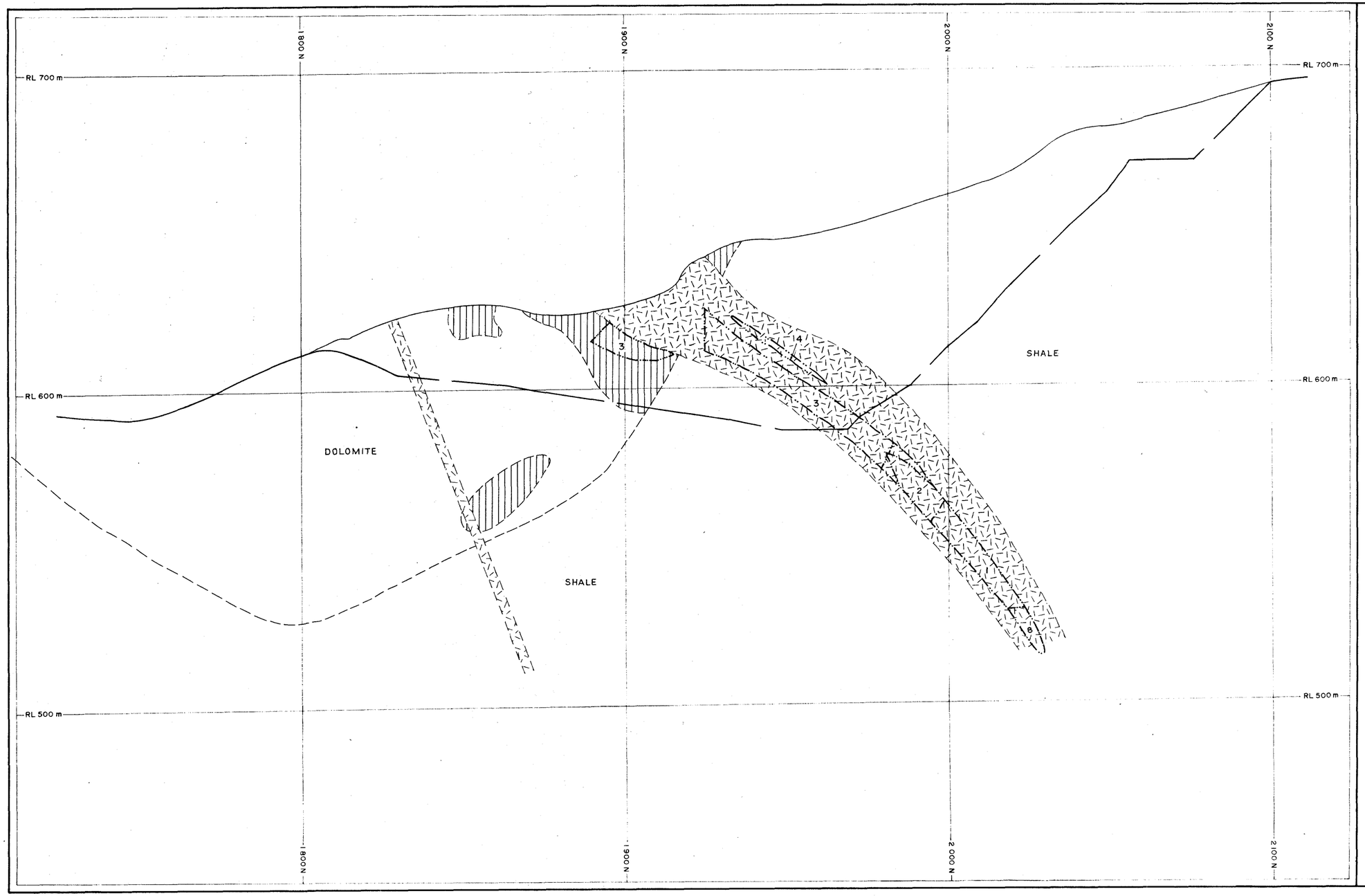
METALS EXPLORATION LIMITED
MT. BISCHOFF JOINT VENTURE

# PIT OPTIMISATION STUDY

## CROSS SECTION 1060 mE

SCALE 1:1000 LOOKING WEST
APRIL 1984 DRG. NO. 84-078.

021A




021B 381022 FIG. 2-3.

## LEGEND

### GEOLOGY

- GEOLOGICAL BOUNDARY
- PORPHYRY
- DOLOMITE SULPHIDE
- 3 ORE BLOCK

### PIT DESIGN

- ORIGINAL SURFACE
- SURFACE AFTER MINING TO ULTIMATE PIT LIMIT

5 cm

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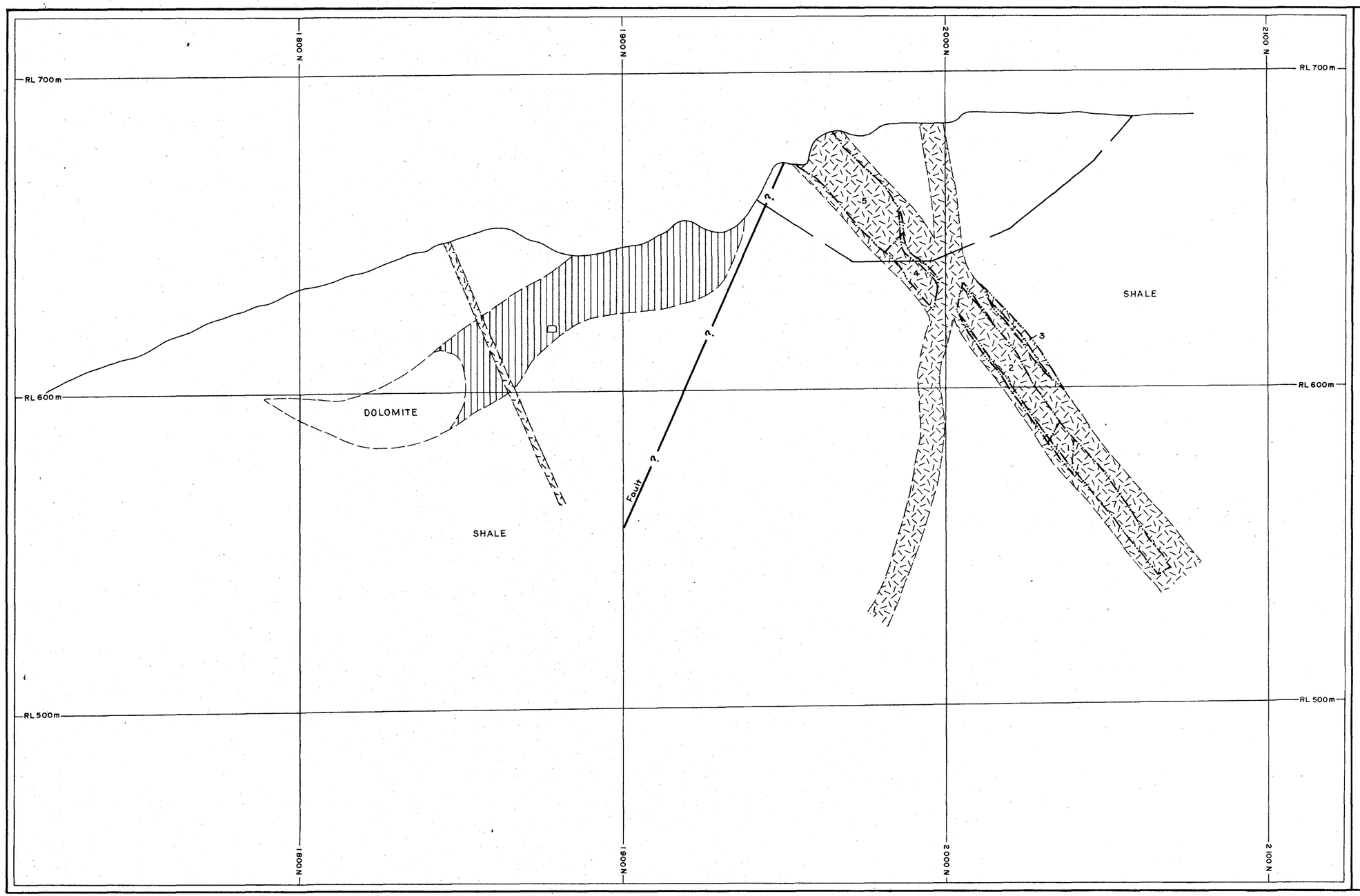
MT. BISCHOFF JOINT VENTURE

# PIT OPTIMISATION STUDY

CROSS SECTION 1140 mE
SCALE 1:1000 LOOKING WEST
APRIL 1984 DRG. NO. 84-077

FIG. 2-4.




## LEGEND

### GEOLOGY

- GEOLOGICAL BOUNDARY
- PORPHYRY
- DOLOMITE SULPHIDE
- (3) ORE BLOCK

### PIT DESIGN

- ORIGINAL SURFACE
- SURFACE AFTER MINING TO ULTIMATE PIT LIMIT

5 cm

METALS EXPLORATION LIMITED

MT. BISCHOFF JOINT VENTURE

# PIT OPTIMISATION STUDY

## CROSS SECTION 1240mE

SCALE 1:1000 LOOKING WEST

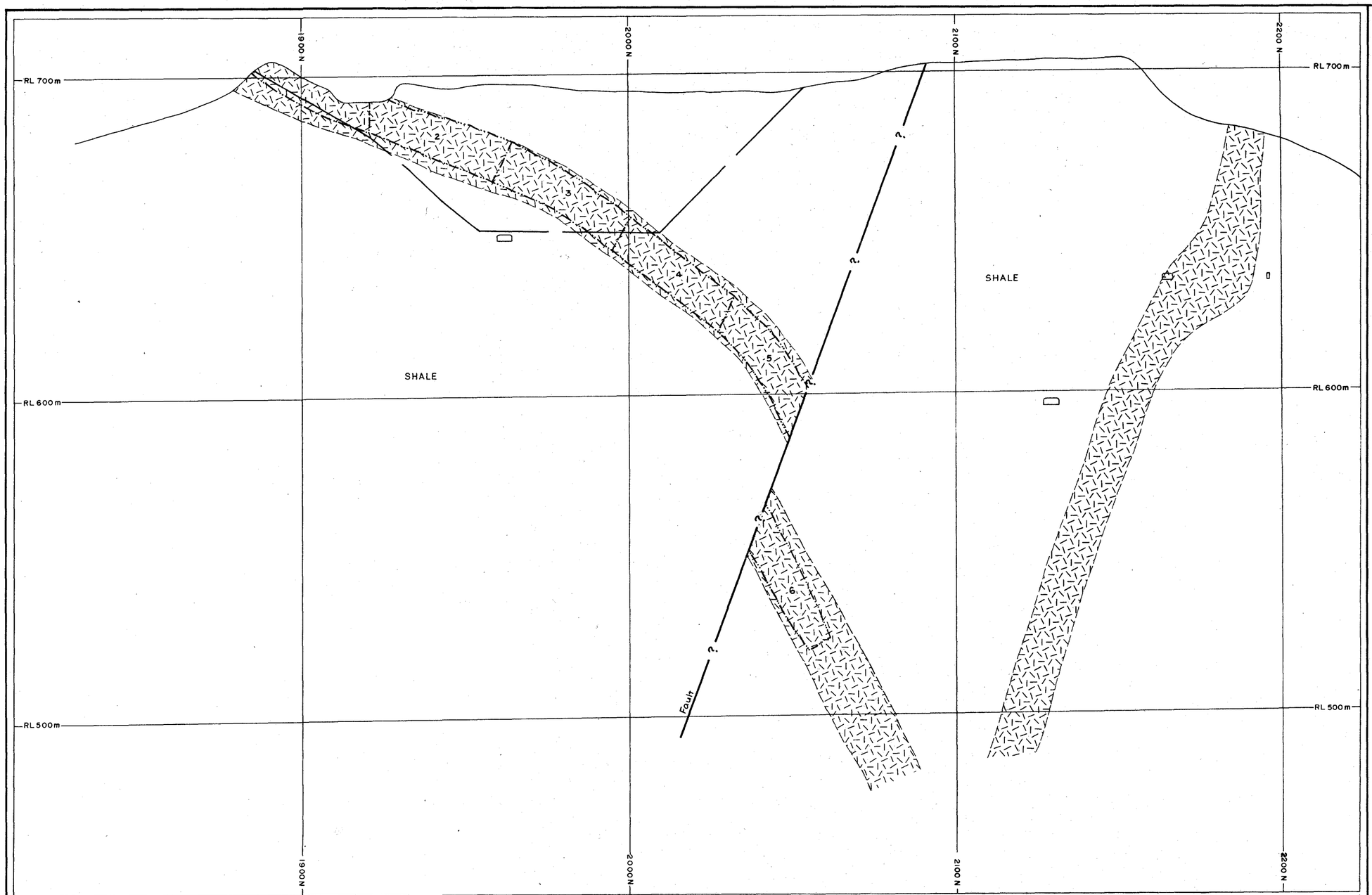
APRIL 1984 DRG. NO. 84-099

023A

381024

023B

FIG. 2-5.




## LEGEND

### GEOLOGY

- GEOLOGICAL BOUNDARY
- PORPHYRY
- DOLOMITE SULPHIDE
- 3 ORE BLOCK

### PIT DESIGN

- ORIGINAL SURFACE
- SURFACE AFTER MINING TO ULTIMATE PIT LIMIT

5 cm

METALS EXPLORATION LIMITED

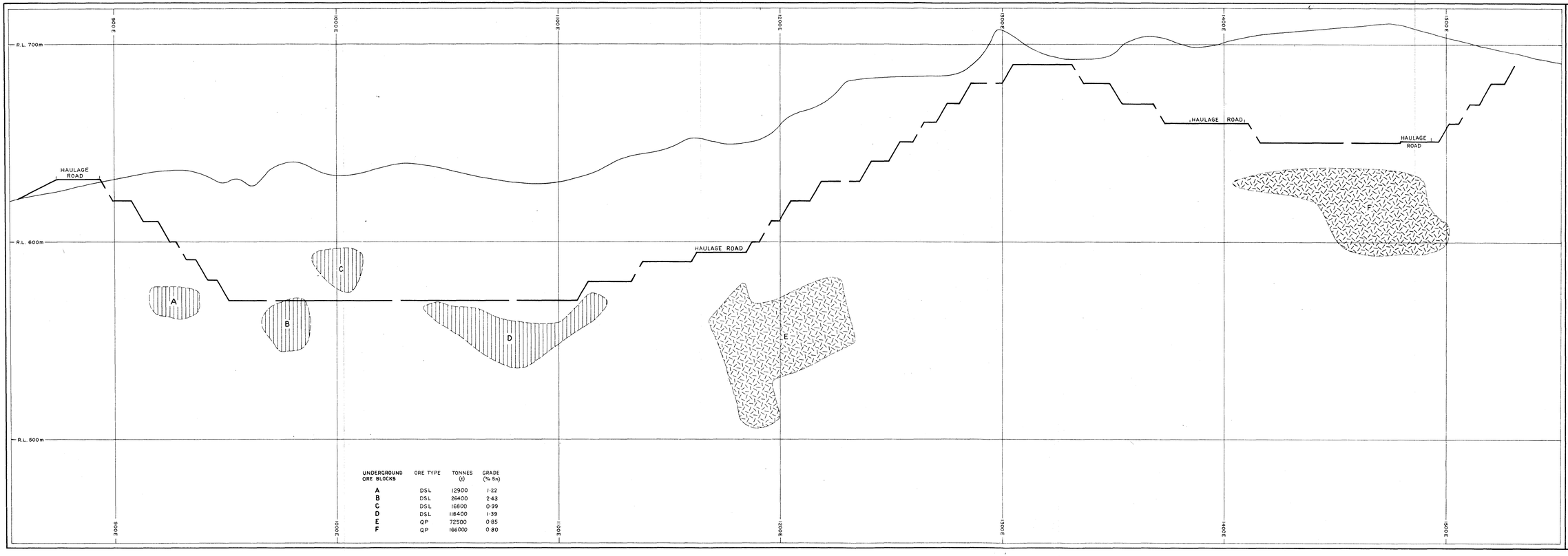
MT. BISCHOFF JOINT VENTURE

# PIT OPTIMISATION STUDY

## CROSS SECTION 1380mE

SCALE 1:1000 LOOKING WEST

APRIL 1984 DRG. NO. 84-098




| UNDERGROUND ORE BLOCKS | ORE TYPE | TONNES (t) | GRADE (% Sn) |
|---|---|---|---|
| A | DSL | 12900 | 1·22 |
| B | DSL | 26400 | 2·43 |
| C | DSL | 16800 | 0·99 |
| D | DSL | 118400 | 1·39 |
| E | QP | 72500 | 0·85 |
| F | QP | 166000 | 0·80 |

FIG. 2-6.

## LEGEND

GEOLOGY

- GEOLOGICAL BOUNDARY
- PORPHYRY
- DOLOMITE SULPHIDE
- 3 ORE BLOCK

PIT DESIGN

- ORIGINAL SURFACE
- SURFACE AFTER MINING TO ULTIMATE PIT LIMIT

5 cm

METALS EXPLORATION LIMITED
MT. BISCHOFF JOINT VENTURE

# PIT OPTIMISATION STUDY

LONGITUDINAL SECTION A-A
LOOKING NORTH-WEST

SCALE 1:1000

MAY 1984 DRG NO. 84-104

Table 2 (i)

MAIN PIT DSL MINEABLE ORE RESERVE
(PROVEN - PROBABLE - POSSIBLE)

| | PROVEN | | | PROBABLE | | | POSSIBLE | | |
|---|---|---|---|---|---|---|---|---|---|
| LEVEL m R.L. | Tonnes | % | t x % | Tonnes | % | t x % | Tonnes | % | t x % |
| Slaughter Yard Area 635 & Above | | | | 7000 | 0.48 | 3360 | 35000 | 1.00 | 35000 |
| 630 | 5232 | 0.91 | 4762 | 29308 | 1.03 | 30248 | 24688 | 0.72 | 17721 |
| 625 | 7104 | 0.95 | 6794 | 57347 | 1.05 | 60264 | 45844 | 0.65 | 29922) |
| 620 | 8785 | 1.10 | 9224 | 68750 | 0.96 | 66272 | 47980 | 0.70 | 33723) |
| 615 | 12117 | 1.07 | 12962 | 72903 | 0.99 | 71988 | 14239 | 1.23 | 17548) |
| 610 | 10584 | 1.21 | 12784 | 77196 | 1.08 | 83730 | - | | - ) |
| 605 | 9992 | 1.12 | 11192 | 64554 | 0.92 | 59580 | - | | - ) |
| 600 | - | - | - | 66845 | 1.01 | 67283 | - | | - ) |
| 595 | - | - | - | 105067 | 0.80 | 83991 | - | | - ) |
| 590 | - | - | - | 75556 | 0.85 | 63857 | - | | - ) |
| 585 | - | - | - | 52420 | 0.72 | 37650 | - | | - ) |
| 580 | - | - | - | 28619 | 0.67 | 19226 | - | | - ) |
| 575 | - | - | - | 28558 | 0.76 | 21791 | - | | - ) |
| 570 | - | - | - | 7599 | 0.95 | 7278 | | | ) |
| TOTAL | 53814t | | 57718 | 741717t | | 676468 | 167751t | | 133914 |
| GRADE | 1.07%Sn | | | 0.91% Sn | | | 0.80% Sn | | |
| COMBINED TOTAL | | | | 963282t | | 868100 | | | |
| GRADE | | | | 0.90% Sn | | | | | |

Table 2 (ii)

MAIN PIT PORPHYRY MINEABLE ORE RESERVE
(PROBABLE - POSSIBLE)

| SECTION m E | Tonnes | PROBABLE % | t x % | Tonnes | POSSIBLE % | t x % |
|---|---|---|---|---|---|---|
| 940 | | Nil | | | Nil | |
| 960 | | Nil | | | Nil | |
| 980 | 22491 | 0.28 | 6331 | | | |
| 1000 | 1656 | 0.37 | 166 | | | |
| 1020 | 1740 | 0.51 | 887 | | | |
| 1040 | 62466 | 0.55 | 97019 | | | |
| 1060 | 81486 | 0.46 | 37639 | | | |
| 1080 | 93348 | 0.39 | 36104 | 3000 | 0.80 | 2400 |
| 1100 | 66432 | 0.38 | 25231 | 3840 | 0.36 | 1382 |
| 1120 | 30420 | 0.32 | 9767 | 15060 | 0.89 | 13403 |
| 1140 | 27252 | 0.24 | 6484 | 18120 | 0.33 | 5906 |
| 1160 | 70884 | 0.54 | 38615 | 11820 | 0.50 | 5910 |
| 1180 | 44709 | 0.66 | 29445 | 14940 | | 3137 |
| 1200 | 70794 | 0.49 | 34568 | | | |
| 1220 | 43637 | 0.47 | 20628 | | | |
| 1240 | 27637 | 0.47 | 13126 | 6720 | 0.32 | 2150 |
| TOTAL | 644951t | | 293544 | 74820t | | 34710 |
| GRADE | 0.46% Sn | | | 0.46% Sn | | |
| COMBINED TOTAL | 719771t | | 328254 | | | |
| GRADE | 0.46% Sn | | | | | |

Table 2 (iii)

MAIN PIT - MINEABLE ORE RESERVE
(PP)

| SECTION m E | BLOCK NO | LEVEL m R.L. | % Sn | tonnes | t x % |
|---|---|---|---|---|---|
| 960 | 17PP | 615 | 0.49 | 8120 | 3979 |
| 980 | | | | | |
| 1000 | 24PP | 620 | 0.59 | 3150 | 1859 |
| 1020 | 21PP | 620 | 0.46 | 3500 | 1610 |
| | 20PP | 625/630 | 0.48 | 17430 | 8366 |
| 1040 | 13PP | 625 | 0.51 | 1820 | 928 |
| | 12PP | 620 | 0.44 | 22330 | 9825 |
| | | 625 | | | |
| | | 630 | | | |
| 1060 | 10PP | 620/625 | 0.91 | 2870 | 2612 |
| 1100 | 10PP | 615/620 | | | |
| | 9PP | 620 | | | |
| 1120 | 7PP | 675 | 0.37 | 5950 | 2202 |
| | 2PP | 640 | 0.73 | 1540 | 1124 |
| TOTAL | | | | 66710t | 32505 |
| GRADE | | | | 0.49% Sn | |

Table 2 (iv)

MAIN PIT MINEABLE ORE RESERVE SUMMARY

| ORE SOURCE | TONNES | % Sn | t x % Sn |
|---|---|---|---|
| DSL | 963282 | 0.90 | 868100 |
| PORPHYRY | 719771 | 0.46 | 328254 |
| PP | 66710 | 0.49 | 32505 |
| TOTAL | 1749763 | 0.70 | 1228859 |

### 3.3.3 Stanhope Extension

Section lines 1300mE - 1500mE establish the east and west boundaries of the mineable ore reserves in the Stanhope area. The following tabulation summarises the Mineable Ore Reserve.

Table 2 (v)

STANHOPE EXTENSION PIT MINEABLE ORE RESERVE

| SECTION mE | TONNES | % Sn | t x % |
|---|---|---|---|
| 1500 | 5640 | 0.28 | 1579 |
| 1480 | 33180 | 0.35 | 11468 |
| 1460 | 44160 | 0.73 | 32237 |
| 1440 | 36360 | 0.45 | 16428 |
| 1420 | 46440 | 0.33 | 15120 |
| 1400 | 65400 | 0.30 | 19785 |
| 1380 | 78720 | 0.39 | 30624 |
| 1360 | 69690 | 0.37 | 25565 |
| 1340 | 56700 | 0.36 | 20537 |
| 1320 | 37920 | 0.39 | 14676 |
| 1300 | 24480 | 0.41 | 10037 |
| TOTAL | 498690t | 0.40 | 198056 |

### 3.3.4 Total

As a result of the Stanhope Extension the total Mineable Ore Reserve is calculated to be as follows:

| | |
|---|---|
| Main Pit | 1749763 tonnes @ 0.70% Sn |
| Stanhope Extension Pit | 498690 tonnes @ 0.40% Sn |
| Total | 2248453 tonnes @ 0.63% Sn |

This total represents 45% of the total Geological Ore Reserve.

3.4 Mining Recovery and Dilution (Fig. 3.1)

1. Recovery - All ore reserve block outlines are established by assay cut-offs (not by geological boundaries). Due to this a certain loss of ore is to be expected and a factor of 10% for both D.S.L. and Porphyry ore is considered suitable.

2. Dilution - The effect of dilution on mineable ore reserves is considered extremely significant, particularly with the D.S.L. ore, for which past studies have estimated 10% to 22% dilution. The Douglas McKenna and Partners Ore Reserves have ore blocked out with different configurations than used in the past and it is therefore necesary to re-assess the dilution factor.

   Three cross sections, representing the ore block configuration for both types of ore, were selected for examination. On these sections 5m (DSL) and 10m (Porphyry) high mining blocks were constructed and the amount of ore loss and waste inclusion were measured and computed (Fig. 3.1).

   In the Porphyry ore blocks, all waste material included within the mining blocks is situated inside the geological porphyry boundaries and a grade of 0.1% Sn has therefore been assigned to this area.

   When considering the DSL ore blocks, a grade of 0.1% Sn has been assigned to the waste. This is due to the limits of the block being assay cut-offs and the waste inclusion will be sulphide material with an assumed tin content.

   Results of the exercise are as follows:

   D.S.L. 90% recovery of mineable ore reserve @ 18% dilution
   Porphyry 90% recovery of mineable ore reserve @ 11% dilution

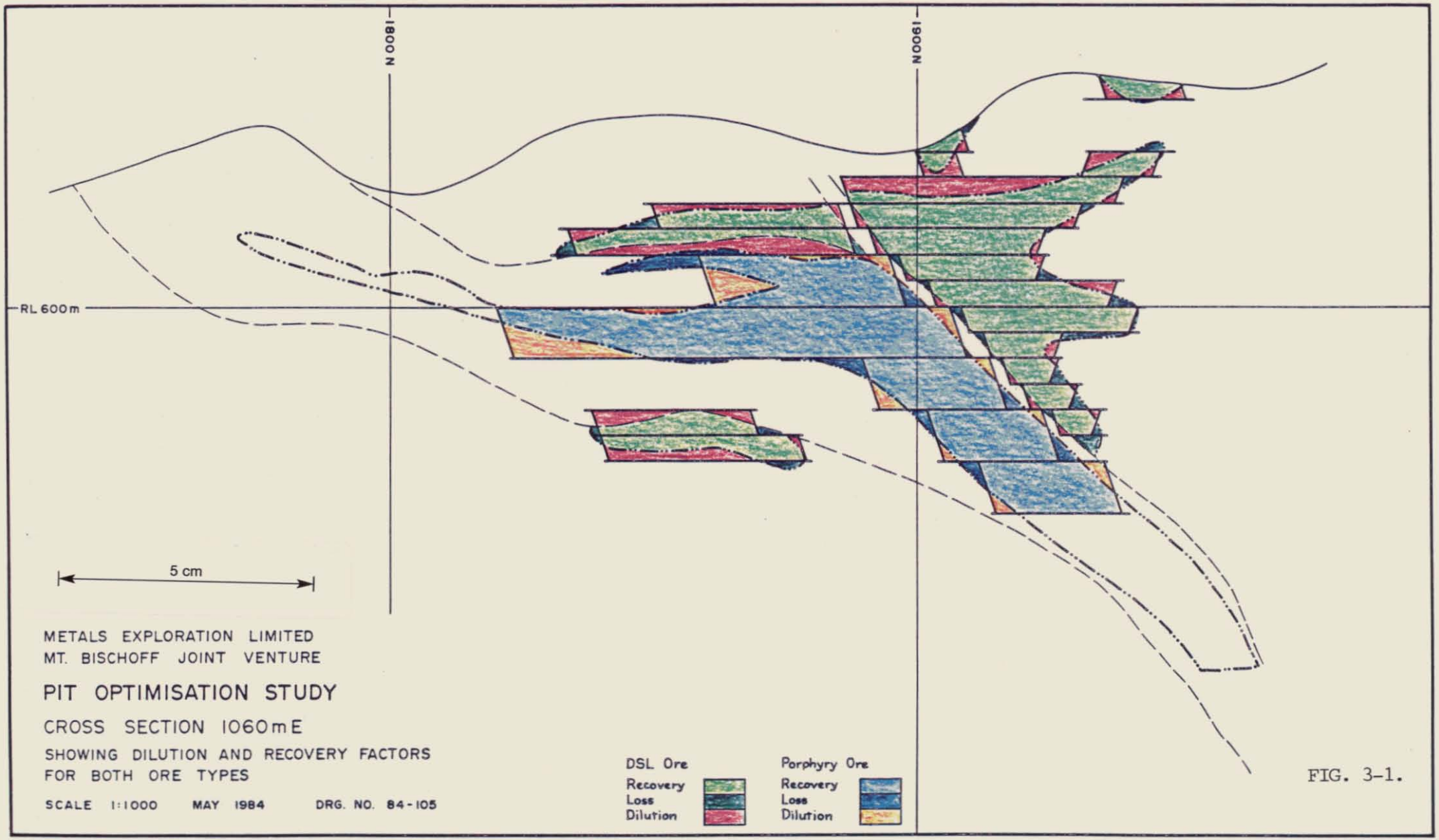
4. OPERATIONAL STRATEGY

4.1 General

The main changes in rationale in operational strategy are:

1. mining/earthmoving contractors are to be used not only for the development work but for the total operation throughout the life of the mine, and

2. all ore will be freighted to the Cleveland concentrator and fed directly into the concentrator via the primary crusher without pre-concentration.




METALS EXPLORATION LIMITED
MT. BISCHOFF JOINT VENTURE

## PIT OPTIMISATION STUDY

CROSS SECTION 1060mE

SHOWING DILUTION AND RECOVERY FACTORS
FOR BOTH ORE TYPES

SCALE 1:1000 MAY 1984 DRG. NO. 84-105

FIG. 3-1.

031

4.2 Main Pit (Pre-Production) (Table 3 (i))

1. Pre-Production (Year - 1)

This period is to allow for general establishment of the operation by the Joint Venture on-site personnel and for the construction of haul roads and the development work necessary by contractors.

(i) Haul Roads - to be constructed around the designed pit perimeter up to the +690m RL. It will be necessary to drill and blast the road to establish the required gradient, with the majority of the waste being dozed down on the western and north western side of the road. It will be necessary to establish safety benches above and on the north side of the road between the designed +645m RL and the +670m RL. Waste material in this area may either be dozed to the west of the haul road or into the existing glory hole. The road should be blasted below the required RL's and then surfaced with road metal from the nearest possible source. Drains 3.0m wide by 2.0m deep should be excavated on the north side of the haul road. This is to redirect any 'run off' water away from the main open pit. The drain will deliver into the river below the +600m RL.

(ii) Bench establishment

Bench 'cut offs' will be excavated concurrently with the haul road. These cut-offs are to extend along the designed mining horizon to a distance far enough away from the road to ensure no damage to the haul road results from blasting on any individual bench.

It will be necessary to remove approximately 200 000 BCM of overburden above the +680m RL to allow access to the Porphyry ore on that horizon.

(ii) Approximately 500 000 BCM of overburden is rippable using a Cat D9 bulldozer. It will be at the discretion of the selected contractor how he mines this material. The only Joint Venture restriction being that production of the 67 000 tonnes of ore available on the surface is maximised and stockpiled.

2. Production (Year 1 onwards)

D.S.L. ore mining will commence on the +610m RL horizon with immediate access through the Pig Flat area. Porphyry ore mining will commence on the +670m RL horizon from the haul road and on the +610m RL horizon when sufficient waste has been removed to expose the 'pay zone'.

Forecast ore production in Year 1 is kept to a minimum, to allow for start up and selling in 'hold ups' and to enable sufficient waste material to be removed to allow for efficient mining of the scheduled tonnes from Year 2 onwards.

Year 2

It will be necessary to cut 'slots' through the waste material on certain benchs to gain access to certain ore blocks to maintain the required ore tonnes at the established stripping ratio.

Ore and waste mining will continue in five distinct areas:
(i) From +680m RL retreating down to +680m RL.
(ii) From +610m RL and +620m RL retreating down to +570m RL.

Care must be taken in (ii) that sufficient waste material is removed annually, on each bench to ensure that subsequent 'drop-cuts' are developed to access ore on lower horizons.

4.3 Main & Stanhope Extension Pits (Table No. 3 (ii))

1. Years 1 and 2

(i) Ore

Immediate access to the Main Pit +610m RL horizon is possible via the 'Pig Flat' area, this allows 70 000 tonnes of high grade (1.06% Sn) D.S.L. ore to be mined.

During the mining of the D.S.L. the permanent access road will be established up to the +700m RL horizon in the Stanhope area in the same manner as described in Section 4.2. Prior to the permanent road establishment all waste above the + 700m RL will be drilled, blasted and dozed into the 'Gloryhole'. A drop cut to the +690m RL will be established on the footwall side of the Stanhope dyke. This will release 60 000 tonnes of Porphyry to be mined in Year 1. Continuation of the drop cut in the footwall allows ore to be exposed quickly on all horizons.

All the rippable surface ore will be excavated in the initial year and totals 67 000 tonnes.

(ii) Waste

Haul road development is the main source of waste in the initial years and where possible will be dozed away from the pit limits.

The waste designated 'rippable' will also be removed in Years 1 and 2.

To maintain DSL recovery from +610m and +620m RL's it is necessary to remove approximately 100 000 BCM from each horizon.

# PRODUCTION SCHEDULE
## MAIN PIT AND PRE-PRODUCTION

TABLE 3(i)

| BENCH | SOURCE | YEAR -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| ROAD | | | | | | | | | | | | | |
| 700 | WASTE | 16700 | | | | | | | | | | | 16700 |
| 690 | WASTE | 37300 | | | | | | | | | | | 37300 |
| 680 | WASTE | 140400 | | | | | | | | | | | 140400 |
| | PP O/RESERVE | 22200 | | | | | | | | | | | 22200 |
| 670 | WASTE | 150000 | 82700 | | | | | | | | | | 232700 |
| | PORPHYRY | | 1100 | | | | | | | | | | 1100 |
| 660 | WASTE | 10000 | 100000 | 100000 | 18800 | | | | | | | | 228800 |
| | PORPHYRY | | | | 4500 | | | | | | | | 4500 |
| 650 | WASTE | 10000 | | 100000 | 100000 | 29900 | | | | | | | 239900 |
| | PORPHYRY | | | | 5000 | 4500 | | | | | | | 9510 |
| | DSL | | | | 3200 | 4000 | | | | | | | 7200 |
| 640 | WASTE | 10000 | | | | 150000 | 149300 | | | | | | 309300 |
| | PORPHYRY | | | | | | 14300 | | | | | | 14300 |
| | DSL | | | | | 4800 | | | | | | | 4800 |
| 630 | WASTE | 10000 | | | | | | 150000 | 150000 | 60000 | 44700 | | 414700 |
| | PORPHYRY | | | | | | | | | 12000 | 5000 | | 17000 |
| | DSL | | | | | | | 13400 | 5200 | 7200 | 1400 | | 16900 |
| 620 | WASTE | 10000 | 50000 | 100000 | 100000 | 100000 | 88700 | | | | | | 448700 |
| | PORPHYRY | | | 10000 | 10000 | 8000 | 5600 | | | | | | 33600 |
| | DSL | | | 10000 | 22400 | 20000 | 15000 | | | | | | 67400 |
| 610 | WASTE | 10000 | 200000 | 120000 | 116500 | | | | | | | | 446500 |
| | PORPHYRY | | 20000 | 17900 | | | | | | | | | 37900 |
| | DSL | | 20000 | 20000 | 13400 | | | | | | | | 53400 |
| 600 | WASTE | | | | 15000 | 50000 | 100000 | 150000 | 28800 | | | | 343800 |
| | PORPHYRY | | | | | 15000 | 10000 | 8500 | | | | | 33500 |
| | DSL | | | | | | 10000 | 30000 | 400 | | | | 40400 |
| 590 | WASTE | | | | | | 15000 | 40000 | 100000 | 40300 | | | 205300 |
| | PORPHYRY | | | | | | | 10000 | 30000 | 900 | | | 40900 |
| 580 | WASTE | | | | | | | | 15000 | 40000 | 60000 | 25870 | 140870 |
| | PORPHYRY | | | | | | | | 2000 | 9200 | 18000 | | 29200 |
| | DSL | | | | | | | | | 2800 | 3800 | 16600 | 23200 |
| 570 | WASTE | | | | | | | | | | 15000 | 63100 | 78100 |
| | PORPHYRY | | | | | | | | | | | 15800 | 15800 |
| | DSL | | | | | | | | | | | 10300 | 10300 |
| TOTALS | WASTE M3 | 404400 | 432700 | 420000 | 350300 | 329900 | 338000 | 340000 | 293800 | 140000 | 119700 | 88970 | 3257770 |
| | PORPHYRY M3 | | 21100 | 27900 | 19500 | 27500 | 29900 | 18500 | 32000 | 22100 | 23000 | 15800 | 237300 |
| | GRADE (% Sn) | | 0.44 | 0.45 | 0.49 | 0.52 | 0.52 | 0.46 | 0.40 | 0.49 | 0.48 | 0.54 | 0.48 |
| | DSL M3 | | 20000 | 30000 | 39000 | 28800 | 25000 | 33400 | 21200 | 31000 | 20200 | 26900 | 275500 |
| | GRADE (% Sn) | | 1.06 | 1.00 | 0.94 | 0.88 | 0.91 | 0.97 | 0.84 | 0.83 | 0.81 | 0.74 | 0.90 |
| | PORPHYRY t | | 63300 | 83700 | 58500 | 82500 | 89700 | 55500 | 96000 | 66300 | 69000 | 47400 | 711900 |
| | DSL t | | 70000 | 105000 | 136500 | 100800 | 87500 | 116900 | 74200 | 108500 | 70700 | 94150 | 964250 |
| DSL | | | | | | | | | | | | | |
| 90% RECOVERY ) | tonnes | | 74340 | 111510 | 144963 | 107050 | 92925 | 124148 | 78800 | 115227 | 75083 | 99990 | 1024036 |
| 18% DILUTION @ 0.1%) | % Sn | | 0.91 | 0.86 | 0.81 | 0.76 | 0.79 | 0.84 | 0.73 | 0.72 | 0.70 | 0.64 | 0.78 |
| PORPHYRY | | | | | | | | | | | | | |
| 90% RECOVERY ) | tonnes | | 63237 | 83616 | 58442 | 82420 | 89610 | 55445 | 95904 | 66234 | 68931 | 47353 | 711192 |
| 11% DILUTION @ 0.1%) | % Sn | | 0.41 | 0.42 | 0.45 | 0.48 | 0.48 | 0.42 | 0.37 | 0.45 | 0.44 | 0.50 | 0.43 |
| TOTAL TONNES | | | 137577 | 195126 | 203405 | 189470 | 182535 | 179593 | 174704 | 181461 | 144014 | 147343 | 1735228 |
| GRADE (% Sn) | | | 0.68 | 0.67 | 0.71 | 0.64 | 0.64 | 0.71 | 0.53 | 0.62 | 0.58 | 0.60 | 0.64 |
| ORE : WASTE M3:M3 | | | 1:10.2 | 1:7.0 | 1:5.8 | 1:5.7 | 1:6.0 | 1:6.3 | 1:5.4 | 1:2.5 | 1:2.7 | 1:2.0 | 1:6.2 |

## PRODUCTION SCHEDULE
## MAIN AND STANHOPE EXTENSION PITS

TABLE NO. 3(ii)

| BENCH | SOURCE | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | m3 | %Sn | m3x% | WASTE M3 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| ROAD | | | | | | | | | | | | | | | |
| 710 | WASTE (S) | 8800 | | | | | | | | | | | | | 8800 |
| 700 | WASTE (S) | 71400 | | | | | | | | | | | | | 71400 |
| | " (WF) | 16750 | | | | | | | | | | | | | 16750 |
| 690 | WASTE (S) | 100000 | 70000 | 25200 | | | | | | | | | | | 195200 |
| | " (WF) | 37300 | | | | | | | | | | | | | 37300 |
| | ORE (S) | 20000 | 6700 | | | | | | | | | 26700 | 0.42 | 11214 | |
| 680 | WASTE (S) | | 100000 | 70000 | 27200 | | | | | | | | | | 197200 |
| | " (WF) | 140400 | | | | | | | | | | | | | 140400 |
| | ORE (S) | | 30000 | | | | | | | | | 30000 | 0.33 | 9900 | |
| | PP O/R | 22200 | | | | | | | | | | 22200 | 0.49 | 10878 | |
| 670 | WASTE (S) | | 50000 | 100000 | 100000 | 76900 | | | | | | | | | 326900 |
| | " (WF) | 150000 | 82700 | | | | | | | | | | | | 232700 |
| | ORE (S) | | | 35000 | 2600 | | | | | | | 37600 | 0.41 | 15416 | |
| | ORE (WF) | | 1100 | | | | | | | | | 1100 | 0.54 | 594 | |
| 660 | WASTE (S) | | | 20000 | 70000 | 57700 | | | | | | | | | 147700 |
| | " (WF) | 10000 | 100000 | 100000 | 18800 | | | | | | | | | | 228800 |
| | ORE (S) | | | | 38300 | | | | | | | 38300 | 0.46 | 17618 | |
| | ORE (WF) | | | | 4500 | | | | | | | 4500 | 0.54 | 2430 | |
| 650 | WASTE (S) | | | | 20000 | 49500 | | | | | | | | | 69500 |
| | " (WF) | 10000 | | 100000 | 100000 | 29900 | | | | | | | | | 239900 |
| | ORE (S) | | | | 3000 | 33000 | | | | | | 36000 | 0.37 | 13320 | |
| | ORE (WF) | | | | 5000 | 4500 | | | | | | 9500 | 0.46 | 2300 | |
| | DSL | | | | 3200 | 4000 | | | | | | 7200 | 0.91 | | |
| 640 | WASTE (WF) | 10000 | | | | 150000 | 149300 | | | | | | | | 309300 |
| | ORE (HF) | | | | | | 14300 | | | | | 14300 | 0.52 | 7436 | |
| | DSL | | | | | 4800 | | | | | | 4800 | 0.91 | | |
| 630 | WASTE (WF) | 10000 | | | | 100000 | | 100000 | 100000 | 60000 | 44700 | | | | 414700 |
| | ORE (WF) | | | | | | | | | 12000 | 5000 | 17000 | 0.50 | 8500 | |
| | DSL | | | | | | 2000 | 5000 | 5000 | 3000 | 1900 | 16900 | 0.89 | | |
| 620 | WASTE (WF) | 100000 | 50000 | 100000 | 100000 | 100000 | 88700 | | | | | | | | 448700 |
| | ORE (WF) | | | 10000 | 10000 | 8000 | 5600 | | | | | 33600 | 0.49 | 16464 | |
| | DSL | | | 10000 | 22400 | 20000 | 15000 | | | | | 67400 | 0.87 | | |
| 610 | WASTE | 110000 | 100000 | 120000 | 116500 | | | | | | | | | | 446500 |
| | ORE (WF) | | 10000 | 17900 | 10000 | | | | | | | 37900 | 0.43 | 16297 | |
| | DSL | 20000 | 10000 | 13400 | 10000 | | | | | | | 53400 | 1.06 | | |
| 600 | WASTE | | | | 15000 | 50000 | 120000 | 150000 | 8800 | | | | | | 343800 |
| | IRE (WF) | | | | | 15000 | 10000 | 8500 | | | | 33500 | 0.55 | 18425 | |
| | DSL | | | | | 500 | 15000 | 15000 | 9900 | | | 40400 | 0.98 | 39592 | |
| 590 | WASTE | | | | | 15000 | 40000 | 100000 | 40300 | | | | | | 195300 |
| | ORE (WF) | | | | | | 10000 | 20000 | 10900 | | | 40900 | 0.39 | 15951 | |
| | DSL | | | | | | 14100 | 10500 | 15000 | 10000 | 2000 | 51600 | 0.82 | | |
| 580 | WASTE | | | | | | | 15000 | 40000 | 60000 | 25870 | | | | 140870 |
| | ORE (WF) | | | | | | | 2000 | 9200 | 18000 | | 29200 | 0.48 | 14016 | |
| | DSL | | | | | | | | 2800 | 3800 | 16600 | 23200 | 0.70 | | |
| 570 | WASTE | | | | | | | | | 15000 | 63100 | | | | 78100 |
| | ORE (WF) | | | | | | | | | | 15800 | 15800 | 0.54 | 8532 | |
| | DSL | | | | | | | | | | 10300 | 10300 | 0.80 | | |
| TOTAL | WASTE | 684650 | 552700 | 635200 | 567500 | 529000 | 498000 | 365000 | 189100 | 135000 | 133670 | | | | 4289820 |
| | PORPHYRY S | 20000/ 0.42 | 36700/ 0.35 | 35000/ 0.41 | 43900/ 0.45 | 33000/ 0.37 | | | | | | 168600 | 0.40 | 67560 | |
| | PORPHYRY WF | 22200/ 0.49 | 11100/ 0.44 | 27900/ 0.45 | 29500/ 0.47 | 27500/ 0.52 | 39900/ 0.49 | 30500/ 0.44 | 20100/ 0.43 | 30000/ 0.49 | 20800/ 0.53 | 259500 | 0.48 | 123820 | |
| | DSL | 20000/ 1.06 | 10000/ 1.06 | 23400/ 0.98 | 35600/ 0.93 | 29300/ 0.88 | 46100/ 0.89 | 30500/ 0.91 | 32700/ 0.87 | 16800/ 0.81 | 30800/ 0.75 | 275200 | 0.90 | 247565 | |
| | | 62200 | 57800 | 86300 | 109000 | 89800 | 86000 | 61000 | 52800 | 46800 | 51600 | 703300 | | | |
| tonnes | PORPHYRY | 126600/ 0.46 | 143400/ 0.37 | 188700/ 0.43 | 220200/ 0.46 | 181500/ 0.44 | 119700/ 0.49 | 91500/ 0.44 | 60300/ 0.43 | 90000/ 0.49 | 62400/ 0.53 | | | | |
| | DSL | 70000/ 1.06 | 35000/ 1.06 | 81900/ 0.98 | 124600/ 0.93 | 102550/ 0.88 | 161350/ 0.89 | 106750/ 0.91 | 114450/ 0.87 | 58800/ 0.81 | 107800/ 0.75 | | | | |
| DSL | | | | | | | | | | | | | | | |
| 90% RECOVERY | | 74340 | 37170 | 86978 | 132325 | 108908 | 171778 | 113369 | 121545 | 62446 | 114483 | 1023342 | | | |
| 18% DILUTION @ 0.1% | | 0.91 | 0.91 | 0.85 | 0.80 | 0.76 | 0.77 | 0.79 | 0.75 | 0.70 | 0.65 | 0.78 | | | |
| PORPHYRY | | | | | | | | | | | | | | | |
| 90% RECOVERY | | 66533 | 143257 | 188511 | 219980 | 181320 | 119580 | 91410 | 60240 | 39910 | 62340 | 1223081 | | | |
| 11% DILUTION @ 0.1% | | 0.49 | 0.34 | 0.40 | 0.42 | 0.41 | 0.45 | 0.41 | 0.40 | 0.45 | 0.49 | 0.41 | | | |
| TOTAL TONNES | | 140873 | 180427 | 275489 | 352300 | 290230 | 291358 | 204779 | 181785 | 152356 | 176823 | 2246420 | | | |
| % Sn | | 0.69 | 0.46 | 0.54 | 0.56 | 0.54 | 0.64 | 0.62 | 0.63 | 0.55 | 0.59 | 0.58 | | | |
| ORE : WASTE | | 1:15,8 | 1:9,5 | 1:7,2 | 1:5,1 | 1:5,8 | 1:6 | 1:5,8 | 1:3,5 | 1:2,8 | 1:2,5 | 1:6,1 | | | |

# PRODUCTION SCHEDULE
## MAIN PIT (NO PRE-PRODUCTION)

TABLE NO. 3(iii)

| BENCH | SOURCE | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | BCM | %Sn | M3XSn | WASTE |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 700 | WASTE | 16750 | | | | | | | | | | | | | 16750 |
| 690 | WASTE | 37300 | | | | | | | | | | | | | 37300 |
| 680 | WASTE | 140400 | | | | | | | | | | | | | 140400 |
| | PP O/R | 22200 | | | | | | | | | | 22200 | 0.49 | 11214 | |
| 670 | WASTE | 150000 | 82700 | | | | | | | | | | | | 232700 |
| | QP | | 1100 | | | | | | | | | 1100 | 0.54 | 594 | |
| 660 | WASTE | 10000 | 100000 | 100000 | 18800 | | | | | | | | | | 228800 |
| | QP | | | | 4500 | | | | | | | 4500 | 0.54 | 2430 | |
| 650 | WASTE | 10000 | 10000 | 10000 | 20000 | 9900 | | | | | | | | | 239900 |
| | QP | | | | 5000 | 4500 | | | | | | 9500 | 0.46 | 2800 | |
| | DSL | | | | 3200 | 4000 | | | | | | 7200 | 0.91 | 6552 | |
| 640 | WASTE | 10000 | | | 150000 | 100000 | 40000 | 9300 | | | | | | | 309300 |
| | QP | | | | | 14300 | | | | | | 14300 | 0.52 | 7436 | |
| | DSL | | | | | 4800 | | | | | | 4800 | 0.91 | 4368 | |
| 630 | WASTE | 10000 | | | | 50000 | 100000 | 100000 | 100000 | 40000 | 14700 | | | | 414700 |
| | QP | | | | | | | | 5000 | 7000 | 5000 | 17000 | 0.50 | 8500 | |
| | DSL | | | | | | 5000 | 5000 | 5000 | 1000 | 900 | 16900 | 0.89 | 15041 | |
| 620 | WASTE | 10000 | 50000 | 100000 | 100000 | 100000 | 88700 | | | | | | | | 448700 |
| | QP | | 10000 | 10000 | 8000 | 5600 | | | | | | 33600 | 0.49 | 16464 | |
| | DSL | | 10000 | 22400 | 20000 | 15000 | | | | | | 67400 | 0.87 | 58638 | |
| 610 | WASTE | 110000 | 100000 | 120000 | 116500 | | | | | | | | | | 446500 |
| | QP | | 10000 | 17900 | 10000 | | | | | | | 37900 | 0.43 | 16297 | |
| | DSL | 20000 | 10000 | 13400 | 10000 | | | | | | | 53400 | 1.06 | 56604 | |
| 600 | WASTE | | | | 15000 | 150000 | 120000 | 50000 | 8800 | | | | | | 343800 |
| | QP | | | | | 15000 | 10000 | 8500 | | | | 33500 | 0.55 | 18425 | |
| | DSL | | | | | 500 | 20000 | 15000 | 4300 | | | 40400 | 0.98 | 39592 | |
| 590 | WASTE | | | | | 15000 | 40000 | 100000 | 40300 | | | | | | 195300 |
| | QP | | | | | | 10000 | 20000 | 10900 | | | 40900 | 0.39 | 15951 | |
| | DSL | | | | | | 14100 | 10500 | 15000 | 10000 | 2000 | 51600 | 0.82 | 42312 | |
| 580 | WASTE | | | | | | | 15000 | 40000 | 60000 | 25870 | | | | 140870 |
| | QP | | | | | | | 2000 | 14200 | 18000 | | 29200 | 0.48 | 14016 | |
| | DSL | | | | | | | | 2800 | 13800 | 6600 | 23200 | 0.70 | 16240 | |
| 570 | WASTE | | | | | | | | | 15000 | 63100 | | | | 78100 |
| | QP | | | | | | | | | | 15900 | 15800 | 0.54 | 3532 | |
| | DSL | | | | | | | | | | 10300 | 10300 | 0.80 | 8240 | |
| | WASTE | 504450 | 432700 | 420000 | 420300 | 424900 | 388700 | 274300 | 189100 | 115000 | 103670 | | | | 3273120 |
| | QP M3 | 22200 | 21100 | 27900 | 27500 | 39400 | 20000 | 30500 | 30100 | 20000 | 20800 | 259500 | | | |
| | t | 66600 | 63300 | 83700 | 82500 | 118200 | 60000 | 91500 | 90300 | 60000 | 62400 | 778500 | | | |
| | % Sn | 0.49 | 0.46 | 0.45 | 0.47 | 0.47 | 0.47 | 0.44 | 0.45 | 0.49 | 0.53 | 0.46 | | | |
| | DSL M3 | 20000 | 20000 | 35800 | 33200 | 24300 | 39100 | 30500 | 27700 | 24800 | 19800 | 275200 | | | |
| | t | 70000 | 70000 | 125300 | 116200 | 85050 | 136850 | 106750 | 96950 | 86800 | 69300 | 963200 | | | |
| | % Sn | 1.06 | 0.97 | 0.94 | 0.93 | 0.89 | 0.91 | 0.91 | 0.85 | 0.76 | 0.77 | 0.90 | | | |
| TOTAL t ORE | | 136600 | 133300 | 209000 | 198700 | 203250 | 196850 | 198250 | 187250 | 146800 | 131700 | 1741700 | | | |
| | | 0.75 | 0.73 | 0.74 | 0.74 | 0.65 | 0.78 | 0.69 | 0.66 | 0.65 | 0.66 | 0.70 | | | |
| DILUTION AND RECOVERY | | | | | | | | | | | | | | | |
| DSL | | | | | | | | | | | | | | | |
| 90% RECOVERY | | 74340 | 74340 | 133069 | 123404 | 90323 | 145335 | 113368 | 102960 | 92182 | 73597 | 1022918 | | | |
| 18% DILUTION @ 0.1% | | 0.91 | 0.84 | 0.81 | 0.80 | 0.77 | 0.79 | 0.79 | 0.74 | 0.66 | 0.67 | 0.78 | | | |
| PORPHYRY | | | | | | | | | | | | | | | |
| 90% RECOVERY | | 66533 | 63237 | 83616 | 82417 | 118082 | 59940 | 91408 | 90210 | 59940 | 62338 | 777688 | | | |
| 11% DILUTION @ 0.1% | | 0.49 | 0.42 | 0.42 | 0.43 | 0.43 | 0.43 | 0.41 | 0.42 | 0.45 | 0.49 | 0.43 | | | |
| TOTAL TONNES | | 140873 | 137577 | 216685 | 205821 | 208405 | 205275 | 204776 | 193170 | 152122 | 135935 | 1800606 | | | |
| % Sn | | 0.69 | 0.65 | 0.66 | 0.65 | 0.58 | 0.68 | 0.62 | 0.59 | 0.58 | 0.59 | 0.63 | | | |
| ORE : WASTE | | 1:11,6 | 1:10,2 | 1:6,4 | 1:6,7 | 1:6,5 | 1:6,3 | 1:4,4 | 1:3,2 | 1:2,6 | 1:2,5 | 1:6,4 | | | |

S00912:CHR/10

(iii) Bench Establishment

It will be necessary to establish 'cut-offs' at each RL when the access road is being established. Bulldozers will be used to establish bench horizons wherever possible and the waste will be dozed to the loaders and haul trucks for removal to the appropriate waste dump.

2. Year 3 onwards

One of the main advantages of this option is the extra ore source which is available due to the Stanhope extension. The intention is to schedule extraction so as to blend D.S.L., Main Pit Porphyry and Stanhope Extension Pit Porphyry ores not only to ensure optimum grade but also to maximise the faces available for mining.

Stanhope ore will be depleted down to +650m RL in Year 5 and will be replaced by ore from benches below the +610m RL benches. Care will have to be taken that development of the main haul road to the +570m RL is kept in advance of the production faces to ensure that continuity of production is maintained. It will be possible to utilise the Stanhope Extension Pit excavation as a waste dump in Years 6-10.

4.4 Main Pit (no Pre-Production) (Table 3 (iii))

The operational strategy of this option is similar to that developed for the previous case, in that no pre-production year is envisaged. It will require greater operational control and scheduling than that required in Section 4.2.

5. OPERATIONAL DETAILS

5.1 Sampling

Due to the very inconsistent nature of the tin mineralisation in the D.S.L. it will be necessary to establish a very strict sampling programme in the Main pit. The following initial concept is considered the minimum requirement and may be improved in the future.

(i) Initial mining blocks to be established by surface drilling.

(ii) A 20m sampling grid to be established on each bench. 50mm diameter percussion drill holes to be drilled to a depth of 20m utilising a contract driller supervised by J.V. personnel. All drill cuttings to be sampled and assayed.

(iii) New mining block outlines established one bench below the current production face and blasts designed according to the new parameters.

(iv) Individual blast holes drill cuttings to be examined and analysed to determine whether the rock yielded by each hole is to be designated ore or waste - within the limits of what is operationally practical.

(v) Broken rock to be selectively mined using front-end loaders and bulk samples to be taken from each truck to ensure stockpile grade is consistent with plant requirements.

(vi) Cost

| | |
|---|---|
| 200 sample holes (20m deep) required/annum | = 4000 |
| Estimated drilling cost = $8.00/m | = $32000/annum |
| 200 holes at 4 samples/hole | = 800 samples/annum |
| Estimated assay cost = $2.50/assay | = $ 2000/annum |
| 1500 ore blastholes/annum | = 1500 samples |
| @ $2.50/assay | = $ 3750/annum |
| Bulk samples say 2/shift and 450 shifts/annum | = 900 samples |
| Cost/annum | = $ 2250 |
| Total Sampling and Assaying Cost | = $40000/annum |

## 5.2 Ore Stockpile

An ore stockpile should be constructed on a site near the existing office block. Use should be made of the natural surface slope and a concrete or wooden (sleeper) retaining wall to be constructed at the base of the slope. Two loading chutes to match the haulage equipment must be constructed into this retaining wall.

Note In the case of off site milling, it may be possible to load ore directly into road transporters ex the open pit, dependent on the operational restrictions at the concentrator, thus saving double handling.

## 5.3 Pit Dewatering

Drains along the 'high' side of the haul road will protect the pit from surface run off above the +610m RL. Below the +610m RL the drains will feed an in-pit pump which will be progressively excavated with each drop cut. A mobile submersible pump will pump the water to the surface and discharge on the western side of the pit.

## 5.4 Waste Disposal

Waste will be hauled to the waste dump area located in a valley (approximately 1.5 km) to the northwest of the Main Pit. The area must be developed and drains constructed to ensure adequate drainage below the waste dump.

## 5.5 Contractor Selection

The final selection of the contractor will obviously be dependent on, cost, reliability, experience, etc. It is suggested that the load and haul category is separated from the sampling/drill and blast category which in turn is separated from the transport category. This diversification obviously maximises negotiation ability and quality of service.

### Contract Mining Rates

Examination of all quotes from the four contractors has resulted in the following costs being assigned to the three basic functions.

| | | |
|---|---|---|
| Ripping and dozing overburden | $1.33/BCM | (Leightons) |
| Drill, blast load and haul overburden | $4.10/BCM | (Leightons) |
| Drill, blast load and haul ore = 590m RL | $1.25/t | (Roche Bros) |

All contractors quotations are appended (Appendix 2).

## 5.6 Manning

Due to the fact that contractors will be used for all operational functions it will be important to all J.V. personnel to be staff employees. All engineering and accounting services and long term planning will be carried out by Head Office personnel on a charge out basis. Short term planning and all operational control will be the responsibility of the Mine Superintendent. The following labour complement is considered adequate:

| Category | $/man/annum | No |
|---|---|---|
| Mine Superintendent | 45,000 | 1 |
| Open Pit Supervisors | 30,000 | 2 |
| Surveyor | 30,000 | 1 |
| Geologist | 30,000 | 1 |
| Survey/Geol. Asst. | 18,000 | 2 |
| Samples | 15,000 | 2 |
| Clerk | 15,000 | 1 |
| Secretary | 15,000 | 1 |
| | 261,000 | 11 |

No allowance has been made for absenteeism or sickness as sufficient excess capacity/function has been allowed for.

## 5.7 Office and Amenities

The following list details additional equipment which will be required.

| | $000's |
|---|---|
| Office/amenities (Atco type) - 2 @ $20,000 ea | 40 |
| Concrete paving (400m x 0.15 thick) | 18 |
| Communications (phone, radio etc) | 10 |
| Fencing | 8 |
| Drainage/Waste disposal | 5 |
| Office furniture | 20 |
| Utilities 3 @ $16,000 ea | 48 |
| Superintendent vehicle | 16 |
| Electrical reticulation | 40 |
| Water reticulation | 15 |
| Site preparation, roads and ore stockpiles | 150 |
| | 370 |

## COST SCHEDULE

### 6.1 MAIN PIT AND PRE-PRODUCTION

| DETAILS | | YEAR -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| B.C.M. ripped and dozed | x $10^3$ | 200 | 200 | 200 | | | | | | | | | 600 |
| @ $1.33/BCM | x $10^3$ | 266 | 266 | 266 | | | | | | | | | 798 |
| B.C.M. Waste | x $10^3$ | 204 | 232.7 | 220 | 350.3 | 329.9 | 338 | 340 | 293.8 | 140 | 119.7 | 89 | 2657.4 |
| @ $4.10/BCM | | 836 | 954 | 902 | 1436 | 1352 | 1386 | 1394 | 1205 | 574 | 491 | 365 | 10895 |
| Tonne Ore | x $10^3$ | | 137.6 | 195.1 | 203.4 | 189.5 | 182.5 | 179.6 | 174.7 | 181.5 | 144 | 147.3 | 1735.2 |
| @ $1.25/t | | | 172 | 244 | 254 | 237 | 228 | 225 | 218 | 227 | 180 | 184 | 2169 |
| Total Mining Cost | ($'000) | 1102 | 1392 | 1412 | 1690 | 1589 | 1614 | 1619 | 1423 | 801 | 671 | 549 | 13862 |
| Sampling | ($'000) | | 40 | 40 | 40 | 40 | 40 | 30 | 30 | 20 | 20 | 15 | 315 |
| Cartage, Milling Admin/Corporate/Town Smelting @ $34/t | ($'000) | 200 | 4678 | 6633 | 6916 | 6443 | 6205 | 6106 | 5940 | 6171 | 4896 | 5008 | 59804 |
| Replacement Equipment | ($'000) | | | | | 100 | | | | 100 | | | 200 |
| Labour | ($'000) | 120 | 261 | 261 | 261 | 261 | 261 | 261 | 261 | 261 | 261 | 261 | 2730 |
| Total Operating Cost | ($'000) | 1422 | 6371 | 8346 | 8907 | 8433 | 8120 | 8016 | 7654 | 7353 | 5848 | 5833 | 75933 |
| $/tonne ore mined | | | 46.30 | 42.78 | 43.79 | 44.5 | 44.5 | 44.6 | 43.8 | 40.5 | 40.6 | 39.6 | 43.76 |

## COST SCHEDULE

### 6.2 MAIN AND STANHOPE EXTENSION PITS

| DETAILS | | YEAR 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| B.C.M. Ripped and dozed | $\times 10^3$ | 300 | 300 | | | | | | | | | 600 |
| @ $1.33/BCM | $\times 10^3$ | 399 | 399 | | | | | | | | | 798 |
| B.C.M. Waste | $\times 10^3$ | 384 | 252.7 | 635.2 | 567.5 | 529 | 498 | 365 | 189.1 | 135 | 133.7 | 3689.2 |
| @ $4.10/BCM | $\times 10^3$ | 1574.4 | 1036.1 | 2604.3 | 2326.8 | 2168.9 | 2041.8 | 1496.5 | 775.3 | 553.5 | 548.2 | 15125.8 |
| Tonne Ore | $\times 10^3$ | 140.9 | 180.4 | 275.5 | 352.3 | 290.2 | 291.4 | 204.8 | 181.8 | 152.4 | 176.8 | 2246.5 |
| @ $1.25/BCM | $\times 10^3$ | 176.1 | 225.5 | 344.4 | 440.4 | 362.5 | 364.2 | 256 | 227.3 | 190.4 | 221 | 2807.8 |
| Total Mining Cost | ($'000) | 2149.5 | 1660.6 | 2948.7 | 2767.2 | 2531.4 | 2406 | 1752.5 | 1002.6 | 743.9 | 769.2 | 18731.6 |
| Sampling | ($'000) | 40 | 40 | 40 | 40 | 40 | 30 | 30 | 20 | 20 | 15 | 315 |
| Cartage, Milling Admin/Corporate/Town) Smelting @ $34/t | ($'000) | 4791 | 6134 | 9367 | 11978.2 | 9866.8 | 9907.6 | 6963.2 | 6181.2 | 5181.6 | 6011.2 | 76381.8 |
| Replacement Equipment | ($'000) | | | | | 100 | | | | 100 | | 200 |
| Labour | ($'000) | 261 | 261 | 261 | 261 | 261 | 261 | 261 | 261 | 261 | 261 | 2610 |
| Total Operating Cost | ($'000) | 7241.5 | 8095.6 | 12616.7 | 15046.4 | 12799.2 | 12604.6 | 9006.7 | 7464.8 | 6306.5 | 7056.4 | 98404.4 |
| $/tonne ore mined | | 51.39 | 44.88 | 45.80 | 42.71 | 44.10 | 43.2 6 | 43.98 | 41.06 | 41.38 | 39.91 | 43.81 |

## COST SCHEDULE

### 6.3 MAIN PIT (NO PRE-PRODUCTION)

| DETAILS | | YEAR 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| B.C.M. ripped and dozed | $\times 10^3$ | 300 | 300 | | | | | | | | | 600 |
| @ $1.33/BCM | $\times 10^3$ | 399 | 399 | | | | | | | | | 798 |
| B.C.M. Waste | $\times 10^3$ | 204.4 | 132.7 | 420 | 420 | 425 | 388.7 | 274 | 189 | 115 | 103.7 | 2673 |
| @ $4.10/BCM | $\times 10^3$ | 838 | 544 | 1722 | 1722 | 1742.5 | 1593.7 | 1123.4 | 774.9 | 471.5 | 425.2 | 10957.2 |
| Tonne Ore | $\times 10^3$ | 140.9 | 137.6 | 216.7 | 205.8 | 208.4 | 205.3 | 204.8 | 193.2 | 152.1 | 135.9 | 1800.6 |
| @ $1.25/BCM | $\times 10^3$ | 176 | 392 | 270.9 | 257.2 | 260.5 | 256.6 | 256 | 241.5 | 190.1 | 169.9 | 2470.7 |
| Total Mining Cost | ($'000) | 1413 | 1335 | 1992.9 | 1979.2 | 2003 | 1850.3 | 1379.4 | 1016.4 | 661.6 | 595.1 | 14225.9 |
| Sampling | ($'000) | 40 | 40 | 40 | 40 | 40 | 30 | 30 | 20 | 20 | 15 | 315 |
| Cartage, Milling Admin/Corporate/Town Smelting @ $34/t | ($'000) | 4791 | 4678.4 | 7367.8 | 6997.2 | 7085.6 | 6980.2 | 6963.2 | 6568.8 | 5171.4 | 4620.6 | 61224.2 |
| Replacement Equipment | ($'000) | | | | 100 | | | | 100 | | | 200 |
| Labour | ($'000) | 261 | 261 | 261 | 261 | 261 | 261 | 261 | 261 | 261 | 261 | 2610 |
| Total Operating Cost | ($'000) | 6505 | 6314.4 | 9661.7 | 9377.4 | 9389.6 | 9121.5 | 8633.6 | 7966.2 | 6114 | 5491.7 | 78575 |
| $/tonne ore mined | | 46.17 | 45.88 | 44.59 | 45.08 | 45.06 | 44.43 | 42.15 | 40.72 | 40.20 | 40.41 | 43.63 |

# APPENDIX 1

630mRL D.S.L.

| BLOCK NO | % Sn | PROVEN TONNES | PROVEN t x % | PROBABLE TONNES | PROBABLE t x % | POSSIBLE TONNES | POSSIBLE t x % |
|---|---|---|---|---|---|---|---|
| 1 | 0.23 | | | 5617 | 1292 | | |
| 2 | 1.32 | | | 6475 | 8547 | | |
| 3/29 | 0.87 | | | 5127 | 4461 | | |
| 4 ) | | | | | | | |
| 5 ) | 0.91 | 5232 | 4762 | | | | |
| 6 ) | | | | | | | |
| 7 ) | | | | | | | |
| 8 | 0.72 | | | | | 332 | 239 |
| 9 ) | 0.95 | | | | | 2955 | 2807 |
| 10) | | | | | | | |
| 11 | 0.64 | | | | | 960 | 614 |
| 12 | 0.51 | | | 962 | 491 | | |
| 13 | 0.37 | | | | | 2450 | 907 |
| 14(1) | 0.44 | | | | | 1610 | 708 |
| 14(11) | | | | | | | |
| 15(1) | 0.67 | | | 420 | 281 | | |
| 16 | 0.82 | | | 1906 | 1563 | | |
| 17(1) | 0.98 | | | | | 2204 | 2160 |
| 18 | 0.83 | | | | | 840 | 697 |
| 19 | 0.77 | | | | | 1505 | 1159 |
| 20 | 0.90 | | | 1801 | 1621 | | |
| 21 | 0.89 | | | | | 4251 | 3783 |
| 22 | 1.89 | | | 6090 | 11510 | | |
| 23) | 0.55 | | | | | 4306 | 2369 |
| 24) | | | | | | | |
| 25 | 1.02 | | | | | 1646 | 1678 |
| 28 | 0.35 | | | | | 1031 | 361 |
| 30 | 0.53 | | | 910 | 482 | | |
| 32(1) | 0.40 | | | | | 598 | 239 |
| Total | | 5232t | 4762 | 29308t | 30248 | 24688t | 17721 |
| Grade (% Sn) | | 0.91 | | 1.03 | | 0.72 | |

Combined Total 59228 t @ 0.89% Sn

S00910:CHR/23

625mRL DSL

| BLOCK NO | % Sn | PROVEN TONNES | PROVEN t x % | PROBABLE TONNES | PROBABLE t x % | POSSIBLE TONNES | POSSIBLE t x % |
|---|---|---|---|---|---|---|---|
| 1 | 0.23 | | | 6772 | 1557 | | |
| 2/4 ) | 1.34 | | | 9170 | 12238 | | |
| 3/38) | 0.87 | | | 6125 | 5329 | | |
| 4 ) | | | | | | | |
| 5 ) | | | | | | | |
| 6 ) | 1.03 | 5775 | 5948 | | | | |
| 7 ) | | | | | | | |
| 8 ) | | | | | | | |
| 9 | 0.55 | | | 717 | 395 | | |
| 10 | 1.08 | | | 10939 | 11809 | | |
| 11 | 0.82 | | | 1485 | 1218 | | |
| 12(1) ) | | | | | | | |
| 13(1) ) | 0.74 | 1365 | 846 | | | | |
| 14(1) ) | | | | | | | |
| 15(1) | 0.47 | | | 1106 | 520 | | |
| 16(1) | 0.86 | | | 2205 | 1896 | | |
| 17 | 0.67 | | | 4550 | 3048 | | |
| 18 | 0.98 | | | | | 2240 | 2195 |
| 19 | 1.18 | | | 1714 | 2023 | | |
| 20 | 0.77 | | | | | 980 | 755 |
| 21(1) | 0.67 | | | 1036 | 694 | | |
| 22(1) | 0.56 | | | 1063 | 595 | | |
| 23/26 | 1.81 | | | 10465 | 18942 | | |
| 24(1) ) | | | | | | | |
| 25(1) ) | 0.53 | | | | | 6342 | 3361 |
| 27(1) | 1.28 | | | | | 1568 | 2007 |
| 28(1) | 1.50 | | | | | 1099 | 1648 |
| 29) | 0.54 | | | | | 20006 | 10803 |
| 30) | | | | | | | |
| 32(1) | 0.39 | | | | | 313 | 122 |
| 33(1) | 1.18 | | | | | 2133 | 2517 |
| 34(1) | 0.45 | | | | | 1890 | 850 |
| 35(1) | 1.48 | | | | | 863 | 1278 |
| 36(1) | 0.42 | | | | | 243 | 102 |
| 37(1) | 0.63 | | | | | 4387 | 2764 |
| 38 | | | | | | | |
| 39 | 0.44 | | | | | 2677 | 1178 |
| 40 | 0.31 | | | | | 1103 | 342 |
| Total | | 7140t | 6794 | 57347t | 60264 | 45844t | 29922 |
| Grade (% Sn) | | 0.95 | | 1.05 | | 0.65 | |

Combined Total 110331 t @ 0.88% Sn

S00910:CHR/24

620mRL DSL

| BLOCK NO | % Sn | PROVEN TONNES | PROVEN t x % | PROBABLE TONNES | PROBABLE t x % | POSSIBLE TONNES | POSSIBLE t x % |
|---|---|---|---|---|---|---|---|
| 1 | 0.23 | | | 7035 | 1618 | | |
| 2 | 1.41 | | | 6387 | 9006 | | |
| 2A | | | | | | | |
| 3 | 0.87 | | | 5792 | 5039 | | |
| 4 | 0.59 | | | 1225 | 723 | | |
| 5 ) | | | | | | | |
| 6 ) | | | | | | | |
| 7 ) | | | | | | | |
| 7A) | | | | | | | |
| 8 ) | | | | | | | |
| 9 ) | | | | | | | |
| 10) | | | | | | | |
| 11) | | | | | | | |
| 12) | 1.1 | 8785 | 9224 | | | | |
| 13) | | | | | | | |
| 14) | | | | | | | |
| 15) | | | | | | | |
| 16) | | | | | | | |
| 17) | | | | | | | |
| 18/50) | | | | | | | |
| 19 | 0.47 | | | 1085 | 510 | | |
| 20/51 | 1.23 | | | 2152 | 2646 | | |
| 21 | 0.47 | | | 2065 | 970 | | |
| 22) | 0.87 | | | 14472 | 12591 | | |
| 23) | | | | | | | |
| 24) | 0.81 | | | 3902 | 3161 | | |
| 25) | | | | | | | |
| 26(1) | 0.91 | | | 4426 | 4028 | | |
| 27 | 1.11 | | | 2011 | 2233 | | |
| 28 | 0.56 | | | 4690 | 2626 | | |
| 29 | 2.00 | | | 9660 | 19320 | | |
| 30/31/53 | 0.61 | | | | | 11480 | 7003 |
| 32) | | | | | | | |
| 33) | 1.30 | | | | | 8890 | 11557 |
| 34) | | | | | | | |
| 35) | | | | | | | |
| 36 | 0.54 | | | | | 7403 | 3998 |
| 37(1) | 0.40 | | | | | 4215 | 1686 |
| 38(1) | 0.34 | | | | | 323 | 110 |
| 39 | 0.61 | | | | | 1890 | 1153 |
| 40 | 0.46 | | | | | 4110 | 1891 |

620mRL DSL (continued)

| BLOCK NO | % Sn | PROVEN TONNES | PROVEN t x % | PROBABLE TONNES | PROBABLE t x % | POSSIBLE TONNES | POSSIBLE t x % |
|---|---|---|---|---|---|---|---|
| 41/55 | 0.49 | | | 3675 | 1801 | | |
| 42 | 1.48 | | | | | 1050 | 1554 |
| 43 | 0.62 | | | | | 1245 | 772 |
| 44 | 0.77 | | | | | 840 | 647 |
| 45 | 0.42 | | | | | 1722 | 723 |
| 46 | 0.42 | | | | | 1638 | 688 |
| 47 | 0.63 | | | | | 2992 | 1885 |
| 49 | 0.31 | | | | | 182 | 56 |
| Total | | 8785t | 9224 | 68750t | 66272 | 47980t | 33723 |
| Grade (% Sn) | | 1.10 | | 0.96 | | 0.70 | |

Combined Total 125515 t @ 0.87% Sn

615mRL DSL

| BLOCK NO | % Sn | PROVEN TONNES | PROVEN t x % | PROBABLE TONNES | PROBABLE t x % | POSSIBLE TONNES | POSSIBLE t x % |
|---|---|---|---|---|---|---|---|
| 1 ) | 0.78 | | | 6860 | 1921 | | |
| 2 ) | | | | | | | |
| 3 ) | 0.95 | | | 5092 | 4838 | | |
| 4 ) | | | | | | | |
| 5 | 0.59 | | | 1750 | 1032 | | |
| 6 ) | | | | | | | |
| 7 ) | | | | | | | |
| 8 ) | | | | | | | |
| 9 ) | | | | | | | |
| 10) | | | | | | | |
| 11) | | | | | | | |
| 12) | 1.17 | 10377 | 12142 | | | | |
| 13) | | | | | | | |
| 14) | | | | | | | |
| +38/15) | | | | | | | |
| 16) | | | | | | | |
| 17) | | | | | | | |
| 18) | | | | | | | |
| 19 | 0.82 | | | 1960 | 1607 | | |
| 20(i) | 0.64 | | | 4655 | 2979 | | |
| 21 | 0.55 | | | 2222 | 1222 | | |
| 22) | 1.06 | | | 16992 | 18012 | | |
| +39/23) | | | | | | | |
| 24) | 0.84 | | | 6510 | 5468 | | |
| 25) | | | | | | | |
| 26 | 1.62 | | | 4900 | 7938 | | |
| 27 | 0.60 | | | 2975 | 1785 | | |
| 28 | 0.56 | | | 5338 | 2989 | | |
| 29 | 2.00 | | | 9362 | 18725 | | |
| 30) | 0.81 | | | 4287 | 3472 | | |
| 31) | | | | | | | |
| 32) | 1.57 | | | | | 9572 | 15028 |
| 33) | | | | | | | |
| 34 | 0.54 | | | | | 4667 | 2520 |
| 36(i) | 0.47 | 1740 | 820 | | | | |
| Total | | 12117t | 12962 | 72903t | 71988 | 14239t | 17548 |
| Grade (% Sn) | | 1.07 | | 0.99 | | 1.23 | |

Combined Total 99259 t @ 1.03% Sn

049

381050

610mRL DSL

| BLOCK NO | % Sn | PROVEN TONNES | PROVEN t x % | PROBABLE TONNES | PROBABLE t x % | POSSIBLE TONNES | POSSIBLE t x % |
|---|---|---|---|---|---|---|---|
| 1 | 0.4 | | | 5180 | 2072 | | |
| 2 | 1.15 | | | 4812 | 5534 | | |
| 3 | 0.59 | | | 1435 | 847 | | |
| 4 ) | | | | | | | |
| 5 ) | | | | | | | |
| 6 ) | | | | | | | |
| 7 ) | | | | | | | |
| 8 ) | | | | | | | |
| 9 ) | 1.21 | 10552 | 12769 | | | | |
| 10) | | | | | | | |
| 11) | | | | | | | |
| 12) | | | | | | | |
| 13) | | | | | | | |
| 14) | | | | | | | |
| 15) | | | | | | | |
| 16) | 0.56 | | | 13370 | 7487 | | |
| 17) | | | | | | | |
| 18 | 1.13 | | | 13772 | 15563 | | |
| 19) | 0.87 | | | 8522 | 7414 | | |
| 20) | | | | | | | |
| 21 | 1.62 | | | 4200 | 6804 | | |
| 22(i)) | 0.55 | | | 1536 | 845 | | |
| 23(i)) | | | | | | | |
| 23(a) | 0.75 | | | 3062 | 2297 | | |
| 24 | 0.56 | | | 3060 | 1713 | | |
| 25 | 2.0 | | | 8225 | 16450 | | |
| 26 | 1.42 | | | 1382 | 1963 | | |
| 27 | 2.25 | | | 5935 | 13353 | | |
| 29(i) | 0.47 | | | | | | |
| 33 | 0.67 | | | 1417 | 950 | | |
| 34 | 0.34 | | | 1288 | 438 | | |
| Total | | 10584t | 12784 | 77196t | 83730 | | |
| Grade (% Sn) | | 1.21 | | 1.08 | | | |

Combined Total 87780 t @ 1.10% Sn

605m RL DSL

| BLOCK NO | % Sn | PROVEN TONNES | t x % | PROBABLE TONNES | t x % | POSSIBLE TONNES | t x % |
|---|---|---|---|---|---|---|---|
| 1 | 0.54 | | | 4042 | 2183 | | |
| 2(i) | 0.8 | | | 518 | 414 | | |
| 3 | 1.31 | | | 2380 | 3120 | | |
| 4 | 0.59 | | | 1715 | 1012 | | |
| 5 ) | | | | | | | |
| 6 ) | | | | | | | |
| 7 ) | | | | | | | |
| 8 ) | | | | | | | |
| 9 ) | 1.12 | 9992 | 11192 | | | | |
| 10) | | | | | | | |
| 11) | | | | | | | |
| 12) | | | | | | | |
| 13) | | | | | | | |
| 14) | | | | | | | |
| 15) | 0.57 | | | 13825 | 7880 | | |
| 16) | | | | | | | |
| 17) | 0.93 | | | 15172 | 14110 | | |
| 18) | | | | | | | |
| 19 | 1.0 | | | 6685 | 6685 | | |
| 20 | 1.62 | | | 3238 | 5246 | | |
| 21 | 1.77 | | | 150 | 266 | | |
| 22) | 0.59 | | | 7927 | 4677 | | |
| 23) | | | | | | | |
| 24 | 1.05 | | | 2825 | 2977 | | |
| 25) | | | | | | | |
| 26) | 2.0 | | | 5197 | 10394 | | |
| 28 | 0.7 | | | 880 | 616 | | |
| Total | | 9992t | 11192 | 64554t | 59580 | | |
| Grade (% Sn) | | 1.12 | | 0.92 | | | |

Combined Total 74546 t @ 0.95% Sn

051

381052

600mRL DSL

| BLOCK NO | % Sn | PROVEN TONNES | t x % | PROBABLE TONNES | t x % | POSSIBLE TONNES | t x % |
|---|---|---|---|---|---|---|---|
| 1 | 0.54 | | | 4340 | 2343 | | |
| 2 | 0.80 | | | 1837 | 1470 | | |
| 3 | 0.56 | | | 882 | 494 | | |
| 4 | 1.31 | | | 1523 | 1994 | | |
| 5 | 0.66 | | | 2677 | 1767 | | |
| 6 | 1.05 | | | 1144 | 1202 | | |
| 7) | 0.65 | | | 5810 | 3777 | | |
| 8) | | | | | | | |
| 9 | 0.89 | | | 2667 | 2374 | | |
| 10 | 0.56 | | | 2600 | 1456 | | |
| 10(i) | | | | | | | |
| 11(i)) | 1.49 | | | 2331 | 3474 | | |
| 12 ) | 0.99 | | | 16590 | 16424 | | |
| 13 ) | | | | | | | |
| 14 | 1.4 | | | 7596 | 10634 | | |
| 16 | 0.68 | | | 6947 | 4307 | | |
| 17 | 1.11 | | | 5127 | 5691 | | |
| 18 | 1.05 | | | 2415 | 2536 | | |
| 19 | 2.77 | | | 1932 | 5351 | | |
| 21 | 0.70 | | | 2842 | 1989 | | |
| Total | | | | 66845t | 67283 | | |
| Grade | | | | 1.01% Sn | | | |

052

381053

595mRL DSL

| BLOCK NO | % Sn | PROVEN TONNES | t x % | PROBABLE TONNES | t x % | POSSIBLE TONNES | t x % |
|---|---|---|---|---|---|---|---|
| 1 | 0.71 | | | 7787 | 5529 | | |
| 2 | 0.80 | | | 1172 | 938 | | |
| 3 | 0.56 | | | 1871 | 1048 | | |
| 6) | 0.79 | | | 8994 | 7106 | | |
| 7) | | | | | | | |
| 8 | 0.67 | | | 4917 | 3294 | | |
| 9 | | | | 10885 | 9688 | | |
| 10 | 0.56 | | | 10955 | 6135 | | |
| 11 | 0.56 | | | 7245 | 4057 | | |
| 12 | 1.49 | | | 6755 | 3522 | | |
| 13 | 1.00 | | | 7577 | 7577 | | |
| 14 | 0.98 | | | 11777 | 11542 | | |
| 15 | 1.33 | | | 7840 | 10427 | | |
| 16 | 0.62 | | | 7367 | 4568 | | |
| 17 | 1.11 | | | 4410 | 4825 | | |
| 18a | 0.61 | | | 4235 | 2583 | | |
| 21 | 0.90 | | | 1280 | 1152 | | |
| Total | | | | 105067t | 83991 | | |
| Grade (% Sn) | | | | 0.80 | | | |

053

381054

590mRL DSL

| BLOCK NO | % Sn | PROVEN TONNES | t x % | PROBABLE TONNES | t x % | POSSIBLE TONNES | t x % |
|---|---|---|---|---|---|---|---|
| 1 | 0.71 | | | 8260 | 5865 | | |
| 2 | 0.47 | | | 1662 | 781 | | |
| 3 | 0.62 | | | 409 | 254 | | |
| 8 | 0.44 | | | 6475 | 2849 | | |
| 10(i) | 0.71 | | | 388 | 276 | 116 | 83 |
| 11 | 0.67 | | | 5917 | 3482 | | |
| 12 | 0.89 | | | 9397 | 8364 | | |
| 13 | 0.56 | | | 5355 | 2999 | | |
| 14) | 1.35 | | | 8820 | 11907 | | |
| 15) | | | | | | | |
| 16 | 1.00 | | | 4690 | 4690 | | |
| 17 | 0.98 | | | 9905 | 9707 | | |
| 18 | 1.33 | | | 2800 | 3724 | | |
| 19 | 1.11 | | | 3482 | 3866 | | |
| 20 | 0.62 | | | 4095 | 2539 | | |
| 21 | 0.82 | | | 2397 | 1966 | | |
| 24 | 0.41 | | | 1172 | 481 | | |
| 26(i) | 0.7 | | | 164 | 53 | | |
| 27(i) | 0.32 | | | 168 | 54 | | |
| Total | | | | 75556t | 63857 | | |
| Grade (% Sn) | | | | 0.85 | | | |

585mRL DSL

| BLOCK NO | % Sn | PROVEN TONNES | PROVEN t x % | PROBABLE TONNES | PROBABLE t x % | POSSIBLE TONNES | POSSIBLE t x % |
|---|---|---|---|---|---|---|---|
| 2/4 | 0.58 | | | 10675 | 6191 | | |
| 3 | 0.62 | | | 2415 | 1497 | | |
| 5(i) | 0.71 | | | 2240 | 1590 | 672 | 477 |
| 7 | 0.59 | | | 1715 | 1011 | | |
| 8 | 0.67 | | | 5932 | 3275 | | |
| 9 | 1.16 | | | 7962 | 9236 | | |
| 10 | 0.98 | | | 5432 | 5814 | | |
| 11 | 1.0 | | | 1592 | 1592 | | |
| 12 | 1.01 | | | 3432 | 3517 | | |
| 16 | 0.41 | | | 3308 | 1356 | | |
| 18 | 0.32 | | | 6545 | 2094 | | |
| Total | | | | 52420t | 37650 | 673t | 477 |
| Grade (% Sn) | | | | 0.72 | | 0.72 | |

580mRL DSL

| BLOCK NO | % Sn | PROVEN TONNES | t x % | PROBABLE TONNES | t x % | POSSIBLE TONNES | t x % |
|---|---|---|---|---|---|---|---|
| 2/15 | 0.52 | | | 3762 | 1956 | | |
| 3 | 1.43 | | | 3759 | 5375 | | |
| 4 | 1.02 | | | 3185 | 3249 | | |
| 5 | 0.98 | | | 2676 | 2623 | | |
| 6 | 1.11 | | | 253 | 282 | | |
| 7 | 0.47 | | | 273 | 128 | | |
| 10 | 0.41 | | | 4375 | 1794 | | |
| 11 | 0.51 | | | 446 | 228 | | |
| 12 | 1.03 | | | 196 | 202 | | |
| 13 | 1.03 | | | 402 | 415 | | |
| 14 | 0.32 | | | 9292 | 2974 | | |
| Total | | | | 28619t | 19226 | | |
| Grade (% Sn) | | | | 0.67 | | | |

575mRL DSL

| BLOCK NO | % Sn | PROVEN TONNES | t x % | PROBABLE TONNES | t x % | POSSIBLE TONNES | t x % |
|---|---|---|---|---|---|---|---|
| 3 | | | | 420 | 479 | | |
| 2 | 1.43 | | | 5862 | 8383 | | |
| 6 | 1.03 | | | 3289 | 3388 | | |
| 7 | 1.03 | | | 3466 | 3570 | | |
| 8 | 0.32 | | | 8797 | 2815 | | |
| 10 | | | | 1277 | 652 | | |
| 12 | 0.46 | | | 5442 | 2504 | | |
| Total | | | | 28553t | 21791 | | |
| Grade (% Sn) | | | | 0.76 | | | |

570mRL DSL

| BLOCK NO | % Sn | PROVEN TONNES | t x % | PROBABLE TONNES | t x % | POSSIBLE TONNES | t x % |
|---|---|---|---|---|---|---|---|
| 3 | 0.97 | | | 4397 | 4265 | | |
| 4 | 1.02 | | | 2642 | 2694 | | |
| 11 | 0.48 | | | 560 | 269 | | |
| Total | | | | 7599t | 7228 | | |
| Grade (% Sn) | | | | 0.95 | | | |

# QP MINING RESERVE

| SECTION mE | BLOCK NO | % Sn | PROBABLE tonnes | PROBABLE t x % | POSSIBLE tonnes | POSSIBLE t x % |
|---|---|---|---|---|---|---|
| 940 | Nil | | | | | |
| 960 | Nil | | | | | |
| 980 | 3 | 0.3 | 15663 | 4699 | | |
| | 4 | 0.23 | 5292 | 1217 | | |
| | 5 | 0.27 | 1536 | 415 | | |
| 1000 | 4 | 0.37 | 1656 | 166 | | |
| 1020 | 3 | 0.51 | 1740 | 887 | | |
| 1040 | 3 | 0.69 | 24810 | 17119 | | |
| | 4 | 0.46 | 35784 | 16461 | | |
| | 5 | 0.52 | 1872 | 973 | | |
| 1060 | 2 | 0.77 | 8304 | 6394 | | |
| | 3 | 0.35 | 41382 | 14484 | | |
| | 4 | 1.52 | 3654 | 5554 | | |
| | 5 | 0.41 | 20820 | 8536 | | |
| | 6 | 0.31 | 5220 | 1618 | | |
| | 7 | 0.50 | 2106 | 1053 | | |
| 1080 | 2 | 0.4 | 24408 | 9763 | | |
| | 3 | 0.8 | | | 3000 | 2400 |
| | 4 | 0.46 | 7200 | 3312 | | |
| | 5 | 0.36 | 39600 | 14256 | | |
| | 6 | 0.41 | 6120 | 2509 | | |
| | 7 | 0.31 | 2650 | 822 | | |
| | 8) | 0.5 | 7850 | 3925 | | |
| | 9) | | | | | |
| | 10 | 0.32 | | | 1320 | 422 |
| | 11 | 0.30 | 2040 | 612 | | |
| | 12 | 0.26 | 3480 | 905 | | |
| 1100 | 2 | 0.4 | 19752 | 7900 | | |
| | 3 | 0.42 | 33300 | 13986 | | |
| | 5 | 0.25 | 13380 | 3345 | | |
| | 6 | | | | 3840 | 1382 |
| Sub Total | | | 329619t | 140911 | 8160t | 4204 |

QP MINING RESERVE

| SECTION mE | BLOCK NO | % Sn | PROBABLE tonnes | PROBABLE t x % | POSSIBLE tonnes | POSSIBLE t x % |
|---|---|---|---|---|---|---|
| 1120 | 2 | 0.40 | 768 | 307 | | |
| | 5 | 0.24 | 16632 | 3992 | | |
| | 4 | 0.89 | | | 15060 | 13403 |
| | 3 | 0.42 | 13020 | 5468 | | |
| 1140 | 3 | 0.24 | 24432 | 5864 | | |
| | 4 | 0.22 | 2820 | 620 | | |
| | 5 | 0.29 | | | 12360 | 3584 |
| | 6 | 0.41 | | | 4980 | 2042 |
| | 7 | 0.36 | | | 780 | 280 |
| 1160 | 4 | 0.86 | 31560 | 27142 | | |
| | 5 | 0.33 | 22620 | 7465 | | |
| | 6 | 0.50 | | | 11820 | 5910 |
| | 7 | 0.24 | 16704 | 4008 | | |
| 1180 | 3 | 0.64 | 3132 | 2004 | | |
| | 4 | 0.66 | 41577 | 27441 | | |
| | 5 | | | | | |
| | 6 | 0.21 | | | 14940 | 3137 |
| 1200 | 2 | 0.32 | 5814 | 1860 | | |
| | 3) | 0.51 | 50580 | 25796 | | |
| | 4) | | | | | |
| | 5 | 0.48 | 14400 | 6912 | | |
| 1220 | 2 | 0.32 | 3917 | 1253 | | |
| | 3 | 0.51 | 10320 | 5263 | | |
| | 4 | 0.48 | 29400 | 14112 | | |
| 1240 | 4 | 0.44 | 3456 | 1520 | | |
| | 5 | 0.48 | 24180 | 11606 | | |
| | 6 | 0.32 | | | 6720 | 2150 |
| Sub Total | | | 315332t | 152633 | 66660t | 30506 |
| Total | | | 644951t | 293544 | 74820t | 34710 |
| Grade (% Sn) | | | 0.46 | | 0.46 | |

# APPENDIX 2

060

MB 4 -7

381061

BRAMBLES INDUSTRIAL SERVICES
A DIVISION OF BRAMBLES HOLDINGS LIMITED INCORPORATED IN N.S.W.

P.O. BOX 580
BURNIE 7320
TASMANIA, AUSTRALIA

TELEPHONES :
MAIN OFFICE (004) 31 6488
BURNIE QUARRY (004) 31 1184
OPERATIONS (004) 31 3844
BELL BAY DEPOT (003) 82 1444

16th May, 1984

Ms. Metal Exploration Ltd.,
80 Collins Street,
MELBOURNE. VIC. 3000

SS

Refer: Mr. S. Solomons

Dear Sir,

We refer to your letter of 18th April regarding the Mount Bischoff joint venture.

Because of limited time to forward our quotation it has not been possible for us to obtain the services of our Mining Engineer who is based in Sydney. Therefore, we have not been as comprehensive in our assessment as we would have liked.

However, because of our previous considerable involvement with C.R.A. we are able to forward the following indicative prices for your venture:-

1. Pre-Production Development - First Year

Establish permanent haul road and strip and dump 400,000 bank cubic metres of over-burden - $2.37 per B.C.M.

2. Production - Second Year and Onward

Drill and blow ore and waste and cart to oppropriate sites - approximately 480,000 bank cubic metres per annum. - $7.89 per B.C.M.

During the initial year of pre-production work we envisage the need for six men to be permanently on the site. Once production begins we expect approximately eleven men will be employed full time.

In both cases we would expect to work 240 days per year each of ten hours.

This would be the basis of our work, but of course, we would expect to work some weekends during the year.

Subject to your requirements, we could be interested in providing some crushing and processing equipment for your Company to process the ore.

If this option is attractive to your Company we would be pleased to discuss it further.

Yours faithfully,
BRAMBLES INDUSTRIAL SERVICES,

R. N. Bligh,
COMMERCIAL MANAGER.

061

Telephone call 12/5/84
Ivan Edwards

381062

99 Nicholson Street
St Leonards

PO Box 49
St Leonards
NSW 2065
Australia

Telephone (02) 436 6000
Telex Opencut AA20790

ELDERS MINING
A DIVISION OF ELDERS CED LIMITED
(Incorporated in New South Wales)

16 May 1984

Mr S S Solomons
Senior Mining Engineer
Metals Exploration Limited
80 Collins Street
MELBOURNE, 3000

Dear Sir,

Re: Mt Bischoff Joint Venture

We refer to your letter of 18 April 1984, and wish to advise that Elders Mining (previously Davis Mining), the contract mining division of Elders IXL Limited, would be interested in tendering for your prospective operation.

Preliminary Quotation

We estimate the unit cost over the life of the deposit to mine both ore and waste at $8.67 per cubic metre. We would be pleased to provide a firm quotation at a later stage when you decide to proceed.

We confirm the prices telephoned to you on Monday, 14 May, as follows:

| | |
|---|---|
| Cost per cubic metre ore or waste drilled | $1.79 |
| Cost per cubic metre ore or waste blasted | $0.68 |
| Cost per cubic metre ore or waste loaded | $1.61 |
| Cost per cubic metre ore or waste hauled | $4.59 |
| Total | $8.67 |

Our approach has been to view the project as a complete mining contract under one contractor, so that ancillary plant (viz dozer, grader, water cart etc) could be allocated as necessary to the four main production functions, i.e. drill, blast, load or haul. For this reason and to simplify our costing calculations we have not dissected separately ore and waste costs. The main variance between the two products, if taken separately, would be on haulage, one less truck for ore. However the truck fleet would have to be geared to waste haulage, and servicing would be carried out when working on ore.

| | |
|---|---|
| Number of people required on site | 46 |
| Number of shifts per day | 2 of 7 hours |
| Number of days per year | 220 |

Qualifications

For your information, Elders Mining is the managing partner of our 50% owned joint venture, Hebden Mining Company, which operates the 2 million tonne per year Swamp Creek Open Cut Coal Mine on behalf of the Electricity Commission of New South Wales. This contract involves the removal of approximately 8 million cubic metres of overburden annually. Hebden recently completed a contract for Blue Circle Southern Cement Limited to mine 1.1 million tonnes of coal and 6.2 million cubic metres of overburden over 2 years near Lithgow, New South Wales. Elders Mining have recently been appointed the contract miner for Kidston Gold Mines Limited in Queensland. This will involve the annual production of about 10 million tonnes of ore and waste.

We have estimated the equipment requirements, production rates and unit costs for operating your mine on the same basis as our current contracts, which require firm prices for the duration of the contract, subject to escalation according to a strict formula. Our Elcom contract, a 15 year contract, recently renewed for 6 years, is thus an excellent example of our extensive experience with long term fixed price contracts.

We believe the price we have quoted on this basis best satisfies your information requirements for your feasibility study.

Alternative Approach

If you are concerned about ore grade control and flexibility on production rates, you may be more interested in a contract arrangement whereby the contractor supplies the equipment, the operators and day to day supervision, and you provide the management. This is precisely the type of contract we have negotiated with Placer for Kidston, and has reduced the Kidston's capital requirements while satisfying Placer's concerns on grade and production control. If you are interested in this approach, we will supply further information on how we could structure a contract.

We would be pleased to discuss our proposal further with you at your convenience.

Yours sincerely,
ELDERS MINING

I.M. DAVIES
GENERAL MANAGER

063

ROCHE BROS. PTY. LTD.
(Incorporated in Victoria)
REPLY TO:
568 ST. KILDA ROAD, MELBOURNE,
VICTORIA, 3004, AUSTRALIA
TELEPHONE: (03) 529 3022 TELEX: AA 33624
FACSIMILE: (03) 529 7739

ROCHE BROS

CIVIL ENGINEERING CONTRACTORS

ARC/bs 9177
21st May, 1984

Metals Exploration Limited,
80 Collins Street,
MELBOURNE. VIC. 3000.

Attention: Mr. S.S. Solomons

Dear Sir,

Re: Mt. Bischoff Joint Venture.

As requested we have carried out a preliminary cost study of your proposed mining at Mt. Bischoff and are pleased to supply prices which we believe to be indicative of the contract mining rates which could be obtained by invitation of tenders for the work.

As explained to you there are a number of factors which would require detailed study before firm prices could be quoted.

We estimate that the total mining could be carried out using one blast-hole drill capable of drilling 160 mm diameter blastholes, one Caterpillar 988 B wheel loader and up to 5-50 tonne rear dump trucks.

We estimate that a total average working time of 40 hours per week would be required. A total manpower requirement including supervisory, plant maintenance and operating personnel of sixteen (16) men is envisaged.

We hope that your optimisation study indicates a viable mining operation at Mt. Bischoff and look forward to being invited to submit a firm proposal in due course.

Please advise us if we can be of assistance in formulating a suitable set of tender and contract documents for any proposed contract.

INDICATIVE MINING PRICES.

1. Overburden.

a. Ripped, loaded and carted to disposal in the gloryhole. $2.82/BCM

/2...

| | | |
|---|---|---|
| b. | Ripped, loaded and carted to disposal in overburden dump. | $3.34/BCM |
| c. | Overburden from above 590 level blasted, loaded and carted to disposal in overburden dump. | $4.00/BCM |
| d. | Overburden from below 590 level. Blasted, loaded and carted to disposal in overburden dump. | $4.17/BCM |
| 2. | Ore. | |
| a. | Ore from above 590 level, blasted, loaded and carted to ore stockpile. | $3.66/BCM |
| b. | Ore from below 590 level, blasted, loaded and carted to ore stockpile. | $3.83/BCM |

Yours faithfully,
ROCHE BROS. PTY. LTD.

A.R. CURRY
Manager

# Leighton Contractors Budget Estimate

1. Rip and doze overburden ($1.5 Mm^3$) — $\$1.33/m^3$

   Total cost = $1,500,000 \times \$1.33 = \$1\,995\,000$

2. Drill ~~[illegible]~~ overburden and ore ($3.4 Mm^3$) — $\$0.64/m^3$

   Total cost = $3\,400\,000 \times \$0.64 = \$2\,176\,000$

3. Blast overburden ($2.75 Mm^3$) — $\$0.55/m^3$

   Total Cost = $2\,750\,000 \times \$0.55 = \$1\,512\,500$

4. Blast ore ($0.65 Mm^3 \div 2\,112\,000 t$) — $\$0.22/t$

   Total cost = $2\,112\,000 \times \$0.22 = \$464\,640$

5. Load, haul and tip ore ($0.65 Mm^3 \div 2\,112\,000 t$) — $\$1.04/t$

   Total Cost = $2\,112\,000 \times \$1.04 = \$2\,196\,480$

6. Load, haul and tip ~~ore~~ overburden ($4.25 Mm^3$) — $\$2.91/m^3$

   Total Cost = $4\,250\,000 \times \$2.91 = \$12\,367\,500$

Total Cost (in 1984 dollars) = $\$20\,712\,120$

# APPENDIX 3

# MT. BISCHOFF TIN PROJECT

## BRIEF FOR

## MINING OPTIMISATION STUDY

19/3/84

General

Previous mine planning for Mt. Bischoff has considered very small pits for the minimum scale production cases or large scale pits which aimed to maximise recovery of in situ reserves, feeding high capacity plant for a maximum life.

The current study requires mine planning to optimise cash flow from the pit, via critical review of stripping requirements to achieve extraction of ore blocks, with possible retrieval of unmined reserves via underground access.

The potential mine at Bischoff should no longer be influenced by agreement requirements or by supply of a specific maximimum tonnage to offsite plant. Rather, an optimal mining rate should be aimed at from the constraints of pit configuration etc. Porphyry ore should be taken as required by the pit (within limits as below) and may be mixed with DSL ore.

It is expected that the optimal mining rate will be in the range of 150,000 to 250,000 tonnes per annum.

Specifics

The study is to include the following:

1. Identification of annual tonnage into:

   DSL Ore Tonnes and Grade

   Porphyry Ore Tonnes and Grade

   Planning should ensure that short term disruptive savings in the ratio DSL/Porphyry do not occur.

2. Optimisation of grade treated.

   2.1 This may require stockpiling of lower grade material (Say, less than 0.4% Sn).

   - To be reviewed during study.

   2.2 This will require sufficient bench space to allow drilling and sampling of a bench 2 weeks ahead of mining and minimisation of dilution through small bench dimensions and appropriate mining equipment.

3. Account is to be taken of area identified as suitable for ripping and dozing in assessment of waste removal and ore blocks to be mined.

4. Pre-stripping requirements are to be identified in terms of time, tonnage and cost. Pre-stripping should only be that required to achieve steady operation.

5. Previous waste disposal areas are to be checked as appropriate.

6. Mining is to be by contractor with sufficient Joint Venture staff and award personnel to ensure effective sampling, planning and control.

7. Ore retrieval from under the pit bottom and higher grade Stanhope areas (probably by underground means) may be via Joint Venture, or Contractor. Basic costing for each is required.

8. Capital and operating costs are to be estimated to ± 15%. Operating costs are to be presented on a yearly and per tonne basis.

9. Drafting of final pit outline and sections indicating potential underground works are to be performed, from sketches, by MEL.

10. Haul roads etc should be specified and identified within the pit. Ore is to be delivered to stockpile in the Mine

Area.

Timing

It is recognised that this work will be performed on a part time basis. However, the report should be available before end May 1984.

Data

All back up data generated for this study is to remain with MEL.

Contractor enquiries and quotes may be obtained through MEL.

R01512:LSW

METALS EXPLORATION LTD. (INCORPORATED IN VICTORIA)
80 COLLINS STREET, MELBOURNE, AUSTRALIA 3000. TELEPHONE (03) 653 7100. TELEX AA32051. TELEGRAM ROCKDOC.

072

27th April, 1984

Mr. G. Jones,
18 Carbine Street,
DONVALE VIC 3111

Dear Geoff,

With regard to your letter of 16th April 1984 on the Bischoff Tin Project, the following points are made:

(i) Mine life need not be 10 years. If extraction at a higher rate than that indicated is practicable during the first half of the pit life this should be indicated.

(ii) Dilution and extraction loss on DSL remain a real problem with respect to plant feed grade indicated. We would appreciate consideration of any reduction in loss and dilution through relaxation of the porphyry/DSL separation and, comment on changes to mining method which may improve grade control. (I have enclosed copy of a report on Nevada gold mining methods).

(iii) Expected economic parameters, as requested, are:

| | | $ per tonne |
|---|---|---|
| Costs: | Mining | ? |
| | Cartage | 4.00 |
| | Milling/Admin/Town | 24.00 |
| | Smelting | 4.00 |
| | Corporate | 2.00 |
| | Total (excl. mining) | $ 34.00 |

| | |
|---|---|
| Plant Metallurgical Recovery | 65% Sn |
| Smelter Payment | 94.6% Sn |
| Total Payment on feed grade = | 61.5% Sn |

Tin Price, m$29.15 per kg
= A$13.88 per kg

Variable costs within the above $34 figure are approximately $26 per tonne giving a decision on carting to mill or waste at approximate 0.3% Sn.

Yours faithfully,
METALS EXPLORATION LTD.,

G. M. Motteram,
MANAGER - METALLURGY

L01341:CHR

# APPENDIX 4

File 5863.01
MHW/ISC

3rd May, 1982

Memorandum to : File No.

From : M.H. Woffenden

Copies to : M.D. Laverty
D. Simpson
G. Bujtor
A. Jannink

Subject : GUIDELINES FOR GEOLOGICAL ORE RESERVE PREPARATION AND PRESENTATION.

1. GENERAL FOR DSL AND PORPHYRY MINERALISATION

1.1 G.O.R. to be calculated by cross-section and by plan

1.2 Where there is waste inclusion bounded by mineralization either side not greater than 2 m true thickness in ore zones then that waste is to be included as internal dilution.

1.3 Where there is doubt concerning the validity of survey or assay data then that data is not to be used in determining the G.O.R.

1.4 Where the assay information from a DDH is not used in G.O.R. calculation or there is poor core recovery from the hole the fact(s) is/are to be highlighted at the DDH collar on the cross-section drawings.

1.5 Total ore reserves for DSL and porphyry ore types are to be specified in proven, probable or possible catagories. Mention is to be made of the tonnage ascribed to holes where core recovery is less than 25% and between 50% and 25%.

1.6 Minimum intersected thickness of greater than cut-off grade that is to be described as ore is 1.5 metres.

1.7 Tin assay results greater than 4% ore to be reduced to 4% for the purpose of ore grade and contained tin calculations.

.../2

1.8 In poor core recovery zones the assay results are to be used but with over-riding consideration of geological interpretation of surrounding data.

1.9 The results of the surface trenching and current reverse circulation drilling programme are to be included, when available, in the total ore reserve calculation.

1.10 Cross section intervals are to be in accordance with the DDH drilling sections. Drawings are to be at a scale of 1:500 and presented as sepia originals.

1.11 Cross sections ore to show DDH traces, Sn assay grades, rock type information, DSL zones and the prescribed tin mineralization zones.

1.12 For cross section ore block calculations, the grade is to be determined on a weighted area of influence basis from the relevant DDH's.

1.13 Tabulation of G.O.R. by section for the DSL and porphyry is to be separate; each is to detail section number, block reference, area (m2), ascribed tonnage and grade, tonnage x grade and DDH reference.

1.14 Plan intervals to be 5 m.

1.15 Plans are to be named according to the R.L. of the plan and refer to the volume above the plan.

1.16 Plans are to show DDH intercept points, major rock types and DSL zones (except on the '5' m levels) and the tin mineralization by block, and to indicate mineralised material in the plan volume where there is no intersection of the plan plane. Scale to be 1:500 with presentation as sepia originals.

1.17 Plan blocks to be numbered on basis of

Plan level - sequential block number,
with suffix of 'P' for Porphyry
" " " 'L' for DSL
with a further suffix '1' to indicate that the block is not full depth of plan

1.18 Porphyry and DSL plan blocks for each level to be shown on separate sheets.

1.19 Tabulation of G.O.R. by plan for DSL and porphyry to be separate, each to detail plan level, block no., area, tonnage, grade, tonnage x grade and relevant DDH.

1.20 Priority is to be given to the DSL and porphyry reserves in the area bounded by 1700N to 2040N and 940E to 1300E and a lower R.L. of 530 metres.

1.21 The ore reserves presentation is to include illustrative long sections.

2. DSL SPECIFICATION

2.1 The mineralization grade boundaries are to be 0.2% Sn, 0.3% Sn and 0.4% Sn.

2.2 Priority must be given to the 0.3% Sn and 0.4% Sn profiles.

2.3 Separate cross sections, plans and reserves tabulations are to be prepared for each grade boundary.

3. PORPHYRY SPECIFICATION

3.1 The mineralization grade boundaries are to be 0.1% Sn, 0.2% Sn and 0.3% Sn.

3.2 Priority must be given to the 0.2% Sn and 0.3% Sn profiles.

3.3 Separate cross sections, plans and reserve tabulations are to be prepared for each grade boundary.

MHW.

M.H. Woffenden

# APPENDIX 5

078

Copies to G.M. 381079
M.L.

**Golder Associates**
CONSULTING GEOTECHNICAL & MINING ENGINEERS

R.H. AMARAL, K.J. ROSENGREN, L.K. WALKER, W.A. PECK, J.R. MORGAN, J.L. SEYCHUK, E. HOEK, R.G. FRIDAY, H.K. SULLIVAN, M.P.A. WILLIAMS, M. KURZEME, I.M. SMITH

81612271
February 4, 1982

The Project Manager,
Tin Division,
CRA Ltd.,
9th Floor, 63 Exhibition Street,
MELBOURNE, 3000.

Attention: Mr. G. Jones

Dear Sir,

MT. BISCHOFF TIN PROSPECT, TASMANIA

At your request, a seismic profiling and core review, as to the extent of rippable materials in the proposed pits at Mt. Bischoff were carried out. The field work and core review were carried out by Mr. Yu Sheng Mou who was at the site from 25 January 1982 to 29 January 1982. This investigation is a follow-up of the earlier investigation carried out by Mr. Warren Peck and Mr. Malcolm Cook in December 1981. A total of 14 seismic traverses of length between 30-60 metres each, was carried out and drill cores from 30 boreholes were reviewed.

The investigation confirms that the distribution of the hard and soft materials at the proposed main pit to be highly irregular. The depths of the rippable materials are highly variable, often with the easily ripped material adjoining hard, unrippable masses of rocks. The table appended shows a summary of the preliminary depths to the unrippable materials present in the pit.

Along some seismic traverses, particularly lines 1000mE 1860N-1900N, and 1040mE 1800-1920N, the seismic profiling carried out did not detect any refraction from layers with seismic velocity in excess of 2500 metres per second, indicating that the thickness of the rippable material along these traverses to be substantial, in excess of 15 metres. Core review and examination also indicate substantial overburden thickness along these profiles.

079

**Golder Associates**
CONSULTING GEOTECHNICAL & MINING ENGINEERS

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M. KURZEME
I.M. SMITH

81612271
February 8, 1982

The Project Manager,
Tin Division,
CRA Ltd.,
9th Floor, 63 Exhibition Street,
MELBOURNE, 3000.

Attention: Mr. G. Jones

Dear Sir,

MT. BISCHOFF TIN PROSPECT, TASMANIA

We refer to our meeting with you on 5 February 1982 whereby you have requested that we submit the results of our seismic investigation and core review to you as soon as possible in its undrafted form. We are pleased to forward under cover of this letter 11 drawings (Cross Section Line Nos: 920, 940, 960, 980, 1000, 1020, 1040, 1060, 1100, 1140 and 1160mE) summarising the results of our seismic investigation and core review.

Please call on us should you have any queries.

Yours faithfully,
GOLDER ASSOCIATES PTY. LTD.
per:

Warren Peck

W.A. Peck

WAP/YSM/blh

We trust that the above statement is sufficient for your preliminary assessment pending the completion of our final report. Please call on us should you have any queries.

Yours faithfully,
GOLDER ASSOCIATES PTY. LTD.
per:

W.A. Peck

WAP/YSM/blh

# THICKNESS OF OVERBURDEN IN MT. BISCHOFF TIN PROSPECT

| Line | | Thickness (metres) |
|---|---|---|
| 920mE | 1750N | 2.5 |
| | 1800N | 2 |
| | 1850N | 5 |
| 940mE | 1750N | 25 |
| | 1800N | 21 |
| | 1850N | 5 |
| | 1900N | 5.5 |
| 960mE | 1740N | 15 |
| | 1780N | 12.5 |
| | 1820N | 11.5 |
| 980mE | 1750N | 12.5 |
| | 1800N | 15 |
| | 1850N | 13 |
| | 1900N | 8 |
| 1000mE | 1850N | 29 |
| | 1875N | 17 |
| | 1900 | 8.5 |
| 1020mE | 1730N | 18 |
| | 1770N | 5 |
| | 1800N | 11 |
| | 1850 | 13.5 |
| 1040mE | 1800N | 15 |
| | 1850N | 24 |
| | 1900N | 20 |
| | 1925N | 13 |

| Line | | Thickness (metres) |
|---|---|---|
| 1060mE | 1850N | 22 |
| | 1900N | 12.5 |
| | 1950N | 3.5 |
| 1100mE | 1830N | 8.5 |
| | 1850N | 1 |
| | 1875N | 16 |
| 1140mE | 1830N | 14 |
| | 1860N | 11 |
| 1160mE | 1825N | 22 |
| | 1850N | 18 |
| | 1875N | 16 |

083

**Golder Associates**
CONSULTING GEOTECHNICAL & MINING ENGINEERS

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M.P.A. WILLIAMS
M. KURZEME
I.M. SMITH

81612271
December 21, 1981

The General Manager
Metals Exploration Limited
29th Floor
Nauru House
Exhibition Street
Melbourne. Vic. 3000.

Attention: Mr. Geoff Jones

Dear Sir,

MT. BISCHOFF TIN PROSPECT TASMANIA

We forward under cover of this letter proof copies of six drawings summarising our preliminary conclusions, from our seismic profiling and core review, as to the extent of rippable material in the proposed pits at Mt. Bischoff. The field work was carried out by Mr. Warren Peck and Mr. Malcolm Cook who were at site for just over 48 hours (Monday, December 14 - Wednesday, December 15, 1981).

The investigation of the Proposed Main Pit shows the distribution of hard and soft materials is very complex with deep pockets of soft, easily ripped materials adjoining hard, unrippable masses of rock with near-vertical sides. Consequently, there is often a wide difference in levels down to which ripping is possible between adjoining borings. This variability has resulted in our seismic traverses along 1000mE and 1960mN missing what, with the benefit of hindsight, now appears to be the larger soft areas in the Proposed Main Pit. For this reason, we propose that a further weeks's site work be carried out consisting of a review of the balance of the available drill core, plus running a further 1.0 - 1.2km of seismic traversing in the most favourable areas for ripping.

This complex distribution of the hard and soft materials makes any precise slope designs difficult to arrive at, since the hard bands, while virtually acting as retaining walls for the soft materials, appear to have an irregular distribution. At this feasibility stage it appears best to assume an overall slope of 45 degrees, but at the detailed design stage, it may prove possible to make some savings by being able to steepen one or two walls of the pit. Based on the available data (see figures 3 and 4) it could be possible to steepen the northern wall of the pit. Since it would be good design to locate haul roads on the hard competent bands wherever possible, it would be valuable to receive a sketch of the likely pit layout (including haul roads) so that we can site the next series of seismic traverses to obtain a good coverage of key areas, and in particular, to locate the hard competent rock masses.

Although the seismic traverses run to date in the area of the proposed Main Pit have located mainly readily rippable and obviously not rippable materials, much shale that appears to be marginally rippable occurs in the proposed Eastern Pit (see Figures 5 and 6). The shale has a well-developed bedding, which is the dominant discontinuity, and also show at least two joint-sets with joints as close as 0.5m. It is rare to find sticks of core of this shale longer than 120mm. Assessments were made as to the depth that ripping might be possible based on the core, and these are shown on Figure 5. However, the seismic data indicates the bedding and joints to be quite tight. The estimated upper velocity for economic ripping for the two Caterpillar tractors is set out below:-

| Machine Model | Caterpillar Handbook | Golder Experience |
|---|---|---|
| D9L | 2700 m/s | 2400 m/s |
| D10 | 3000 m/s | 2600 m/s |

Estimated Economic Ripping imits in Shales from Seismic Velocities

The extent of the marginal material is further underlined by the fact that the seismic velocity measured in refraction traverses is then immediately below each interface, and that mostly the seismic velocity can be expected to steadily increase with depth. Thus material with a velocity of 2100 m/s at 3m depth could easily have a velocity of 3000 m/s at 10m to 15m depth.

In discussion with Dr. Tim Hagan, our blasting specialist, two possible methods of handling the shale emerged. The first method is to rip off the rippable overburden and then excavate by benched drill and blast techniques. A typical powder factor for jointed shales would be around 0.25 kg. per cubic metre which, for 102mm diameter blast holes, gives an approximate burden of 4m and a spacing of 4.5m. The second method would be to loosen the rock mass with lighter blasting and then rip the loosened shale with a Caterpillar D9L or D10. The notional powder factor for such a possibility is about 0.125 kg. per cubic metre, but a small on-site blast and rip trial would be necessary to confirm this figure. Should 0.125 kg. per cubic metre be appropriate, the burden would be 5.7m and the spacing 6.5m for 102mm diameter shot holes. Where open diamond drill holes are available, these would be charged with explosives thus reducing the number of shot holes needed to be drilled by percussion rigs. The blasting aspect would repay more detailed investigation early next year.

We recommend that a further week's field work by seismic traversing and diamond drill core evaluation be carried out early next year to delineate the extent of soft rippable zones in the proposed Main Pit, and to establish stable areas on which to site the haul roads. For this latter purpose, we will need a sketch of the proposed layout of the Main Pit.

Yours faithfully,
GOLDER ASSOCIATES PTY. LTD.
per:

for Warren Peck.

Encs: 6 Figures

WP/YSM/es

600 E 700 E 800 E 900 E 1000 E 1100 E 1200 E 1300 E 1400 E 1500 E 1600 E 1700 E

2200 N 2100 N 2000 N 1900 N 1800 N 1700 N 1600 N

GLORY HOLE

ROAD

HAULAGE

ROAD

HAULAGE

PIT ENTRANCE

A

A

750 740 730 720 710 700 690 680 670 660 650 640 630 620 610 600 590 580 570 560

N

METALS EXPLORATION LIMITED
MT. BISCHOFF JOINT VENTURE

PIT OPTIMISATION STUDY

MAIN PIT PLAN

086

381087

SCALE 1:1000

5 cm

MAY 1984

DRG. NO. 84-107

FIG. 1-2.

680

GLORY HOLE

HAULAGE ROAD

HAULAGE ROAD

HAULAGE ROAD

PIT ENTRANCE

A

A

600 E 700 E 800 E 900 E 1000 E 1100 E 1200 E 1300 E 1400 E 1500 E 1600 E 1700 E

2200 N 2100 N 2000 N 1900 N 1800 N 1700 N 1600 N

METALS EXPLORATION LIMITED
MT. BISCHOFF JOINT VENTURE

# PIT OPTIMISATION STUDY

## MAIN AND STANHOPE EXTENSION PITS PLAN

SCALE 1:1000

MAY 1984

DRG. NO. 84-108

381088 087

FIG. 1-3.

087

GLORY HOLE

HAULAGE

ROAD

ROAD

HAULAGE

HAULAGE

PIT ENTRANCE

A

A

| UNDERGROUND ORE BLOCKS | ORE TYPE | TONNES (t) | GRADE (% Sn) |
|---|---|---|---|
| A | DSL | 12900 | 1·22 |
| B | DSL | 26400 | 2·43 |
| C | DSL | 16800 | 0·99 |
| D | DSL | 118400 | 1·39 |
| E | QP | 72500 | 0·85 |
| F | QP | 166000 | 0·80 |

381089

METALS EXPLORATION LIMITED
MT. BISCHOFF JOINT VENTURE

# PIT OPTIMISATION STUDY

## MAIN AND STANHOPE EXTENSION PITS PLAN

SHOWING POSSIBLE UNDERGROUND STOPING BLOCKS

SCALE 1:1000

MAY 1984

DRG. NO. 84-109

088

FIG. 1-4.