# HEEMSKIRK TIN

# SCOPING STUDY

**For**

# STELLAR RESOURCES LTD

| | | |
|---|---|---|
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| Prepared by: | Phil Bremner | |

Mining One Pty Ltd
Level 9, 50 Market Street
Melbourne VIC 3000
Ph: 03 9600 3588
Fax: 03 9600 3944

FINAL REPORT

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# EXECUTIVE SUMMARY

A hard rock tin prospect located near to Zeehan in Tasmania is currently under preliminary study for commercial development by Stellar Resources Limited (Stellar).The deposit was explored and evaluated by Aberfoyle Ltd (Aberfoyle) in the 1970 / 80`s. in JV with Gippsland Ltd.

The mineralised zones, known as ***Queen Hill, Severn and Montana***, are complex and refractory and Aberfoyle put considerable effort into finding process routes to exploit the deposit after they had established the resource, but for a number of reasons it was not sufficiently attractive for Aberfoyle to proceed further and development were **halted in 1984** largely due to **imposition of export quotas** on tin concentrates by the **Association of Tin Producers.**

## Geology

Exploration drilling work carried out by Aberfoyle delineated some 7.3 million tonnes of mineralization at an average grade of 0.66% Sn. Higher grade zones within this mineralised envelope were reported as 3.61 Mt @ 1.21% Sn.

This has been re-estimated by Mining One based on recent drilling results supplied by Stellar and has been reported at 4,360,000 Mt tonnes @1.1% Sn for a cut-off grade of 0.6% Sn.

## Mining & Geotechnical Engineering

Mining One has conducted a geotechnical assessment of the drill core and mineralised outcrops to determine an initial overall slope pit angle for any surface mining along with comment on underground stope stability. This assessment was conducted on the Queen Hill orebody only and with limited information could only be considered a high level estimate. The work has recommended an overall pit slope angle of 49 to 50 degrees for a pit height of approximately 100m. Queen Hill orebody has a weak slate hangingwall and therefore a limited exposure is recommended. Mining One would therefore recommend a mechanised cut and fill as the underground mining method.

An economic cut-off grade of 0.6 % Sn was calculated when reporting the ore resources; this was confirmed through a more detailed calculation after determining the chosen underground mining method.

A whittle pit optimisation was run on the deposit and it was shown that a small pit producing 171,599 ore tonnes @ 0.96% Sn would be profitable on the Queen Hill orebody. The pit optimisation was based on a net Sn price of $22,500 / tonne following smelting costs as well as a set of defined geotechnical, mining and metallurgical parameters.

The tonnes per vertical metre were calculated for the 3 orebodies and the combined figure would suggest that an underground mining operation could sustain a production rate of 600,000 tpa.

Mining One designed a set of stope shapes around a cut and fill mining method and a cut-off grade of 0.6% Sn. Once mining parameters were added to the reported mining shapes the underground mine inventory was calculated at 4.442 million tonnes @ 0.93% Sn. The breakdown between the three orebodies is shown in Table 1.

**Table 1: Breakdown Between the Three Orebodies**

| Method | Orebody | Tonnes (kt) | Grade (%) |
|---|---|---|---|
| Open Pit | Queen Hill | 172 | 0.96 |
| Underground | Queen Hill | 1,651 | 0.93 |
| Underground | Severn | 2,382 | 0.86 |
| Underground | Montana | 409 | 1.25 |
| | **Total** | **4,613** | **0.93** |

The mine inventory shown in Table 1 assumes that maximum extraction can be achieved from each level and therefore wider areas of the orebody will require a primary and secondary pillar extraction sequence which relies on a cemented paste fill for consolidated support.

Mining One has designed individual decline development for each orebody with a minimum standoff distance of 50m. This will allow sufficient room to access a minimum of 4 levels from each access ranging in gradient from -1:6.5 to +1:6.5. It has been assumed that all underground mining will be carried out by contractor using twin boom jumbos, 10 yd loaders and 55 tonne haulage trucks.

Mining one has developed a mine schedule in Microsoft excel which will ramp up production to approximately 600,000 tpa by year 3 as is shown in Table 2.

**Table 2: Mine Schedule**

| | Yr -1 | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr 6 | Yr 7 | Yr 8 | Yr 9 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Pit Ore (Kt) | | 172 | | | | | | | | | **172** |
| Grade (%) | | 0.96 | | | | | | | | | **0.96** |
| U/G Ore (Kt) | 30 | 230 | 579 | 601 | 601 | 601 | 600 | 599 | 400 | 200 | **4,442** |
| Grade (%) | 1.16 | 1.00 | 0.97 | 0.99 | 0.97 | 0.95 | 0.88 | 0.81 | 0.88 | 0.86 | **0.93** |
| **Total Ore (Kt)** | **30** | **402** | **579** | **601** | **601** | **600** | **600** | **599** | **400** | **200** | **4,613** |
| **Grade (%)** | **1.16** | **0.98** | **0.97** | **0.99** | **0.97** | **0.95** | **0.88** | **0.81** | **0.88** | **0.86** | **0.93** |

## Processing

Aberfoyle's original premise for developing matte fuming changed when more amenable ores were discovered at Queen Hill, and offered an option of processing by conventional mineral processing methods such as those used at Cleveland Tin and Renison Tin mine.

The maximum mining rate determined by Aberfoyle, between 150,000-200,000 TPA, imposed a major constraint to building a dedicated process plant, as operating costs would be high due to no benefit from economies of scale. However, at current tin prices and a higher mining and treatment rate of 600,000 tpa determined in this study the economics are much improved.

The ores respond to Heavy Media Separation, and although not pursued by Aberfoyle could be employed to pre–concentrate the ore ahead of down stream beneficiation.

An estimate has been made to determine CAPEX and OPEX at scoping study level of accuracy for a conventional mineral processing plant using operating parameters at 600,000 TPA @ 1.1 % Sn. tin recovery of 70 % and final concentrate grade at 50% Sn.

These parameters were based on the ore characterisation and metallurgical test work on drill core from the deposit carried out by Aberfoyle, and the 1983 pre–feasibility study report, and preliminary results of ore characterisation work on core from the recent drilling program. Using a process flow diagram developed from the test work, costs for a plant incorporating the unit processes shown was developed using standard engineering procedures.

## Costs

Mining One has assumed an open pit cost of $4 / tonne for all material excavated from the pit, this is based on contractor rates for a small pit of less than 4 million tpa. As the pit is to be mined by contractor who will supply all equipment Mining One has assumed that all costs will be operating.

Underground operating costs have also been developed assuming a contractor will complete the works. These have been built up from a series of fixed and variable rates applied to the scheduled physicals.

Processing operating costs have been built up from analysis of labour, reagents, power and maintenance and have been estimated at $26.59 / tonne with an additional $0.5 / tonne allowed for distribution of tailings. Environment and other site cost not included in the processing costs have been allowed for in administration and management wages.

Operating costs are shown in Table 3.

**Table 3: Operating Costs**

| Major Operating Costs | Total cost | Unit Costs ($/ore tonne) | Unit Costs ($/ Sn recovered) |
|---|---|---|---|
| Mine Production (Contractor) | $206M | $44.63 | $6,879 |
| Mill | $125M | $27.09 | $4,176 |
| Mine Management Wages | $27M | $5.76 | $887 |
| Power | $14.6M | $3.18 | $490 |
| Administration | $6.0M | $1.31 | $202 |
| Transport | $2.4M | $0.52 | $80 |

| Major Operating Costs | Total cost | Unit Costs ($/ore tonne) | Unit Costs ($/ Sn recovered) |
|---|---|---|---|
| Light vehicles & Surface loader | $1.4M | $0.33 | $51 |
| Other Items | $0.4M | $0.09 | $14 |
| **Total** | **$382.4** | **$82.90** | **$12,779** |

Capital costs for major equipment have been derived from either budget quotes or a Mining One database of capital equipment costs. Major capital costs are shown in Table 4.

**Table 4: Capital Costs**

| Major Capital Items | Estimated Cost | Unit Costs ($/ore tonne) | Unit Costs ($/ Sn recovered) |
|---|---|---|---|
| Mill (including tailings dam) | $80M | $17.42 | $2,685 |
| Mine Development | $44M | $9.53 | $1,468 |
| Geology | $12M | $2.59 | $399 |
| Pastefill Plant | $7M | $1.52 | $234 |
| Mine Ventilation Fans | $1.75M | $0.38 | $58 |
| Office / Change room / Workshop | $700,000 | $0.15 | $24 |
| Light vehicles & Surface loader | $800,000 | $0.17 | $27 |
| Power Installation (Mill to UG) | $800,000 | $0.17 | $27 |
| Mine Rescue & Safety Equipment | $400,000 | $0.08 | $12 |
| Other Items | $1.2M | $0.26 | $40 |
| **Total** | **$148.9M** | **$32.28** | **$4,976** |

## Financial Model

The capital and operating costs have been applied to the mine schedule to create a financial model as shown in Table 5. The adjusted revenue has allowed for smelter costs and state royalties.

**Table 5: Financial Model**

| | Yr -1 | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr 6 | Yr 7 | Yr 8 | Yr 9 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|

| | Yr -1 | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr 6 | Yr 7 | Yr 8 | Yr 9 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Cost (US$,000,000) | $94.1 | $26.3 | $10.6 | $3.7 | $7.0 | $0.75 | $0.95 | $0.75 | $0.5 | $0.25 | **$148.9** |
| Operating Cost (US$,000,000) | $6.5 | $41.2 | $46.1 | $48.3 | $47.8 | $47.0 | $46.8 | $46.5 | $32.5 | $19.7 | **$382.4** |
| Adjusted Revenue (US$,000,000) | | $65.0 | $95.7 | $96.4 | $95.2 | $93.2 | $86.4 | $79.5 | $57.6 | $30.4 | **$699.4** |
| Cash Flow (US,000,000) | **-$100.6** | **-$1.5** | **$40.6** | **$38.8** | **$45.2** | **$47.0** | **$40.0** | **$33.6** | **$25.6** | **$10.9** | |

The split between pre production capital and working capital is shown in the table below.

| | Yr -1 | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr 6 | Yr 7 | Yr 8 | Yr 9 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Pre Production Capital (US$,000,000) | $94.1 | $14.3 | | | | | | | | | **$108.4** |
| Working Capital (US$,000,000) | | $8.0 | $10.6 | $3.7 | $7.0 | $0.75 | $0.95 | $0.75 | $0.5 | $0.25 | **$32.5** |

## NPV and Sensitivities

The NPV for the project is $70M and the IRR 25% for the base case Sn price of $25,000 / tonne and a discount rate of 10%. A number of sensitivities on Sn price, grade and recovery have been performed to produce a range of NPV's as shown in Table 6.

**Table 6: NPV Sensitivities**

| NPV | | | |
|---|---|---|---|
| **Change** | **$Sn** | **Grade** | **Recovery** |
| **-10%** | $ 27M | $ 27M | $ 27M |
| **0%** | $ 70M | $ 70M | $ 70M |
| **+10%** | $ 113M | $113M | $113M |

**Phil Bremner**
**Senior Mining Engineer**

**Bill Frazer**
**Director**

# 1 INTRODUCTION

Mining One Pty Ltd (Mining One) was commissioned by Stellar Resources to undertake a scoping study on the economic extraction of the Heemskirk Tin deposits. The deposits are located on the west coast of Tasmania outside the township of Zeehan.

Earlier work by Mining One had produced a geological resource report and block model in Surpac mining software which was used as the basis for the study. The geological report had identified three deposits which make up the Heemskirk Tin resource, Montana, Severn and Queen Hill the latter of which has the more intense drilling.

Mining One has also carried out a site investigation to develop an understanding of the geotechnical and site based issues which will affect a mining operation in the area.

The scoping study will provide the following deliverables as defined in the scope of works.

- A comparison of cost options considering capital and operating options while mining via open pit v underground methods;
- An open pit optimisation and if required a preliminary surface mining design and mining schedule;
- An underground mine layout, schedule and costing taking into consideration service and environmental requirements; and
- A comparison of the three processing options and their likely operating and capital costs; and
- A report summarising the findings and any recommendations for future work or activity.

While the following areas will be excluded from the study

- Future exploration drilling requirements to expand the quality or quantity of the resource;
- Environmental assessment and rehabilitation requirements at end of mine life;
- Owner operator option for either underground or open pit mining or processing;
- Freight and shipping costs of concentrate to any optional smelting option; and
- Surface hydrology and hydrogeology aspects required for dewatering of the mine.

# 2 GEOLOGY

The Queen Hill, Zeehan Montana and Severn tin deposits are deposits of cassiterite in shear zones and as replacement deposits. Tin occurs principally as cassiterite and is associated with base metal sulphides, pyrite and pyrrhotite.

The Queen Hill mineralisation outcrops, strikes more or less north-south, dips about 60$^{o}$ to the east, has a strike length of over 300m, a width of 1 to 50m and a down dip extent of over 300m.

The Zeehan Montana mineralisation is known below a depth of 75m below surface, strikes just north of east, dips about 55$^{o}$ to the south, has a strike length of just over 100m, a width of 1 to 15m and a down dip extent of over 300m.

The Severn mineralisation is known below a depth of 120m below surface, strikes north-south, dips from 70$^{o}$ to 20$^{o}$ to the east, and has a strike length of about 400m, a width of 1 to 50m and a down dip extent of over 400m.

The mineralisation in the Zeehan Cassiterite Deposits is of two styles:

- Mineralisation in faults with cassiterite associated with siderite, quartz, galena, sphalerite, and minor chalcopyrite, pyrite, stannite and fluorite. This is the style of mineralisation associated with Clarke's Lode and Stormsdown Lode.
- Replacement mineralisation with cassiterite associated with siderite, quartz, pyrite and pyrrhotite with minor sericite and fluorite or sellaite. This is the style of mineralisation associated with replacement of beds in the Montana volcanics at Queen Hill, perhaps in the replacement of the Poverty Point beds at Zeehan Montana and in the replacement of the dolomite in the Success Creek at Severn

The estimates of mineral resources were made using diamond drill hole assays within the interpreted mineralisation. All samples were composited to 1 metre lengths and no top-cuts were applied. Bulk densities were based on estimated sulphur grade, where this was available, or were set to 3.3 tonnes per cubic metre for Queen Hill, 3.9 tonnes per cubic metre for Zeehan Montana and 3.2 tonnes per cubic metre for Severn. The grade estimates of the Mineral Resources were made using an inverse distance squared algorithm.

The Mineral Resources were based on a cut-off grade of 0.6% Sn which was based on a tin price of US$30,000 per tonne and reasonable assumptions for exchange rate, costs and modifying factors including mining recovery, mining dilution and metallurgical recovery.

All tonnages are dry metric tonnes.

| Mineral Resources | | |
|---|---|---|
| **0.6% Sn cut-off grade** | | |
| **Indicated Mineral Resources** | | |
| Queen Hill | 1,600,000 tonnes | 1.2% Sn |
| All Indicated Mineral Resources | 1,600,000 tonnes | 1.2% Sn |
| | | |
| **Inferred Mineral Resources** | | |
| Zeehan Montana | 360,000 tonnes | 1.6% Sn |
| Severn | 2,400,000 tonnes | 0.9% Sn |
| All Inferred Mineral Resources | 2,760,000 tonnes | 1.0% Sn |
| | | |
| **Indicated + Inferred Mineral Resources** | | |
| All Mineral Resources | 4,360,000 tonnes | 1.1% Sn |

# 3 GEOTECHNICAL

## 3.1 Site Inspection

Mining One geotechnical engineer Trent Collins conducted a site visit of the Queen Hill area in March 2011 to collect data for the geotechnical assessment of mining in the area.

The area was inspected for its size, topography, population and development spread, flora, fauna and waterways. Historic mine workings and regional infrastructure were all noted for input into overall scoping study considerations.

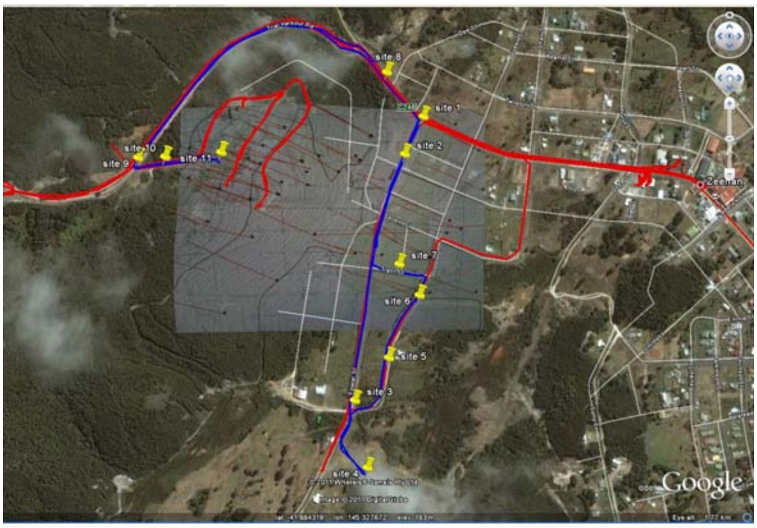



**Figure 3-1: Site Area with Inspection Routes and Overlay of Drill Sites and Ore Bodies.**

Figure 3-1 above shows a summary of the areas inspected by vehicle and the yellow markers indicate areas that were more extensively inspected on foot. The locations of Queen Hill and Severn ore bodies and the previous exploration holes have been overlaid on the satellite photo.

Part of the inspections was directed by geologist, Ray Hazeldene whose local knowledge aided the information collection. Ray outlined much of the mining history and local conditions.

## 3.2 Geotechnical Characteristics

The local rock was inspected in outcrop and in the core stored by Stellar on site. Most of the core inspected was older than thirty years and had deteriorated beyond useful geotechnical interpretation. It therefore did not present an accurate picture of the expected geotechnical characteristics of the rock mass. More recent boreholes in the Queen Hill ore body did present more relevant information for inclusion in this study. The core from a total of 8 historic and 2 recent boreholes was inspected for broad ranging geotechnical parameters.

The outcrop around Queen Hill was also inspected on the northern and western sides, especially where existing workings were present.

Geological logs of the core inspected were collected to support other information as well as local and regional surface geology maps.

The ore bodies are believed to lie near the contact between two units of rock. The lower of these is a volcaniclastic rock and the upper are highly deformed and brecciated shale called the “black shale” on site.  This Black Shale unit as logged in the recent Queen Hill boreholes varies from outcrop to 13 m deep and to be 45 to 86 m thick. Its full extent was not recorded in this report from the historical logs.

## 3.3 Empirical Pit Slope Design

From the observed drill core and outcrops made during the site inspection, typical rock structure and joint conditions were assigned to rate the anticipated rock mass conditions. The empirical Mining Rock Mass Rating System (MRMR) was used to determine an appropriate overall pit wall slope angle for design purposes. The rock mass conditions and mining adjustments applied are summarised in Table 3-1.

**Table 3-1: Summary of typical slope rock mass conditions.**

| MRMR Laubscher (1990) | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| IRS | FF | Calcs | | | | | $RMR_L$ | Mining Adjustments | | | | | MRMR |
| | | Small Joint Expression | Large Joint Expression | Fill | Wall Alt. | x 40 | | Weathering | Orientation | Blasting | Stress | Total Adj. | |
| 12 | 15 | 0.90 | 0.95 | 0.85 | 1.00 | 29 | 56 | 0.94 | 0.98 | 0.94 | 1 | 0.87 | 49 |

The RMRL value of 56 rates the expected rock mass conditions as being fair. The mining adjustments further down grade the rock mass to take into account weathering, structure orientation to the slope, blasting effects and stress field.

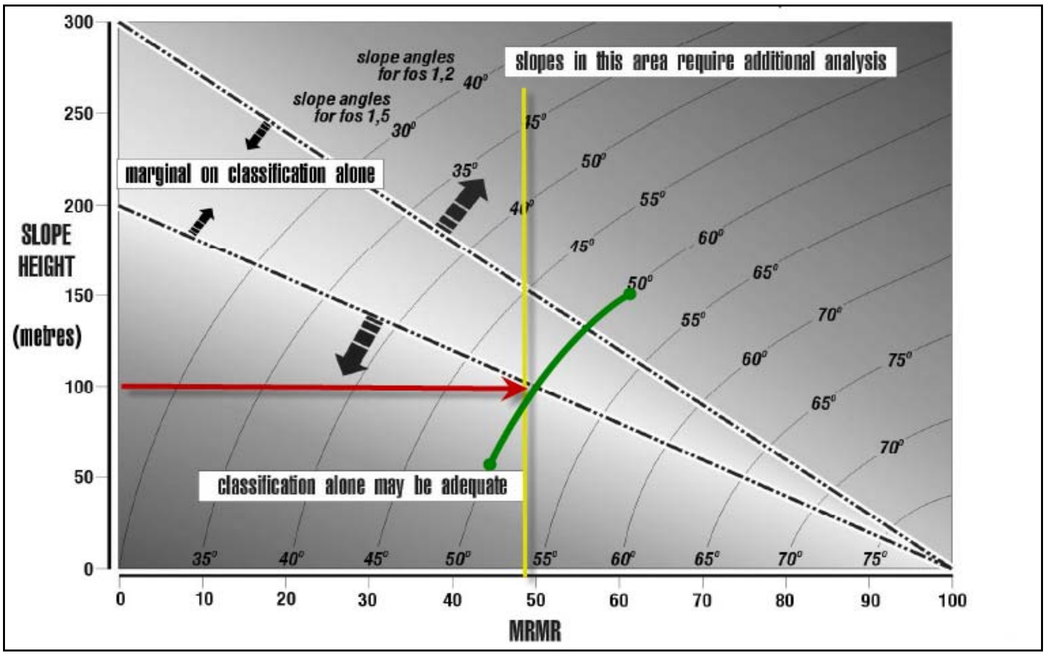



**Figure 3-2: Empirical Overall Pit Slope Angle Design Chart (Haines & Terbrugge)**

Using the empirical slope design chart compiled by Haines and Terbrugge, a 100m deep pit would require an overall slope angle of 49° to 50 to be considered stable for a slope height of 100m (Figure 3-2).

## 3.4 Underground

The underground mining methods based on the dip of the ore body and variation in mineralised width (1m to 95m), lend itself to a cut and fill mining method. The hangingwall shale/slate unit, which varies in thickness between 45m and 86m, is friable and Poor to Very Poor in rock mass quality. It is recommended that the access drives intersect the Queenscliff orebody from the footwall to avoid having to manage poor ground conditions at the shale intersection.

### 3.4.1 Ground Support

From the ground conditions observed in the limited drill core, conditions in the ore body are expected to be typically Fair. For the majority of the access and ore drives a standard ground support of 2.4m bolts on a 1.2m by 1.2m spacing with mesh installed grade to grade is considered to be appropriate. An Excavation Support Ratio (ESR) of 1.6 was selected for the planned drives, which classifies the drives to be permanent mine openings. This is considered to be a conservative approach, as temporary mining openings (ore drives) can be assigned an ESR of 3-5 with further study.

The equivalent excavation dimensions with the applied ESR are plotted against calculated Q values obtained from the inspected core, on the ground support design chart shown in Figure 3-3.

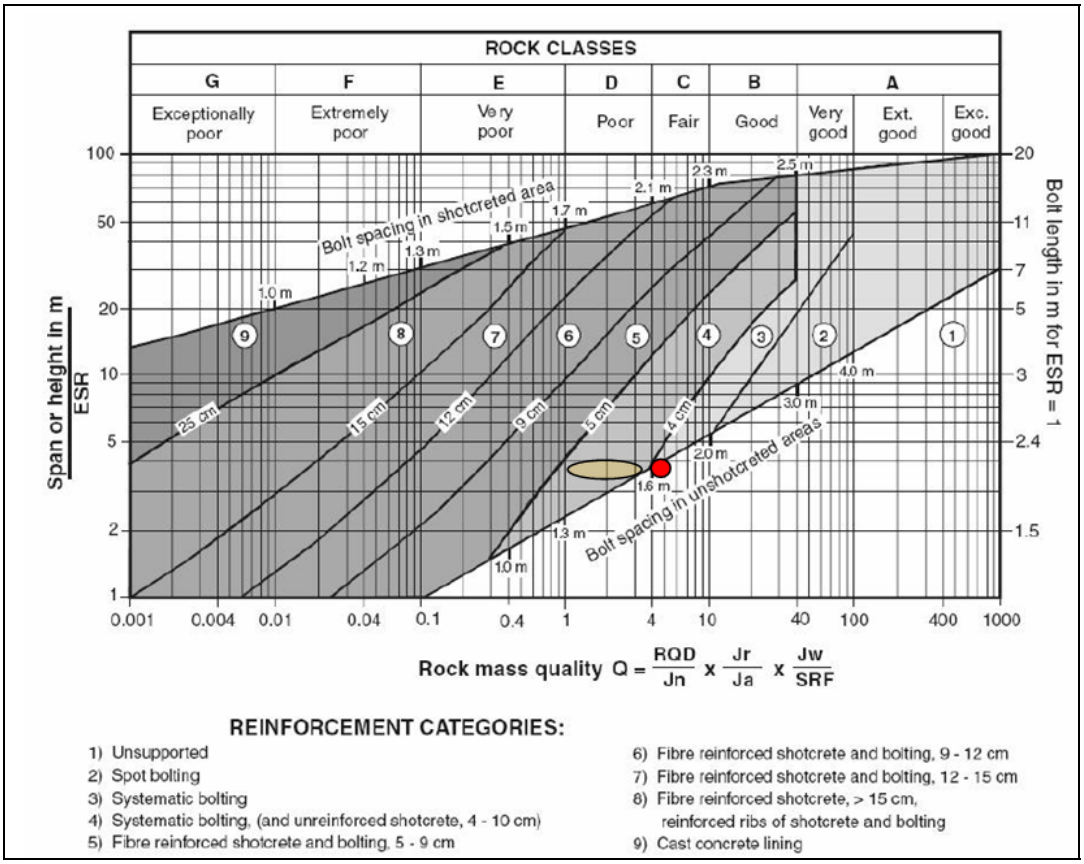



**Figure 3-3: Empirical Ground Support Design Chart (Grimstad and Barton, 1993)**

The red dot indicates typical expected ground conditions, while the orange eclipse indicates Poor ground conditions which may require additional ground support in the form of fibrecrete.

It is anticipated a contingency of 10% be included in the ground support costing to account for drives requiring fibrecrete as additional ground support.

Fibrecrete will also be required for the portal entrance and fibrecrete ribs for a distance of approximately 15m. This is dependent on final portal location and depth of weathering. A more detailed geotechnical study should be completed once the portal position is finalised.

Paste fill should be considered as an alternative backfill material to waste rock, as this will lead to improved ground control and potential increase in mining extraction percentage.

## 3.5 Geotechnical Conclusion

A wide range of information was collected for input into models to assist in the scoping study of the Queen Hill and nearby ore bodies. While the geotechnical information available was lacking a good overview of the area was gained and potential siting for a mine and infrastructure was discovered.

An overall pit slope design angle of 50°for slope height of 100m is considered to be appropriate for the anticipated rock mass conditions.

A cut and fill mining method is considered to be applicable for the ore body dip, variation in width and ground conditions.

It is recommended that access development intersect the ore bodies from the footwall to avoid poor ground conditions associated with the hangingwall shale.

The recommended ground support for access and ore drives in typical ground conditions is 2.4m long bolts on a 1.2m by 1.2m pattern with mesh grade to grade.

A 10% contingency should be included in the ground support costs for fibrecrete to be used in Very Poor to Poor ground conditions that may be encountered in the drives.

The portal entrance will require fibrecrete and ribs through the weathered zone. A detailed geotechnical evaluation and design should be complete in the next stage, once the portal location is finalised.

# 4 OPEN PIT

## 4.1 Optimisation

Open pit optimisations were conducted using Whittle software to produce theoretical pit shells based upon the geological block model and estimated financial, geotechnical, and productivity parameters.

A summary of the parameters used in the initial optimisations are contained in Table 4-1 below.

**Table 4-1: Initial Optimisation Parameters**

| | |
|---|---|
| Tin price | $30,000 / tonne |
| Overall slope angle | 40 degrees |
| Mining cost | $4.00 / tonne mined |
| Mining recovery | 90% |
| Mining dilution | 10% |
| Processing cost | $40.00/ ore tonne |
| Processing recovery | 70% |

These parameters have been estimated within the accuracy levels for a scoping study and will need to be reviewed prior to going to a feasibility study.

Given the level of detail of the study and the shallow design of the pit there has been no time discounting or specific allowance for an increased mining cost due to RL (depth) included in the optimisations.

The initial optimisation resulted in only the Queens Hill deposit being mined via an open pit method. The optimum pit shell had an estimated total cash value of $16.9M for a total pit size of approximately 3.3 Mt. These figures are shown in Table 4-2.

After the initial optimisation it was decided to use a more conservative tin price of $25,000 / tonne and discount this by 10% for smelting charges to give a net value of $22,500 / tonne. This optimisation is considered the base case and the results are shown in Table 4-2 along with those for a range of higher tin prices.

Where a larger pit was optimal the mining costs, dilution and mining recoveries were adjusted to reflect the scale of the operation. It was estimated that pit shells exceeding 50 Mt would have a reduced mining cost of $3.50 per tonne, a recovery of 95% and a dilution of 5%.

**Table 4-2: Optimisation Summary**

| Net Sn value | Mining cost | dilution | Recovery | Undiscounted profit | Total pit tonnage | Waste tonnage | Ore tonnage | Mined Sn Grade | Recovered Sn (tonnes) | $/ Ore tonne | $ / Sn tonne (recovered) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **$22,500** | **$4.00** | **10%** | **90%** | **$7,021,764** | **2,631,857** | **2,484,944** | **146,913** | **1.01%** | **1,041** | **$112** | **$15,758** |
| $25,000 | $4.00 | 10% | **90%** | $9,735,743 | 3,045,804 | 2,882,395 | 163,409 | 0.99% | 1,138 | $115 | $16,450 |
| $30,000 | $4.00 | 10% | **90%** | $15,659,789 | 3,252,499 | 3,073,038 | 179,461 | 0.95% | 1,195 | $112 | $16,894 |
| $30,000 | $3.50 | 5% | **95%** | $61,052,371 | 54,612,263 | 52,426,653 | 2,185,610 | 0.74% | 11,321 | $127 | $24,606 |
| $36,000 | $3.50 | 5% | **95%** | $131,637,816 | 60,282,831 | 57,919,443 | 2,363,388 | 0.73% | 12,143 | $129 | $25,161 |
| $40,000 | $3.50 | 5% | **95%** | $180,409,578 | 60,501,659 | 58,077,255 | 2,424,404 | 0.72% | 12,229 | $127 | $25,246 |

High grades at depth have resulted in a large increase in pit size with only an incremental change in cash value when the net tin value reaches $30,000/ tonne.

The base case pit shell using a net tin value of $22,500 is shown in Figures 4-1 & 4-2.

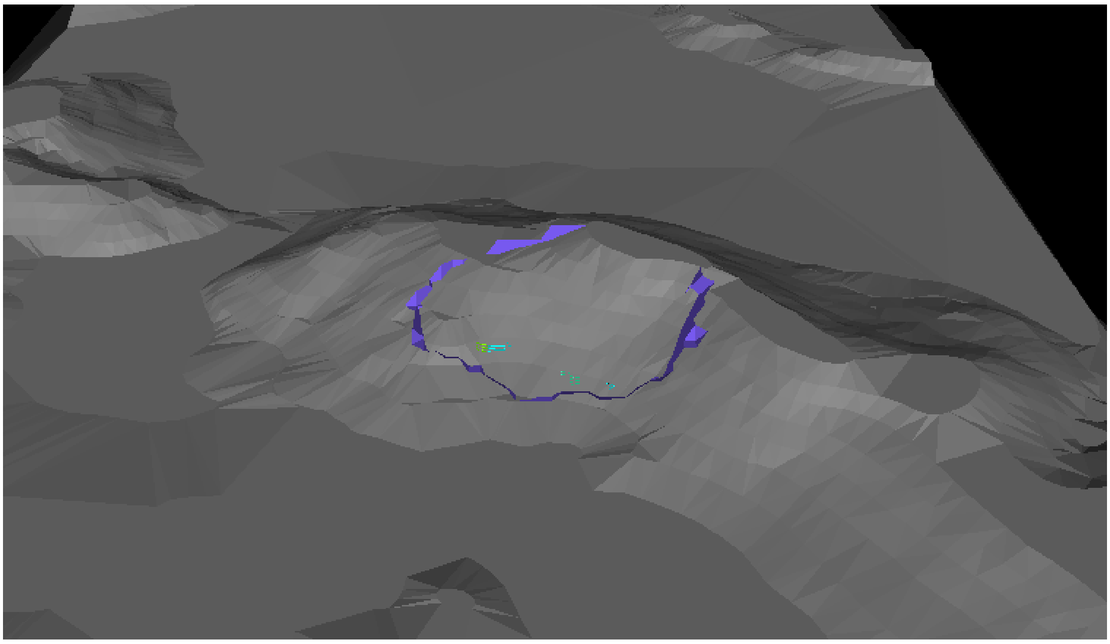


**Figure 4-1: $22,500/t Pit Shell, Viewed from the West**

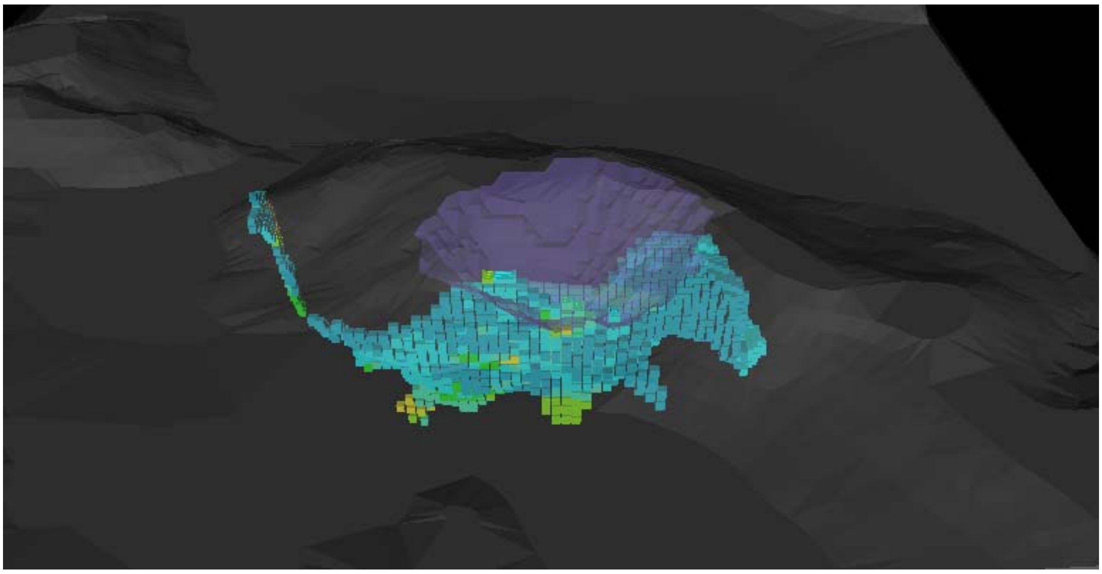


**Figure 4-2: $22,500/t Pit Shell, Viewed from the West (Transparent Wireframes)**

The optimal transition point between mining the orebody from either open pit or underground has not be determined however there would appear to be a low tonnage area below 1150 RL that could be left as a barrier pillar between the open pit and the underground.

## 4.2 Pit Design

Mining One designed an open pit on the base case optimisation of $22,500/t pit shell. The design parameters are in Table 4-3 below.

**Table 4-3: Pit Design Parameters**

| | |
|---|---|
| Berm width | 5m |
| Batter angle | 60° |
| Bench height | 15m |
| Road width | 22m |
| Road gradient | 11.1%, 1:9 |

The wall on the eastern side was made as steep as possible in an attempt to minimise the total amount of material movement, by putting the haul ramps in a switchback configuration on the western side of the pit.

The haul roads are 22m wide at a gradient of 1 in 9 (11.1%). This haul road configuration was based upon Caterpillar 777 (or equivalent) sized trucks.

The initial pit design can be seen below in Figure 4-3.

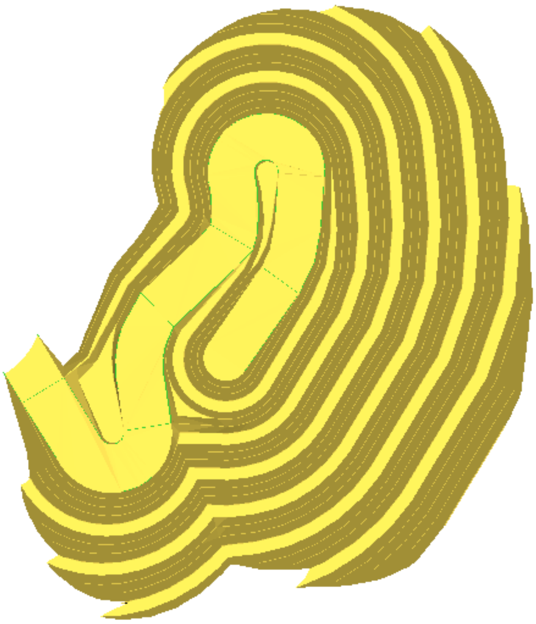


**Figure 4-3: $22,500/t pit design, plan view**

**Figure 4-4: $22,500/t pit design and Whittle shell, plan view**

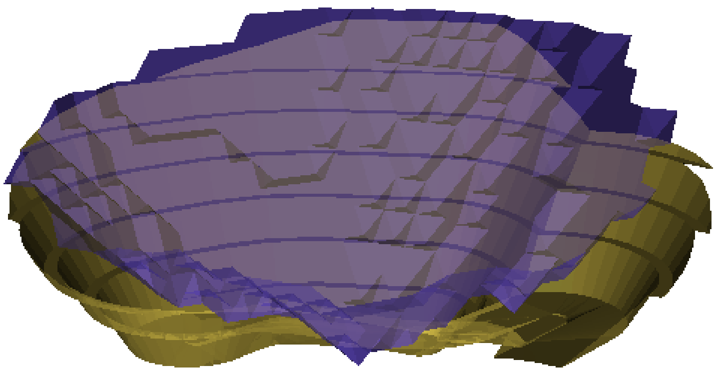


**Figure 4-5: $22,500/t pit design and Whittle shell, viewed from the west**

There is no 'goodbye' cut in the design – this means that there may be an upside potential value to the overall value of the pit when mining has almost been completed. A cut is done via a retreating excavating method to extract the maximum amount of ore from the pit. The advantage of doing this is somewhat negated if a 'glory hole' is to be mined via underground methods to break into the pit.

Due to the large amount of switchbacks in the western wall and the flat areas required for the corners, the pit design has been pushed outside the whittle shell. This has impacted on the profitability of the design.

A summary of the pit design results can be seen below in Table 4-4.

**Table 4-4: Pit Design Results**

| RL from | RL to | Total tonnes | Ore tonnes (insitu) | Tin tonnes (insitu) | Average grade (insitu) |
|---|---|---|---|---|---|
| **1170** | **1280** | **2,977,602** | **171,599** | **1,647** | **0.960** |
| 1270 | 1280 | 34,598 | 0 | 0 | 0 |
| 1260 | 1270 | 97,632 | 0 | 0 | 0 |
| 1250 | 1260 | 170,878 | 0 | 0 | 0 |
| 1240 | 1250 | 263,015 | 0 | 0 | 0 |
| 1230 | 1240 | 374,166 | 0 | 0 | 0 |
| 1220 | 1230 | 444,067 | 5953 | 72 | 1.210 |
| 1210 | 1220 | 515,195 | 12383 | 106 | 0.859 |
| 1200 | 1210 | 511,118 | 39098 | 255 | 0.652 |
| 1190 | 1200 | 337,409 | 44883 | 340 | 0.758 |
| 1180 | 1190 | 184,685 | 47322 | 505 | 1.066 |
| 1170 | 1180 | 44,840 | 21960 | 369 | 1.682 |

## 4.3 Pit Schedule

The schedule was constructed based upon a single Hitachi EX1200 Excavator. It is anticipated that there may be impacts to the maximum production rate due to difficult terrain and relatively small working benches, especially at the start and end of the mine life. Hence, the production rate was limited to 4 Mtpa and bench advance was limited to a maximum of 2 benches per month. The schedule can be seen below in Table 4-5.

**Table 4-5: Proposed Schedule at 4Mtpa**

| RL from | RL to | Month 1 | Month 2 | Month 3 | Month 4 | Month 5 | Month 6 | Month 7 | Month 8 | Month 9 | Month 10 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1270 | 1280 | 34,598 | | | | | | | | | |
| 1260 | 1270 | 97,632 | | | | | | | | | |
| 1250 | 1260 | | 170,878 | | | | | | | | |
| 1240 | 1250 | | 162,455 | 100,560 | | | | | | | |
| 1230 | 1240 | | | 232,773 | 141,393 | | | | | | |
| 1220 | 1230 | | | | 191,940 | 252,127 | | | | | |
| 1210 | 1220 | | | | | 81,206 | 333,333 | 100,656 | | | |
| 1200 | 1210 | | | | | | | 232,677 | 278,441 | | |
| 1190 | 1200 | | | | | | | | 54,892 | 282,517 | |
| 1180 | 1190 | | | | | | | | | 50,816 | 133,869 |
| 1170 | 1180 | | | | | | | | | | 44,840 |
| Ore tonnes: | | | | | 2,573 | 5,332 | 8,012 | 20,218 | 28,601 | 50,602 | 56,261 |

## 4.4 Pit Costs

Mining and haulage costs of $4 / tonne have been based on using a contractor utilising a Hitachi EX1200 excavator loading into 5 Caterpillar 777 trucks or equivalent.

The total mining cost of the pit is $12.6M which equates to a unit cost of $73.41 / ore tonne.

# 5 UNDERGROUND

## 5.1 Assumptions

### 5.1.1 Cut-off Grade

The following assumptions have been used to calculate the mining cut-off grade:

- Sn price of AU$25,000 per tonne
- Mining & administration operating cost estimate of $60 per tonne
- Mill recovery of 70%
- Mill throughput of 600,000 tpa with associated milling cost of $40/t
- Concentrate grade of 50% Sn
- Smelter Charges of 5%

These assumptions were used to calculate a cut-off grade of 0.60% Sn which coincides with the cut-off grade calculated for reporting geological resources, and were used as the basis for stope designs.

### 5.1.2 Accuracy

In line with the level of accuracy determined for a scoping study Mining One has determined the accuracy of this report to be +/- 30%.

## 5.2 Mine Layout

### 5.2.1 Mullock Dump

For the study Mining One has designated the area to the north of the mine as the location of the mullock dump. Although a specific location has not been chosen a haulage distance of 1.2km has been used to calculate haulage costs.

### 5.2.2 ROM Pad & Mill

Ideally the mill will be located on flat ground approximately 250m x 250m in area. Mining One has not identified a specific location but has assumed the mill will be located to the north near the mullock dump. This location is convenient to the mullock dump to allow haul trucks to back load waste into the mine for backfill.

### 5.2.3 Tailings Storage Facility

The milling process will produce approximately 2.3 M $m^3$ of tailings of which 0.4 M $m^3$ will be used as pastefill, this will leave 1.9 M $m^3$ for placement in a tailing storage facility. The open pit has a volume of 1.2 M $m^3$ and could be used to store part of the remaining tailings. As the pit is built into the side of Queen Hill its capacity to hold saturated tailings will be limited.

The old Zeehan Zinc pit located 4 km along the trial harbour road would serve as a suitable location to deposit the remaining tailings however this area is at an elevation 220m higher than the proposed mill site.

Rather than pump the tailings to the Zeehan Zinc pit it may be a better option to dewater the tailings so they could be transported via truck. Tailings destined for the Queen Hill pit could also be treated in this way and therefore alleviates any water problems in the underground mine associated with wet tailings as well as increasing the capacity of the pit.

### 5.2.4 Site Offices & Paste Plant

Mining One has allocated an area between the underground portal and open pit as a proposed location for site offices and a pastefill plant. This area has not been fully investigated for suitability for such a site but has been selected purely for its close proximity to the mine workings.

## 5.3 Mining Method & Production Rates

### 5.3.1 Method Selection

Geotechnical constraints have restricted the open stope spans and therefore the mechanised cut and fill method has been chosen as the preferred mining method. For the case of the scoping study and to comply with geotechnical ground support recommendations the width of the drives has been restricted to a maximum of 6m. Sections of the orebody which are wider than 6m will be developed as cross cut panels as shown in Figure 5-1.

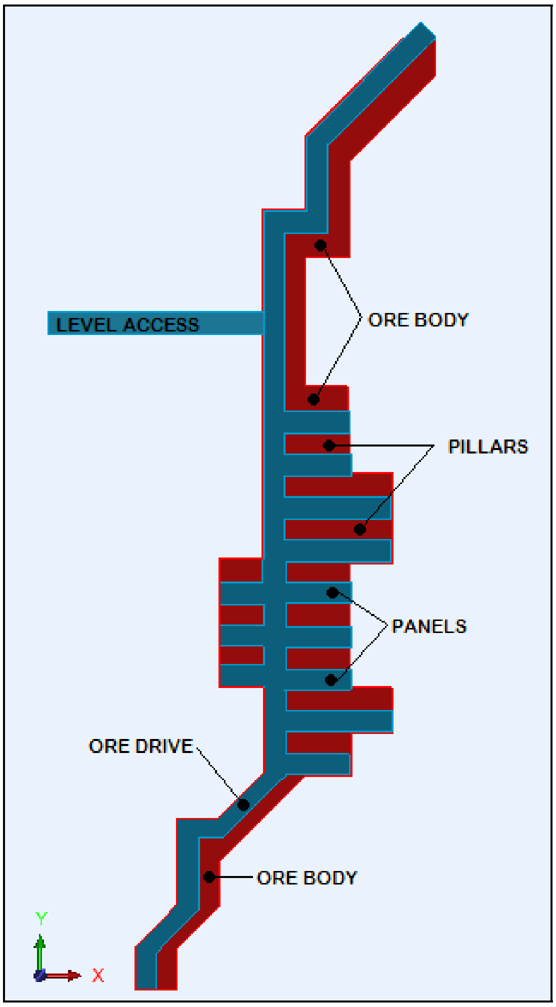



**Figure 5-1: Level Plan Development Design**

To maximise extraction, panels will be filled with a consolidated fill allowing the pillars to be extracted prior to mining the lift above. This sequence is shown in Figure 5-2. Mining One has estimated that 70% of the ore will be extracted prior to pillar recovery.

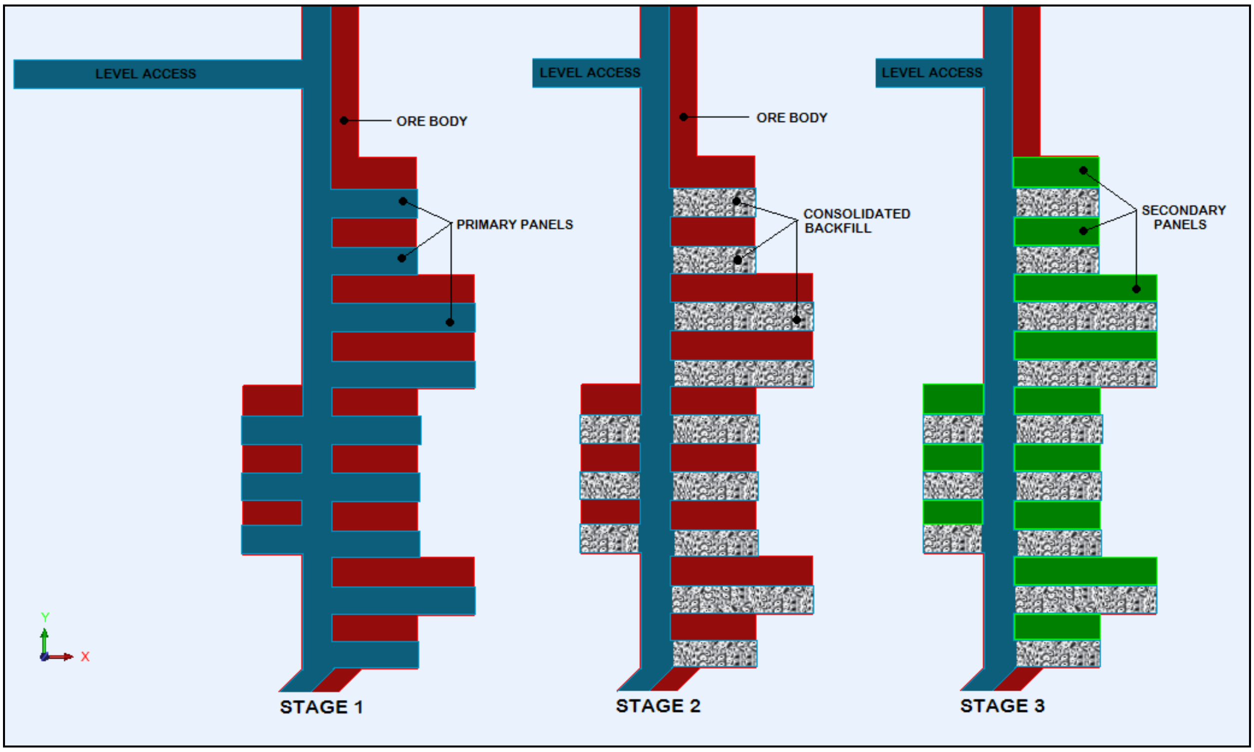



**Figure 5-2: Pillar Recovery Sequence**

### 5.3.2 Cut & Fill

Cut & fill mining is based on taking consecutive horizontal slices through the orebody between 4 & 5 metre high and filling the level to produce a working platform for the lift above as well as offering wall support.

Whilst this method introduces a short drill and blast cycle and does not have the productivity advantages of up-hole stoping, it does maintain far better ground conditions for the backs and walls. This is particularly the case when combined with the use of suitable low powered explosives for perimeter holes.

Each lift higher in the sequence is accessed by stripping the backs of the access as shown in Figure 5-3. To ensure the maximum number of lifts can be achieved from each access the initial development is declined at a gradient of -1:6.5.

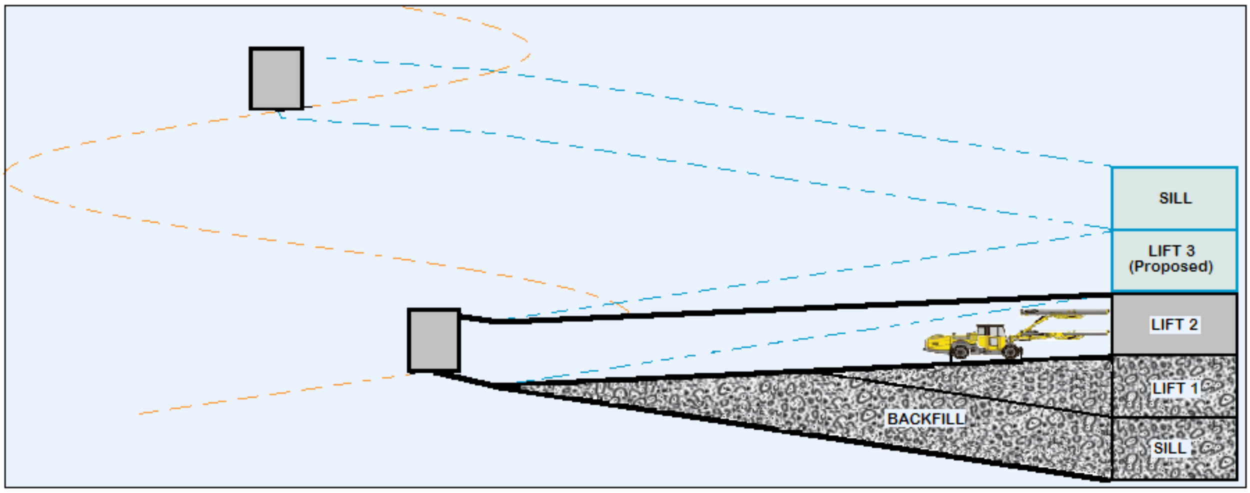



**Figure 5-3: Extraction Sequence Cross Section**

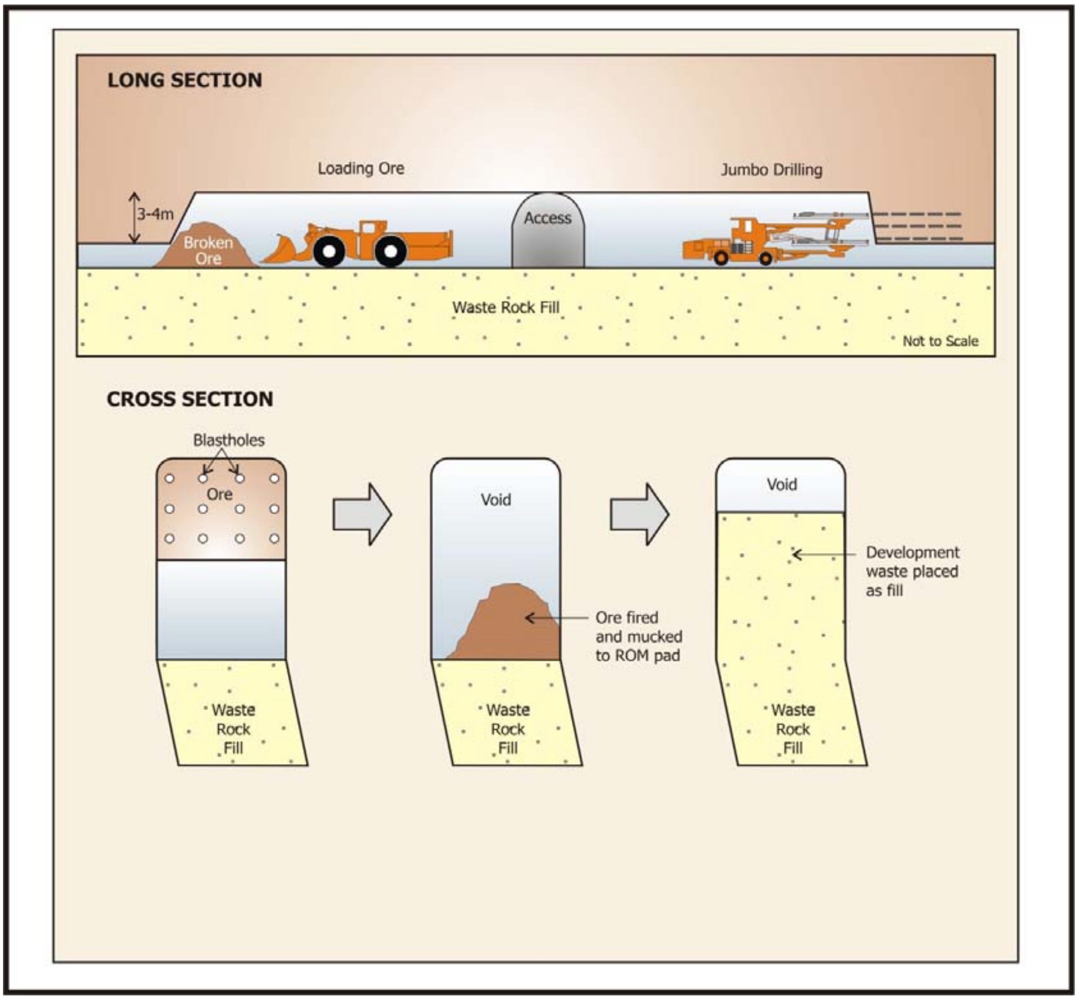



**Figure 5-4: Extraction Sequence Long Section**

Ventilation of stopes will be by positive pressure ventilation fans exhausting by flexible plastic ducting back to the main decline and eventually through the return ventilation raises.

### 5.3.3 Crown Pillar Recovery

As each block is mined up underneath the level above a 10m crown pillar will remain in place eventually being recovered towards the end of mine life. The final pillar thickness will need to be determined through geotechnical investigation. This pillar will be recovered by mining in an uphole stoping method retreating towards the access. The ground conditions and the presence of unconsolidated fill above will not allow 100% extraction of this pillar therefore Mining One has assumed that only 70% will be recovered.

### 5.3.4 Production Rates

All development and stoping will be completed using a twin boom jumbo and diesel loader. Maximum development rates have been estimated at 270 metres / month and maximum stoping / flat backing rates at 300 metres / month. At a rate of 80 tonnes/metre and 65 tonnes/ metre respectively this equates to 20,000 tonnes / month for both methods.

Mining One has calculated the average tonnes / metre of the three deposits to be 8700 t/ m and with an industry accepted advance rate of 70 vertical metres / year have determined the overall mine production rate to be 600,000 tpa.

### 5.3.5 Backfill

Cut and fill mining relies on backfill to offer wall support as well as providing a working platform for the lift above. In the majority of cases unconsolidated fill in the form of development waste can be used as backfill. However in wider parts of the orebody where it is economical to aim for 100% extraction a consolidated fill is required to act as an artificial pillar.

#### 5.3.5.1 Waste Backfill

Development waste or waste mined in the open pit can be used as unconsolidated fill in stopes. This will be either trucked directly from development headings or back loaded from the surface as required. For the purpose of creating a cost model Mining One has assumed that all backfill required will be back loaded from the surface mullock dump. Waste will be dumped in stockpiles close to the stope and trammed in with diesel powered loaders.

#### 5.3.5.2 Consolidated Backfill

Mining One has recommended that cross cut panels in the ore be backfilled with a consolidated fill to act as artificial pillars thereby allowing extraction of ore pillars within the stope. This fill can be either a cemented pastefill or rock fill. Mining One has allowed for a pastefill plant as part of the infrastructure and a pastefill operating cost in the financial model.

## 5.4 Mine Inventory

In order to use the majority of the resource in the scoping study Mining One has included inferred material in the mine design. This will exclude the reporting of a JORC compliant reserve and therefore Mining One will refer to the mine design tonnes and grade as the mine inventory.

### 5.4.1 Mine Design

The mine design has been based on the geological block model using a standard drive profile of 5m wide x 5m high. This has simplified the design process as the standard geological model block size is 10m x 10m x 10m. The actual lifts have been designed and reported as 10m high stopes in fitting with the model. Each 5m lift has then been reported as 50% of each design stope. The relationship between development size and geological block size is shown in Figure 5-5.

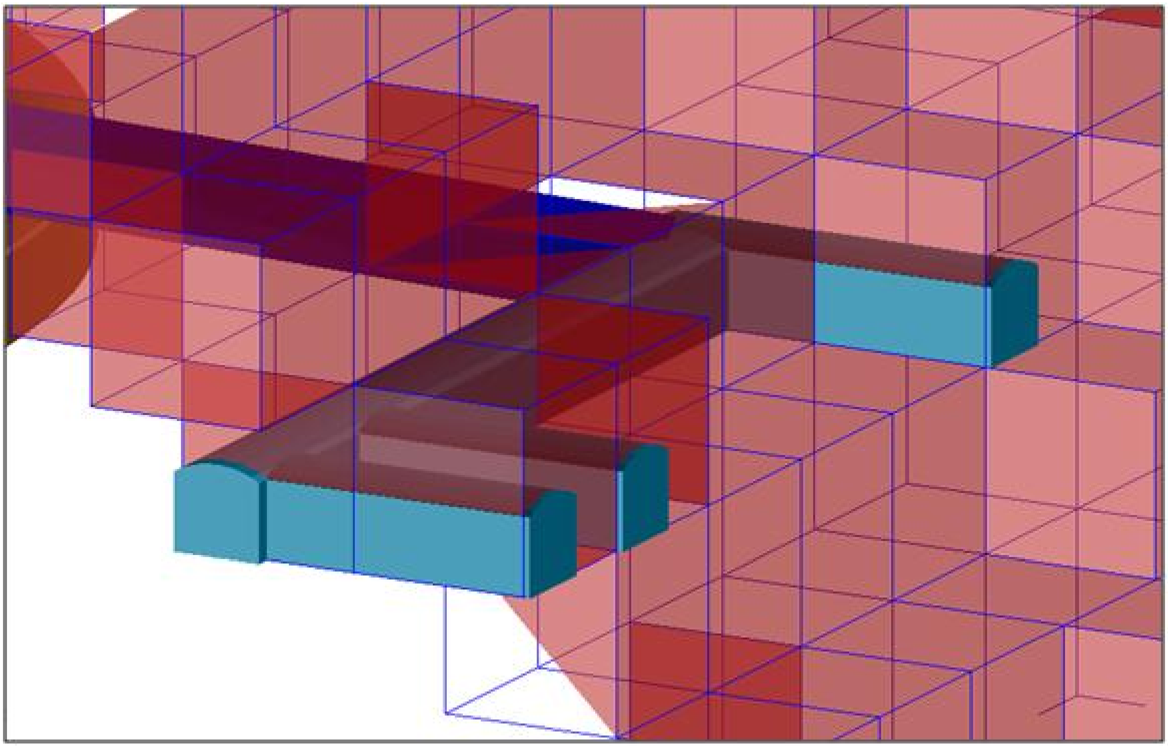


**Figure 5-5: Relationship Between Drive Profile and Block Size**

### 5.4.2 Recovery & Dilution

Mining One has assumed that the recovery of each cut & fill lift will be 90% with ore losses resulting from irregular wall contacts and ore mixing with waste fill, recovery of the crown pillar extraction will be 70% due to poor ground conditions and the presence of unconsolidated fill above.

Dilution for all extraction methods will be 10% at 0.3% Sn grade resulting from hangingwall overbreak and the bogging of unconsolidated waste fill off the stope floor.

**Table 5-1: Underground Mine Inventory**

| Orebody | Tonnes (kt) | Grade (%) |
|---|---|---|
| Queen Hill | 1,651 | 0.93 |
| Severn | 2,382 | 0.86 |
| Montana | 409 | 1.25 |
| **Total** | **4,442** | **0.93** |

## 5.5 Mine Development & Productivity

The mine will be developed by a series of declines independent to each orebody but accessed through a single portal. Mining One has investigated the possibility of accessing the Queen Hill and Severn orebodies from the same decline however the separation distance would have resulted in an excessive amount of access development.

The decline portal will commence at an elevation of 1190 RL at the northern end of the deposit and decline down at a gradient of 1:7. The decline will split at the 1175 RL with the eastern branch accessing the Severn and Montana orebodies while the western branch will access the Queen Hill orebody as shown in Figure 5-6. The decline will have a minimum standoff distance of 50m from the orebody to allow a minimum of 4 lifts from each access.

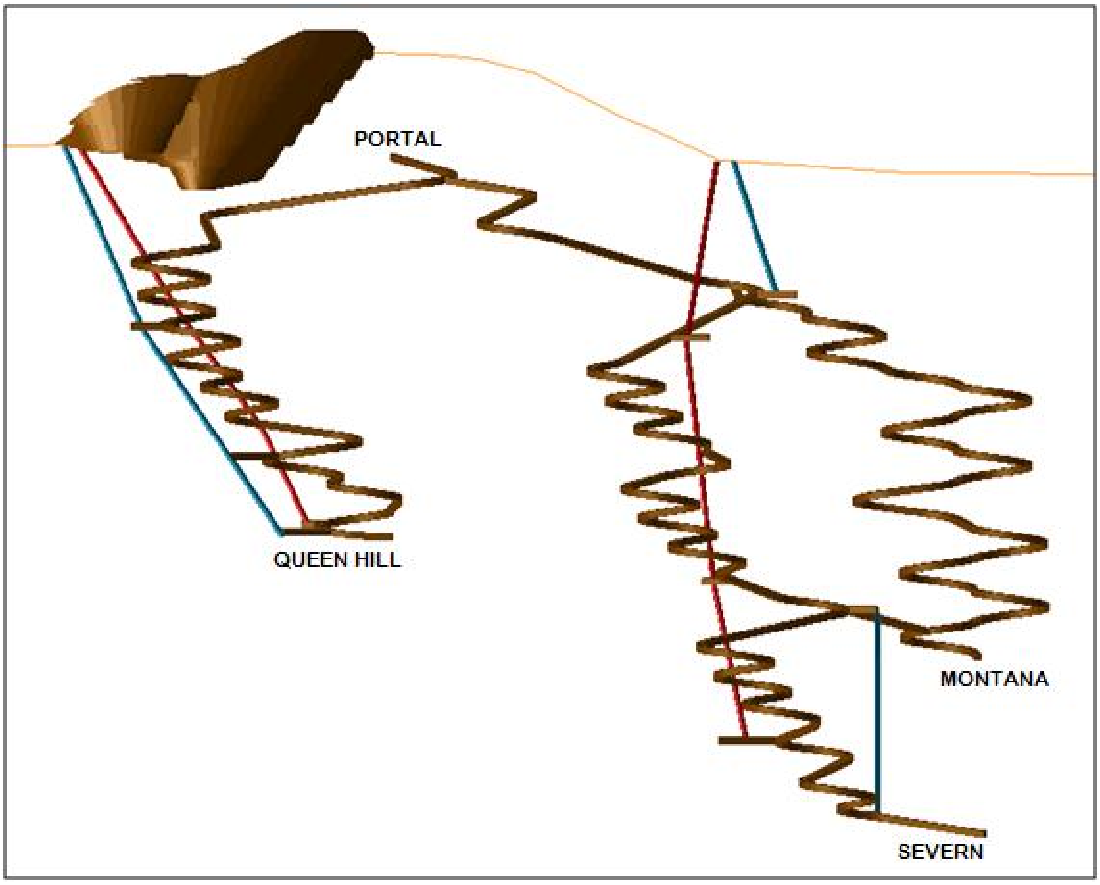



**Figure 5-6: Decline Design**

The mine will be developed using twin boom jumbos with a maximum productivity of 270 m / month / jumbo. This rate has assumed that new equipment will be used and a standard bolting and meshing pattern can be used for ground support. The declines have been designed at 5m wide x 5.5 m high to allow sufficient room for 55 tonne trucks to operate.

Ventilation rises of 4m diameter have been included in the design and these will be excavated by raise bore supplied and operated by the mining contractor.

## 5.6 Manning & Equipment

Mining One has assumed that all underground development and production will be completed by contractor and mine management and technical support will be supplied by the principal.

The contractor will be required to supply mine personnel, all mobile equipment and certain items of fixed plant. Mining one has recommended the following equipment and personnel to effectively run the mine at the required rate. Mine costs have been development according to these recommendations.

### 5.6.1 Underground Equipment

Mining One have recommended drive sizes of 5.0mW x 5.0mH in the ore drives and access development and 5.0mW x 5.5mH in the decline development. These drive sizes enable suitably sized equipment to fit in the development which will provide the desired production rates for the ore body. The following equipment was considered in selecting the drive sizes.

#### 5.6.1.1 Drilling Jumbo

Two boom jumbos will be used for both development and production. A range of equipment would be suitable for this application and three models are shown in Table 5-2 along with profile restrictions.

**Table 5-2: Drilling Jumbo Profile Capacities**

| Equipment | Max Drive Width (m) | Max Drive Height (m) |
|---|---|---|
| Tamrock DD320 | 8.7 | 5.9 |
| Tamrock DD420 | 10 | 6.4 |
| Atlas Copco M2C | 9 | 6.45 |

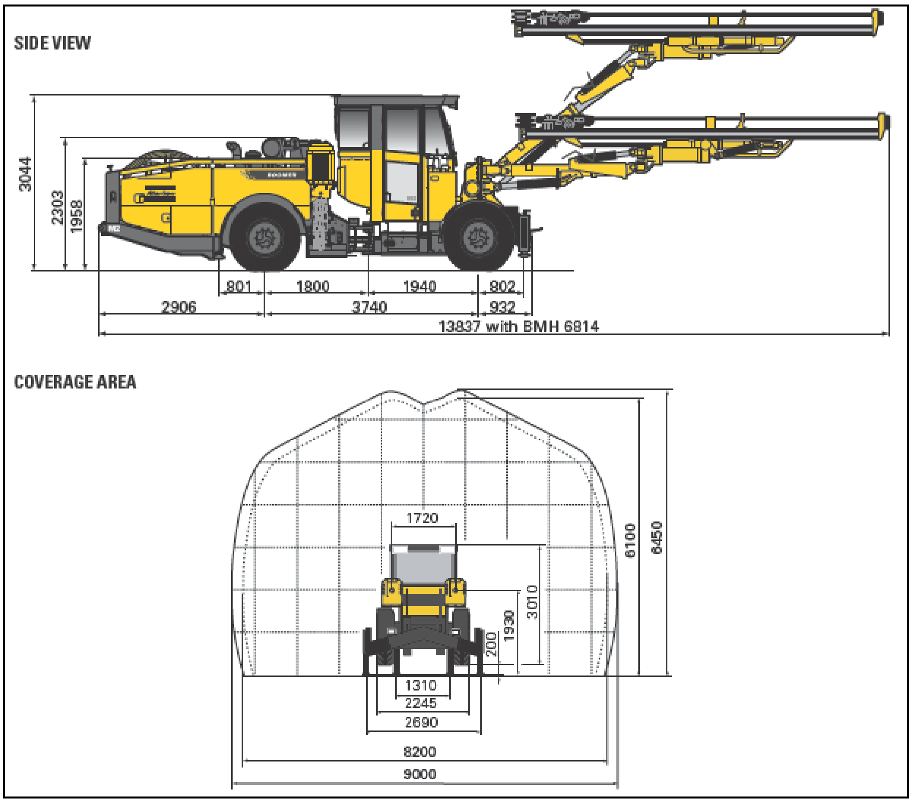


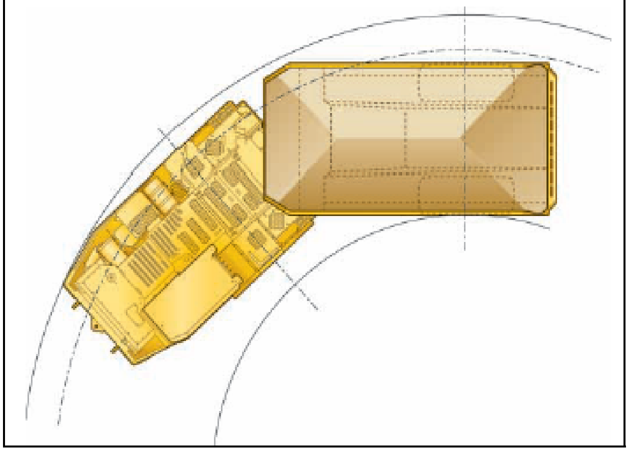

**Figure 5-7: Details of Atlas Copco Boomer M2C**

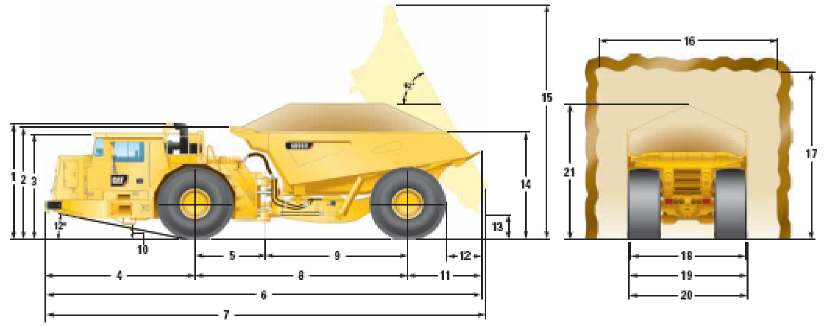
#### 5.6.1.2 Haul Trucks

Haulage will be via a diesel powered low profile underground mining truck of 55 tonne capacity. Mining One has based the mine design and productivities on the Caterpillar AD55B truck.

**Turning Dimensions**

| | | |
|---|---|---|
| Outside Clearance Radius | 10 005 mm | 393.9 in |
| Inside Turning Radius | 5540 mm | 218.1 in |
| Frame Oscillation | 10° | |
| Articulation Angle | 42.5° | |






| | 313-4730 (Standard Body) | 312-1395 | 307-6655 | 311-4721 |
|---|---|---|---|---|
| Dump Body | 26.9 m³ (35.2 yd³) | 32.6 m³ (42.6 yd³) | 33.8 m³ (44.2 yd³) | 36.6 m³ (47.9 yd³) |
| 1 Overall Height – Body Empty | 3278 mm (10'9") | 3418 mm (11'3") | 3557 mm (11'8") | 3560 mm (11'8") |
| 2 Height to Top of Body | 3202 mm (10'6") | 3418 mm (11'3") | 3556 mm (11'8") | 3560 mm (11'8") |
| 3 Height to Top of ROPS | 3000 mm (9'10") | 3000 mm (9'10") | 3000 mm (9'10") | 3000 mm (9'10") |
| 4 Front Bumper to Centerline of Front Axle | 4164 mm (13'8") | 4164 mm (13'8") | 4164 mm (13'8") | 4164 mm (13'8") |
| 5 Centerline of Front Axle to Centerline of Hitch | 1920 mm (6'4") | 1920 mm (6'4") | 1920 mm (6'4") | 1920 mm (6'4") |
| 6 Overall Length | 12 040 mm (39'6") | 12 040 mm (39'6") | 12 040 mm (39'6") | 12 040 mm (39'6") |
| 7 Max Overall Length | 12 064 mm (39'7") | 12 180 mm (39'11") | 12 222 mm (40'1") | 12 241 mm (40'2") |
| 8 Wheelbase | 5900 mm (19'4") | 5900 mm (19'4") | 5900 mm (19'4") | 5900 mm (19'4") |
| 9 Centerline of Hitch to Centerline of Rear Axle | 3980 mm (13'1") | 3980 mm (13'1") | 3980 mm (13'1") | 3980 mm (13'1") |
| 10 Ground Clearance | 393 mm (1'3") | 393 mm (1'3") | 393 mm (1'3") | 393 mm (1'3") |
| 11 Centerline of Rear Axle to Tail | 1976 mm (6'6") | 1976 mm (6'6") | 1976 mm (6'6") | 1976 mm (6'6") |
| 12 Rear Wheel to Body Raised | 857 mm (2'10") | 857 mm (2'10") | 857 mm (2'10") | 857 mm (2'10") |
| 13 Dump Clearance | 514 mm (1'8") | 514 mm (1'8") | 514 mm (1'8") | 514 mm (1'8") |
| 14 Loading Height | 3045 mm (10'0") | 3326 mm (10'11") | 3426 mm (11'3") | 3473 mm (11'5") |
| 15 Overall Height – Body Raised | 6969 mm (22'10") | 7202 mm (23'8") | 7322 mm (24'0") | 7334 mm (24'1") |
| 16 Tunnel Clearance Width* | 5000 mm (16'4") | 5000 mm (16'4") | 5000 mm (16'4") | 5000 mm (16'4") |
| 17 Tunnel Clearance Height* | 5000 mm (16'4") | 5000 mm (16'4") | 5000 mm (16'4") | 5000 mm (16'4") |
| 18 Overall Tire Width | 3250 mm (10'8") | 3250 mm (10'8") | 3250 mm (10'8") | 3250 mm (10'8") |
| 19 Overall Width (Including Body) | 3346 mm (11'0") | 3480 mm (11'5") | 3480 mm (11'5") | 3480 mm (11'5") |
| 20 Overall Width (Excluding Body) | 3346 mm (11'0") | 3346 mm (11'0") | 3346 mm (11'0") | 3346 mm (11'0") |
| 21 Height to Top of Load (SAE 2:1) | 3848 mm (12'7") | 4178 mm (13'8") | 4183 mm (13'9") | 4333 mm (14'3") |

**Figure 5-8: Details of Caterpillar AD55B Haul Truck**

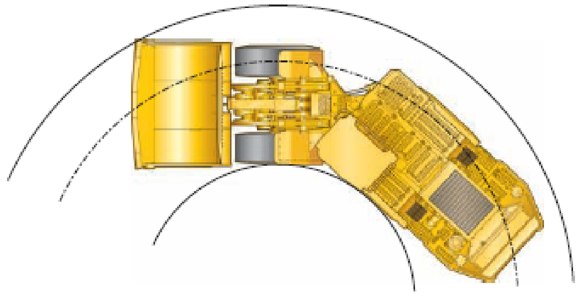
### 5.6.1.3 Load Haul Dump (LHD)

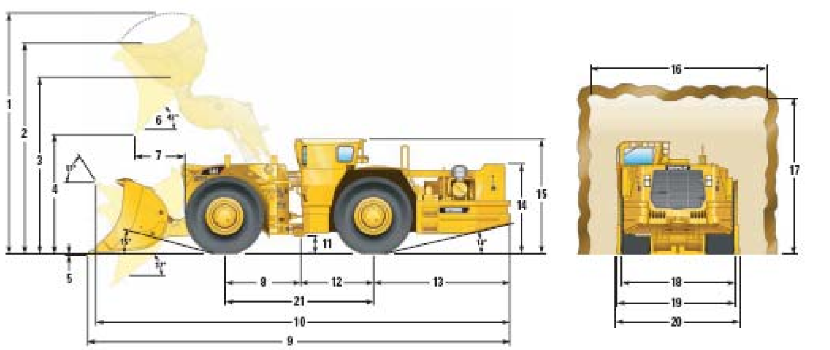
Mining One has based the productivities and design on using one type of loader for all underground activities. The Caterpillar R2900G with a standard bucket capacity of 7.2 m$^3$ has been selected.

## Turning Dimensions

| | | |
|---|---|---|
| Outside Clearance Radius** | 7323 mm | 288.3 in |
| Inner Clearance Radius** | 3383 mm | 133.2 in |
| Axle Oscillation | 8° | |
| Articulation Angle | 42.5° | |

** Clearance dimensions are for reference only.






| | | 249-4899 (Standard Bucket)* | | 249-4892* | | 249-4893* | |
|---|---|---|---|---|---|---|---|
| | Bucket capacity | 7.2 $m^3$ | (9.4 $yd^3$) | 8.3 $m^3$ | (10.9 $yd^3$) | 8.9 $m^3$ | (11.6 $yd^3$) |
| | Bucket width over cutting edge | 3054 mm | (10 ft) | 3154 mm | (10 ft 4 in) | 3354 mm | (11 ft) |
| 1 | Overall height – bucket raised | 6179 mm | (20 ft 3 in) | 6232 mm | (20 ft 6 in) | 6232 mm | (20 ft 6 in) |
| 2 | Maximum dump height | 5427 mm | (17 ft 10 in) | 5427 mm | (17 ft 10 in) | 5427 mm | (17 ft 10 in) |
| 3 | Bucket pin height at maximum lift | 4539 mm | (14 ft 11 in) | 4539 mm | (14 ft 11 in) | 4539 mm | (14 ft 11 in) |
| 4 | Dump clearance at maximum lift | 2868 mm | (9 ft 5 in) | 2735 mm | (9 ft) | 2724 mm | (8 ft 11 in) |
| 5 | Digging depth | 52 mm | (2 in) | 65 mm | (2.6 in) | 65 mm | (2.6 in) |
| 6 | Dump angle at maximum lift | 42° | | 42° | | 42° | |
| 7 | Reach | 1656 mm | (5 ft 5 in) | 1803 mm | (5 ft 11 in) | 1817 mm | (6 ft) |
| 8 | Centerline of front axle to centerline of hitch | 1890 mm | (6 ft 2 in) | 1890 mm | (6 ft 2 in) | 1890 mm | (6 ft 2 in) |
| 9 | Overall length (digging) | 11 302 mm | (37 ft 1 in) | 11 507 mm | (37 ft 9 in) | 11 525 mm | (37 ft 10 in) |
| 10 | Overall length (tramming) | 10 949 mm | (35 ft 11 in) | 11 073 mm | (36 ft 4 in) | 11 083 mm | (36 ft 4 in) |
| 11 | Ground clearance | 465 mm | (1 ft 6 in) | 465 mm | (1 ft 6 in) | 465 mm | (1 ft 6 in) |
| 12 | Centerline of back axle to centerline of hitch | 1890 mm | (6 ft 2 in) | 1890 mm | (6 ft 2 in) | 1890 mm | (6 ft 2 in) |
| 13 | Length – rear axle to bumper | 3572 mm | (11 ft 9 in) | 3572 mm | (11 ft 9 in) | 3572 mm | (11 ft 9 in) |
| 14 | Height to top of hood | 2371 mm | (7 ft 9 in) | 2371 mm | (7 ft 9 in) | 2371 mm | (7 ft 9 in) |
| 15 | Height to top of ROPS | 2886 mm | (9 ft 6 in) | 2886 mm | (9 ft 6 in) | 2886 mm | (9 ft 6 in) |
| 16 | Tunnel clearance width** | 4500 mm | (14 ft 9 in) | 4500 mm | (14 ft 9 in) | 4500 mm | (14 ft 9 in) |
| 17 | Tunnel clearance height** | 4500 mm | (14 ft 9 in) | 4500 mm | (14 ft 9 in) | 4500 mm | (14 ft 9 in) |
| 18 | Overall tire width | 2898 mm | (9 ft 6 in) | 2898 mm | (9 ft 6 in) | 2898 mm | (9 ft 6 in) |
| 19 | Overall width excluding bucket | 3010 mm | (9 ft 10 in) | 3010 mm | (9 ft 10 in) | 3010 mm | (9 ft 10 in) |
| 20 | Overall width including bucket | 3176 mm | (10 ft 5 in) | 3176 mm | (10 ft 5 in) | 3176 mm | (10 ft 5 in) |
| 21 | Wheelbase | 3780 mm | (12 ft 5 in) | 3780 mm | (12 ft 5 in) | 3780 mm | (12 ft 5 in) |

**Figure 5-9: Details of Caterpillar R2900G Load Haul Dump (LHD)**

#### 5.6.1.4 Service Vehicles

The mine will require service vehicles for general servicing and charging. It is envisaged that two service vehicles would be included in the underground mining fleet. Appropriate service and charge up baskets would be attached to a Volvo L90 / L120 "type" vehicle or Caterpillar IT. A grader will also be required to maintain underground road ways.

#### 5.6.1.5 Light Vehicles

Six 4WD underground light vehicles for staff and supervisor transport have been included in the ventilation design for the mine. It is assumed that the contractor will supply their own light vehicles however an additional 4 vehicles have been included in the cost model for use by mine management and technical staff.

#### 5.6.1.6 Equipment Summary

Table 5-3 shows a summary of the equipment recommended for underground development and production. The quantity of each piece of equipment has been calculated from the mine schedule. All costs and schedules have been based on new equipment of this type and size.

**Table 5-3: Equipment Summary**

| Description | Typical Supplier | Quantity |
|---|---|---|
| Twin Boom Jumbo | Atlas Copco or Sandvik | 3 |
| Load Haul Dump (LHD) (R2900G) | Caterpillar | 4 |
| Truck (AD55B) | Caterpillar | 3 |
| Service Vehicle (IT) | Caterpillar or Volvo | 2 |
| Grader | Caterpillar | 1 |
| 4WD Light Vehicles | Toyota | 6 |

### 5.6.2 Manning

Mine management, planning and technical support will be supplied by the principal while the mine will be operated by a contractor and the following manning figures are estimates with the final contractor numbers to be determined once the preferred contractor has been selected. It is envisaged that the successful underground mining contractor would source a local workforce and no provisions have been made for the cost of flights or accommodation.

**Table 5-4: Mining Personnel**

| Category | Day shift | Typical Shift Crew (x4 crews) | Total |
|---|---|---|---|
| **Underground Mine Principal Staff** | | | |
| General Manager / Mine Manager | 1 | | 1 |
| Admin Assistant | 1 | | 1 |
| Mining Superintendent | 1 | | 1 |
| Mining Engineer | 1 | | 1 |
| Senior Geologist | 1 | | 1 |
| Geologist | 3 | | 3 |
| Senior Surveyor | 1 | | 1 |
| Survey Assistant | 1 | | 1 |
| **Total Principal – U'G Mine Personnel** | | | **10** |
| **CONTRACTOR PERSONNEL** | | | |
| **Management** | | | |
| Project Manager | 1 | | 1 |
| Project Engineer | 1 | | 1 |
| Safety and Training Officer | 1 | | 1 |
| Mine Foreman | 1 | | 1 |
| Maintenance Foreman | 1 | | 1 |
| Maintenance Scheduler | 1 | | 1 |
| Mine Clerk | 1 | | 1 |
| Shift supervisor | 4 | | 4 |
| **Total - Contractor Management** | | | **11** |
| **UG Production** | | | |
| Jumbo Operators | 3 | 4 | 12 |
| Loader Operators | 4 | 4 | 16 |
| Truck Drivers | 3 | 4 | 12 |
| Jumbo Offsider / Charge up | 3 | 4 | 12 |
| Service Crew | 3 | 4 | 12 |
| **Total - UG Production** | | | **64** |

| Category | Day shift | Typical Shift Crew (x4 crews) | Total |
|---|---|---|---|
| **UG Maintenance and Support** | | | |
| Leading Hand Fitter | 1 | 4 | 4 |
| Workshop Fitter | 1 | 4 | 4 |
| Electrician | 1 | 4 | 4 |
| Jumbo Fitter | 2 | 4 | 8 |
| **Total - Maintenance** | | | **20** |
| **GRAND TOTAL - MINE PERSONNEL** | | | **105** |

## 5.7 Mine Ventilation

### 5.7.1 Ambient Temperature

The project area is within a temperate climate area with mild summer and cold, wet winters, with the average rainfall being 2440.5 mm. Mean maximum temperatures are around 19°C for summer and 11°C in winter with minimums of around 9°C and 3.5°C for summer and winter respectively.

### 5.7.2 Mine Development and Stoping Considerations

The mine is to be accessed via a 5.0mW x 5.5mH decline (arched profile) with all level access and ore drive and waste development being 5.0mW x 5.0mH. The decline surfaces will be blasted rock supported with bolts and mesh and mine ore production is planned to be around 600,000 tonnes per annum hauled up the three declines. The mine will require compliance with ventilation standards, including dust suppression and occupational health and safety programs.

### 5.7.3 Mining Equipment and Ventilation Requirements

The list of rubber tyred diesel powered mining equipment that might be expected at the underground operation is set out in the table below:

**Table 5-5: Diesel Powered Equipment Fleet and Operating Power**

| Unit Description | No. of Units Onsite | Engine Power (kW) | Max. Units Operating Together | Operating Engine Power (kW) |
|---|---|---|---|---|
| Twin Boom Jumbo | 3 | 74 | - | - |
| LHD (R2900G) | 4 | 321 | 2 | 642 |
| Truck (AD55B) | 3 | 600 | 1 | 1,200 |
| Grader | 1 | 108 | - | - |

| | | | | |
|---|---|---|---|---|
| Service Vehicle (IT38H) | 2 | 147 | 1 | 47 |
| Light Vehicles | 6 | 96 | 4 | 384 |
| **Total** | **19** | **4,284** | **9** | **2,373** |

Mining One has considered that the maximum number of units working at any one time within the mine is as shown in Table 5-5. The total installed engine power is 4,284 kW which is used as a basis for primary ventilation network design. Also considered is the maximum operating engine power that may be within a single decline which is shown in Table 5-6.

**Table 5-6: Typical Operating Engine Power per Decline**

| Typical Unit (1) From Fleet | Engine Power kW | Maximum Units Operating Together | Operating Engine Power kW |
|---|---|---|---|
| Twin Boom Jumbo | 74 | - | - |
| LHD (R2900G) | 321 | 2 | 642 |
| Truck (AD55B) | 600 | 2 | 1,200 |
| Grader | 108 | 1 | - |
| Service Vehicle (IT38H) | 147 | 1 | 47 |
| Light Vehicles | 96 | 4 | 384 |
| **Total** | | | 2,481 |

Note: Units specified are typical units that are likely to be used. Actual units have not yet been selected.

### 5.7.4 Primary Ventilation System Design Criteria (Total Mine)

The primary ventilation required has been selected based on the total underground engine power at a 75% operating factor.

- Total installed engine power — 4,284 kW
- Typical Operating factor — 75%
- Typical Operating engine power — 3,213 kW
- Ventilation Capacity Required [as per Section 4.3.1 (a)] — 0.06 $m^3$/sec/kW
- Ventilation Required for Installed Power — 257 $m^3$/sec
- Ventilation Required for Operating Power — 193 $m^3$/sec
- Minimum Ventilation Required for a Workplace — 19 $m^3$/sec

  (Workplace requirement based on 1 LHD per workplace)

### 5.7.5 Primary Ventilation System Design Criteria (Per Decline)

The primary ventilation required has been selected based on the underground engine power likely to be in a single decline area at a 75% operating factor, which is close to the calculated operating engine power on Table 5-6.

| | |
|---|---|
| ➢ Total installed engine power | 2,481 kW |
| ➢ Typical Operating factor | 75% |
| ➢ Typical Operating engine power | 1,861 kW |
| ➢ Ventilation Capacity Required [as per Section 4.3.1 (a)] | 0.06 $m^3$/sec/kW |
| ➢ Ventilation Required for Installed Power | 149 $m^3$/sec |
| ➢ Ventilation Required for Operating Power | 112 $m^3$/sec |
| ➢ Minimum Ventilation Required for a Workplace | 19 $m^3$/sec |
| (Workplace requirement based on 1 LHD per workplace) | |

### 5.7.6 Primary Ventilation Circuit

The primary ventilation circuit assumes the following:

- Intake airflow is via the decline portal, Queen Hill intake shaft and Montana intake shaft;
- Exhaust airflow is via two x 4m diameter return air rises to surface
- Each intake air raise will double as an emergency egress and will be equipped with an escapeway;
- Barricades and restrictions will be required at specific locations from time to time and these will be planked timber wall stoppings or similar;
- Stopes and development headings will be ventilated by secondary fan and ducting systems as required;
- Total ventilation capacity required for each decline is designed to be between 110 $m^3$/sec and 130 $m^3$/sec to allow for leakage and workplace flows of 19 $m^3$/sec; and
- Primary ventilation fans designed to be installed on surface over the return air rises.

A simulation of the primary ventilation circuit at full mine development using VentSim Version 3.9.1 to assess mine flow distribution has been undertaken. The aim was to assess airflow volumes within the network and to assist in finding appropriate fans.

The analysis was carried out assuming raise bore rises of 4.0m diameter circular profile as vertical intake and return airways, combined with 5.0mW x 5.5mH development profiles.

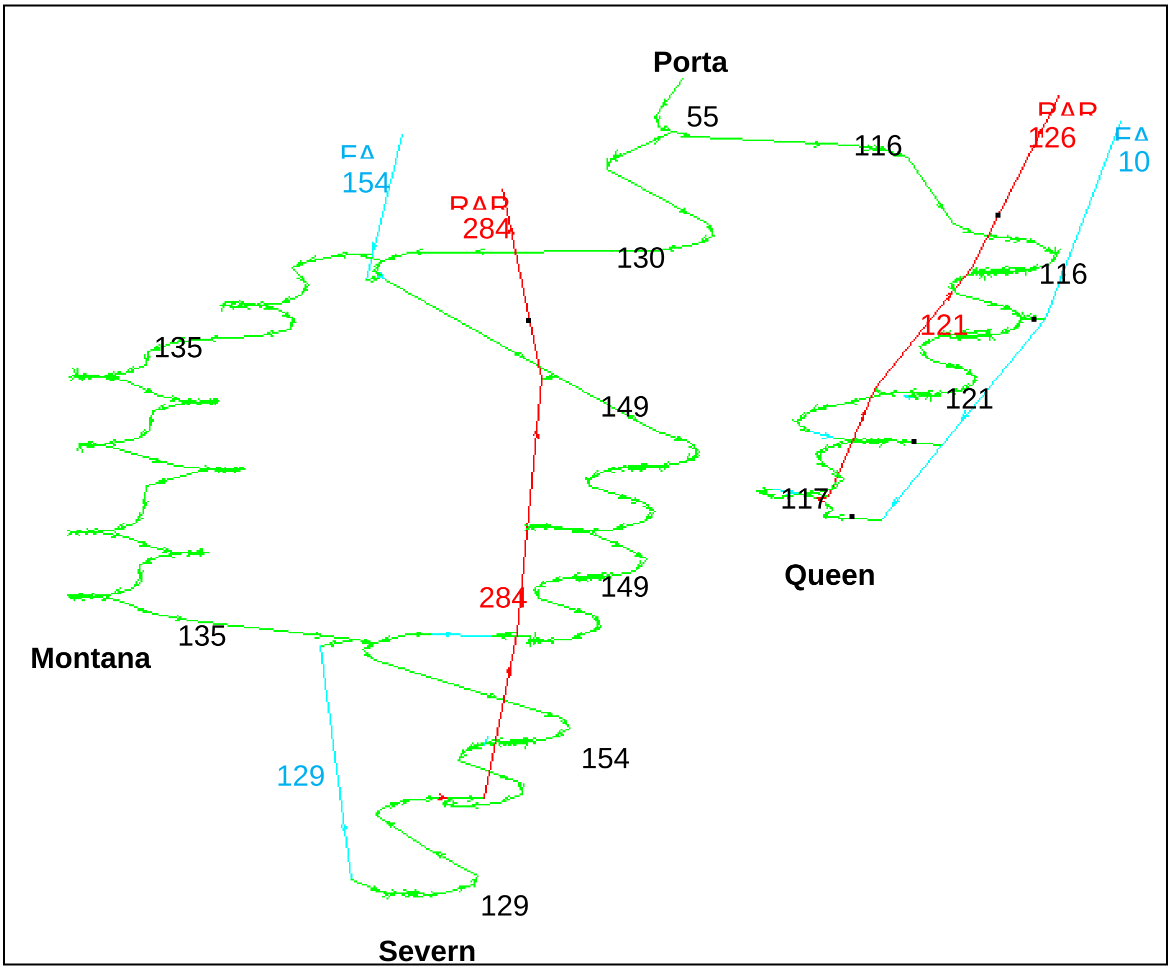



**Figure 5-10: Ventilation Network (Airflow in m$^3$ / sec)**

Summaries of the calculated airflow velocity and quantity throughout the network are shown in Table 5-7.

**Table 5-7: Air Velocities and Quantities**

| Location | Air Velocity (m/s) | Airflow (m$^3$/s) |
|---|---|---|
| Queen Hill Decline upper section | 4.2 | 116 |
| Queen Hill Decline middle section | 4.4 | 121 |
| Queen Hill Decline lower section | 4.2 | 117 |
| Queen Hill RAR | 10 | 126 |
| Severn Decline upper section | 5.4 | 149 |
| Severn Decline middle section | 5.6 | 154 |

| | | |
|---|---|---|
| Severn Decline lower section | 4.7 | 130 |
| Severn RAR (also RAR for Montana) | 22.6 | 284 |
| Montana Decline | 4.9 | 135 |
| Montana FAR | 12.2 | 154 |

### 5.7.7 Primary Fan Selection

Fan selection for the primary ventilation circuit has been determined by inputting known fan curves into VentSim and running a simulation. This has not been an optimized ventilation simulation, but has been completed to give an indication of what will be required and what equipment is available to ventilate the mine.

To achieve the required flow from the Severn RAR, a 400kW centrifugal fan would be required. The fan curve used in the simulation was for a Richardson CY2430 fan. Using this fan gives an air flow of greater than 130 $m^3/s$ down each of the declines.

An 180kW fan was used for the simulation of the Queen Hill decline ventilation circuit achieving an airflow of 126 $m^3/s$.

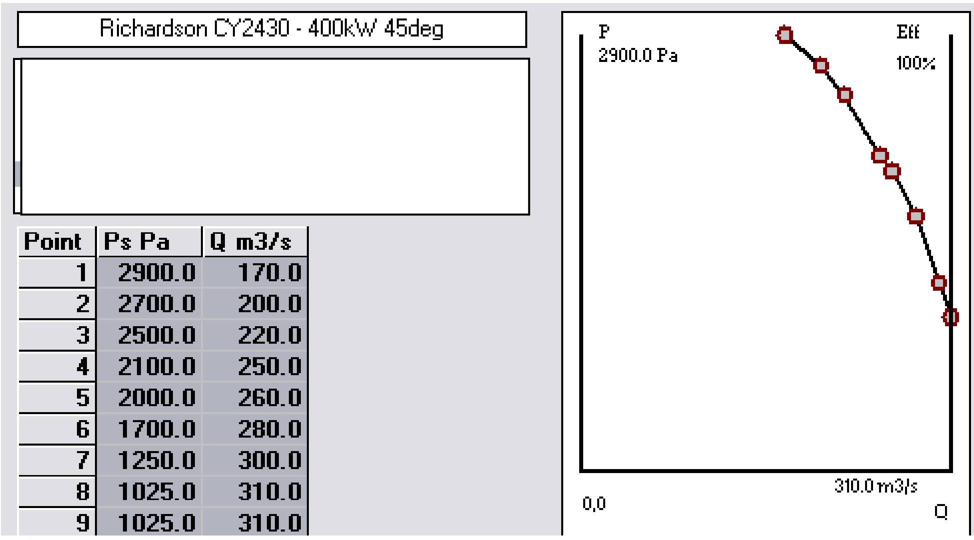



**Figure 5-11: Fan Curve for the 400kW Fan used to ventilate the Severn Return Air Rise (RAR)**

Surface Fan (180kW)

| Point | Ps Pa | Pt Pa | Q m3/s | Efc% |
|---|---|---|---|---|
| 1 | 2100.0 | 2131.0 | 45.0 | 40 |
| 2 | 2000.0 | 2042.0 | 52.0 | 45 |
| 3 | 1900.0 | 1956.0 | 60.0 | 50 |
| 4 | 1750.0 | 1826.0 | 70.0 | 60 |
| 5 | 1550.0 | 1649.0 | 80.0 | 68 |
| 6 | 1325.0 | 1451.0 | 90.0 | 74 |
| 7 | 1100.0 | 1256.0 | 100.0 | 71 |
| 8 | 850.0 | 1038.0 | 110.0 | 65 |
| 9 | 550.0 | 774.0 | 120.0 | 60 |

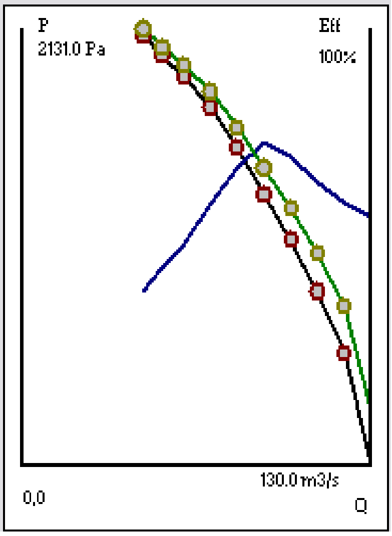



**Figure 5-12: Fan Curve for the 180kW Fan used to ventilate the Queen Hill Return Air Rise (RAR)**

### 5.7.8 Ancillary Fans

Auxiliary fans will be required to ventilate all stoping areas and development headings in the mine; these fans will be located in the decline and will supply ventilation to the working faces. The working areas will generally be restricted to loaders, and trucks will be loaded in the main decline the majority of the time. This reduces the required secondary ventilation at working faces. All secondary ventilation is based on single and twin stage 55 kW fans or similar.

## 5.8 Mine Services

### 5.8.1 Power

Power will be established at the mill and an 11kv feeder will run from the mill to the mine site substation which will transform the 11kv to 1000v through a 2.5 MVA transformer for reticulation underground. As the mine gets deeper the substation will be relocated underground with the 11kv either reticulated down a ventilation rise or service hole.

### 5.8.2 Compressed Air

Compressed air will be required throughout the mine for charge up, fibrecreting and pneumatic hand tools. Mining One has assumed a 132kw compressor delivering 400 l/sec at 7 Bar will be adequate to power all pneumatic equipment and have included costs accordingly. Compressed air will be reticulated throughout the mine through 100mm polyethylene pipe.

### 5.8.3 Mine Dewatering

There is no hydrology information regarding water flows underground therefore Mining One has not calculated pump requirements. It is however assumed that flow will be significant and therefore dewatering will need to be analysed in detail in later studies. For the scoping study it is assumed that each declined heading (including accesses) will require a 20 kW electric flygt pump which will feed to a series of 45kw mono pumps. These pumps will pump water out of the mine to a settling pond on the surface where the water can be decanted for cleaning and reused in the mine.

### 5.8.4 Water

Raw water can be sourced from recycled mine water as discussed in section 5.8.3. Supplies of potable water have not been investigated but will be required for drinking purposes.

## 5.9 Mine Schedule

Mining One has created a mine schedule in Microsoft Excel and has assumed that underground mine development will commence at the same time as the open pit. This can be achieved as the portal and underground mine can be developed independently to the open pit. As can be seen from Table 5-8 this will result in maximum production being achieved in year 3.

The schedule shows underground development completed by year 5 approximately 5 years prior to the end of mine life. This development could be smoothed out in a more sophisticated scheduling software package such as Enhanced Production Scheduler (EPS) and it is recommended that this be the case in later studies.

The underground mine schedule has been based on the following assumptions.

- Maximum single heading development rate of 100m / month
- Maximum jumbo development rate of 270m / month @ 80t/metre
- Maximum jumbo flatback (production) rate of 300m / month @ 65t/metre
- Maximum single level production rate of 20,000t / month

Table 5-8 shows the yearly physicals for mine development production and backfilling. The cumulative waste balance takes into account waste produced from the open pit and the underground and that which is back loaded to the mine for backfill. The figure shown in red of 3,414 Kt is the maximum tonnes to be placed on the waste dump.

**Table 5-8: Mine Schedule**

| Description | Year -1 | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Open Pit Waste (Kt) | | 2,978 | | | | | | | | | **2,978** |
| Open Pit Ore (Kt) | | 172 | | | | | | | | | **172** |
| Pit Ore Grades (%) | | 0.96 | | | | | | | | | **0.96** |
| U/G Development (m) | 2,730 | 5,758 | 2,121 | 2,293 | 1,191 | | | | | | **14,093** |
| U/G Flatback (m) | 0 | 2,018 | 8,967 | 9,448 | 9,459 | 10,010 | 9,760 | 9,607 | 6,262 | 3,155 | **68,686** |
| U/G Ore Production (Kt) | 30 | 230 | 579 | 601 | 601 | 601 | 600 | 599 | 400 | 200 | **4,442** |
| U/G Ore Grades (%) | 1.16 | 1.00 | 0.97 | 0.99 | 0.97 | 0.95 | 0.88 | 0.81 | 0.88 | 0.86 | **0.93** |
| U/G Waste (Kt) | 171 | 320 | 116 | 134 | 55 | | | | | | **796** |
| Waste Backfill Required (Kt) | 0 | 55 | 251 | 266 | 265 | 280 | 280 | 280 | 187 | 94 | **1,957** |
| Cumulative Waste Balance (Kt) | 3,149 | **3,414** | 3,279 | 3,147 | 2,937 | 2,657 | 2,377 | 2,097 | 1,910 | 1,816 | |
| Pastefill Required ($m^3$) | 0 | 11,751 | 53,734 | 57,090 | 56,706 | 60,058 | 60,029 | 59,940 | 40,023 | 20,055 | **419,385** |
| Total Ore (Kt) | **30** | **402** | **579** | **601** | **601** | **600** | **600** | **599** | **400** | **200** | **4,613** |
| Ore Grades (%) | **1.16** | **0.98** | **0.97** | **0.99** | **0.97** | **0.95** | **0.88** | **0.81** | **0.88** | **0.86** | **0.93** |

# 6 PROCESSING

## 6.1 Introduction

Ore characterisation and metallurgical test work on drill core from the Queen Hill deposit carried out by Aberfoyle for a 1983 pre–feasibility study report, and preliminary results of test work on core from the recent drilling program have been used to develop a process flow sheet, Costs for a plant incorporating the unit processes included in that process flow sheet has been determined using standard engineering procedures.

The estimate of CAPEX and OPEX is at scoping study level of accuracy for a conventional mineral processing plant using parameters of 600,000 TPA @ 1.1 % Sn. tin recovery 70 %, and final concentrate grade of 50% Sn.

## 6.2 Preamble

*Its important to put in context Aberfoyle`s presence and activities in Tasmania and the mainland in the 70`s and 80`s compared to the current situation as this will have a bearing on determining the best way forward for the Queen Hill prospect.*

The exploration effort by Aberfoyle in the 1970`s & 80`s on the West Coast of Tasmania was a natural extension to their other activities in the State where they operated the Cleveland Tin Mine at Luina, a process plant at Rossarden treating tin – tungsten ores from Storeys Creek and Aberfoyle mines on the East Coast, and mining of copper-lead-zinc ore from Que River Mine for processing by E Z Co Ltd (Pasminco) at Rosebery. Aberfoyle also owned the Ardlethan Tin Mine near Wagga in NSW and were the second largest tin producer after Renison in Australia at that time.

The Queen Hill discovery caused some disappointment when characterisation of ore from the upper lode showed it to comprise sulphides, (mainly pyrite), carbonates, fluorite and silicates. The tin mineral was mainly cassiterite, but occurred in extremely fine particles (a median D 50 of 15 microns), disseminated throughout the ore, 60% in the sulphide and the remainder in the other gangue.

To liberate the cassiterite in Upper Queen Hill ore requires a very fine grind; consequently losses to sulphide flotation concentrate were very high due to mechanical entrainment and some un-liberated composites. The cassiterite in the remaining gangue (sulphide flotation tail) was too fine for gravity separation even at natural grain size, and fine grinding had reduced it further - the only route left was cassiterite flotation which requires considerable de-sliming of the feed, hence resulting in losses to slimes. Cassiterite flotation is not very selective even using the best collectors available, so that saleable concentrates could not be made. The selective tin flotation collectors are also very expensive.

Aberfoyle showed that high recoveries could be achieved by treatment of the whole ore by a pyro-metallurgical process, in which the cassiterite (tin oxide) converts to tin sulphide, which volatilises from the melt, and is converted back to tin oxide + SO2 by means of oxygen, then captured in a bag house as a high grade tin fume ( +60%Sn).The SO2 passes on and has to be dealt with in some way, e.g. stack to atmosphere, sulphuric acid plant, wet scrubbing, wet scrubbing and neutralisation Aberfoyle referred to this as ***Matte Fuming***.

At CSIRO Aberfoyle successfully demonstrated on a small scale that submerged lance smelting could be used to treat the Queen Hill Ore, and then moved on to pilot the process. A 4 tph unit was built at the Kalgoorlie Nickel smelter. This program in Kalgoorlie was technically very successful.

The potential for use of matte fuming at Cleveland Tin changed Aberfoyle focus, in that they had an operating mine that may have been able to use the technology. In addition the exploration program at Queen Hill had identified ore below Queen Hill and other adjacent ore bodies called Severn and Montana. Metallurgical characterisation test work on these showed them to be more amenable to conventional mineral dressing than the Upper Queen Hill ore. (Amenability was judged on cassiterite grain size and ease of liberation, and response to gravity and flotation separation). In particular the Severn ore responded better than some of the fine grained ores at Renison Bell Tin Mine when subjected to similar unit processes employed in the Renison Concentrator. (Severn had a D50 grain size at 65 microns some of the Renison Fault ores are 50 microns).This offered the option to process these ores by standard mineral dressing methods and produce a saleable gravity concentrate, a low grade tin concentrate to be further upgraded by fuming, and had the advantage of reducing fuming plant feed tonnage for a given output. It also offered an option to considerably reduce SO2 emission in that concentrates would be much lower in sulphur than the whole ore, in fact sulphur would be added to for the SnO2 to be converted to the volatile SnS –Aberfoyle referred to this variation as ***Slag Fuming*** as the process was employed at some world smelters to recover tin from primary slag in the tin smelting process. Other smelters such as Freiberg, East Germany, Novosibirsk, Russia and Capper Pass UK had used it for many years upgrade low grade concentrates. Capper Pass treated low grade tin concentrates from various parts of the world including Renison Tin and Cleveland Tin – as such was well proven technology.

Consequently Aberfoyle adopted a development strategy to carry out a feasibility study into the use of fuming within the Group, the location and whether to be matte or slag fuming would be part of the study. A major part of the study would be what to do with the SO2 generated. Adding on a sulphuric acid plant would be expensive and could not be economic on this small scale even if markets could be found. Discharge to atmosphere via a tall stack was not seen as an option for matte fuming, and wet scrubbing would create a weak acid for disposal, neutralisation of this with lime would be expensive, even discharge into sea water was considered. Location at Cleveland Tin had some advantages in that the tails were alkaline and could be used to aid neutralising of an acid effluent from the wet scrubber.

In regard to the Queen Hill ores three options were considered in November 1982 in a document prepared by Dr Simon Meik :-

- A flow sheet along the lines of Cleveland Tin and Renison Tin i.e. Staged crushing, Heavy Media Separation to remove light barren gangue as *floats*, primary grinding of HMS *sinks*, sulphide flotation with inter-stage regrind, hydraulic classification, of the sulphide tails, treatment of the coarse fraction by gravity (spirals and tables), with inter-stage regrind, treatment of the fine fraction by cassiterite flotation. This circuit would result in a saleable gravity concentrate + 50% Sn and a saleable flotation concentrate +25%Sn

- A flow sheet along the lines of the above - Staged crushing, Heavy Media Separation to remove light barren gangue as floats , primary grinding of HMS sinks , sulphide flotation with inter-stage regrind, hydraulic classification, of the sulphide tails, treatment of the coarse fraction by gravity (spirals and tables) with inter-stage regrind, treatment of the fine fraction by cassiterite flotation. This circuit would result in a saleable gravity conc. + 50% Sn and a flotation conc. 10% Sn to be sent to a fuming plant together with 1st cleaner tails from the sulphide circuit.

- A flow sheet along the lines of the above - Staged crushing, Heavy Media Separation to remove light barren gangue as floats , primary grinding of HMS sinks , sulphide flotation with inter-stage regrind, further grinding of the sulphide tails, cyclone de-sliming, treatment of the coarse fraction by cassiterite flotation. This circuit would result in a flotation conc. 10% Sn to be sent to a fuming plant together with 1st cleaner tails from the sulphide circuit.

A Pre–feasibility study was completed in May 1983 which considered the above options but without HMS plant as this was deemed to complicate the process. Treatment rate in the study was 150,000 TPA @ 1.37 % Sn – achieved by mining the higher grade zones; a fuming plant was included in the study and rated at 4tph with a preferred location at Cleveland Tin although other options were considered.

All of this work came to an abrupt halt when the International Tin Council collapsed in 1984 and was replaced by a committee made of members the ***Association of Tin Producing Countries*** ATPC. To stabilise tin prices an export quota was imposed which affected new projects around the world.

Cleveland Tin, Ardlethan Tin and Rossarden were all coming to the end of their life and Aberfoyle had discovered the Hellyer lead-zinc-copper deposit in Tasmania and focussed efforts on developing this. A pilot plant for Hellyer was set up at Cleveland tin plant which was close by. Subsequently all of Aberfoyle tin activities ceased and they concentrated on other mineral and base metal prospects.

The situation therefore has changed in the intervening years, and the Queen Hill tin prospect needs to be considered without advantages available when Aberfoyle was a force in tin production. However, the work carried out by Aberfoyle has provided the platform to progress the project.

## 6.3 Process Considerations

### 6.3.1 Metallurgical Test Work

For this hard rock primary tin prospect, the objective is to produce saleable tin concentrates at an acceptable recovery. The early stage of investigation comprised basic ore characterisation tests necessary prior to metallurgical beneficiation tests to develop a process flow sheet. This initial test work was regarded as sighter or scout testing to characterise the ore and comprised:-

- Sizing and assay of crushed samples of the drill core to determine particle and mineral distribution within sized fractions.
- Specific Gravity
- Subjecting the fractions to heavy liquid separation to give a guide to valuable mineral liberation size and amenability to gravity concentration.

- Mineralogical examination of selected fractions to determine the mineral suite present, mineral association and degree of association (composites of gangue and valuable minerals).

- Assay for a range of elements to determine if other metals were present at a level which could be economic.

Following this further work aimed at determining the amenability to beneficiation processes was carried out on comprising liberation and separation testing using grinding, flotation, and gravity methods together with assay and mineralogy of separated products,

The practical mineral dressing options to achieve a separation using the properties of minerals are size, specific gravity, magnetic susceptibility, conductivity, colour, floatability and solubility;

From the initial characterisation work the best options to achieve the targeted outcome was regarded as the use of certain combinations of sulphide flotation, gravity and oxide flotation following communication to liberate the valuable mineral.

The drill core was reduced by crushing and heavy liquid separation carried out on sized fractions

Beneficiation test work carried out comprised removal of sulphides by flotation, assessment of cassiterite liberation in the sulphide tail and gravity response, de-sliming and flotation of cassiterite from the de-slimed fraction. Test work is on-going with current efforts being directed to the fines recovery system employing a combination of fine grinding, selective tin flotation, acid leaching, and magnetic separation.

This test methodology will need to be applied to drill core from any future drilling program, with Severn being of particular interest.

### 6.3.2 Process Logic and Design Philosophy

Based on samples tested Queen Hill ores may be described as complex, of medium hardness. The tin occurs mainly as cassiterite (SnO2) and there are significant iron sulphides mainly as pyrite FeS2 - other gangue present, fluorite, iron carbonate (siderite) and silicates. There are no other minerals present in commercial quantities. The cassiterite is fine grained and disseminated throughout the ore, needs to be ground initially below 200 microns to achieve sufficient liberation to allow concentration to saleable tin grades. Further reduction of the ores needs to be carefully staged to minimise over-grinding and maximise the amount of cassiterite which can be recovered by natural gravity methods, (spirals and tables). Cassiterite particles below 30 microns need to be recovered by flotation with final upgrading by centrifugal gravity methods. Centrifugal concentrator were not available when Aberfoyle did their test work – since then a number of centrifugal devices have been commercialised and allow recovery of finer cassiterite by gravity means to be considered, such as Mozley Falcon, Knelson, or Kelsey separators. Of these Falcon is probably the most appropriate. The upgrading of cassiterite flotation concentrate was not finalised in the Aberfoyle test work and there may be other options such as acid leaching, or high intensity magnets, if iron carbonates are present.

To achieve a reasonable recovery of the tin, a process involving 3 stage crushing to – 12 mm followed by heavy media separation, rod and ball mill grinding to reduce the sulphides to a size for flotation and liberate cassiterite from the rest of the gangue. To minimise over-grinding of cassiterite the ball mill needs to be closed with fine screens to ensure early removal of liberated cassiterite from the grinding circuit. Flotation of the sulphides, and concentration of the tin by a combination of classification, gravity separation, and tin flotation, with inter-stage regrinding should result in a medium grade concentrate, acceptable to tin smelters.

Aberfoyle test work indicated that **pre-concentration** was a possibility with Queen Hill ore but as the heavy liquid work was done at fine sizes it could not be considered in this exercise, and test work at coarser size has been carried out which indicates pre-concentration between the crushing and grinding circuits is possible.

### 6.3.3 Process Plant

Based on the characterisation work and the requirements mentioned above the **Process Plant** proposed would comprise

- 3-stage crushing of Run-of-Mine ore to – 12mm particle size.
- Heavy Media Plant based on cyclone separation and Ferro-silicon media to remove a barren siliceous fraction prior to grinding.
- Rod and Ball mill grinding to a particle size which liberates the sulphide and tin mineral from the gangue and to a size where the sulphides will float (< 200 microns).
- Flotation to remove pyrite which will interfere with downstream gravity processes, cleaning and re-cleaning with inter-stage regrind to minimise tin loss
- Hydraulic classification to separate the sulphide flotation tails into feed for natural gravity, tin flotation and enhanced circuits, plus a slime tail (-6 microns) which would be discarded as being too fine for gravity or tin flotation beneficiation methods.
- Acid leaching to remove carbonates from tin flotation concentrates
- De-watering of the tin concentrates in thickeners, followed by filtration to within transportable moisture limits, and storage in a concentrate shed
- Tailings from the process plant thickened and delivered to a tailings storage facility – sulphides may need to be stored separately.

### 6.3.4 Assumptions

For the purpose of this preliminary cost study the following assumptions have been made:-

The treatment plant will be located on a level Greenfield's site

- Access to the site to be via roads suitable for the use of heavy lifting equipment
- The treatment rate is 600,000 tonnes per annum @ 1.1% Sn, 50 % Sn Combined gravity and tin flotation concentrate at 70 % overall tin recovery.
- The basis of the design has been made based on practice at a number of other operations - equipment sizing and selection is preliminary only.
- The treatment plant will operate 24 hours a day for 365 days a year and 91% availability.

- Workforce numbers have been determined to cover for 4 operating crews working 12 hour shifts on a suitable roster; an appropriate number of management, supervisory and technical staff has been included, and a site organisation suggested.

### 6.3.5 Exclusions

To develop the capital and operating cost estimates the following exclusions have been made:

- Metallurgical test work and further process investigations
- Geotechnical test work. Cost of process equipment foundation is based on suitable ground conditions available for large equipment.
- Allowance for the variation in head grade, ore type and hardness.
- Mining, geology, administration, environment, occupational health and safety costs
- Tailings storage facility, raw water dam, and water reticulation
- Site Power and water supply.
- Handling and transportation costs
- Royalties
- Personnel accommodation costs
- Refinery and treatment charges
- Marketing and warehousing
- First Fill Consumables
- Any scope changes
- Owner's costs
- Fire fighting equipment
- Plant mobile equipment, new or used – loaders, trucks, forklift, cranes
- Interest during construction, project financing costs and financing fees
- Import duties, Corporation taxes and Royalties
- Permitting costs
- Schedule delays
- Sunk costs Escalation

- Preparation of Development Proposal and Environmental Management Plan Any other statutory legal requirements

### 6.3.6 Battery Limits

- ROM pad
- Delivery side of tailings disposal pumps
- Concentrate storage at the process plant
- Power supply to on-site sub-station
- Process water from the tailings storage facility and the raw water dam

### 6.3.7 Data and Information

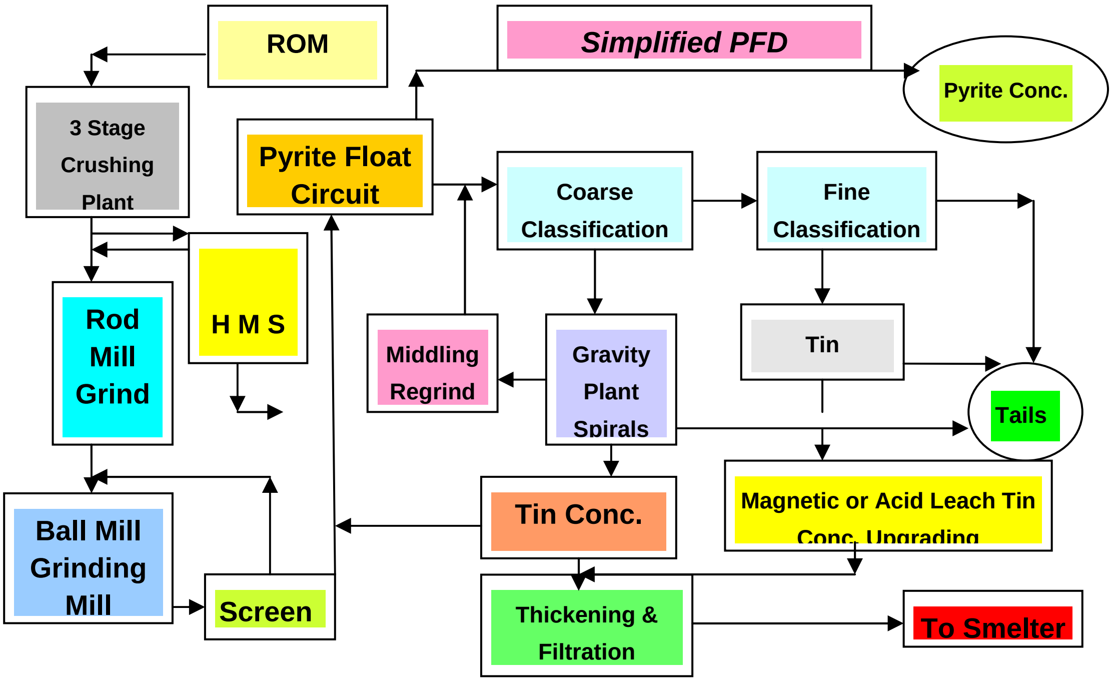
The basis of the capital and operating cost estimate encompasses the following:

- Basic design criteria, current labour and power costs, operating consumables from test work and other similar operations, maintenance as a percentage of the installed equipment.
- Process description from conceptual flow sheet based on test work at Burnie Research Laboratories (ALS AMMTEC)
- Major Mechanical Equipment List comprising: -
  - ROM Stockpile Feeders
  - 3-stage Crushing plant including screens, conveyors - assumes a top size of 600mm for the ROM Ore -- important as this affects the primary jaw crusher selection a smaller size is better in this case as will result in a cheaper crushing plant
  - Fine ore storage
  - Cyclone Heavy Media plant
  - Rod Mill
  - Ball Mill
  - Fine Screens
  - Hydro-cyclones / Slurry Pumps
  - Sulphide flotation
  - Sulphide regrind Mill
  - Spirals
  - Shaking Tables
  - Tin Flotation
  - Falcon Centrifugal Gravity Separator
  - Regrind Ball Mill

- Acid Leach Plant
- Concentrate Thickener
- Concentrate Filter
- Tailings thickener

### 6.3.8 Conceptual Process Flow Sheet

The conceptual process flow sheet below shows 3 stage crushing to – 12 mm followed by, cyclone heavy media plant, primary rod and ball mill grinding to liberate the cassiterite from the gangue, sulphide flotation with inter-stage regrind to remove pyrite, and concentration of the tin by a combination of classification, gravity plant and tin flotation and acid leach plant to produce concentrates acceptable to tin smelters.



### 6.3.9 Design Parameters

***Ore Characteristics***

| | | |
|---|---|---|
| Specific Gravity | t/m3 | 3.0 |
| Run-of-Mine moisture | % H2O | 5.0 % |
| Bulk Density broken rock | t/m3 | 1.6 |
| Compressive Strength | Kg/cm2 | n/a |
| Work Index | kWhr/t | Assumed 18.0 |
| Abrasion Index | | n/a |
| Run-of-Mine top size | mm | Say 600mm |

***Crushing Plant Operations (3 stage crushing)***

| | | |
|---|---|---|
| Annual throughput target | Tonnes per annum | 600,000 |
| Hours per shift | - | 10 |
| Shifts per day | - | 1 |
| Days per week | - | 7 |
| Weeks per year | - | 52 |
| Plant utilisation | % | 40 |
| Nominal Throughput | Tonnes per hour | 200 |
| ROM Top size | mm | 600 |
| Crusher Product size | P80 mm | 12 |
| ROM ore storage | Tonnes | 10,000 |
| Crushed ore storage | Tonnes | 2000 |

***HMS Operations***

| | | |
|---|---|---|
| Annual throughput target | TPA | 600,000 |
| Operations | Hours per shift | 12 |
| “ | Shifts per day | 2 |
| “ | Days per week | 7 |
| “ | Weeks per year | 52 |
| Utilisation | % | 91% |
| Treatment rate | Tonnes per day | 1800 |
| Treatment rate | Tonnes per hour | 75 |
| HMS Floats Reject | % Wt | 18 |
| HMS Floats Reject | Tonnes per annum | 108,000 |
| HMS Sinks + Fines | Tonnes per annum | 492,000 |
| HMS Tin recovery | % | 97 |

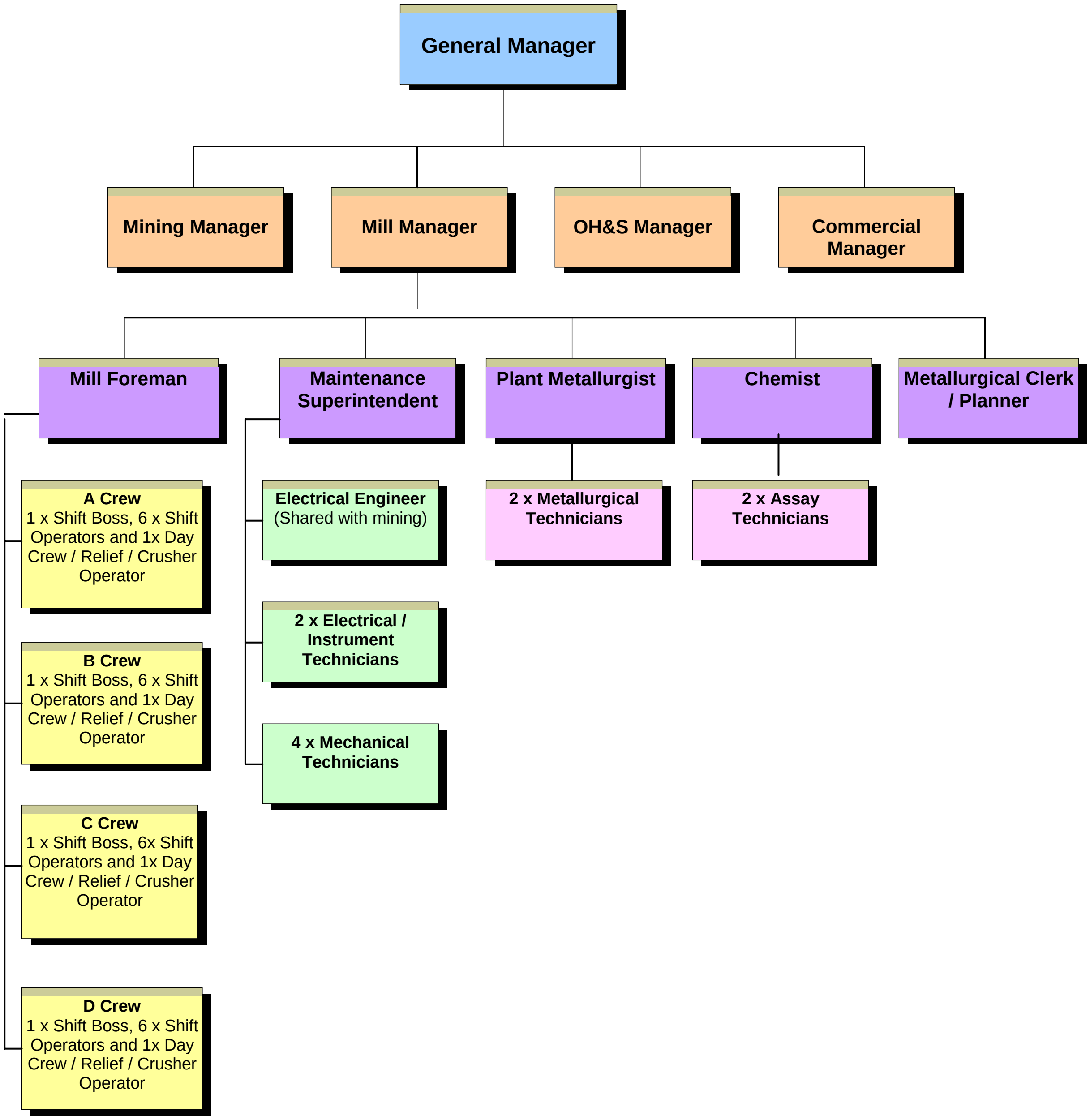
***Concentrator Operations***

| | | |
|---|---|---|
| Annual throughput target | TPA | 492,000 |
| Operations | Hours per shift | 12 |
| “ | Shifts per day | 2 |
| “ | Days per week | 7 |
| “ | Weeks per year | 52 |
| Utilisation | % | 91% |
| Treatment rate | Tonnes per day | 1480 |
| Treatment rate | Tonnes per hour | 62 |

***Metallurgical Parameters***

| | | |
|---|---|---|
| Ore Grade | % Sn | 1.1 |
| Annual Feed Tonnes | TPA | 600,000 |
| HMS Floats | % Sn | 0.18 |
| HMS Sinks + Fines | % Sn | 1.30 |
| Mill Feed | TPA | 492,000 |
| Mill Feed Grade | % Sn | 1.30 |
| Overall Tin Recovery | % | 70.0 |
| Final Conc. Tin grade | %Sn | 50 |
| Final Concentrate | Dry TPA | 9240 |
| Total Tin metal in Conc. | TPA | 4620 |

## 6.4 Proposed Site Organisation Chart



# 7 CAPITAL & OPERATING COSTS

## 7.1 Mining Capital Costs

The underground development is envisaged to be undertaken using a suitably experienced underground mining contractor. The pricing has been based on the contractors supplying the required mobile capital equipment to satisfactorily perform the mining schedules provided. It is envisaged that the principal would supply surface infrastructure items and specialist services and Mining One has provided price estimates for these items. Major capital items have been itemised in Table 7-1.

**Table 7-1: Mining & Administration Capital Costs – Life of mine**

| Major Capital Items | Estimated Cost | Unit Costs ($/ore tonne) | Unit Costs ($/ Sn recovered) |
|---|---|---|---|
| Mine Development | $44M | $9.53 | $1,468.46 |
| Geology | $12M | $2.59 | $399.32 |
| Pastefill Plant | $7M | $1.52 | $233.89 |
| Mine Ventilation Fans | $1.75M | $0.38 | $58.47 |
| Office / Change room / Workshop | $700,000 | $0.15 | $23.72 |
| Light vehicles & Surface loader | $800,000 | $0.17 | $27.50 |
| Power Installation (Mill to UG) | $800,000 | $0.17 | $26.73 |
| Mine Rescue & Safety Equipment | $400,000 | $0.09 | $12.35 |
| Other Items | $1.2M | $0.26 | $40.26 |
| **Total** | **$68M** | **$14.86** | **$2,290.70** |

### 7.1.1 Mining Operating Costs

Operating costs have been calculated by applying unit rates to physicals in the schedule. The majority of these costs are attributed to the mine production which has been calculated using a set of contractor's rates deemed suitable for this size operation.

**Table 7-2: Mining Operating Costs – Life of Mine**

| Major Operating Costs | Total cost | Unit Costs ($/ore tonne) | Unit Costs ($/ Sn recovered) |
|---|---|---|---|
| Mine Production (Contractor) | $206M | $44.6 | $6,879 |
| Mine Management Wages | $27M | $5.76 | $887 |
| Power | $14.6M | $3.18 | $490 |
| Administration | $6.0M | $1.31 | $202 |
| Transport | $2.4M | $0.52 | $80 |
| Light vehicles & Surface loader | $1.5M | $0.33 | $51 |
| Other Items | $0.4M | $0.09 | $14 |
| **Total** | **$257M** | **$55.81** | **$8,603** |

### 7.1.2 Open Pit V Underground Cost Comparison

Mining One have made a comparison between mining the upper resource by open pit as shown in section 4 or extracting the same resource by underground.

Underground mine operating costs are $55.8 / tonne which is less than the operating cost calculated for the open pit of $73.41 / tonne (section 4.4). However if the ore was to be mined totally by underground methods there would be a shortfall in backfill of 1.16M tonnes. The overall costs comparison is shown in Table 7-3.

**Table 7-3: Extraction Comparison**

| Major Operating Costs | Underground | Open Pit |
|---|---|---|
| Ore Tonnes | 171,599 | 171,599 |
| Mining Costs ($ / ore tonne) | $55.8 | $73.41 |
| Additional Backfill Required (tonnes) | 1,160,000 | 0 |
| Mining Backfill Costs ($ / tonne) | $4 | 0 |
| **Total Costs** | **$14.2M** | **$12.6M** |

The table below shows that once the allowance for mining backfill is included the overall costs of mining the upper resource from underground is an additional $1.6M.

This has been a fairly simple comparison and does not take into account the time frame to mine the resource and the flexibility to mine additional material in the pit both which would further support the decision to adopt the open pit option.

The financial model has therefore been based on a small pit followed by underground mining.

## 7.2 Process Plant Capital Cost Estimate

| Major Equipment | Capital Cost Estimate ($M) |
|---|---|
| **A 1 - Crushing Plant** | **12.0** |
| **A2 Heavy Media Plant** | **5.0** |
| **A3 Concentrator** | 23.9 |
| **A3 Other Items**<br>• *Instrumentation, samplers, reagent feeding*<br>• *Plant services Compressors water pumps etc*<br>• *Concentrate handling system*<br>• *Main Process building Auxiliary Buildings*<br>• *Electrical installation Piping Installation*<br>• *Equipment Installation* | **27.24** |
| **Total Installed Cost** | 68.14 |
| EPCM – Engineering Procurement & Construction Management | **10.22** |
| **Grand Total** | **78.36** |

### 7.2.1 Process Equipment Inclusions

The estimate includes crushing, heavy media plant, grinding, fine screening, sulphide flotation, cyclone classification, gravity, tin flotation, concentrate acid leach plant, thickening and filtration of concentrates, thickening of plant tails sized to treat 600,000 TPA. Major process equipment costing is from recent vendor quotations for similar projects based on preliminary design parameters and test data.

These costs need to be firmed up in later study stages – it should be noted that with the current global activity in the mining mineral and metal industry all project costs are escalating on a short term basis.

### 7.2.2 Engineering, Procurement and Construction Management

Based on in-house information for a number of projects similar in scope and scale.
The EPCM costs do not cover geotechnical testing, surveying, material testing, warehousing and the contractor's direct construction supervision and other owner's costs.

## 7.3 Process Plant Operating Cost Estimate

*Preliminary Operating Cost Estimates exclude:*

- Tailings storage facility wall lifts
- Transport handling charges
- Marketing, refinery and treatment charges for concentrates

- Effluent treatment
- Royalties
- Interest charges and tax
- Mining, geology, administration, environment, occupational health and safety costs.

*Estimated Operating Costs composition*

*Consumables*

- Crusher wear parts
- Heavy Media
- Grinding mill liners
- Grinding media
- Flotation reagents
- Filter Cloths
- Labour
- Power (installed estimated at 5 MW)
- Maintenance

### 7.3.1 Labour

The list below is an estimate of the Management and Labour required to operate the proposed concentrator – wages, salary and on-costs based on current rates in Tasmania

| Number | Position | Salary | On Costs 40% | Annual Cost $ |
|---|---|---|---|---|
| 1 | Mill Manager | 150,000 | 210,000 | 210,000 |
| 1 | Planner / Met Clerk | 60,000 | 84,000 | 84,000 |
| 1 | Plant Met | 120,000 | 168,000 | 168,000 |
| 1 | Mill Foreman | 100,000 | 140,000 | 140,000 |
| 4 | Shift Boss | 90,000 | 126,000 | 504,000 |
| 24 | Shift Operators | 80,000 | 112,000 | 2688,000 |

| Number | Position | Salary | On Costs 40% | Annual Cost $ |
|---|---|---|---|---|
| 4 | Day Team Crushing | 60,000 | 84,000 | 336,000 |
| 1 | Maintenance Supt | 120,000 | 168,000 | 168,000 |
| 1 | Electrical Supt | 120,000 | 168,000 | 168,000 |
| 2 | Elect /Instr Tradesmen | 80,000 | 112,000 | 224,000 |
| 4 | Mechanical Tradesmen | 80,000 | 112,000 | 448,000 |
| 1 | Chemist | 80,000 | 112,000 | 112,000 |
| 2 | Assay Technicians | 60,000 | 84000 | 168,000 |
| 2 | Met Technicians | 60,000 | 84000 | 168,000 |
| **49** | **Total** | | | **5,586,000** |
| | **Cost per Tonne** | | | **9.31** |

### 7.3.2 Consumables

The reagent and grinding media costs were estimated in $ per tonne based on similar operations.

| Item | Tonnes per annum | Cost per tonne $ | Total Cost $ |
|---|---|---|---|
| Ferro-silicon | 300 | 1750 | 525,000 |
| S Float Collector | 60 | 3800 | 228,000 |
| Copper Sulphate | 90 | 3000 | 270,000 |
| Frother | 36 | 3375 | 121,500 |
| Tin Float Reagents | 220 | 9500 | 2,090,000 |
| Leach Plant Acid and Lime | | | 1,200,000 |
| Flocculants | 45 | 5000 | 225,000 |
| Grinding Media | 500 | 2000 | 1000,000 |
| Jaw Crusher liners | | | 80,000 |

| Item | Tonnes per annum | Cost per tonne $ | Total Cost $ |
|---|---|---|---|
| Cone Crusher Liners | | | 150,000 |
| Screen Panels | | | 100,000 |
| Ball Mill Liners | | | 250,000 |
| Cyclone Parts | | | 100,000 |
| Filter Cloths | | | 50,000 |
| **Total $** | | | **6,389,500** |
| **Cost per tonne treated** | | | **10.65** |

### 7.3.3 Power

Power demand has been based on an estimate of the installed equipment as shown in the equipment list above. The estimate of the consumed power has been made to determine the annual power costs based on equipment operating hours. Unit cost of Power (10 Cents per kWh) and is typical for this region.

| Item | MW | Cost per kWhr $ | Total Cost $ |
|---|---|---|---|
| Installed Power | 5.00 | | |
| Total consumption = Load Factor 0.70 and plant running Time 91 % x Installed Power | 3.5 | 0.10 | 2,782,416 |
| **Cost per tonne treated** | | | $4.63 |

### 7.3.4 Maintenance

| Item | Basis | Total Cost $ |
|---|---|---|
| **Maintenance Materials** | **4% of installed equipment** | **1,200,000** |
| Cost per tonne treated | | **$2.00** |

### 7.3.5 Operating Cost Summary

| Area | Unit Costs $ / Tonne | Total costs per annum $ |
|---|---|---|
| Labour | 9.31 | 5,586,000 |
| Consumables | 10.65 | 6,389,500 |
| Power | 4.63 | 2,782,416 |
| Maintenance | 2.00 | 1,200,000 |
| **TOTAL COSTS** | 26.59 | 15,957,916 |

## 7.4 Financial Model

The capital and operating costs have been applied to the mine schedule to create a financial model as shown in Table 7-4. Items excluded from the processing cost estimated have been included as part of mine services. The adjusted revenue has allowed for smelter costs and state royalties.

**Table 7-4: Capital and Operating Costs**

| | Yr -1 | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr 6 | Yr 7 | Yr 8 | Yr 9 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Capital Cost (US$,000,000) | $94.1 | $26.3 | $10.6 | $3.7 | $7.0 | $0.75 | $0.95 | $0.75 | $0.5 | $0.25 | **$148.9** |
| Operating Cost (US$,000,000) | $6.5 | $41.2 | $46.1 | $48.3 | $47.8 | $47.0 | $46.8 | $46.5 | $32.5 | $19.7 | **$382.4** |
| Adjusted Revenue (US$,000,000) | | $65.0 | $95.7 | $96.4 | $95.2 | $93.2 | $86.4 | $79.5 | $57.6 | $30.4 | **$699.4** |
| Cash Flow (US,000,000) | **-$100.6** | **-$1.5** | **$40.6** | **$38.8** | **$45.2** | **$47.0** | **$40.0** | **$33.6** | **$25.6** | **$10.9** | |

The split between pre production capital and working capital is shown in the table below.

| | Yr -1 | Yr 1 | Yr 2 | Yr 3 | Yr 4 | Yr 5 | Yr 6 | Yr 7 | Yr 8 | Yr 9 | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Pre Production Capital (US$,000,000) | $94.1 | $14.3 | | | | | | | | | **$108.4** |
| Working Capital (US$,000,000) | | $8.0 | $10.6 | $3.7 | $7.0 | $0.75 | $0.95 | $0.75 | $0.5 | $0.25 | **$32.5** |

## NPV and Sensitivities

The NPV for the project is $70M and the IRR 25% for the base case Sn price of $25,000 / tonne and a discount rate of 10%. A number of sensitivities on Sn price, grade and recovery have been performed to produce a range of NPV's as shown in Table 7-5.

**Table 7-5: NPV Sensitivities**

| NPV | | | |
|---|---|---|---|
| **Change** | **$Sn** | **Grade** | **Recovery** |
| **-10%** | $ 27M | $ 27M | $ 27M |
| **0%** | $ 70M | $ 70M | $ 70M |
| **+10%** | $ 113M | $113M | $113M |

A second financial model was run which included a contract crushing component. This reduced the capital cost of the mill by $14M but increased the operating cost by $6 / ore tonne. The NPV and IRR for this option were $66M and 26% respectively.

# 8 CONCLUSIONS & RECOMMENDATIONS

1. Since the top of the Severn and Montana orebodies are located approximately 80 meters below the Queen Hill orebody there is a delay of up to 12 months in bringing this ore into the schedule. To minimise mill ramp up the underground development has commenced 12 months prior to open pit mining and 17 months prior to milling first ore. This has an impact on pre production capital which may be reduced if more ore was to be realised higher in the Severn and Montana. This could therefore be a focus for future drilling.
2. The model is sensitive to grade and any increase in the tonnage of the high grade Montana orebody will benefit the project providing the same grades are maintained.
3. The option of contract crushing in the mill had a $14M saving in capital but resulted in a $27M additional in operating cost. The advantage is a lower pre production capital by $14m but also a lower NPV of $66M.
4. Credits for lead, silver and zinc in the pit were not taken into account however there was insufficient information to model this material.
5. For this project to move from scoping study to pre feasibility the following conditions need to be met:
    - A Measured or Indicated Mineral Resource leading to a refined block model and interpretation;
    - More detailed geotechnical assessment of all 3 orebodies including near to surface ground conditions;
    - Completion of a hydrogeology study for underground and surface mining to better define the potential impact of water on the project;
    - More detailed stope designs taking into account the revised geology and ground conditions;
    - Establishment of potential sites for a processing plant, tailings facilities, ROM pad, mullock dump/s and other site surface infrastructure;
    - Budget contractor open pit and underground mining costs along with confirmation of the processing and refining costs;
    - Budget quotes for major capital items such as fans, buildings, processing plant, tailings facilities, etc.; and
    - Further consideration of the economic and environmental impact of mining, processing and transportation of concentrate on the township of Zeehan.

Yours Sincerely

**Phil Bremner**
Senior Mining Engineer
**MINING ONE PTY LTD**

Yours Sincerely

**Bill Frazer**
Managing Mining Director
**MINING ONE PTY LTD**

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