104001

THE ANCHOR MINE

ˉ PROJECT

88 - 2835

FEASIBILITY REPORT

55n/89.

MINES

File Ref.

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RL 8713

SPECTRUM RESOURCES AUSTRALIA PTY LTD

JUNE 1988

Restricted file

ANCHOR MINE PROJECT

I N D E X

1.   **INTRODUCTION**

The Anchor Mine lies some 20 road kilometres west of St Helens in north-east Tasmania. The main access route is by travelling 16km along the Tasman highway towards Launceston from St Helens and then by good gravel road 4km to the mine site.

Primary tin mineralisation was discovered around 1881 during alluvial mining operations. Hard rock mining commenced at the Anchor Mine and was carried out up to 1918. The workings now consist of a series of abandoned open pits known as the Anchor opencut occupying an area of approximately 5 hectares.

The area of interest is covered by R.L. 8713. The licence is held by Hellyer Mining and Exploration Pty Limited for a Joint Venture involving Spectrum Resources Australia Pty Limited with a 60 percent interest, Hellyer with 38 percent, and Nargun Pty Limited with 2 percent.

In 1964 Aberfoyle commenced the first of several drilling programmes and between 1964 and 1966 they completed 39 short vertical holes, mainly in the north east area of the opencut.

In 1968, the Tasmanian Department of Mines drilled one hole to test for an extension to the north.

Over the period 1977-1978 Renison completed twenty-nine drillholes aimed at exploring for lateral and vertical extensions of the mineralisation defined by Aberfoyle. A further 42 holes were drilled by Renison over the period 1979-1981. A further 15 large core holes were also drilled for metallurgical samples.

Two high grade lenses, A lens and B lens, were located and assessed at 425 000 tonnes at 0.57% Sn, and 208 400 tonnes at 0.45% Sn respectively at a cut off of 0.2% Sn. A further 2 878 000 tonnes at 0.23% Sn were assessed at 0.1% Sn cut off.

Metallurgical testing was carried out and it was concluded that a recovery of 85 percent was achievable from the ore at low head grades. Simple gravity methods were adequate.

All mining proposals envisaged an open pit mining method with some deeper extensions being mined by underground methods.

The various studies carried out by Renison concluded that insufficient ore resources were present to justify development of a mine.

Spectrum Resources Limited became interested in the property in 1986, and considered that a small scale underground operation to mine the high grade A and B lenses could be viable.

A preliminary feasibility report was prepared in 1987 which examined the concept of a six year life underground mining project based on a 100 000 tpa production using highly mechanised mining methods and a milling plant employing gravity concentration methods. This report concluded that there were advantages in increasing the production rate, and that a review of costs should be undertaken.

This preliminary report was sufficiently encouraging for Spectrum to enter into negotiations for the acquisition of the 38 percent held by Hellyer Mining and Exploration Pty Limited. This acquisition has now been completed.

Spectrum has re-examined the ore resource status, and has had an evaluation carried out by an independant consulting company, Barratt Fuller and Partners. Metallurgical testwork has been undertaken by the Tasmanian Department of Mines, Launceston, to confirm the Renison work and to evaluate the upgrading of tin concentrates.

Additional work was carried out to assess the crushing response of the ore by Tidco Limited, and chemical characterisation of ore and metallurgical products by Miller and Associates for environmental impact assessment.

The work has resulted in the preparation of this Feasibility Report for presentation to the Tasmanian Department of Mines to support an application for a mineral lease over the area of R.L. 8713. Water licence applications will also be made.

John Miedecke and Partners Pty Limited have been retained to prepare an Environmental Impact Statement Addendum as required by the Tasmanian Department for the Environment.

Barratt Fuller and Partners have prepared a report on the tailings dam, and water storage dam for approval by the Tasmanian Department of Mines.

Planning approval application is to be made to the Portland Municipality for the project.

583000mE   84   85   586000mE

000   104000

CRYSTAL HILL

5436000mN

GROOM

RIVER

ANCHOR CREEK

GOUGHS HILL

GIBSONS CREEK

R.L. 8713

35

34

33

5 cm

0   1 kilometre

1:25 000

5432000mN

LOCALITY AREA

TASMANIA

HOBART
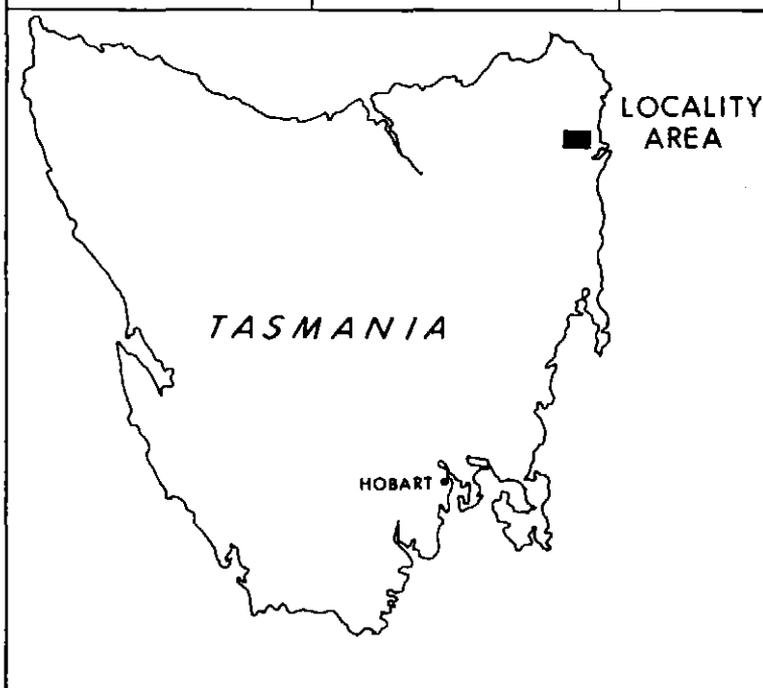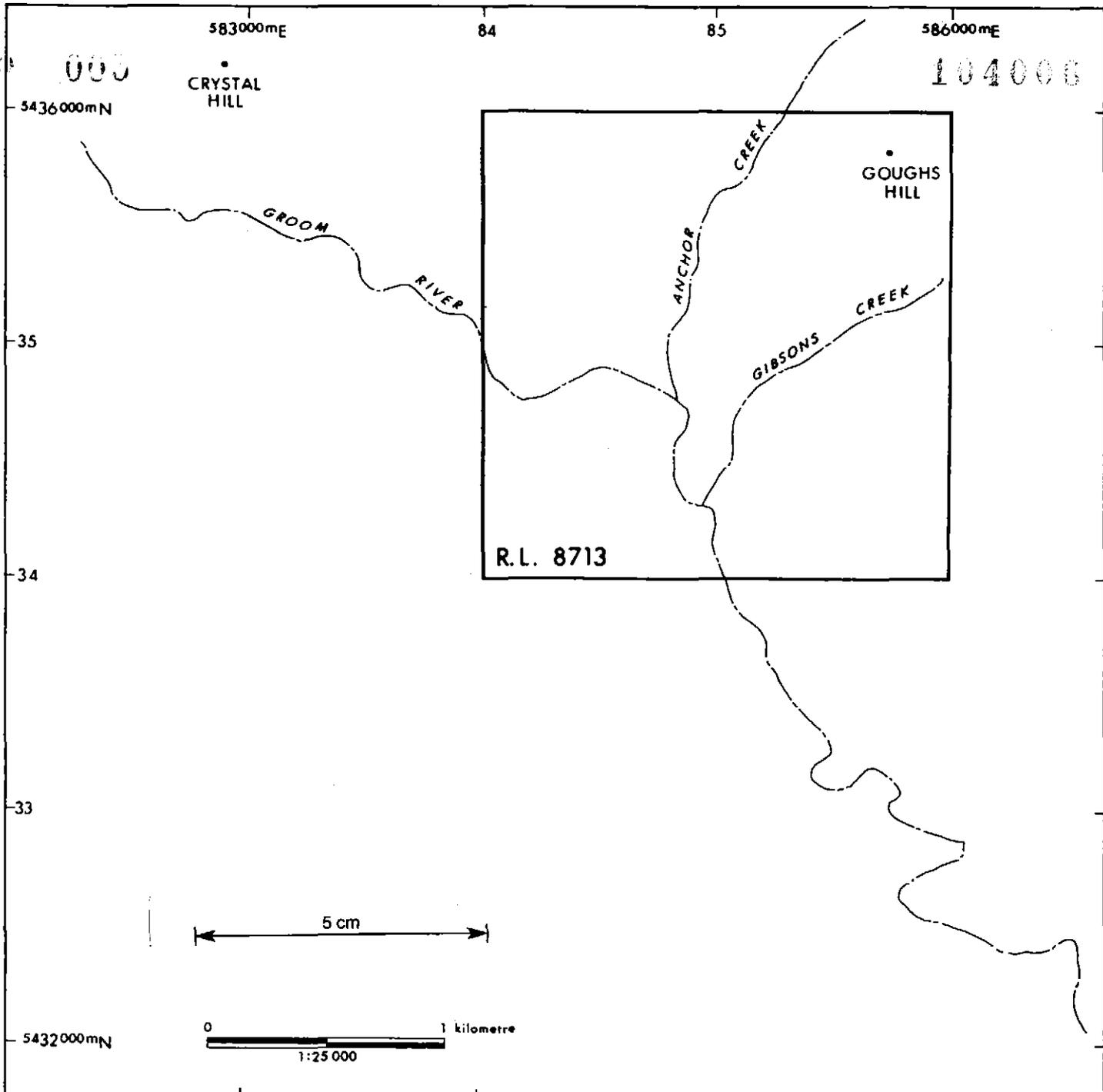
TASMAN HIGHWAY

SPECTRUM RESOURCES AUSTRALIA

ANCHOR MINE
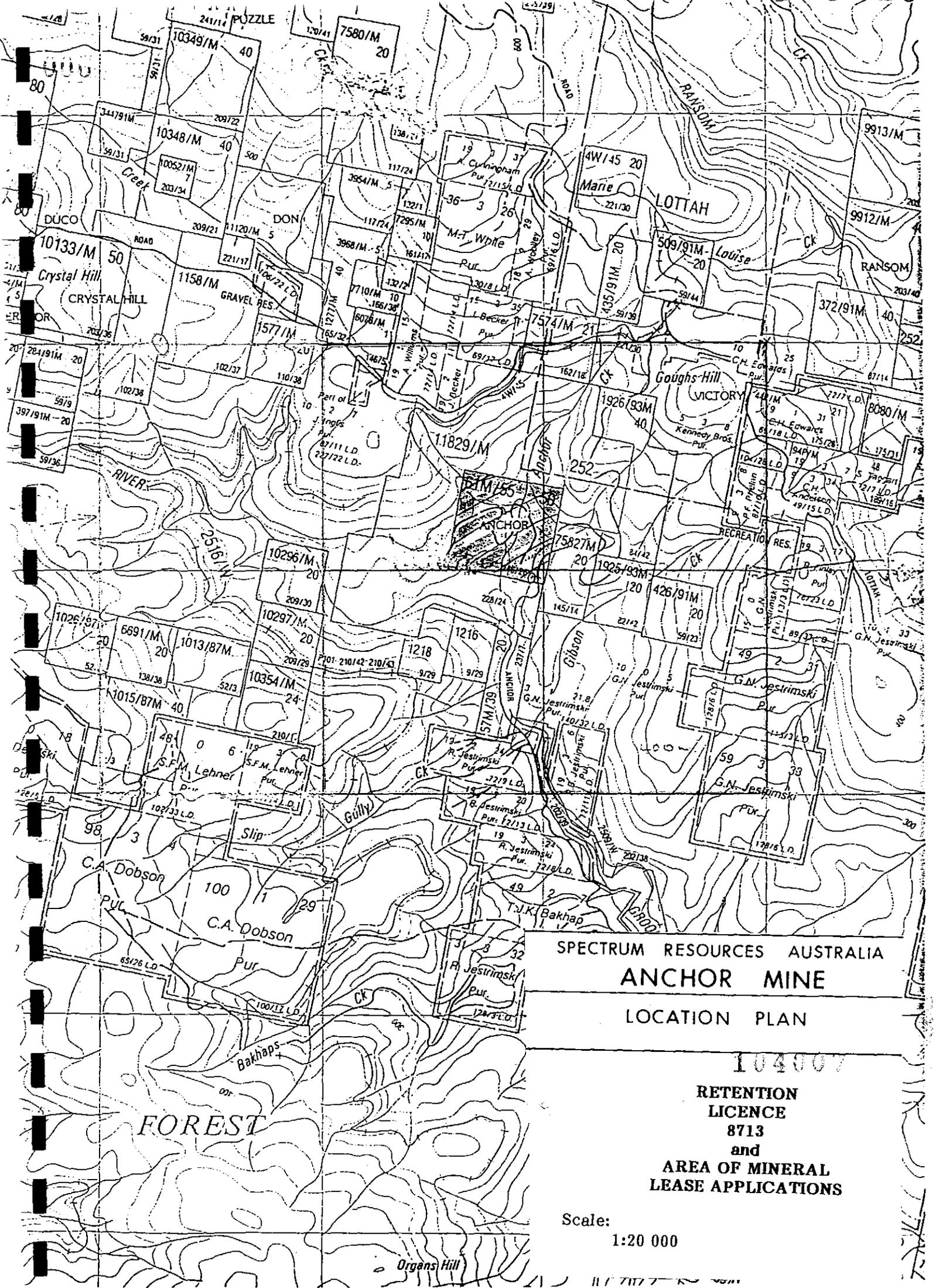
LOCATION PLAN

SPECTRUM RESOURCES AUSTRALIA

ANCHOR MINE

LOCATION PLAN

104007

**RETENTION
LICENCE
8713
and
AREA OF MINERAL
LEASE APPLICATIONS**

Scale:

1:20 000

Boundary old 61M/55

5435 400 N

5435 200 N

104000

5 cm

0   20   40   60   80   100
METRES
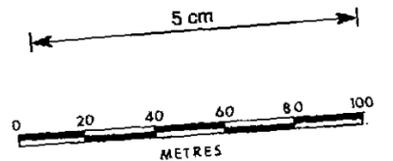
SPECTRUM RESOURCES AUSTRALIA
ANCHOR MINE

AREA OF OLD
61M/55 MINERAL LEASE
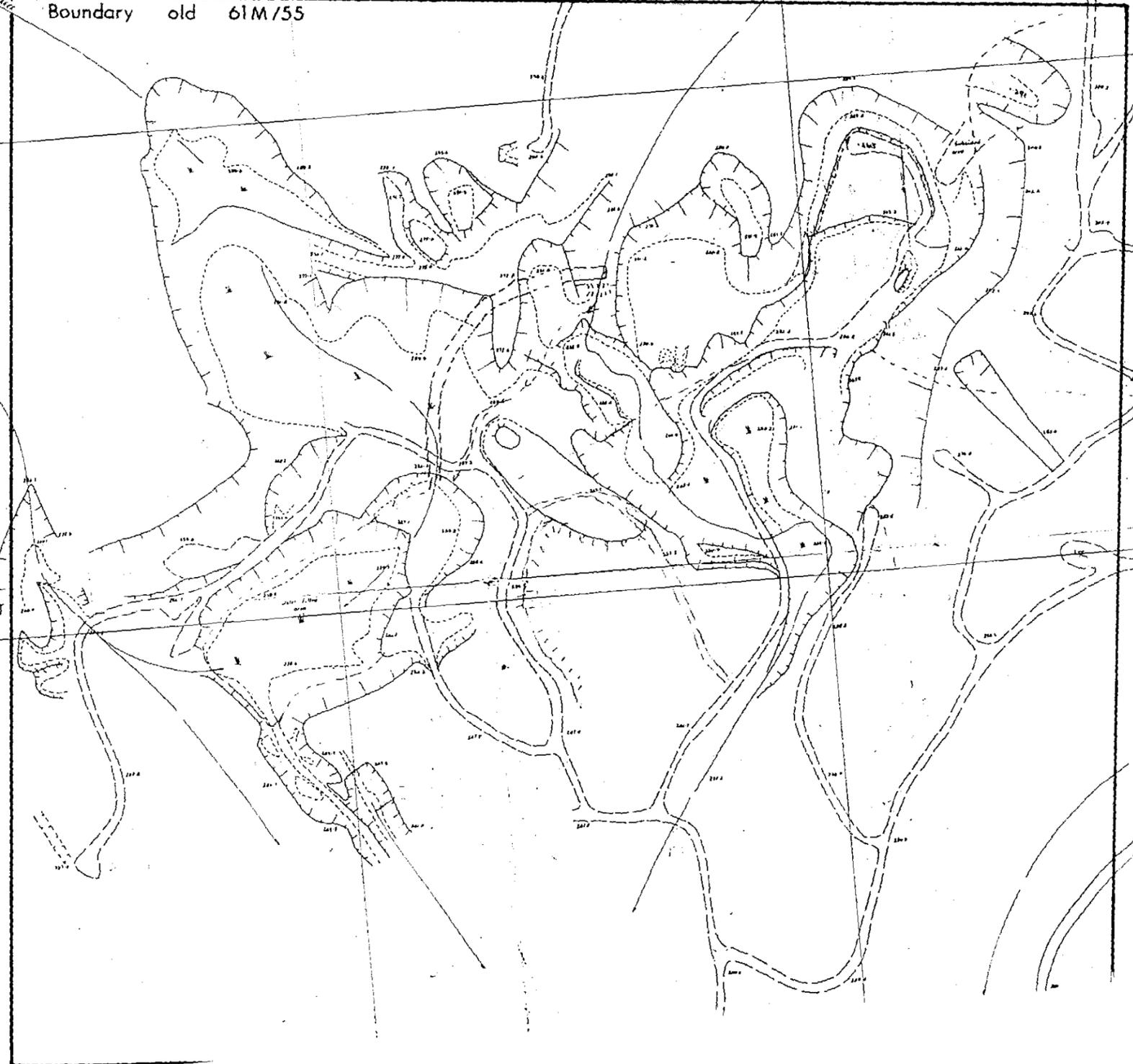
2. SUMMARY

Ore Resources

A **Spectrum** evaluation of ore resources resulted in totals of:

446 000 tonnes at 0.62% Sn	0.4% Sn cut off
611 000 tonnes at 0.54% Sn	0.3% Sn cut off

Barratt Fuller assessed the resources as:

Indicated resources of:

472 000 tonnes at 0.67% Sn	0.4% Sn cut off
511 000 tonnes at 0.60% Sn	0.3% Sn cut off

in addition

Inferred resouces of:

378 000 tonnes at 0.48% Sn

The Spectrum evaluation of indicated resources leaned more towards continuity for the mining plan giving higher tonnage and lower grade than Barratt Fuller who used a stricter interpretation of individual intercepts.

MINING

Mining Reserves

A mining plan was derived from the Spectrum indicated resources, and Barratt Fuller inferred resources. These reserves were calculated as:

795 000 tonnes at 0.52% Sn

Mining Schedule

The mine life of seven years envisages a production rate of 100 000 tpa rising to 125 000 tpa to maintain tin production at around 600 tpa despite falling grade.

Year	1	2	3	4	5	6	7	Total
Tonnes	100	100	110	115	120	125	125	795
Grade % Sn	0.60	0.60	0.55	0.52	0.50	0.46	0.43	0.52
Contained Sn	600	600	605	598	600	575	538	4 116

Mining Method

The stoping method will be room and pillar using 6 metre rooms and 4 metre pillars to give 84 percent extraction. Stope height will be 3.0 metres. Stoping will progress in lifts of 2 metres to extract ore from the hanging wall with mill tailings introduced as fill.

Drilling will be carried out by a single boom electro-hydraulic jumbo, and hauling of ore from face to ore bin by diesel LHD. A mining crew of eight miners will perform all mining work with an extra two men being required at a later stage in the mine life. Mining operations will be carried out on a two shift basis.

Milling Operations

Milling will entail primary crushing to 50mm and then feeding a Barmac crusher to produce a minus 500 micron product in closed circuit with Mogensen and sieve bend screens. Successive cuts from sieve bend screens at 200 micron and 75 micron will be fed to small jigs. The minus 75 micron fraction will be discarded.

Jig concentrates will be treated by table flotation for the separation of sulphides. Tin concentrates will be dried and cleaned by high tension and magnetic separation to produce 70 percent tin concentrates. Overall tin recovery is estimated as 85 percent. Concentrates will be shipped to Malaysia under a marketing arrangement with Tennant Trading (Australia) Pty Ltd.

The mill labour force will be six men operating a two shift, five day roster. This will eventually be expanded to a six day week.

The production of tin in concentrates is tabled as follows:

Year	1	2	3	4	5	6	7	Total
Tonnes contained Tin	510	510	515	510	510	490	455	3 500
Tonnes concentrates	730	730	735	730	730	700	650	5 005

Sulphide concentrates should contain 25 000 oz silver, 50 tonnes zinc and 50 tonnes copper per year.

Mill tailings will be stored in an adjacent tailings dam. A rock starter dam will be constructed with higher lifts being constructed from the coarse sand fraction. The finer fraction will be deposited to the rear of the dam.

A water storage dam will be expanded in storage capacity. A new overflow system will allow the depth of water to be increased with the overflow discharging along the present creek to the Groom river. Anchor Creek will be diverted into the water storage dam to free the tailings dam area.

A small maintenance workshop and office/changehouse would be required on the site to support the operation and the total workforce of 15-17 people.

Environmental Impacts

No acid leachates are likely to be derived from either ore, or tailings storage as the ore contains very low levels of sulphide which will be removed during the flotation process. Fine sediment settlement will be carried out in the tailings pond. Testwork has indicated excellent settling characteristics for this material.

The tailings dam and water management has been engineered for long term performance following abandonment at the end of mine life.

The principal requirement for rehabilitation will be the surface of the tailings dam where stabilisation and re-vegetation will be undertaken.

Financial Aspects

The direct mining and milling and overhead costs are estimated at $16.19 per tonne falling to $15.13 with increased production. This does not include a contingency figure.

A capital cost of $2.995 million is estimated for establishment to the 100 000 tpa production rate. This figure includes a 10% contingency factor.

Three financial cases were run on a simplified basis:

> Base case
> Tin price + 10%
> Operating and capital costs + 10%

$M (Life of mine)	Base Case	+ 10% Tin Price	+ 10% Cap. & Op. Costs
Revenues	29.7	32.6	29.7
Costs	16.1	16.1	17.7
Operating Profit	13.6	16.5	12.0
Profit after Tax	7.9	9.6	6.9

The first financial year would be somewhat less favourable on detailed analysis as allowance has to be made for some development costs several months in advance of production. There will also be a lag time for receipts of tin sales. This would involve some $0.3M for working capital.

UNDERGROUND
WORKINGS

MILL
AND
FACILITIES
AREA

ANCHOR CREEK
DIVERSION

WATER
STORAGE

TAILINGS
AREA

5 cm

0 20 40 60 80 100
METRES
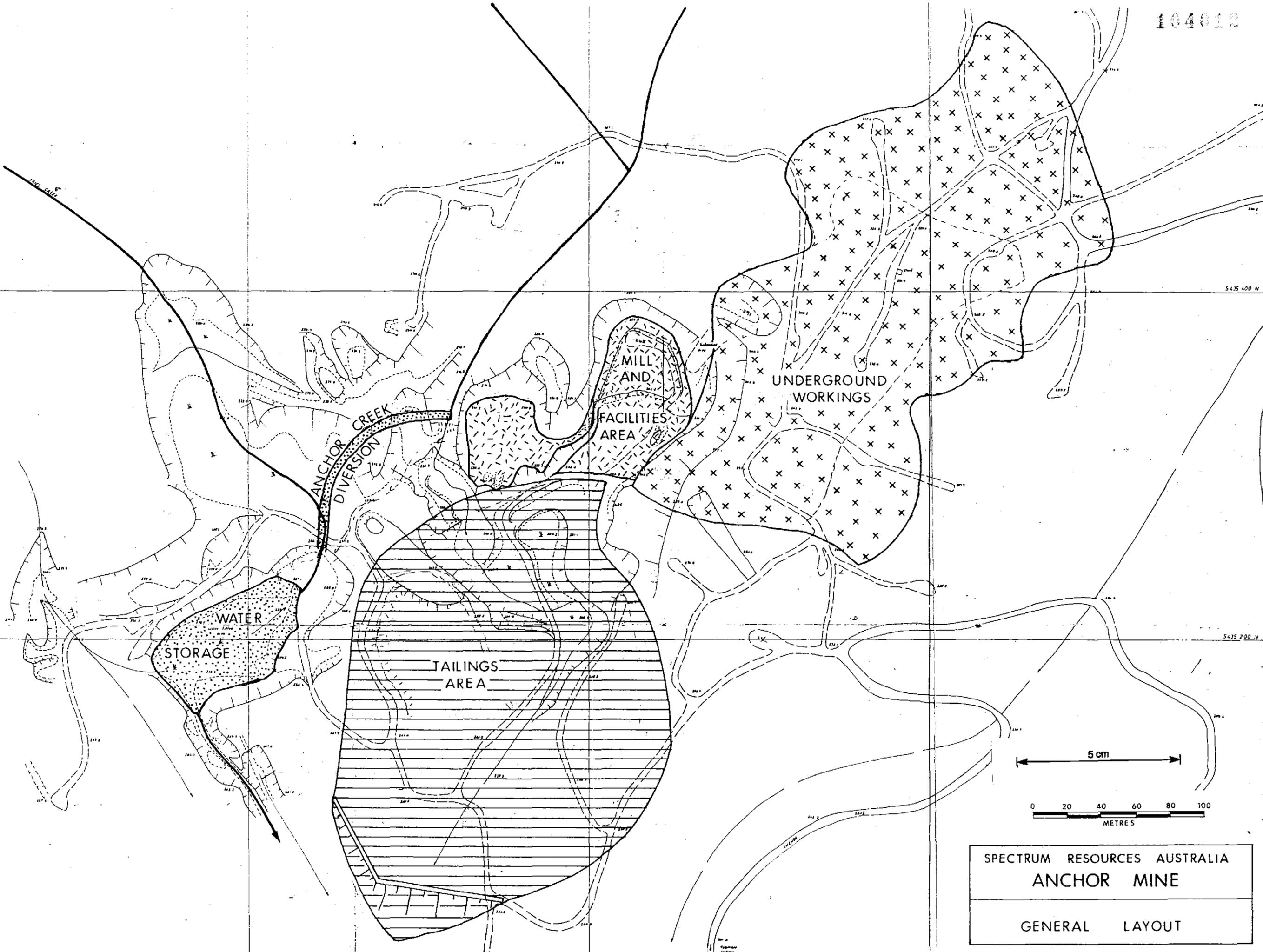
SPECTRUM RESOURCES AUSTRALIA
ANCHOR MINE

GENERAL LAYOUT

3. CONCLUSIONS

The development of an underground mine at 100-125 000 tpa with a highly mechanised operation and small workforce would give a productivity of 26 tonnes per man shift overall and a mine productively of 50 tonnes per man shift. The mining productivity is high, but is comparable with similar mining methods at Renison and King Island in Tasmania.

The ore grades will be variable over short distances and time spans particularly in the undercut stoping operation where grade control will be more difficult. While this is not likely to have any adverse effect on milling performance tin production could be erratic in the first year.

The crushing and screening plant is the heart of the milling circuit and the plant will be operating at its design rate at the 400 TPD rate. Expansion of production to an eventual 500 TPD requires a six day week. While the milling circuit is based on proven technology the required hourly product rate does have to be demonstrated in practice.

The additional metallurgical testwork has confirmed that it is possible to produce high grade tin concentrates with very low levels of impurities. Also by-product sulphide concentrates containing high grades of silver, copper and zinc can be produced. Optimisation of the sales will have to await actual production as separate zinc and copper concentrates may be desirable.

The additional testwork did demonstrate that a liberation size of less than 500 micron is required as suggested by the Renison work. Tests at coarser sizes have indicated that recoveries are too low.

Site investigations of dam sites have not disclosed any unfavourable foundations conditions, in fact the permeable nature of the subsoil is advantageous to the dam design. The tailings have been shown to settle very quickly. They are very low in sulphides and acid generation is most unlikely. The environmental impacts appear to be small, and acceptable.

The preliminary financial indicators of capital requirements and operating costs together with anticipated revenues at the current low tin price give a good return on capital, and an acceptable profit margin.

As mining projects go the Anchor Mine project is simple in mining concept with minimal access development, the metallurgical processing is basic and uncomplicated. At current tin prices the downside risk in pricing is low, and expectations are that 1989 could see an upward movement of the order of ten percent. This expectation is anticipating the tin stockpile overhanging the market to come into balance with demand and supply of tin could become tighter.

4. HISTORICAL SETTING

Mining is believed to have commenced in the Blue Tier area in 1874 with interest centering on the rich deposits of alluvial ground. These operations flourished and extended from the valley bottoms into in-situ detrital deposits overlaying the primary tin bearing greisens. Large scale sluicing operations denuded the surface exposing the greisen bodies. Mining operations were then established to undertake hard rock mining by open pit techniques.

Operations around the Anchor Mine were the Anchor itself, Crystal Hill, the Liberator and the Don, and then to the Australia group of workings.

The estimated hard rock production of the Anchor area is 2 000 000 tons at an average recovered grade of 0.2 percent tin. The Anchor Mine was the predominant producer with an estimated production of 1 750 000 tons. The period of greatest activity extended from 1895 to 1914.

The Geological Survey Bulletin No 38 Blue Tier Tin Field by McIntosh Reid and Henderson published in 1928 contains extremely useful information on ore genesis, structure, mining and treatment. The authors drew upon the experience of J B Lewis who was manager of the Anchor Mine for many years. This information is very relevant to the proposed establishment of a new mining venture when assessing the diamond drilling programme and determining the mining and metallurgical processes to be adopted.

Some general commentaries were that the cassiterite was as a rule of a coarse grain size, and generally of a similar grain size to the gangue minerals. The host rock – the greisen – is soft and granular and is easily disintegrated and pulverised. A smaller portion described as quartz greisen and quartz-mica greisen is harder. Further reduction was necessary to liberate the fine tin enclosed within the mica and feldspar components. Felspathic granite is described as very soft and clayey with coarse cassiterite.

In discussion on the optimum metallurgical treatment of the Anchor ore various observations are made. Cassiterite within pegmatite occuring between the greisen and the overlying porphyritic granite is coarse. Just below the pegmatite it is of equal grain size to the gangue minerals and further away it occurs in part as coarse grains, but in a greater proportion as minute grains filling cleavages in the crystals of mica and feldspar.

Lewis had conducted various tests to determine the cassiterite distribution and the recoveries from certain size ranges. The broad conclusion was that up to 20 percent of the ore was plus 30 mesh, 70 percent plus 120 mesh and 30 percent minus 120 mesh. Losses of cassiterite in the plus 120 mesh range were said to be very small but high in the fine fraction.

Conclusions from testwork and operations indicated that the bulk of the cassiterite is plus 100 mesh (147μ); the bulk of the gangue consists of feldspar and kaolin with quartz as the other important constituent; the cassiterite is free in crystal aggregates; and the sulphide minerals are insignificant.

Deductions from the above were as follows:

* The ore material should not be crushed finer than the cassiterite grains.

* Pulverising (grinding) is not desirable.

* No difficulties should be encountered in the concentration of cassiterite from deslimed ore feed.

* Classification through all stages is desirable.

* Slime portion of the slurry would present difficulties in separation of cassiterite.

The conclusions were that the most satisfactory process was to reduce the ore to 20 mesh (833μ), classify the material and undertake concentration. It is also remarked that the ore was wet and sticky causing choking of crushers.

Production records from the Anchor 1906-07 indicate that the tin content of the concentrates was 71 percent. This contrasts with realisation returns giving 68.8 percent tin in concentrates. The feed expressed as recovered grade was 0.105 percent tin.

The nearby Liberator Mine working similar but higher grade ore dressed concentrates to 73 percent tin.

These observations and data are of use when considering the flowsheet proposed for treatment of the ore from the new mining proposal.

PORPHYRITIC   ADELLAMITE

PEG

AG   ZONE

MICRO
ADELLAMITE

A   LENS

LG A LENS

LG   ZONE

LG   ZONE

B   LENS

?

?

?

LG   ZONE

?

200m R.L.
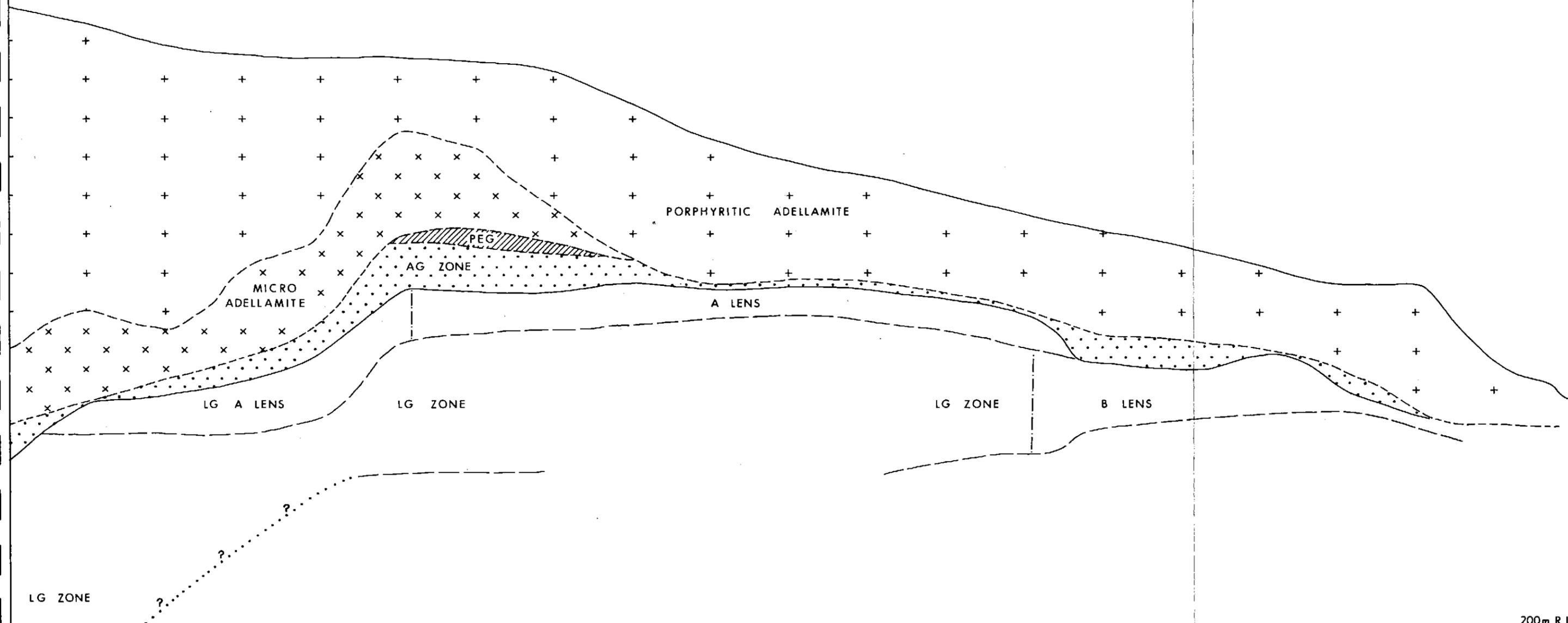
1 200 mNE

1 100 mNE

1 000 mNE

900 mNE

5 cm

0   50 metres

1 : 1000

SPECTRUM RESOURCES AUSTRALIA
ANCHOR MINE

LONG SECTION L-L

SPECTRUM RESOURCES AUSTRALIA
ANCHOR MINE

FOOT WALL CONTOURS
OF ORE BODY

———·—— 280   A Lens
———·—— 240   B Lens
· 265   Spot height

0 _____ 50 metres
1:1000

5 cm

250 contour of
Adellamite Sill

Slip

5. THE GEOLOGICAL SETTING

The tin mineralisation is hosted in an altered Alkali Granite beneath a generally horizontal roof contact with overlying Adamellites. A stanniferous zone of plus 500ppm Sn has been delineated by old workings and more recent drilling. This zone trends north-east and has dimensions of 800 metres by 100-200 metres. The thickness is 40-60 metres. The intensity of alteration decreases with depth beneath the contact and two alteration types have been recognised. These have been described as Greisenised Granite and Granular Greisen. The Greisenised Granites retain their original feldspar, and granitic texture while Granular Greisen consists of granular aggregations of quartz-topaz-mica.

Alteration occurred in two phases. The first is believed to be late magmatic and involved the replacement of feldspars and primary micas by topaz and yellow-dark green siderophyllite, accompanied by cassiterite, purple fluorite and apatite. The second stage is regarded as post-magmatic involving a low temperatre hydrothermal phase where some topaz and siderophyllite was replaced by fine sericite, siderite and trace sulphides.

Many features of the Anchor Deposit are consistent with magmatic and post-magmatic processes of rare-element formation (Sn, W, Nb, Ta, Be, Mo, U, REE). These deposits are typically formed in the apical zones of the late stage intrusives from polyphase intrusive complexes.

The Anchor mineralisation is the major focus of tin deposition in the area. Other deposits in the locality that have been examined are the Liberator-Crystal Hill-Don group, and the Australia group. These are also hosted by alkali granite.

5.1 STRUCTURAL GEOLOGY

The higher tin grade sections of the Alkali granite lie immediately below the anticlinal or cupola structure of the alkali granite-adamellite granite contact and these orebodies form the target for the mining operation. The contact is often occupied by a band of pegamatite, and this generally forms the hanging wall limit to mineralisation. The thickness of ore is 7-9 metres below the hanging wall in the highest cupola section of A lens, but can reach greater intervals in areas of B lens and lower A lens.

There appears to be a small secondary cupola above A lens where there is an enrichment of silver-copper-zinc but only marginal tin grades. This zone also has the greatest development of the roof pegmatite.

Previous operators postulated that cassiterite deposition was related to isotherms created by successive cooling stages of the alkali granite. These isotherms would to some degree at least reflect the overlying adamellite undulating contact. This deposition mechanism was held to account for the irregular distribution of cassiterite over vertical intervals, and the irregularity of grade has been confirmed by diamond drilling.

A number of authors have examined the Anchor deposit and their reports are available. A greater reliance has been placed on the Renison Limited work, and the thesis by A.F. Ross on the Anchor deposit (1983), all of which is available for reference.

5.3 ORE RESOURCES

Spectrum carried out an evaluation of ore resources based on the A and B lenses as basically delineated by Renison Limited. A polygon method was used which seemed appropriate to the ore body slope. The summarised results were:

0.4% Sn Cut off

A Lens	332 320 tonnes	at	0.63% Sn
B Lens	113 650 tonnes	at	0.61% Sn
Total	445 980 tonnes	at	0.62% Sn

0.3% Sn Cut off

A Lens	390 760 tonnes	at	0.58% Sn
B Lens	220 610 tonnes	at	0.48% Sn
Total	611 370 tonnes	at	0.54% Sn

Barratt Fuller and Partners were then approached to carry out a critique of the methodology and the results. They were provided with complete cross sections, the Renison Limited geological reports, and the Spectrum report.

The BFP report examined the grade variability and considered resource grades from a normal and log-normal perspective.

The indicated resource grade for both A and B lenses as calculated by BFP and Spectrum lay within the 95 percent confidence level.

The summary of the BFP report is as follows:

0.4% Cut off (Log-normal distribution)

A Lens	350 000 tonnes	at	0.62% Sn
B Lens	122 000 tonnes	at	0.82% Sn
Total	472 000 tonnes	at	0.67% Sn

0.3% Sn Cut off (Log-normal distribution)

A Lens	366 000 tonnes	at	0.59% Sn
B Lens	143 000 tonnes	at	0.62% Sn
Total	511 000 tonnes	at	0.60% Sn

The Spectrum resource was slightly lower in tonnes and grade at the 0.4% Sn cut off. The latter is probably due to the BFP figures being based on a stricter interpretation of individial intercepts while the Spectrum figures lean more towards continuity for the mining plan leading to the inclusion of additional lower grade material.

Inferred resouces were calculated by BFP who concluded that the following categories could be delineated.

A lens extension to the north east

Inferred Resource	8 000 tonnes	at	0.32% Sn
	114 000 tonnes	at	0.38% Sn
	122 000 tonnes	at	0.35% Sn

SW extension – B Lens

Inferred Resource	150 000 tonnes	at	0.6% Sn

NE extension – B Lens

Inferred Resource	106 000 tonnes	at	0.45% Sn
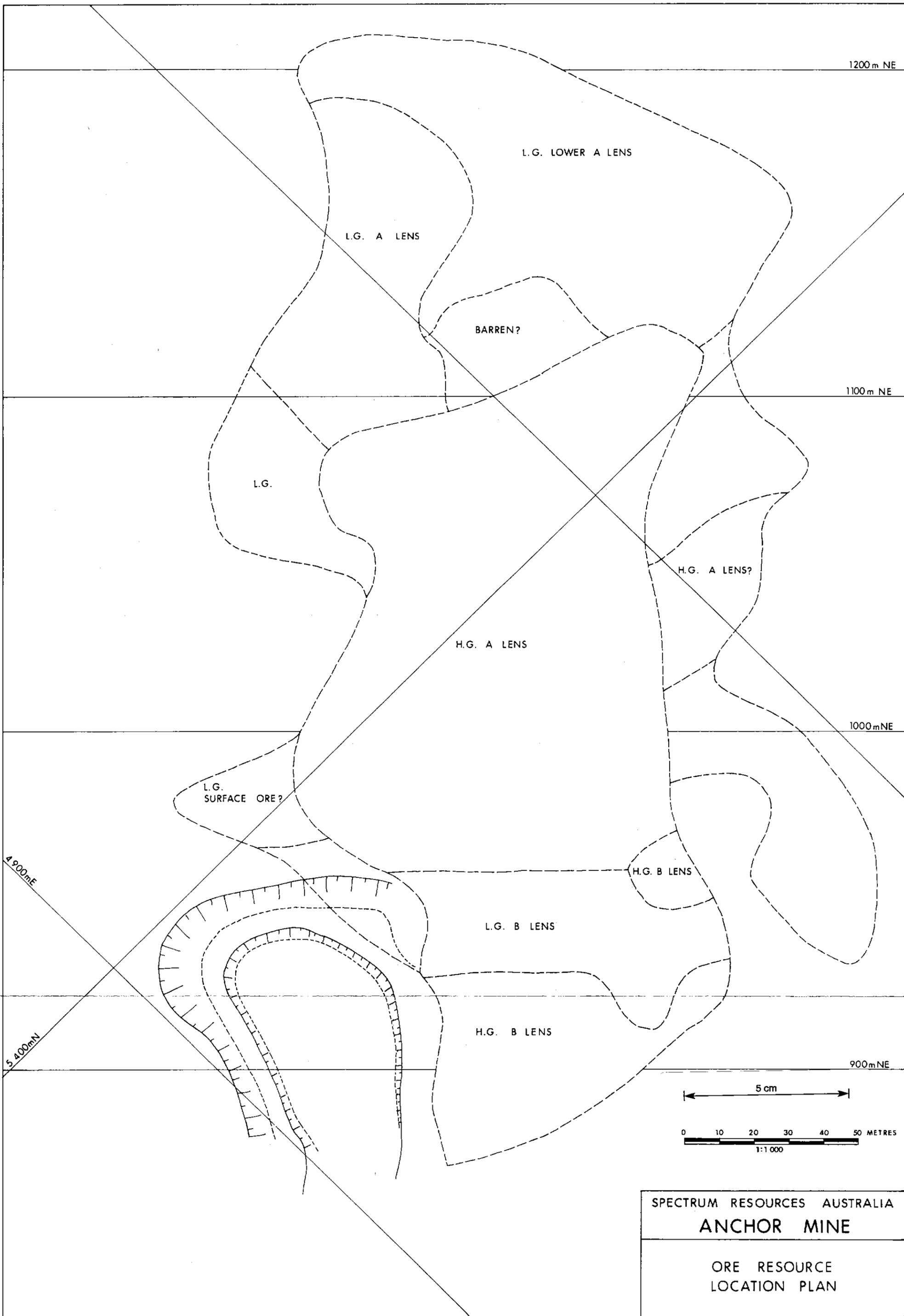
1200 m NE

L.G. LOWER A LENS

L.G. A LENS

BARREN?

1100 m NE

L.G.

H.G. A LENS?

H.G. A LENS

1000 m NE

L.G.
SURFACE ORE ?

H.G. B LENS

L.G. B LENS

4 900 m E

5 400 m N

H.G. B LENS

900 m NE

5 cm

0   10   20   30   40   50 METRES
1:1 000

SPECTRUM RESOURCES AUSTRALIA

ANCHOR MINE

ORE RESOURCE
LOCATION PLAN

5.3 MINING RESERVES

Mining reserves have been assessed from the indicated and inferred resources as determined by Barratt Fuller and Partners and the determination of indicated resources carried out by Spectrum.

A mining plan was based initially on the indicated resources using a room and pillar mining method with 6 metre rooms and 4 metre square pillars for an extraction of 84 percent. This resulted in the following scenerio:

At 0.3% Sn Cut off

	Tonnes	Grade % Sn
A and B Lenses	611 370	0.54
Extraction 84%	513 550	0.54
Dilution 5%	25 680	0.10
	539 230	0.52

At 0.4% Sn Cut off

	Tonnes	Grade % Sn
A and B Lenses	445 980	0.62
Extraction 84%	374 620	0.62
Dilution 5%	18 730	0.10
	393 350	0.60

The BFP study suggested that at least 378 000 tonnes at 0.48% Sn could be available from inferred resources.

The orebody grade distribution made it possible to consider two basic extraction plans.

A. A five year life at 100 000 tpa at 0.52% Sn at a cut off of 0.3% Sn

B. A five year life at 75 000 tpa at 0.60% at a cut off of 0.4% Sn

It was also possible to combine these alternatives as access would naturally be obtained to higher grade blocks in the earlier stages of the operation which would be unaffected by the different cut-off grades.

The earlier operations in the areas of the indicated resources would give opportunity to confirm the inferred resource tonnages. A plan based on a seven year life operation was considered feasible by assuming that a reasonable proportion of the inferred resources would prove to be mineable. The plan envisages a falling grade over time and this would be off-set by increasing the production rate. The objective would be to hold a steady production of tin in concentrates for as long as possible.

The mining reserves were taken as 539 230 tonnes at 0.52% Sn at 0.3% cut off, plus a further 317 000 tonnes at 0.47% Sn from inferred resources. The latter assumes an 80% extraction and 5% dilution at 0.1% Sn.

A mining plan has been drawn up as follows:

Year	1	2	3	4	5	6	7	Total
Tonnage	100	100	110	115	120	125	125	795
Grade	0.60	0.60	0.55	0.52	0.50	0.46	0.43	0.52
Contained Sn	600	600	605	598	600	575	538	4116

6. THE MINING OPERATION

6.1 THE MINING METHOD

The relatively flat lying ore body, and ease of access from the old open pit area indicates that a highly mechanised stoping system with very limited development is applicable.

The method selected is room and pillar with fill which becomes the post pillar mining system. The method is very flexible and barren or low grade areas can be left in situ. The method has been used successfully in other Tasmanian mines notably Renison and Dolphin Mine at King Island. The method involves a room and pillar floor plan wih an undercut being made on the footwall. Fill is then introduced, and forms the floor for the next stoping level. A further stoping cut is made in the roof, ore extracted, and another level of fill introduced. This is continued until the hanging wall is reached.

The method can be highly mechanised and requires a minimum of labour. High productivities are normal, and it is one of the lowest cost methods. Dilution is low and in this case would consist of some low grade, or barren material being taken up when following an assay cut off in the footwall, and also barren fill when in the folling phase.

It is proposed to develop rooms 6 metres wide to leave 4 metre pillars. This will give an 84 percent extraction of the ore. Generally an extra percent or two results from rounding of pillars and overbreak even when closely controlled.

The relatively tight pattern of pillars will give good safety factors for pillar resistance, and reduce the open roof spans at drive and cross cut intersections to give minimal areas thus reducing the need for rock bolt support, or eliminating this entirely.

It is proposed to mine the undercut at a height of 3 metres with subsequent lifts of 2 metres. A height of 1.5 metres would be maintained between the fill floor and the back after filling to allow free movement of personnel.

Drilling would be carried out by a single boom hydraulic jumbo. A drillhole length of 4.1 metres will be utilised for both undercut, and flat back stoping. Mucking will be carried out by a 5 tonne capacity LHD unit hauling driectly to the crusher bin or stockpile. Haulage distances are generally under 200 metres and trucking is not required.

The minimum of services are required underground as all working places are within walking distance of the portals. Light power cables only are required for power to the hydraulic jumbo and to ventilation fans together with a small transportable air compressor. Explosives and assorted supplies can be transported on small trailers pulled by horticultural type tractors. Larger items can be transported by the Cat 920 utility unit.

The manpower requirements are very small with an actual mining crew of eight miners to perform the drilling, blasting, mucking and general production work. With a larger haul approaching 200 metres and increasing tonnage, an extra LHD shift would be required, necessitating an extra two men. This would be in later stages of the mine life.

L.G. LOWER A LENS

L.G. A LENS

BARREN?

1200 m NE

1100 m NE

L.G.
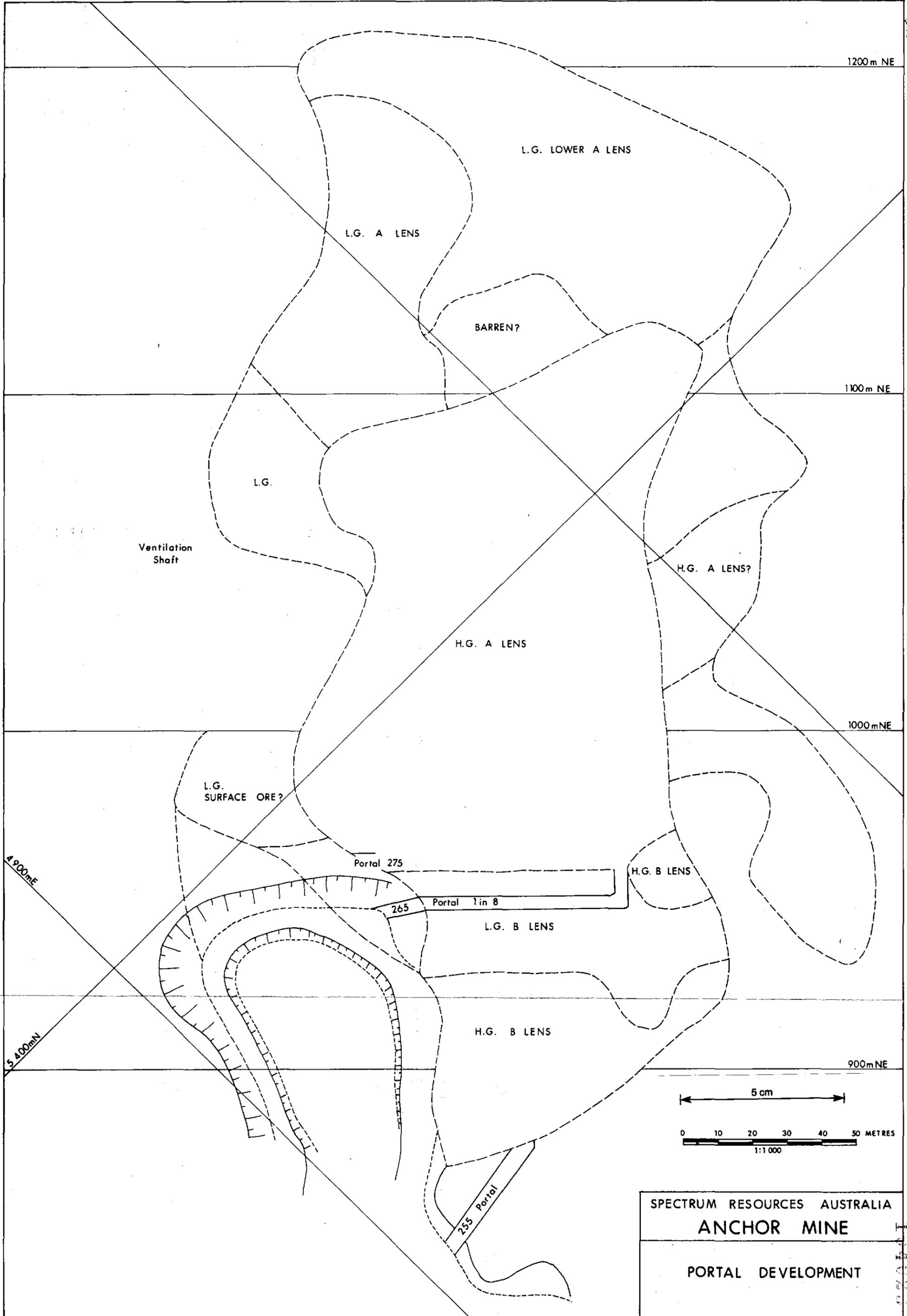
Ventilation
Shaft

H.G. A LENS?

H.G. A LENS

1000 m NE

L.G.
SURFACE ORE?

Portal 275

H.G. B LENS

265   Portal 1 in 8

L.G. B LENS

H.G. B LENS

900 m NE

4 900 mE

5 400 mN

255 Portal

5 cm

0   10   20   30   40   50 METRES

1:1 000

SPECTRUM RESOURCES AUSTRALIA

ANCHOR MINE

PORTAL DEVELOPMENT

6.2 MINE DEVELOPMENT

Access to the orebody can be obtained by a short adit at the 225 RL to drive north on the B lens footwall. This can be portalled in from an open pit face in low grade ore for 35 metres before encountering higher grade material. Undercut stoping would then commence with connections to the open pit face for ventilation.

Access to the A lens section will be gained from an 80 metre long 1 in 8 incline portalled at the 265 RL to intersect the footwall of A lens at the 275 RL. This development will be through low and higher grade ore. Stope undercutting will commence with priority to establish a haulage exit at 275 RL above the crusher bin. Additional openings will be made to the west of A lens to maintain a ventilation return.

The adit openings would have 5 x 3 metre dimensions giving development tonnages of:

A Lens access   3 180 tonnes at 0.30% Sn.
B Lens access   1 390 tonnes at 0.20% Sn

6.3 PILLAR SUPPORT

The design layout is 6 metre rooms and 4 metre pillars to give an overall extraction of 84 percent. Previous reports had considered 8 metre rooms with 4 metre pillars to give 89 percent extraction.

It was decided to take a more conservative approach to roof control by reducing the room size to 6 metres. This has a beneficial effect in reducing roof spans, and in particular the area of unsupported ground at room and cross-cut intersections.

It was also thought to be more efficient to take a single pass drill and blast operation on the undercut than the two pass operation previously considered.

The superincumbent ground over the footwall of the proposed mining zone varies from some 25 metres at the SW area to 50 metres in the centre increasing to a possible maximum of 100 metres at the NE margin.

Considering that the 4 x 4 metre pillars may have an effective bearing area of 3 x 3 metres the following compressive loads would apply, taking an average rock strength from test samples of 14 425 psi.

Depth m	25	50	100
Load psi	92	185	370
Pillar stress psi	1 020	2 055	4 110
Safety factors	14	7	3.5

These safety factors are quite acceptable for incorporation into the mining plan. The pillar height-width ratio become of some significance above 4:1 ratio, which could occur in thicker parts of the deposit.

The mining system relies upon pillar strength being maintained by the confining action of the fill that is introduced around the pillars. It is important for good drainage to be maintained particularly as pillar heights increase so as to avoid the possibility of liquefaction on blasting. This risk will be low due to the rapid drainage anticipated from the coarse tailings used for fill.

1200 m NE

L.G. LOWER A LENS

L.G. A LENS

BARREN?

1100 m NE

L.G.

Ventilation
Shaft

H.G. A LENS?

1000 mNE

L.G.
SURFACE ORE?

Portal 275

265   Portal 1 in 8

900 m NE

255 Portal

5 cm

0   10   20   30   40   50 METRES

1:1 000
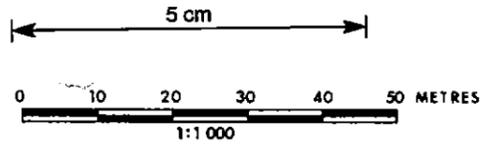
SPECTRUM RESOURCES AUSTRALIA
ANCHOR MINE

TYPICAL STOPING LAYOUT
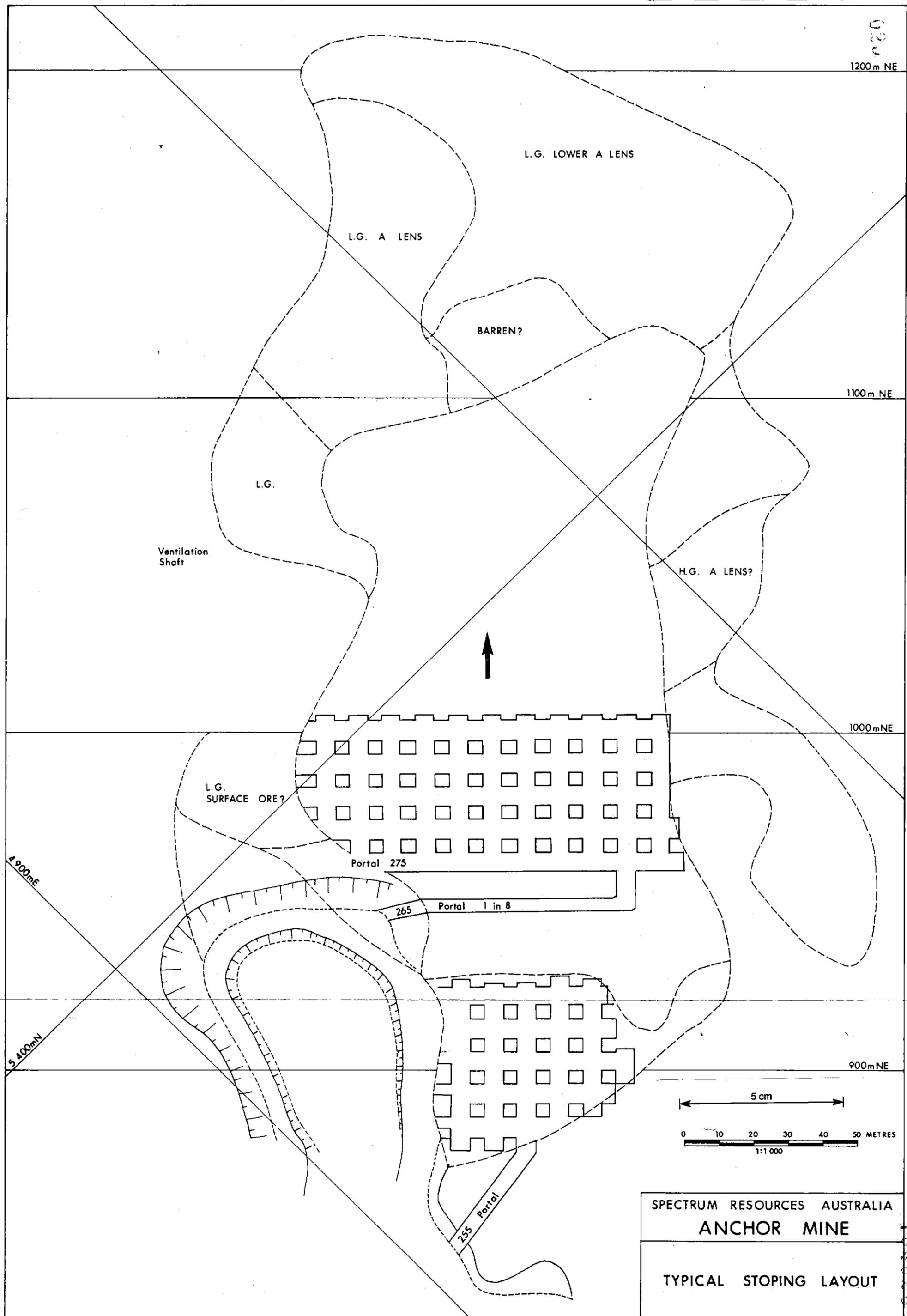
6.4   MINING EQUIPMENT

The economics of the operation require very low mining costs, and this can only be achieved by minimum labour requirements and highly productive mining units.

These major units of equipment would be required:

* A single boom electrohydraulic drilling jumbo.
  An Eimco-Secoma Mercury 14 unit mounting a Hydrastar 300 hydraulic drifter.

* An Eimco 913D diesel powered LHD unit with 2.2 cu.metre, 5.4 tonne, capacity bucket.

* A Caterpillar 920 loader, converted to underground use. This would have the primary use as a work platform for scaling down and charging up faces and building up fill walls.

Drilling Jumbo

An analysis of the drilling duty required indicated that a two boom unit was not necessary. A modern compact single boom unit with a powerful hydraulic drill would be capable of fulfilling the amount of drilling required. The unit can be operated by one man, and can deal with the tunnelling type operations involved in the undercut stoping, and the cut and fill operation.

A 4.1 metre drilling depth is selected as the face widths are capable of producing a long round. This gives benefits in detonator and explosive consumption, and larger muck piles. There are also benefits in minimising non-productive time through extra travelling and set-up time for shorter faces.

Load-Haul-Dump Unit

An Eimco 913D diesel powered unit has been selected as the loading and hauling unit. Productivity calculations indicate that with an oversize bucket capable of carrying 5 tonnes of material one unit can handle the mucking requirement. Initially this should be achievable on short hauls, but in later stages of the mine life two units will be required.

The greisenised granite is very light with a S.G. of 2.65 and an oversized bucket is permissable as long as the load remains within the design parameters of the unit.

There are significant numbers of Eimco 913 units working in Australia and Eimco has a dealership and spare parts centre in Burnie on the north coast of Tasmania.

Caterpillar 920 Loader

A second-hand loader will be procured and modified for underground operations.

A primary function will be to act as a backup loader for the Eimco 913 in the event of prolonged breakdown. A modified bucket of nominal capacity of 1.5cu.metre can be fitted for loading and haulage. It can also be utilised in a loader capacity from the stockpile and in clean-up activities.

The unit will be designed for quick attachment fittings which can accommodate a work platform for scaling down high faces and charging of faces with explosives. A bulldozer blade can be fitted for road maintenance and light bulldozing, and a forklift and crane boom attachment for work around the mill area.

6.5 MINE MANPOWER

The basis of the mining operation rests upon a high level of mechanisation and a minimum manpower component.

The operations manning would consist of the following:

>   Mine Manager
>   Technical Assistant
>   Diesel Mechanic/Fitter
>   Underground Miners – 8-10 men
>   Mill Operators   – 4 men

The crew would be supervised by the Mine Manager who would be responsible for operations and safety. He would be assisted by a technical assistant who would carry out surveying and grade control and generally assist the Manager.

A diesel mechanic/fitter would be responsible for scheduled maintenance and breakdown. He would be assisted by other operators as required. Major breakdowns, or repairs would be handled by outside contract. Electrical servicing would be carried out by contract on a call-out basis using a local electrician.

The mill attendance requirements are small with the removal of concentrates to the drier, and to the dry separation section being the main man handling requirements. It would be anticipated that two men would work on day shift, and two men on afternoon shift.

Extra LHD shifts would be required as haul distances become extended after the first two years of the mine life and production increases. Some overtime hours would handle the problem for a time, but a second full shift will be necessary eventually together with a second LHD.

Costs have been assessed on a payroll basis, but it would be intended to contract the work to separate parties for mining and milling work, and they would look after their own on-costs such as leave, levies etc. A high daily rate would result, but there should be substantial benefits to the Company.

6.6   LOAD-HAUL-DUMP OPERATION

The short haul distances from the working face to the crusher bin, or stockpile area do not require the use of a haul truck. The loading and haulage duty can be handled by an LHD unit.

The incline ramp to gain access to the southern end of A Lens is some 70 metres long. A further 70 metres undercut drivage will be required to daylight at 275 RL which would become a future portal/dump point. This will shorten the haul distance significantly. Virtually the whole of A Lens will be within 200 metres haul from this point. Some two thirds of the higher grade mining resource are within 100 metres haul.

In the case of B Lens, an initial portal will be created at RL 255. This is 100 metres from the crusher bin. All the B Lens mining reserve will be within 100 metres of this point. This would indicate a minimum 100m haul and a maximum of 200m. Once openings are made to the old open pit face on the exterior access ramp the haul distance will shorten significantly to 150 metres and become progressively less.

After a brief development period haul distances for A Lens will be within 100 metres, and B Lens beyond 100 metres. B Lens will shorten to within the 100 metre range, and later A Lens haul will extend beyond 100 metres.

The NE A Lens extension would require a haul of beyond 200 metres and LHD productivity would drop away. A second LHD unit will be required before production commences in this area and capacity should be available even at the 500 tpd level.

6.7   **PRODUCTION STOPING**

CUT AND FILL

On completion of the undercut over sufficient working area filling operations can begin. Fill will be introduced to a depth of 1.5 metres leaving 1.5 metres between the fill surface and the back. This leaves sufficient room for personnel to move around on the fill to move fill lines. On completion of the filling phase drilling and blasting of the back can commence.

A 2.0 metre slice will be taken across the full 6m width of the face using horizontal holes. These will be drilled in two rows of 7 holes per row to a depth of 4.1 metres. An advance of 90 percent, or 3.69 metres would be anticipated.

The holes would be blasted using half-second detonators and Amex explosives with AN60 primers. A blast should result in 117 tonne muck pile.

At a mechanical availability of 80 percent over a nominal 7 hour shift 5.5 hours should be available for drilling. The productivity should be more than 4 faces after allowing for face moves.

6.8   **FILLING**

The sources of available fill are tailings, small amounts of waste mined underground, and surface waste from an area of possible open pit ore adjacent to the 275 RL portal.

Tailings will be the main source of fill with any available waste used for fill walls prior to introducing fill. Deposited fill can be rehandled by the Cat 920 using a bucket, or its bulldozer blade to also build fill walls. Some dry fill can be back loaded by the Eimco 913.

Tailings will be pumped from the tailings launder to a cyclone to thicken the tailings to 70 pecent solids. This will be basically a dewatering exercise as the minus 37 micron fraction will not be used and the fine fraction will be deslimed to minus 37 micron.

It is estimated that 67 percent of the tailings will be plus 75 micron and 84 percent plus 37 micron. The total of usable tailing from 795 000 tonnes of processed ore will be 668 000 tonnes. This indicates that the tailings dam must have a capacity for 398 000 tonnes of tailing with the other 396 000 tonnes being returned as fill.

Fill will probably be required some 6 to 7 months after ore production begins for the first lift of 2 metres. Fill will then be required on a regular basis as undercut stoping, breast stoping and filling can be scheduled in various sectors of the mine.

Filling would be carried out over the two shifts of the milling operation. Only a periodic inspection would be required to move the discharge pipe. Discharge rates will be about 200 litres per minute. This should be achievable via 50 mm polypipe.

6.9 VENTILATION

It is proposed to use the 255 portal and the 265 portals as main intake airways with a ventilation exhaust at 280 RL to the north. Workings will extend north from the 255 portal and north from the 265 portal.

The development undercut phase of mining will give priority to establish the connection. The ventilation will be eased by making breakthroughs to the open pit for temporary ventilation of B Lens section.

As workings extend eastwards and northwards the ventilation can be extended on the panel system with temporary or permanent stoppings to control the airflow.

The main ventilation demand will be for the Eimco 913D diesel LHD with a 10 000 cfm requirement. The Caterpillar 920 unit will require 8 000 cfm.

The mine is small in area with wide openings with a low friction factor. An economical low water gauge fan delivering around 40 000 cfm will be adequate for the main circuit once established.

Small blower fans will be required for the earlier stages of development before a circuit is established.

6.10   **MINE DRAINAGE**

The open pit workings have created a low point in the topography which has been dewatered down to the level of the lower open pit benches.

The lowest level of the proposed mine workings is about 250 RL which is the same level as some of the nearby open pit benches. The topography then falls away towards the Groom River.

The mine will be developed by adit entrances and generally worked up the dip. The make of water into the workings is expected to be principally rain water seepage from surface. Ground water inflow is expected to be minimal as the ground water mound is expected to be below the level of the workings. The workings are expected to be dry.

Water used in the mine during drilling and filling operations will discharge from the mine via the adit entrances. The initial entry will be made on the 255 RL and it is anticipated the water will drain from this entrance throughout the mine life. The tailing dam settling pond is below 255 RL in early years and mine drainage can discharge directly to the tailings pond.

Drilling operations with the jumbo will require 60 litres per minute while working. The filling operations will take tailings as fill and dewater to 60 percent solids before pumping underground. The rate of filling will be 40 TPH of dry solids which will require water at 430 lpm. Drainage from the workings will be at a slower rate and is not likely to exceed 200 lpm. Thus maximum mine drainage from water used in operations is likely to be 250-300 lpm.

6.13 EXPLORATION DRILLING PROGRAMME

A diamond drilling programme will be required to test for orebody extensions, and to further test low grade zones.

A small surface programme will be undertaken to test an area to the north of the old open pit face and west of A Lens. A potential 25 000-30 000 tonnes open pittable ore could be available. Some 200 metres of NQ drilling is indicated.

A few surface holes would be useful to close up the grid spacing in a few areas of A and B Lenses. These would give additional grade, and footwall information to guide the undercut stage of stoping. Some 400-500 metres of drilling may be required.

Underground diamond drilling will be carried out to test the lower grade A Lens mineralisation at the northern end of the orebody, and seek out any higher grade portions suitable for mining. A programme of 10 holes averaging 50 metres drilled from underground opening would be contemplated towards the end of year 2 of production.

6.14 PRODUCTION SCHEDULE

The mining reserves based on indicated and inferred resources are assessed at 856 230 tonnes at 0.51% Sn.

A production plan at a stoping cut off of 0.3% Sn has been drawn up.

Year	Tonnes	Grade	Contained Sn
1	100 000	0.60	600
2	100 000	0.60	600
3	110 000	0.55	605
4	115 000	0.52	598
5	120 000	0.50	600
6	125 000	0.46	575
7	125 000	0.43	538
	795 000	0.52	4 116

6.15 AVERAGE PRODUCTIVITIES AND COSTS

The productivity and costs of the two stages of the mining operations are quite different. The undercutting operation is basically tunnelling while the cut and fill stage breaks to a free face with consequent savings in drilling and blasting costs, and much greater productivity for the same activities.

The undercut stage would theoretically produce a total of 131 220 tonnes from the mineable reserve with some 243 420 tonnes being produced from the cut and fill stage.

Once the undercut has progressed sufficiently to permit filling to commence, both undercut and cut and fill operations will be carried out concurrently. Planned scheduling would then only require a blast from an undercut and 2-3 breasting rounds to produce the required daily tonnage of 400-500 tonnes.

The proposed drilling jumbo would be able to handle the drilling requirement on a single shift once undercutting and cut and fill stoping are in balance. In the undercutting phase two drilling shifts would be required.

The average costs will range from the highest level on undercutting only through an average of undercutting and breast stoping to breast stoping only in the cut and fill cycle.

7.1   METALLURGICAL BACKGROUND

The Anchor Mine operated for some 20 years at a significant level of activity at less than 0.2% Sn and was moderately successful.

The G S Bulletin No 38 on the Blue Tier Tin Field by McIntosh Reid and Henderson drew upon work by Lewis, the Anchor Mine manager, in the area of metallurgical treatment. Observations included the following:

*   The cassiterite was as a rule of a coarse grain size, and of a similar grain size to the gangue minerals.

*   The greisen host rock is soft and granular and is easily crushed and pulverised.

*   The felspathic granite is soft and clayey.

*   The ore is wet and sticky causing choking of crushers.

*   A general conclusion was that 20 percent of the ore was plus 30 mesh, 40 percent plus 120 mesh and 30 percent minus 120 mesh.

*   The bulk of the cassiterite is plus 100 mesh (147 micron) and free in crystal aggregates.

*   The bulk of the gangue consists of feldspar and kaolin together with quartz.

*   Sulphide minerals are insignificant.

The various recommendations put forward were:

*   The ore material should not be crushed finer than the cassiterite grains.

*   Pulverising is not desirable.

*   Classification through all stages is desirable.

*   Ore feed should be deslimed before concentration.

*   Difficulties are likely to be encountered in recovering cassiterite from the slimes.

Lewis concluded that reduction to 20 mesh should be sufficient followed by classification before concentration. McIntosh suggested reduction to 10 mesh with the use of rod or tube mills for any further reduction if found desirable.

The production records for 1906-07 indicated a tin content of concentrates of 71 percent (68.8 percent on realisation) from a feed of 0.105 percent Sn. The adjacent Liberator Mine dressed concentrates to 73 percent.

Renison carried out metallurgical investigations as part of a Pre-Feasibility Study. Crushing, grinding and gravity separation were examined from four bulk samples (August 1981).

The results from this work are summarised as follows:

* Heavy liquid testwork on 4 bulk samples and drill core samples indicated considerable variation in tin distribution between size fractions, between specific gravity products for the total sample, and between specific gravity products within individual size fractions of the sample.

* Fifteen sets of data were plotted as mean particle size against tin distribution to the product using a specific gravity of 3.3 within that size fraction. The liberation sizes range from 190 micron to 560 micron. The conclusion was that the ore should be reduced to 80 percent passing 350 micron for gravity separation.

* The predicted recovery from a circuit based on spirals and tables was 89 percent into a concentrate assaying 52.8 percent Sn. The principal contaminant was topaz with some base metals.

* Desliming was considered unnecessary before spiral concentration and could result in a loss of tin.

* Loss of tin to slimes were indicated at 7 percent at -150 micron and 5 percent at minus 106 micron.

* Good recoveries and grades were made from samples that were considerably coarser than that indicated from liberation size.

The results of Renison's work is of variance in some respects to the observations of Lewis and McIntosh. Lewis indicates a reduction to 20 mesh (833 micron) was sufficient, while McIntosh recommended 10 mesh (1655 micron). They were also firmly of the opinion that classification was required becuase of the slimes content.

The Renison bulk test work was carried out on the less intensely greisenised granite which was also less well mineralised. Thus grades were between 0.15-0.25 percent tin. The grain size would be expected to be smaller in these lower zones. The ore high grade sample 3A was not used, but this was very much coarser - as would be expected.

A matter open to question is the actual treatment of the bulk samples and the diamond drill cores used to determine liberation size. Cassiterite is notorious for fragmenting and sliming during grinding, and over-grinding is to be avoided at all costs. There is no indication of the passing size adopted for the grinding prior to the heavy liquid tests. Some cassiterite could have been ground rather than simply liberated. A pointer in this direction is that bulk sample 3B have good grades and recoveries despite the grind being 80 percent passing 400 micron rather than the indications of 190 micron from the liberation size tests.

Historically in tin concentrators staged grinding and separation have been the rule to avoid over-grinding and excessive fine tin losses. The proposed Renison flowsheet does not adequately consider this problem and emphasises a single liberation size figure in the interests of plant simplicity. Classification is suggested as unnecessary but this again goes against normal practice, and also ignores the fact that the bulk tests were not carried out on the more highly altered upper levels of the greisen where more fines could be anticipated.

The Renison tests were valuable in that good recoveries of 80 percent and upwards were made from low grade feed. Only three tests were carried out and some variation could be anticipated as the mass of cassiterite involved in the test is quite small, and tends to collect in the equipment. The tailings results may well be in the threshold zone in that tailings value have reached the lowest level attainable. In tests 1, 2 and 3B these are 0.023, 0.048 and 0.022 percent Sn. A value of 0.025 percent Sn would be regarded as an excellent result.

Renison Limited report August 1981 reported jigging tests carried out at Mines Department Launceston on Bulk Sample 1. Further testwork was planned, but no record of testwork can be located.

The results of the first series of tests can be summarised as follows:

Size (μ)	Recovery %		Tailings % Sn	
	Test 1	Test 2	Test 1	Test 2
+425	84.84	69.77	0.04	0.07
+300	87.69	83.60	0.03	0.04
+212	94.47	89.32	0.02	0.04
+150	95.54	93.70	0.02	0.03
+ 75	80.60	94.09	0.10	0.03

Head grade 0.35% Sn.

Test 1   used $\frac{1}{2}$" ragging
Test 2   used $1\frac{1}{4}$" ragging
Top size was not indicated, but believed to be 1.2mm.

These tests suggest that the +425 fraction was not completely liberated, but overall the results were excellent demonstrating a very favourable response.

Renison used spirals as the main concentrating equipment followed by tables for up-grading. Jigs were considered, but rejected for no particular reason. The expectations from a circuit based on spirals and tables was 80 percent recovery into 52.8 percent tin concentrate from a feed of 0.27 percent tin. The predicted analysis was as follows:

Element	Assay %	Range
Sn	52.8	50-55
Fe	3.6	1.2-5.9
S	1.5	<0.1-3.0
As	<0.01	
Cu	0.6	0.03-1.2
Pb	0.1	0.07-0.12
Zn	1.1	0.08-2.1
Ag	86	5-166
Mo	0.2	0.14-0.26
Bi	0.22	0.09-0.35
Wo_3	1.0	0.4-1.5

Renison did not contemplate any further upgrading before sales. The operation now being examined would be prejudiced in the sales area and further ugrading is seen as essential. Drying of the concentrate and the use of high tension would remove the major contaminates of topaz, and biotite, magnetite and wolfram would be removed by magnetic separation. Sulphides may be removed either by table agglomeration during concentration or by cleaning the tin concentrate by flotation prior to drying. A final concentrate grade of 70 percent tin should be achievable with very low levels of contaminants.

By products would be a somewhat mixed sulphide concentrate containing silver, and small quantities of wolfram.

Spectrum carried out a number of tests at the Mines Department laboratories in Launceston. Samples were obtained from low grade B lens, low grade A lens, high grade A lens, and the high silver zone above A lens. The samples were obtained from half core stored in Burnie and at Renison.

The objective of the testwork was to explore the following aspects of the proposed flowsheet.

* Jig concentration at 2.4mm and 1.2mm passing sizes.

* Table concentration of jig concentrates.

* Table flotation of sulphides from jig concentrates.

* Magnetic cleaning of table concentrates.

* Differing responses of ore from the four zones.

Two tests have been completed on low A and low B lens material. The overall Sn recovery was 65.6% and 47.9% respectively.

The jig tailing at 2.4mm passing size was reground to pass 1.2mm and further concentration took place. The recovery on first pass for low A was 49.7% and a further 15.9% on retreatment. For low B the figures were 28.6% and 19.3% respectively.

The final concentrate grades achieved after jigging, tabling, table flotation and magnetic separation were:

	Passing Size	Concentrate Grade % SN
Low A	2.4mm	70.7
	1.2mm	60.2
Low B	2.4mm	68.6
	1.2mm	62.0

Higher levels of contaminants particularly topaz were reported in the retreatment runs.

The bulk sulphide concentrates produced from table flotation were assayed with the following results:

	Low A	Low B
Pb	0.15%	0.07%
Bi	0.64%	0.84%
Cu	15.4%	10.7%
Zn	19.1%	22.2%
Ag	1080ppm	1140ppm
Sn	0.33%	0.18%
Mo	0.72%	0.11%
Sb	170ppm	170ppm
As	1850ppm	1450ppm

Conclusions

The results from these initial tests lead to the following conclusions.

Jigging at coarse sizes of +653 micron is recovering substantial amounts of the available tin cassiterite which can be subsequently upgraded to near shipping grade. The primary tailings were in excess of 0.1% Sn indicating lack of liberation, and that the figure of 330μ for an adequate liberation size as determined by Renison would appear correct.

Table flotation to remove sulphides into a bulk concentrate should be successful with high concentration of Ag, Cu and Zn. High Bi and Mo may cause problems in sales of mixed concentrates. The economic advantages of making selective copper and zinc concentrates should be examined.

The desired shipping concentrate grade of 70% Sn would appear achievable following sulphide removal, magnetic separation, and high tension separation.

Jigging tests conducted by Renison indicated very good recoveries on well sized material even down to 74 micron. This suggests that separate units be used on a more closely sized range of feed than a single unit treating a wide size range.

7.3   FLOWSHEET DERIVATION

General Concept

The ore is likely to be sticky due to alteration products such as kaolin, but reasonably easy to crush. The slimes content is significant and slimes removal should take place prior to concentration. Overcrushing, or grinding of the cassiterite should be avoided. Wet screening will be required in the circuit. A simple and inexpensive circuit is required to minimise capital and operating costs.

The critical question is the required liberation size. The Renison testwork indicated that the ore should be reduced to 80 percent passing 350 micron for gravity separation. The Spectrum jigging tests indicated that a large proportion of the cassiterite is free at much coarser sizes. It was decided to adhere to the Renison recommendations of 80 percent passing 350 micron, which translates to 100 percent passing 500 micron.

The circuit proposed would involve crushing through a jaw crusher, and feeding to a Barmac Mk II Duopactor through a 50mm grizzly. The Barmac crusher will be in closed circuit with Mogensen screens screening at 1.0mm. The undersize will flow to a sieve bend screen designed to pass an 80 percent minus 350 micron undersize. The actual product should be all passing 500 micron indicating a bar spacing of 550 micron. The oversize from the sieve bend screen will join with the oversize from the Mogensen screens to return to the Barmac crusher.

The minus 500 micron product will pass to a second sieve bend screen to effect a separation at 200 micron. The oversize will pass to a small jig. The undersize will pass to a large sump.

The minus 200 micron product from the sump will be pumped to a rapifine sieve bend screen to effect a separation at 75 micron. The oversize product will again pass to a jig with the oversize going to tailings.

The jig concentrates from each jig will go to a table having first been conditioned for table flotation. The sulphides will be removed and stored. The tin concentrates will be dried.

The dried tin concentrates will be screened into various sizes and batch treated by high tension separation followed by magnetic treatment of the conductors. An 85 percent recovery of cassiterite should result. By product bulk sulphide concentrates of copper-zinc-silver may be saleable as is, or a selective flotation into separate copper-silver, and zinc concentrates may be worthwhile.

The concentrator plant will have a nominal capacity of 26 TPH throughput on two shifts to process 400 TPD of ore.

Increases in production beyond 400 TPD will require extra working time. This can probably be achieved by working a Saturday shift.

7.4   **FLOWSHEET DESCRIPTION**

Primary Crushing

Good fragmentation is anticipated from undergrouhd mining and oversize is not thought to be a problem. A 15x30 Blake type jaw crusher with 50-60 HP motor with 50mm openside setting would handle the duty.

The crusher would be fed from the coarse ore bin via a feeder to control the feed rate.

Screening

The jaw crusher product will be fed to a grizzly screen via a conveyor belt. This screen will pass 50mm material. Oversize will return to the jaw crusher. Provision will be made to bypass minus 50mm material to a stockpile for reclaiming on a weekend shift.

Secondary Crushing

The minus 50mm material will be fed to a Barmac Mk II Duopactor via a feed bin. This is a type of impact crusher depending upon the impact of rock on rock. The product size is determined by the closed circuit screen as the crusher has no setting adjustments. The product size is theoretically infinitely small as rock is recirculated until it passes the desired size. This leads to very high circulating loads and high installed horsepower. The principal attraction is that a ball mill will not be required as the Barmac can produce the desired sizes.

The horsepower required will be 200 HP.

Screening

The Barmac product will be elevated to twin Mogensen screens which will wet screen at 1.0mm. The oversize will be recirculated.

The Mogensen undersize will pass directly to a sieve bend screen screening at 500 micron. The oversize will be recirculated to the Barmac. The undersize will pass to a second sieve bend screen to screen at 200 micron. The oversize will then pass to a small jig. The undersize pulp will pass to a concrete sump. The minus 200 micron pulp will be pumped to a third sieve bend screen of the Rapifine type to screen at 75 micron. The 75 micron oversize will pass to a second jig and the undersize to waste.

Sieve bend screens are preferred to cyclones as a sized feed is produced rather than a classified feed. This will enhance the efficiency of jigging as the heavy minerals can take full advantage of the specific gravity difference during jigging which is itself a classification process.

Testing has indicated that very little tin has been found in the minus 75 micron fraction. Should plant experience indicate that there is economically recoverable tin present a cyclone and slime table treatment could be installed.

Jigging

Two small jigs are preferred to treat the minus 500+200 micron fraction, and the minus 200+75 micron fraction. Improved recovery can be anticipated on more closely sized feeds, and jig settings can be altered to suit the feed material.

The jig concentrates will consist of cassiterite, sulphides and heavier gangue minerals. Concentration ratios are expected to be high in the region of 50:1. The jig concentrates will pass to table concentration.

Flotation and Tabling

The jigging concentrate will be conditioned with xanthate and cresote, or similar products. The conditioned feed will be fed to the table where sulphides will float off on the water surface to report to the bottom of the table. Cassiterite will be concentrated in the tabling process together with accessory heavy minerals. Quartz tailing will be rejected to waste with middlings being recirculated.

The cassiterite concentrates will be sent to the drier for further upgrading by dry processes. The sulphide concentrates will be stockpiled for sale, or further treatment into selective copper and zinc concentrates.

Drying and Screening

The concentrates amounting to about 6 tonnes per day will be dried in a small oil or gas fired drier. They will be collected and then screened at 600, 300 and 200 and 150 micron. Sufficient stocks of each size will be accumulated for batch treatment.

High Tension Separation

Sized batches of concentrate will be teated by high tension separation to separate the conductor minerals from the non-conductor minerals. The cassiterite, wolfram and magnetite will report as conductors while quartz, topaz and other silicates will report as non-conductors. The smallest commercial unit should have sufficient capacity for this duty.

Magnetic Separation

The conductor concentrate from the high tension separation will be treated over a small magnetic separator of twin disc type to give a final cassiterite concentrate, wolfram by product, and discard magnetite.

Cassiterite Concentrates

Concentrates should be produced at about 3 tonnes per day at plus 70 percent tin content. Batches of final concentrates will be sampled. When assayed and found up to specification batches will be mixed, and then weighed out into one tonne bulker bags for shipment. Each one tonne lot will be sampled as a shipping sample.

Mill Control

Tailings streams will be sampled and visually assessed for cassiterite using the zinc block assay. Shipping grade samples will be assessed with the burette assay and zinc block with final shipping samples being sent away for laboratory assay before shipping out. Check tailing assays will also be performed by an outside laboratory.

Tailings Production

Tailings production will be at the rate of 26 TPH or 208 TPS.

The estimated grading size of this material is as follows:

Size micron	%
−500 + 300	16.7
−300 + 200	16.7
−200 + 100	25
−100 + 75	8.3
− 75	33.3
	100

SIMPLIFIED FLOWSHEET

R. O. M.

Jaw   Crusher

o/s

Grizzly (50mm)

Barmac
Crusher

o/s   Mogensen
Screen   (1000 micron)

o/s   Sieve   Bend
Screen   (500 micron)

Sieve   Bend
Screen   (200 micron)

o/s ———————————————————— u/s

JIG   SUMP

Conc   Tail   Rapifine
Sieve   Bend
Condition   Screen   (75 micron)

o/s   u/s
Table Flotation
Jig   Waste

Tin   Sulphide   Tail   Conc   Tail
Conc   S/Pile

Condition

Table   Flotation

Tin   Sulphide   Tail
Conc   S/Pile

Dry ——— To   Dry   Tailings
Separation   Dam

Dry

Screen

Batch Treatment

High   Tension
Separation

Non Conductors   Conductors

Magnetic   Separation

Magnetics   Non Mags

Final Sn. Concentrate

7.5   MILL LAYOUT

The coarse ore bin and stockpile will be situated close to the 265 portal to A lens so as to maintain as short an underground haul as possible. The crushing and screening plant will be situated at a lower elevation to take advantage of the fall in gravity.

The crushed ore will be pumped from the sump to the concentrator section which will be situated on a flat area just below the crushing and screening plant.

The ore feeding and crushing sections will be uncovered with overhead cover on the jig and table section. The dry concentration section will be totally enclosed. A weighing and tin concentrate store will be adjacent to the dry separation area.

7.6 METALLURGICAL PRODUCTS

Cassiterite Concentrates

The concentrate is anticipated to assay a minimum of 70 percent tin with impurities being trace topaz and quartz. The concentrates would be packaged into one tonne non-returnable bulker bags. Annual production is expected to be:

	Tonnes Concentrate	Tonnes Tin in Concentrate
1	730	510
2	730	510
3	735	515
4	730	510
5	730	510
6	700	490
7	650	455
	5 005	3 500

Sulphide Concentrates

The silver values are quite high in some areas of A lens but relatively insignificant in others. The average grade for A and B lens is 10.5g per tonne. The values of copper and zinc average about 0.05% for each metal. There is also some pyrite present with trace amounts of bismuth, molybdenite and tungsten.

Bulk concentrates are likely to be in the range Copper 10-15%, Zinc 19-22%, Silver 1100g/tonne, Bismuth and Molybdenum levels may prove a problem.

The feed to the mill at 100 000 TPA rate would contain in excess of 33 000 oz silver, 50 tonnes zinc and 50 tonnes copper per year. The recoveries are not known, but likely to be high based on flotation response and a 75 percent recovery could be anticipated.

Tungsten

Tungsten is noted in the ore, but the mineral has not been specifically identified. It will be of high S.G. and carry through to the dry concentration section where it would be separated.

04

R.L.
(m)

— 265·0
— 260·0
— 255·0
— 250·0
— 246·5

262 RL

Pond Area

260

←— Groom River

STARTER DAM

SEEPAGE
COLLECTION
DAM

5 cm

0   50 metres

1:1 000
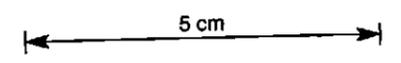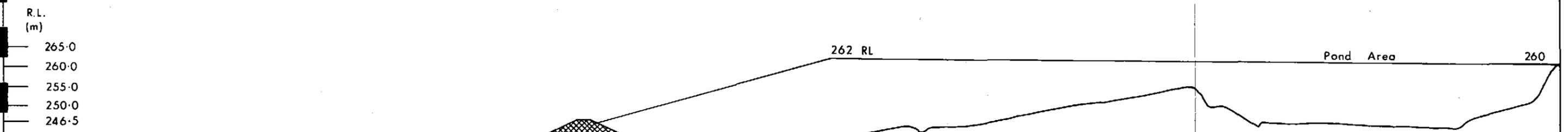
SPECTRUM RESOURCES AUSTRALIA

ANCHOR MINE

TAILINGS DAM

LONGITUDINAL SECTION
THROUGH DAM

8.   THE TAILINGS DAM

8.1   TAILINGS DAM DESIGN

The tailings dam design has considered a number of factors the most important being:

*   Final abandonment considerations.

*   Integrity during operational phase.

*   Method of tailings deposition.

*   Water management.

*   High intensity storm events.

*   Drainage of the tailings.

*   Revegetation.

*   Tailings characterisation.

All these matters are inter-related to a greater or lesser degree.

The basis of design consideration is to plan for final abandonment at the outset. In its simplest terms this requires the structure to be self-draining to avoid any build up of the phreatic surface where the tailings become saturated and lose their cohesiveness. This can be achieved by adequate drainage under the base of the tailings structure, and drainage from the surface of the tailings area during high intensity rainfall events. Similar considerations also apply during operation.

The structural stability is improved by appropriate tailings deposition techniques, so as to increase the density of the material and reduce the percentage of pore water. This reduces the chances of liquefaction under shock loading from seismic events, and also under intense rainfall.

Water management requires that excess water from storm events, and run off is appropriately channelled to discharge ditches, or decant towers to avoid increasing the saturation level of the tailings, or overtapping of the dam. Ideally, the minimum size of pond should be maintained on the dam, but this is modified by the desire to draw process water from the dam and maintain a closed circuit.

The character of the tailings is of considerable importance. An assessment of the size grading will influence the height to which dams can be raised and the slope angle of the face. The chemical composition will affect the quality of the discharge waters.

Tailings characterisation

The predicted tailings grading from crushing tests, and the required liberation size has been determined as:

Micron	%
−500 + 300	16.7
−300 + 200	16.7
−200 + 100	25.0
−100 + 74	8.3
− 74	33.3
	100.0

This is in the relatively coarse range for tailings disposal and storage. Samples of the −74 micron fraction from the crushing tests were tested for settling behaviour as it was suspected that as the site waters were near neutral pH slimes would be reluctant to flocculate and settle. This was found to be the case. Treatment with alum and lime were found to be effective in altering the pH, and effecting precipitation. Alum treatment was not well regarded as this creates an acidic environment. Lime treatment at an addition of 150ppm raised the pH to 10.4 and gave effective flocculation and precipitation.

Sedimentation rates of 2.5 metres per hour were obtained with 80 ppm solids in the supernatant after 20 minutes and zero after one hour. Final pulp densities of 58 percent solids were achieved after 15 minutes.

These tests indicate that excellent depositional behaviour can be expected from the fine fractions. Admixture with the coarse fractions and beach deposition would further improve the storage characteristics by giving higher densities.

The chemical characteristics are notable for the very low levels of sulphide minerals, and of base metals. Copper and zinc do not exceed 0.1% combined. There are low levels of other metals and some pyrite. The sulphide content will be substantially removed during processing by flotation with the products being sold for their silver, copper and zinc content.

The results of this will be to make the tailings virtually sulphide free, and there will be no opportunity for acid leachates containing heavy metals to be created. This supposition is to be confirmed by analysis of tailings material from testwork and process water. The results will be discussed more fully in the EIS Addendum.

Drainage of the tailings structure

The depth of tailing on completion of the project should not exceed 20 metres above original surface. The shallow depth and coarse size grading will create good permeability and drainage to the base of the structure.

The underlying land surface is mainly solid open pit bench with a shallow cover of silt and vegetation close to the final open pit faces. Proceeding downstream the Anchor Creek bed is progressively filled with rubble and overlying fine material. When the old miners worked their way up from the Groom River by sluicing the eluvial deposits the fines were washed downstream with rocks being picked and stacked in terrace layers adjacent to the ground sluices. As work progressed upstream fines were deposited over the coarse cobbly material. This has created excellent drainage, and soil profiles which have contributed to the rapid revegetation of the area.

Advantage is to be taken of the natural site conditions to enhance the base drainage of the tailings dam and create a flow through structure for eventual safe abandonment. The objective is to encourage vertical drainage through the tailings pile to the base. Water would then flow along the base taking advantage of the site conditions. Additional drainage will be provided by removing the silty layer adjacent to the proposed starter dam and laying down a rock drainage filter which will pass beneath the starter dam wall. The starter dam will be of rock. It will not be sealed on the upside face other than by progressive deposition of the coarse tailing fraction. The objective here is to encourage horizontal drainage beneath the tailings and the dam wall and minimise any build up of the phreatic surface.

It is believed that the flows through the structure will be rapid and that some difficulty may exist in maintaining a tailings pond. This may be particularly difficult in the early period of the operation when a large proportion of the process water could be lost. To obviate this to some extent two measures would be taken.

Sediment Collection Dam

This will be a small dam downstream from the main starter and tailings dam wall. Its function is to intercept water draining from the tailings dam and to collect any sediment eroding from the tailings dam face. It will also double as a sediment trap during the construction phase. This small dam will be designed to hold water. Discharge will be via a spillway. This will be a discharge point for monitoring purposes.

The ponded water can also be pumped back to the processing plant for reuse.

Temporary Fines Bund

In the initial start up of the operation and early management of the tailings dam it is very desirable to ensure that the coarse tailings fraction is deposited adjacent to the starter dam wall and the fine fraction deposited upstream to the back of the dam area. Again, advantage will be taken of the site topography to construct a bund across a narrow part of the Anchor Creek between two old pit benches. It is proposed to impound the fines fraction of the tailings upstream of the bund. This will give opportunity for the beach deposition of the coarse fraction to become well established upstream of the starter dam wall to ensure a good drainage profile. The bund and impounded fines will eventually be overtaken and buried beneath the rising level of the whole tailings dam.

It is expected that a pond of water can be maintained behind the temporary bund for plant use until incorporated into the fabric of the whole dam structure.

The Tailings Pond and Run off Control.

A tailings pond will be maintained at the rear of the dam area for recirculation of plant water and to assist in precipitation of ultra fines. Even though the pond will be in the finer fraction area of the dam difficulties would eventuate in maintaining the pond due to rapid drainage. The pond will be adjacent to the discharge outlet from the dam. This will discharge surplus water in winter and heavy rainstorm events.

A discharge channel will be cut through a narrow rock abutment at 246 RL on the western side of the tailings area to discharge to the water storage area. As the tailings dam increases in height the discharge will be raised by a stop log weir. This discharge side will be progressively raised by broken rock and concrete fill to leave a permanent structure on abandonment. At higher elevations a new channel can be cut further to the rear of the dam to form the permanent discharge on abandonment.

Tailings Dam Construction

The main structure, the starter dam, will be a rock construction across the Anchor Creek. A trench will be excavated across the incised portion of the creek bed and filled with rock fill. This is to ensure that no fines layers be immediately beneath the dam structure which could be subject to piping.

The base of the rock dam will be RL 238, and the crest will be 246.5 RL. Some 10 000 cu.metres of rock fill will be required for the construction. Placement will be by end dipping trucks and consolidation by bulldozer. The rock fill will be obtained from a small quarry opened in one of the existing open pit benches.

A rock drainage filter will be incorporated into the structure that will be placed upstream of the dam and pass under the dam to the downstream side. This will ensure drainage under the dam wall, and enhance the good drainage already existing.

The starter dam is a simple structure designed to impound tailings. It is intended to be permeable to ensure good drainage during operations and after abandonment.

The dam will be raised above the 246.5 RL by placement of coarse sand tailing in successive small lifts. Sand can be deposited by gravity pipe line in earlier years with pumping probably being required in later years as the gradient decreases with increasing dam height.

Tailings Deposition

The tailings will be divided into coarse jig and table tailings, and a dilute fine fraction. The latter will be of the order of 10% solids and minus 75 micron.

The coarse tailings will be directed by gravity or pumped pipeline to the upstream side of the starter dam. Sands will be placed so that a beach is built up allowing water to drain to the pond. The beach will be built up in thin layers, which will improve the sand density and dam stability by improved drainage and air drying. A packing factor of 1.4 tonnes per cu.metre should be achievable.

The discharge point will be moved around the wall as necessary to create the beach and also to maintain the desired slope angle as the height of the dam face is increased. The final height should be approximately 262 RL at the crest to maintain a gradient back to the tailings pond, and to the decant channel.

The finer fraction will initially be discharged behind a low bund at the rear of the tailings area to give an opportunity for the coarse tailings beach to become well established. When this is achieved the finer fraction can be discharged downstream of the coarse material to give a reasonable admixture.

Revegetation and Abandonment

The design and management of the tailings dam during the operational life is to permit abandonment to take place smoothly and with minimal additional work. The tailing dam structure is designed for abandonment at any time with stormwater discharge decants or channels in place, the total structure acting as a flow through structure with permanent vertical and horizontal drainage in place, and sediment trap dam in place with bypass spillway.

The starter dam rock face can be progressively vegetated and extended upwards on to the coarse tailings sand face. The outer edge of the face will have layers of straw and biograde textile netting incorporated to reduce erosion and provide a foothold for vegetation. This will be planted in the first instance with fast spreading ground cover which in turn will provide a medium for planted and naturally introduced species. Hydromulch and fertilisers may also be helpful.

On abandonment the surface of the tailings dam will require stabilisers to prevent erosion, and to provide a growing medium. The surface area will be roughly oval measuring 210 metres by 170 metres sloping from the wall to the pond and discharge. An area of roughly 3 hectares will require rehabilitation.

The area is already sloped for drainage but some contouring may be required to establish drainage lines which can be surface reinforced for permanent drainage by rock lining and other materials. The surface would then be mulched and seeded for a fast growing surface cover for stablisation and planted with native species of plants and shrubs. The tailings pond area adjacent to the decant will be dry in summer and reasonably wet in winter as the tailings depth will be shallow. This will become a seasonal wet land area and the appropriate plantings can be made.

Two years would probably suffice for development of vegetation to a point where formal abandonment handover can be requested.

9. WATER STORAGE AREA

A dam has been created by an old mining excavation on the western side of the old open pit. Access to the mining site was by means of an adit connecting with the base of the excavation. The adit was used for haulage and drainage. The upstream end of the adit is now partially blocked with rock debris. This has the effect of creating a dam and impounding water to a level of 238.6 RL. The outlet is 236.2 RL and implies a head difference of 2.4 metres. Water can be observed flowing from the top of the obstruction within the adit which would confirm this head difference in approximate terms.

Soundings in the dam indicate that some 5 metres of water exists with shallowing to 4 metres towards the sides. The floor of the dam is probably 223.6 R.L.

It is intended to utilise this existing water storage area and to increase the storage capacity for plant water requirements.

Ariel Creek flows into the depression and discharges through the mine adit. It is intended to divert Anchor Creek into Ariel Creek with the combined flow discharging into this water storage area. In addition, the decant from the tailings dam will discharge to the same area. The initial decant level will be 246 TL and this would limit the level of water storage at least in the early stages of operations.

It is intended to plug the adit and increase the impounded water level to 243.5 RL at which point water would overflow through an open pit gully to rejoin the creek below the adit outflow. This will result in an increase in level of 5 metres to give a total of approximately 10 metres. The storage will be approximately 26 000 cu.metres with a surface area of 2 550 sq.metres.

This proposal results in a lower storage than that proposed in the pre-feasibility study. This lower storage is quite adequate for plant requirements. The water management plan envisages that plant water will be reclaimed from the tailings dam in a closed circuit operation, and that settlement of fines will be successfully achieved in the tailings pond. It is further envisaged that only in the most extreme circumstances will the tailings decant be operative. Its function will be for operational contingency, and long term abandonment security.

In this way the Anchor Creek-Ariel Creek-water storage-overflow discharge will preserve the natural waterflow to the Groom River. The water storage itself will act as a sediment trap for the Ariel and Anchor Creek catchments. It will also act as a contingency sediment trap for any sediment escaping from the tailings dam.

The water storage area will be created to provide additional water storage for mill use particularly in dry summers, and during the early start up period.

10. SURFACE WATER MANAGEMENT

Existing Water Flows

There are two permanent water courses that lie within the area affected by the proposed mining operations. The first of these is Ariel Creek which discharges into the deep depression formed by previous mining activity. This discharges via an adit to eventually flow into the Groom River. It is intended to increase the storage capacity of the existing dam in the depression by plugging the adit. The overflow at 243.5 RL will be higher, but will rejoin the existing creek downstream of the adit to resume the same course.

The Anchor Creek has to be diverted high on the side of the open pit workings to join with Ariel Creek before discharging into the storage dam. The existing Anchor Creek will be required for the tailings dam.

Other water courses in the area flow and discharge away from the mine area.

The Anchor Creek diversion will be permanent as will the increased dam storage and overflow.

Tailings Dam

The tailings dam will impound tailings and process water for recirculation. Mine drainage water will also flow into the tailings dam.

Water may discharge from the tailings dam in extreme rainfall events via the decant which will prevent any overtopping of the dam wall. Otherwise the dam water will be in closed circuit.

Leakage is anticipated from the tailings dam structure as it is designed as a flow through structure. This leakage will find its way beneath the dam to the sediment trap dam downstream from the tailings dam wall. Water will then discharge via the overflow weir. Provision will also be made to pump back water from this collection dam to the tailings pond to maintain the tailings pond level.

The normal discharge point for water from the mining operation will be the spillway from the sediment trap dam. A second discharge point for contingencies is the decant discharge to the water storage dam. The tailings dam area is approximately 3 hectares, and the additional discharge under storm conditions should be relatively insignificant.

The proposed general water management plan achieves several objectives:

*   It frees the Anchor Creek for tailings storage.

*   Natural waterflows are maintained by permanent short diversions.

*   A water storage is created for plant make up water.

*   All water from mining activities is directed to the tailings pond.

*   A single operational discharge point is established.

*   Diversions, storages and discharges are of a permanent nature and planned for long term abandonment.

WATER MANAGEMENT

SCHEMATIC

Ariel
Creek

Anchor
Creek

MINE

Mine
Drainage

Mine
Fill

Precipitation

Plant
Water

MILL

Contingency Decant

Tailings
Slurry

WATER
STORAGE

TAILINGS
POND

Make Up Water

Make up
Water

Seepage

SEDIMENT
DAM

Groom
River
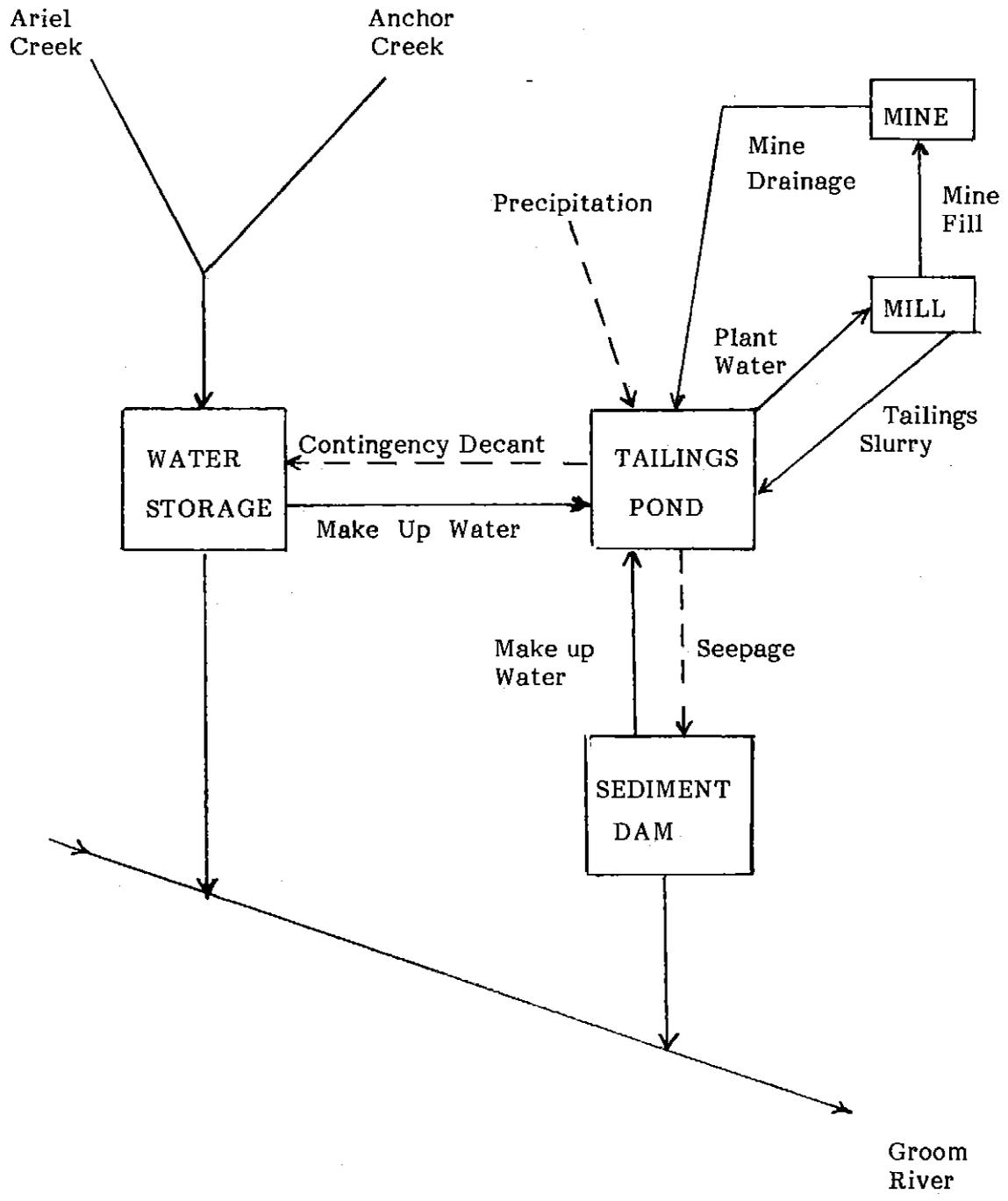
SPECTRUM RESOURCES AUSTRALIA
ANCHOR MINE

CATCHMENT AREAS
IN MINE VICINITY

0   100   200   300 metres
1:5 000

5 cm
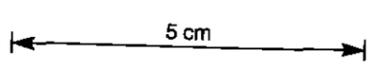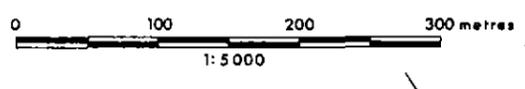
Anchor Creek Catchment
67 ha

Anchor Creek

Ariel Creek

Ariel Creek Catchment
15 ha

54 36 500 mN
54 36 000 mN
54 35 500 mN

5 85 500 mE
5 85 000 mE
5 84 500 mE

11. SITE FACILITIES

Access Roadway

The main access to the mine will be by an existing class C gravel road from the Tasman Highway known as the Anchor Road. The road is in a good condition and easily transitable.

A new access track will be formed off the Anchor Road on the flanks of Goughs Hill to access the mill and mine portal area. Existing tracks can be used early in the construction phase, but these will be redundant following construction of the tailings dam.

Water Supply

Water will be required for milling operations, mining and personnel usage. The demand for water for milling will be predominantly for jigging and tabling which is estimated at 225 lpm, mining usage for drilling and filling about 250 lpm while personnel use will be quite small. No internal mill recirculation will be attempted and clean feed water used from the tailings pond.

A 10 000 litre capacity head tank will be established above the mine site. Water would be diverted from upper Anchor Creek to gravity feed the tank with a back up pump supply from the water storage area being installed. This will be used for ablutions.

Power Supply

Power will be supplied by the HEC by means of an overhead line along the Anchor Road. A power line exists which comes from the main transmission line on the Tasman Highway some 2 km along the Anchor Road to supply two farm houses. This line will be upgraded and extended a further 2 km to the mine site.

Transmission will be 11 000 kv and a substation and transformer will be established on site.

Installed power demand is anticipated as about 375 KW. The bulk of the load will be the mill circuit with the Barmac crusher dominating with 40 percent of the load.

Administration

The administration of the mine will be carried out by the manager with technical and policy directions from the owners. The manager will be provided with a utility vehicle for his own use, and as a stores vehicle.

The stores holding will be under the control of the manager and the inventory will be held at minimum levels as almost all supplies will be readily available from north coast centres.

Accounting functions will be carried out by a local accountant for onward transmission to the owners.

Telephone and fax facilities will be available for ease of communication and a photocopier will be rented.

Basic draughting facilities will be supplied to enable the manager and his assistant to draw up and maintain mine plans for recording and control purposes.

Technical direction will be supplied by the technical director, and a bi-monthly visit will be the norm.

The simple mining and milling operations make it unlikely that any outside technical expertise will be required after the mill commissioning has been bedded dcwn.

A small office for mine records, plan drafting and plan storage and general administration will be required. A lunch room and changehouse facilities will occupy another section.

12. __MAINTENANCE__

The two principal areas of maintenance cover the mine equipment, and milling plant.

Mine equipment will be serviced by the diesel mechanic and operators on a daily basis which will attend to operating requirements. Machines will be steam cleaned and checked over on a Saturday maintenance shift when a full maintenance check will be carried out and repairs-replacements carried out. Major services will be carried out by the suppliers service team from Burnie and arrangements will also be made for exchange service parts.

Drill jumbo maintenance will also be carried out on site with the hydraulic drifters being serviced by the suppliers on an exchange unit basis from the depot in Burnie.

Any serious breakdown will require the services of the suppliers. This would require about a four hour travel time from Burnie.

Mill maintenance will be daily servicing, and end of shift and weekend maintenance. Principal areas will be sizing screens and jig screens, and cleaning of jig beds, replacement of pump liners, and wearing parts on crushers. Spare pump units will be held on site.

As a general rule stores inventory will be held to a minimum as virtually all components can be obtained on a 12-24 hour basis either from Tasmania or Melbourne.

The maintenance facilities will be very basic. It will consist of a concrete wash down pad where servicing will be carried out for mine units. A small service workshop with compressed air, welding facilities, power hacksaw, drill press, hydraulic press, and work bench facilities. A drill sharpening bench will also be required. The minimum of facilities will be provided initially which will be reviewed as experience dictates.

13.   MINE DEVELOPMENT SCHEDULE

The mining plan will be finalised by the time the necessary permits and licences are in hand. Also the milling plant layout will have been completed. A decision on financing will have been taken and if loan funds are required arrangements will have been made to put them in place.

Milling Plant

The crushing and screening section will be put out as a package and will probably go to the Barmac manufacturers. Fabrication is estimated to be 8-10 weeks with shipment to site taking another month. Erection would be rapid as equipment would be shipped in large modules.

The remaining equipment will be sourced locally and shipped in for erection in the fourth or fifth month.

Mine Equipment

The major mining plant would arrive in month 5 when a start will be made on the portal development while the mill is being erected.

Site Works

The first priority after road access will be construction of the water storage to store sufficient water for start up.

The Anchor Creek diversion cannot be carried out until this is completed. After completion of the Anchor Creek diversion, work can begin on the starter dam and sediment dam. Mill site levelling can be completed over this period.

Power Line

The power line extension and upgrading will be carried out by the Tasmanian Hydro Commission. They will also provide a suitable transformer and main switch. Power will be needed before buildings arrive on site.

Site Buildings

These will be sourced in the local area and will be the prefabricated type on steel bearers. Erection would be completed in 2-3 days. The exception will be the mill building and the maintenance bay building. These will have to be erected on site. Suitable standard storage shed/barns may be available.

Quarry-Open Pit

A small quarry will be required for provision of rock for the various dam constructions and site levelling. Utilisation of old open pit benches will be adequate. Some ore may be available from one of the open pit benches which requires confirmation by drilling. A combination of waste rock quarry and ore open pit may prove possible. Diamond drilling would be required very early in the construction schedule to confirm the presence of any open pit ore.

MINE DEVELOPMENT SCHEDULE

Activity	Month	N 1	D 2	J 3	F 4	M 5	A 6	M 7	J 8	J 9	A 10	S 11	O 12
Milling Plant		order			ship	erect							
Mine Equipment		order				portals							
Site Works			construct										
Power Line		erect											
Site Buildings		order		construct									
Production Stoping													
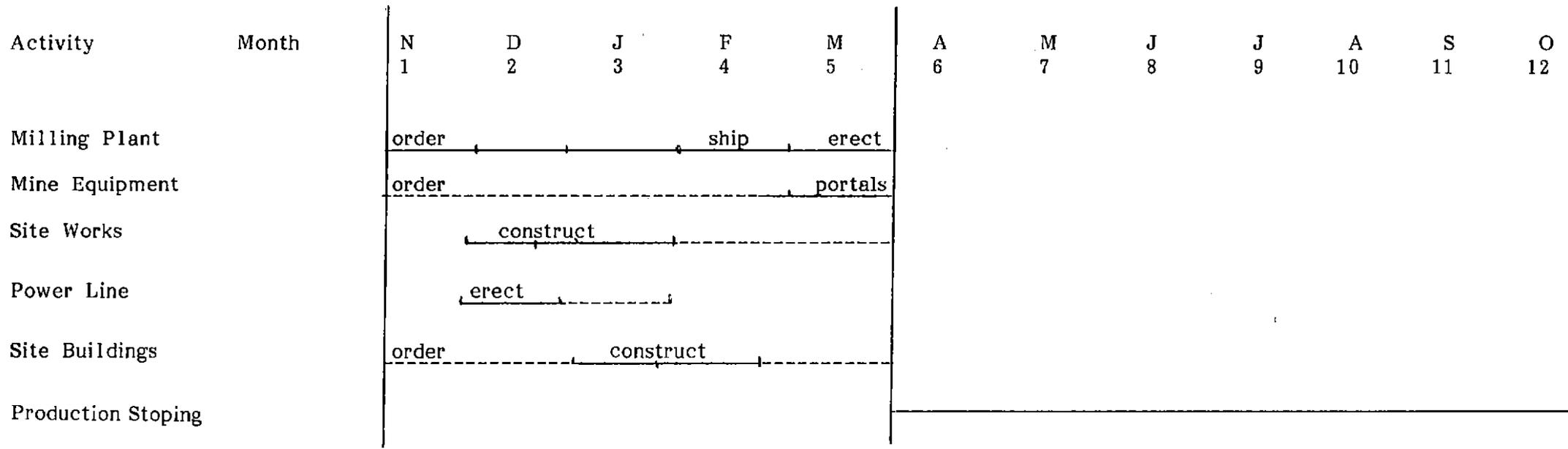
14. CAPITAL COSTS

The operation is of an unsophisticated nature with a simple mining method, and equally simple metallurgical treatment.

The requirements of site preparation, earth and rock works for the starter dam for the tailings dam, and sediment dam are relatively small. The Anchor Creek diversion is slightly more complicated from an access angle, but this and the other works can be handled by local contractors with site supervision.

The limited life of the operation does not justify permanent structures for buildings. The requirement will be for the prefabricated type for office and changehouse. The mill building and the maintenance shed will utilise the farm implement shed construction.

The mining equipment comprises three major items - the LHD unit, the drilling jumbo, and the Caterpillar 920 FEL unit. Prices are based on firm quotations from suppliers.

Milling equipment costs are based on quotes for the secondary crushing and screening plant with M & S indexed costs derived from CIM Mining and Mineral Processing Equipment costs 1982 for other items.

Installation of machinery apart from the orebin and jaw crusher will not require formed concrete foundations, only provision for bolting down as most will be free standing.

The contingency factor has been held to 10 percent due to the preponderance of the three major quoted pieces of mining equipment. Similarly, the same figure is used for milling as the flowsheet is finalised and equipment is estimated individually.

There is considerable scope for the use of used equipment in the milling circuit. This applies to the orebin-feeder-jaw crusher section together with conveyor belts. There is also every likelihood of acquiring jigs, tables and dry separation equipment at used equipment prices.

Arrangements can also be made for extended rental with option to purchase on the LHD and drilling jumbo to reduce front end costs.

CAPITAL COST SUMMARY

		$'000
14.1	Earthworks and structures	2 036
14.2	Power Supply	79
14.3	Surface Structures	315
14.4	Mine Equipment	904
14.5	Milling Equipment	1 494
		2 995

CAPITAL REQUIREMENTS

14.1 EARTHWORKS AND STRUCTURES

1.	Access Road. 100 m of new formation including 1 culvert and side drainage. <u>say</u> 15 days work at $650 per day	9 750
2.	Haulage ramp from 255 portal to orebin 100 m regrading and widening. <u>say</u> 3 days work at $650 per day	2 000
3.	Rock Quarry Preparation To clean up one of the existing open pit faces for use as a small quarry for rock for dam works <u>say</u> 2 days work at $650 per day	1 300
		13 050

TAILINGS DAM

4.	Starter Dam. Excavation of site to firm base rock. 5 days work at $650 per day	3 300
	Haulage and placement of 10 000 cu.m. of blasted quarry rock. Placement by rear dumping from a truck with consolidation by bulldozer. Use $8.00 per cu.m.	80 000
	Rock Drainage Filter Haulage and placement of 420 cu.m. of rock. 35 m long 12 m wide, 1 m deep. Use $8.00 per cu.m.	3 400
	Decant channel-drill and blast 20 m slot	6 000

5. WATER STORAGE AREA

Gravity Weir (if necessary) Site preparation say 2 days work at $650 per day	1 300
Placement and compaction of 500 cu.m. of graded rock Use $10.00 per cu.m.	5 000
Placement of concrete and rock facing requiring 25 cu.m. of concrete at $150 cu.m.	3 800
Sealing of old adit. Site preparation 3 days at $500 per day	1 500
Block work and placement of 12 cu.m. of concrete at $500 per cu.m. placed	6 000
	17 600

6. **ANCHOR CREEK DIVERSION**

Excavation – drill, blast, ripping to create channel
100mx10mx2m.   40 000

7. **MILL AREA PREPARATION**

Levelling and filling
2 days work at $650 per day   1 300

8. **GENERAL TIMBER REMOVAL**

Cutting loggable trees, clearing vegetation,
burning winrows, and cleaning up.
40 days at $400 per day   20 000
                                                              _____
                                                               184 650
                    Contingency 10%   18 470
                                                              _____
                                                               203 120

14.2   POWER SUPPLY

1.   **HCB Overhead Power Line from Tasman Highway**

2km of new line	18 000
2km of upgraded line	5 000
	23 000

2.   **Sub Station**

750 KVA Transformer. HEC supply	10 000
Main Isolating switch. HEC supply	2 500
Switch room	10 000
Busbars	10 000
Breakers for sub circuits (4)	10 000
	42 500

Total	$65 500
Contingency 20%	13 100
	$78 600

14.3   SURFACE STRUCTURES

1.   Office – Changehouse Buildings

Changeroom – showers, lockers, toilets
allow facilities for 17 persons

17 clean lockers	700	
17 dirty lockers	700	
4 showers	8 000	
2 toilets	4 000	
1 urinal	1 000	
benches	1 500	
heaters	1 200	
handbasins	500	
water heater	1 000	
Floor Area 30m² in 10m x 3m building	15 000	33 600

2.   Lunchroom

Tables	400	
Chairs – Benches	1 200	
Sink	500	
Refrigerator	1 200	
Hot plate	600	
Hotwater heater	500	
Kettle	100	
Cupboards	300	
Heaters	600	
Floor Area 18m² 6m x 3m room	9 000	14 400

3.   First Aid Room

Bed	300	
Stretchers	2 000	
Handbasin	200	
First Aid box	300	
Heaters	300	
Floor Area 9m² 3 x 3m room	4 500	7 600

4.   **Administration Office**

2 x Desks and chairs	2 400
2 x Visitors chairs	600
Light table	2 000
Draughting tables	3 000
Stools	1 000
Vertiplan	2 000
Plan drawers	3 000
Draughting Equipment	1 000
Filing cabinets	2 000
Shelves	1 000
Stationery cupboard	1 000
Typewriter	3 000
Facsimile unit	4 000
Heaters	600

Area 36m²   12m x 3m building   18 000
                                                   ‾‾‾‾‾‾
                                                   45 600

                                                               101 200

Site Improvements	5%	5 000
Electrical	10%	10 000

                              Total   116 200
                                                   ‾‾‾‾‾‾‾

5.   **Maintenance Bay**

Washdown Pad
15m x 5m concreted   15 000

Service Workshop

Corregated iron clad workshop
15m x 6m with sliding door   27 000
plus 6m lean to one side   20 000
Concrete Floor   36 000

Equipment:

Compressor	5 000
Steam cleaner	7 000
Electric welder	8 000
Gas welder	4 000
Power hacksaw	3 000
Drill Press	5 000
Hydraulic Press	8 000
Work benches	5 000
Drill sharpening bench	7 000
Hand tools	5 000

                                                   57 000
                                                   ‾‾‾‾‾‾
                                                              140 000
                                                              155 000

Electrical	5%	7 000

                              Total   162 000

14.4 MINE EQUIPMENT

<div align="right">Cost '000</div>

Mobile

Load-haul Dump Unit – Eimco 913D		225
Eimco-Secoma Mercury 14		310
Cat 920 FEL (used model)	65	
Attachments: Bucket adapter	2	
Work basket	7	
Dozer blade	10	
Fork lift	4	
Crane boom	3	
	91	91

Mini tractor	15	
Explosives trailer–equipped	10	
Lube trailer – equipped	10	
Manager/store vehicle	20	
	55	55

Fixed

Ventilation fans 25 HP 3 off	24	
Ventilation Ducting – flexible	10	
Face dewatering pumps 7.5 HP 2 off	7	
Water hoses – rubber	2	
Water pipes – polypipe	5	
	48	48

Underground Power

Gate end boxes 2 off	10	
Power cables 200m	10	
Lamp rack and Lamps	5	
	25	

Working Spares

LHD tyre and rim	5	
Cat 920 tyre and rim	3	
Drill Jumbo tyre and rim	2	
Trailer tyre and rim	1	
Drifter – Hydraster 300	42	
	53	53

Miscellaneous equipment		40
Total		822
Contingency 10%		82
		904

A second Eimco 913D unit will be required in the third year of operations		225

14.5 MILL EQUIPMENT – CAPITAL COST

	Cost $'000
Ore Bin 25T capacity	30
Apron feeder	26
Conveyor Belt 20m	27
Primary Crusher	100
Conveyor Belt 20m	27
Vibrating Grizzly	20
Return Conveyor 20m	27
Barmac – Mogensen screen package	330
Sieve bend screens 2 off	15
Sieve bend screen Rapifine	6
Pump sump/hopper	20
Slurry pump Warman 3/2 2 off	15
Jigs 2 off	20
Conditioners and reagent feeders 2 off	9
Tables 2 off	48
Tailings pump Warman 4/3	10
Middlings pumps Warman $1/1\frac{1}{2}$	5
Driers	20
Dry screens	15
High Tension separators	40
Magnetic separator	40
Mill water supply pump	10
Mill head tanks	15
Fill dewatering cyclone	7
Mine fill pump Warman 4/3	10
Platform scales, benches, miscellaneous	40
Mill building 18mx14m	125
	1 057
Shipping	50
Freight	50
Installation	60
Electrical	75
Instrumentation	15
Layout drawings	20
Site supervision	40
	1 367
Continegncy 10%	137
	1 494

15. COST ESTIMATES – OPERATING

Mining

The underground operation as proposed is extremely simple in concept and in operation. There are significant economic advantages acruing from complete mechanisation with large capacity equipment and a minimal labour force.

Room and Pillar mining with fill is a well proven method and productivity rates for this method are amongst the highest of any method. Costs are favoured by the very short haul distances from the stoping areas to the mill crusher enabling haultrucks to be dispensed with.

Load-haul-dump costings on Eimco 913D have been supplied by the manufacturers as being indicative of operations at other sites. The level of costs taken is conservative, and lower levels of cost could be anticipated.

Drilling costs are estimated from probable wear life and prime cost and case history. Estimated drilling capacity per shift has been discounted from the theoretical levels. A cautionary point is that no case history results are available for drilling in greisenised granite. Diamond drilling penetration was said to be very rapid.

Blasting costs are relatively high and alternative systems should be examined.

All indications are that roof conditions should be very good, but an allowance has been made for rock bolting in the stoping costs.

The provision of fill is a simple matter due to the proximity of the mill, and the fact that tailings will be deslimed as part of processing. The tailings pump will double as the fill pump when fill will be required.

Personnel costs have been estimated on an annual basis together with on-cost overheads. Costs are based on ruling rates.

Administration costs will be low on site. Certain accounting functions will be carried out at month end by a local accountant. Close liaison will be maintained with the corporate office.

Stores inventory will be minimal as all operating consumables are available on short notice from Burnie, or Melbourne.

Milling

The milling circuit is simple with the critical aspect being secondary crushing and screening to produce an acceptable liberation size.

Power costs represent a major proportion of costs and the cost is based on the HEC tariff. A conservative costing has been adopted based on installed kw, not on kw actually drawn when in operation. This is of significance for the two major units, the jaw crusher and the Barmac crusher.

Other costs are based upon wear rates, replacements and repairs.

Transport and Shipping

These costs are based on current rates as indicated by Tennant Trading for tin concentrate shipments.

15.1 SUMMARY – Operating Costs

Mining Costs   Base 100 000 TPA.

	Undercut Stoping $/Tonne	Cut & Fill Stoping $/Tonne
Drilling	1.81	0.84
Blasting	1.27	0.60
Mucking	0.60	0.60
Filling	–	0.10
Ground Control	0.30	0.30
Grade Control	0.08	0.08
Power Costs	0.23	0.23
General Supplies	0.40	0.40
	4.69	3.15
Estimated tonnage stoped	265 000	530 000
Total Cost	$1 243 000	$1 669 000
Average Cost/Tonne	$3.66	

Milling Costs

Operation and Maintenance	0.97
Power consumption	3.00
	3.97

General Maintenance   0.21

Administration   0.61

Personnel Costs   6.24

Freight and Shipping   1.50

Total Site Cost $16.19

MINING COSTS

15.2 UNDERCUT HEADINGS

The undercut is based on a single pass 6 x 3 metre heading leaving 4 x 4 square pillars for support.

Data:

Ore density 2.65 tonnes/m^3
Area of face 6 x 3 = 18m^2
Anticipated advance 4.1 x 0.9 = 3.69m
Tonnes broken per round 176

Drilling

Drilling round of 39 holes plus 4 cut holes
Drilling depth 4.1 metres
Advance pulled 3.69 metres

Metres drilled 39 x 4.1 at 45mm = 159.9
             4 x 4.1 at 76 mm = 16.4
Average drilling productivity 1.0 metre per minute.
Time to drill round 176 minutes.

Drill String

	Life	Cost	Cost/m
Shank life	2 000	220	0.11
Steel life 4.100 238.32	1 300	275	0.21
Coupling	1 000	75	0.07
Bit 45mm	150	150	1.00
Bit 76mm	200	400	2.00

Costs

159.9m at (0.11+0.21+0.07+1.00 = 1.39) = 222.26
16.4m at (0.11+0.21+0.07+2.00 = 2.30) =   39.20
                                      261.46

Average cost per metre $261÷6 ÷ 176.3 $1.48
                      261.46

Bit Sharpening

Grinding wheels – 100 sharpenings per wheel
@ $50.00 per wheel = $0.50 per sharpening
Say 10 sharpenings per bit
Cost per metre $5.00 ÷ 200 = $0.025

Drilling Jumbo

Use operating cost $0.30 per metre drilled (Secoma)

Drill string costs	1.48
Bit sharpening	0.03
Drilling Jumbo and drifter	0.30
	$1.81

Cost per round		$318.92
Cost per metre advanced	-	$86.43
Cost per tonne blasted		$1.81

Availability

Literature case histories of various mines indicate availabilities of 75-90 percent. Programmed maintenance has been strictly adhered to in order to obtain the higher figures. In the case of the Anchor Mine the rig will be easily accessible, will be close to the maintenance area, which is close to the working area. An availability of 80 percent or better could be anticipated. On a nominal 7 hour shift because of easy access 5.5 hours should be available for drilling on average. This should equate to 1.83 rounds drilled, or 3.6 over 2 shifts. The daily demand is 400 tonnes, or 2.30 rounds per day rising to 500 tpd or 2.85 rounds per day. Thus one drilling jumbo should be able to cope on a double shift allowing for one or two drill jumbo moves.

Blasting

Amex explosives will be used with primers and half second delays.

Explosive duty is 0.4kg per tonne.
176 tonnes broken gives 70.4kg per round.

Blasted holes	39
Detonators	39
Primers	39

Costs per Blast

70.4kg of Amex at $1.50kg	105.60
39 Detonators @ 2.50 ea	97.50
39 Primers @ 0.35 ea	13.65
Wire	15.00
Cable	1.00
	$222.75
Cost per metre advanced (3.69)	$60.37
Cost per tonne broken (176)	$1.27

15.3 CUT AND FILL

Data.

Ore density 2.65 tonnes/M^3
Area of face 6x2 = 12M^2
Anticipated advance 3.69m
Tonnes broken per round 117

Drilling

Drilling round of 14 holes of 45mm
Drilling depth 4.1m
Advance   3.69m

Metres drilled 14 x 4.1 = 57.4m
Average drilling productivity 1.0 metre per minute
Time to drill round 57.4 minutes

Drill string costs as before at	$1.39 per metre
Bit sharpening	0.03
Drilling Jumbo and drifter	0.30
	$1.72

Cost per round 57.4 x 1.72	$98.73
Cost per tonne blasted	$0.84

Blasting

Amex explosives with primers and half second delays

Explosive duty 0.3kg per tonne
117 tonnes broken requires 15.1kg per round

Blasted holes	14
Detonators	14
Primers	14

Costs per Blast

15.4kg Amex at $1.50kg	23.10
14 detonators at $2.50 ea	36.00
14 primers at $0.35 ea	4.90
Wire	5.00
Cable	1.00
	$70.00

Cost per tonne broken (117)	$0.60

15.4 **EIMCO 913D LHD UNIT**

Calculations of Productivity.

The commonly accepted formula $R = (T_h \times L)/(t + T_v)$

is used where

R = Tonnes per hour
T_h = 50 minute hour
L = Bucket load – tonnes
t = Fixed time
T_v = Variable time.

T_h uses a 50 minute hour to express the real time spent on work for each full hour utilised.

L is the actual load carried in the bucket. In this case 5 tonnes corrected for a bucket factor of 0.9.

t is the fixed time involved in filling the bucket, maneuvering and dumping the load.

T_v is the variable time which is the time taken to travel from the loading place to the dump point and varies with distance and travelling conditions.

The travel speed depends on the standard of the roadway, the number of corners, gradients on the road. Higher speeds possible under the conditions are not achievable under very short hauls due to the decleration and acceleration stages.

An average speed of 5kph has been used for calculation. This is a relatively slow speed for a machine capable of over 12kph on the flat.

For 100 metre one way haul

T_h = 50
L = 5 x 0.9
t = 1.10 minute
T = 1.2 x 2 minute

$$= \frac{50 \times 5 \times 0.9}{1.10 + 1.2 \times 2}$$

$$= 64 \text{ TPH}$$

For 200 metre one way haul

T_h = 50
L = 5 x 0.9
t = 1.10 minute
T = 2.4 x 2 minute

$$R = \frac{50 \times 5 \times 0.9}{1.10 + 2.4 \times 2}$$

$$= 38 \text{ TPH}$$

Using a 6.5 hour working shift and a 75 percent mechanical availability the following production is anticipated.

| 100 metre haul | 312 tonnes |
| 200 metre haul | 185 tonnes |

At an annual rate of 100 000 tonnes a daily production of 400 tonnes is required. This is achievable on two shifts within a 100 metre haul distance and on two shifts if operating at about the 180 metre range. Beyond this range two units would be required, or a haul truck introduced.

Annual hours are 1 200 actually working on single shift.

A single shift operation could be contemplated in the early months of the mine operation with hours of work extending with longer average haul distances to a full two shift working schedule. The age of the unit will also play a part in availability. With a contemplated seven year life of mine total operating hours would be in the region of 12-13 000 hours. Operating at maximum of 200 metres over two shifts may be a marginal proposition for one unit towards the end of the second year. A second unit will be required at the beginning of year 3.

Cost of Operation

Suppliers indicate a cost of $25-$30 per hour including tyres, oil, grease, fuel depending on age and severity of operation.

Taking the lower figure for early life and shorter haul operational cost would be 6.24 actual working hours x $25.00 = $156.00.
With a productivity of 400 tonnes the cost per tonne would be $0.39.

With a longer 200 metre haul and aging machine for the same 400 tonnes a total working hours of 400 ÷ 185 = 2.16 shifts x 4.875 hours = 10.53 hours.
Operational cost would be 10.53 x $30.00 = $315.9.
This equates to $0.79 per tonne for 400 tonnes production.

Raising the production to 500 tpd will require 7.8 actual working hours at 100 metres and 13.16 hours at 200 metres. This would be achievable with one unit on two shifts at 100 metres, but requires two units on two shifts at the 200 metre range.

The LHD cost per tonne would be a minimum of $0.40 per tonne rising to $0.80 per tonne. An average figure of $0.60 is used for costing purposes.

15.5 CATERPILLAR 920 UNIT

The Cat 920 utility unit will be used for general purposes. One application will be charging the breast stoping faces although these can be reached by ladder quite easily. Charging will take about 45 minutes therefore 1.5 hours could be spent on charging. Barring down will be from the muck pile with occasional use of the work platform for rechecking.

Other work will be grading the haul road and assisting around the mill area. The estimate of usage is as follows:

Charging	2.75 hour
Scaling	0.50
Road maintenance	0.75
General	1.00
	4.50

Only one shift is envisaged.

The cost per hour is estimated as $20.00 per hour as for half the time the unit will only be used as a platform. The cost per shift would thus be $90.00 or $0.22 per tonne.

15.6 GROUND CONTROL

Ground conditions are expected to be excellent based on rock conditions viewed in the old workings and from diamond drill core. The reduction of the roof span from 8 metres to 6 metres will assist in minimising open spans at intersections. No specific pattern of roof bolts is contemplated, but there will be occasions when roof reinforcement may be necessary. A contingency allowance of 0.30 per tonne is set aside for roof control.

15.7 FILLING COSTS

It is not possible to estimate the amount of material used for fill walls whatever its source being rehandled placed fill, or waste rock. For the purpose of costing it is assumed that 20 percent of the fill is placed in walls. This would amount to 40 000 tonnes over life of mine. The Cat 920 can easily handle this and if it is assumed that the machine can rehandle 50 tonnes per hour by loading or bulldozing some 800 machine hours would be required. As the machine will be working fairly hard a cost of $30.00 hour could be used. This results in a cost of 800 x $30 or $24 000.

The operating cost of dewatering and pumping fill at a rate of approximately 17 tonnes per hour over two shifts is not likely to exceed $0.15 per tonne. Only 398 000 tonnes are required over life of mine assuming tight filling is required. This gives a cost of 398 000 x $0.15, or $59 900.

Total costs for placement of 398 000 tonnes of fill should be $59 700+$24 800 or $83 700. This equates to $0.21 per tonne of fill, and $0.10 per tonne of ore extracted.

18.5 GENERAL SUPPLIES

These supplies would be flexible ventilation duct, ANFO hoses, explosive boxes, protective clothing, marker paint, suspension hooks etc. Use a cost of $40 000 per year, or $0.40 per tonne of ore.

POWER CONSUMPTION

Ventilation

The A and B Lens sections of the ore body will be worked as separate sections for some time and will have to be ventilated separately.

The ventilation requirement is not large and is primarily to cater for the Eimco 913D (USEPA 10 000 cfm), and the Cat 920 80 HP, or nominal 8 000 cfm. A total ventilation requirement of 40 000 cfm would seem satisfactory. Mine resistance would be low, and so low horsepower fans would be quite adequate. The same fans could be used for blowing the face for the short development period and used for exhausting when a return circuit is established.

Two 25 HP fans should be adequate for the duty, operating over an 8 hour shift. The latter period of production will be concentrated in A Lens, and two shift mucking would be required, but probably only one fan.

> Fan demand 2 x 25HP for 16 hours
> or 155 000 KWH   (neglecting actual HP drawn)
> Using $0.14 per KWH unit
> Cost per year is $21 720

Jumbo Operation

Power cost is included in the cost of operating the jumbo.

Ancilliary Power

An allowance of $2 000 should be included for miscellaneous usage underground over the year.

The total power cost underground is likely to be $23 720 per year, or $0.23 per tonne extracted.

15.10 GRADE CONTROL

Costs

2 000 metres of percussive drilling at $1.81/m	3 620
400 metres of diamond drilling at $100/m	40 000
2 400 metres of sample assaying at $7.50	18 000
	$61 620
Cost per tonne of ore	$0.08

15.11 EXPLORATION PROGRAMME

Surface

Potential open pit ore
    200m diamond drilling at $120m $24 000

Delineation of A and B Lens
    500m at $120m $60 000

Underground

Exploration of Lower A Lens
    500m at $120 $60 000
Scheduled in year 3.

15.2 MINING COSTS

Summary

	Undercut Stoping $/Tonne	Cut and Fill Stoping $/Tonne
Drilling	1.81	0.84
Blasting	1.27	0.60
Mucking	0.60	0.60
Filling	–	0.10
Ground Control	0.30	0.30
Grade Control	0.08	0.08
Power costs	0.23	0.23
General Supplies	0.40	0.40
	4.69	3.15

Estimated tonnage stoped	265 000	530 000
Total Cost $	1 243 000	1 669 000
Average Cost/Tonne $	$3.66	

15.3 **MILLING – OPERATING AND MAINTENANCE COSTS**

based on 100 000 TPA

	$ Annual

1. Ore Handling

A stockpile of run of mine ore will be maintained on surface. Normally ore will be dumped directly into the ore bin by LHD and reclaimed by apron feeder.

Grizzly and Bin Maintenance	3 000

2. Ore Feeder

Apron Feeder, Grease, oil, repairs	2 000

3. Primary Crushing

Feed conveyor	1 000
Metal Detector	
Jaw crusher –Grease, oil, repairs	3 000
Liner wear and replacement	
0.3 kg/tonne @ $0.50 /kg steel	15 000
	19 000

4. Screening

Feed conveyor	1 000
Vibrating grizzly – repairs	3 000
Return conveyor to jaw crusher	1 000
	5 000

5. Secondary Crushing

Feed chute	1 000
Barmac Crusher	
Repairs	5 000
Wear parts	4 000
Grease and oil	2 000
	12 000

Screening

Feed elevators – repairs	4 000
Mogensen screens	
– repairs	1 000
– screen mesh	15 000
Sieve Bend Screens	
– repairs	1 500
– screen mesh	1 500
Pump	
– impeller wear	1 000
– liner wear	1 000
Return conveyor	1 000
	26 000

Jig Plant

General maintenance	500
Jig screens	1 000
Ragging	500
Hutch diaphrams	1 000
	3 000

Flotation

Conditioners – maintenance	1 000
Reagents – xanthate 0.10 kg/T @ $2.00 kg	500
– cresote 0.10 kg/T @ $0.50 kg	125
	1 625

Tabling

Sand Tables – repairs and grease	500
Fine sand table – repairs and grease	500
	1 000

Jig Tailing Pump

Impeller wear	2 000
Liner wear	2 000
	4 000

Drier

Maintenance	500
Fuel	2 500
	3 000

Dry Separation Equipment

Maintenance	1 000

Water Supply Pump

Maintenance pump and lines	1 500

General Supplies and Maintenance   15 000

Total Cost   97 125

Cost per tonne milled $0.97

MILL POWER CONSUMPTION AND COSTS

Unit	Installed HP	Annual KWH	Annual Cost $
Apron Feeder	10	54 000	7 560
Conveyor Belt	3	16 000	2 240
Jaw Crusher	60	324 000	45 360
Grizzly Feeder Conveyor	3	16 000	2 240
Vibrating Grizzly	5	27 000	3 780
Return Conveyor	3	16 000	2 240
Barmac Crusher	200	1 080 000	151 200
Elevators	18	96 000	13 440
Elevator feed conveyors	8	42 000	5 880
Mogensen screens	2	10 000	1 400
Return conveyors	8	42 000	5 880
Slurry pump	10	54 000	7 560
Jig Plant	4	21 000	2 940
Flotation conditioner	1	5 000	700
Sand Tables	6	32 000	4 480
Jig tailing pump	15	80 000	11 200
High Tension separator	3	16 000	2 240
Magnetic separator	3	16 000	2 240
Water supply pump	15	80 000	11 200
General (LIGHTING + HEATING)	25	134 000	15 876
	402	2 161 000	299 656

Based on 4 000 hours per year operation.
Increase pro rata for additional hours.
Processing rate 100 000 TPA   Cost per tonne $3.00

POWER COST CHECK: 300 KW AV. × 4000 H × .15 = $180,000. ?
SPECTRUM   CALC LOOKS TOO HIGH.

15.15 **MILL OPERATING COSTS**

SUMMARY

Operating and Maintenance	$0.97
Power Costs	$3.00
	$3.97 per tonne

Based on 100 000 TPA
and 4 000   Hours PA.

15.16 PERSONNEL COSTS

Site Manpower

	Year						
	1	2	3	4	5	6	7
Manager	1	1	1	1	1	1	1
Managers Assistant	1	1	1	1	1	1	1
U/G Miners – Jumbo	2	2	2	2	2	2	2
LHD	2	2	4	4	4	4	4
Other	4	4	4	4	4	4	4
Diesel Mech/Fitter	1	1	1	1	1	1	1
Mill Operator	4	4	4	4	4	4	4
	15	15	17	17	17	17	17
Contract Electrician	1	1	1	1	1	1	1

Annual Shifts

	1	2	3	4	5	6	7
Manager	–	–	–	–	–	–	–
Managers Assistant	–	–	–	–	–	–	–
Underground Miners	2 000	2 000	2 500	2 500	2 500	2 500	2 500
Diesel/Mechanic/Fitter	250	250	250	250	250	250	250
Overtime 20%	50	50	50	50	50	50	50
Mill Operators	1 000	1 000	1 000	1 000	1 000	1 000	1 000
Overtime 5%-20%	50	50	200	200	200	200	200
Contract Electrician	50	50	50	50	50	50	50

Annual Expenditure $'000

	1	2	3	4	5	6	7
Manager	45	45	45	45	45	45	45
Managers Assistant	30	30	30	30	30	30	30
Underground Miners $125	250	250	312	312	312	312	312
Diesel Mechanics/Fitter	31	31	31	31	31	31	31
Overtime X 1.5	10	10	10	10	10	10	10
Mill Operators X $100	100	100	100	100	100	100	100
Overtime X 1.5	8	8	20	20	20	20	20
	474	474	547	547	547	547	547
Contract Electrician $150	8	8	8	8	8	8	8

Add

On-cost figure to cover annual leave loading, A.C. insurance, payroll base, sick leave and long service leave of 30% to payroll employees.

	1	2	3	4	5	6	7
30% on-cost	142	142	164	164	164	164	164
Total Cost $000	624	624	711	711	711	711	711
Annual tonnes '000	100	100	110	115	120	125	125
Cost per tonne $	6.24	6.24	6.46	6.18	5.92	5.67	5.67

The manning and costs are based on two shifts on a five day week in the mine with a two shift five day week in the mill moving to a two shift six day week in the mill in year 3.

15.17 ADMINISTRATION COSTS

General Administration Supplies	6 000
Post and telephone	6 000
Insurance	28 000
Accounting fees	7 000
Licencing fees	1 000
Manager's vehicle	8 000
Access road maintenance	5 000
	61 000

Cost per tonne $0.61

15.18 MAINTENANCE COSTS

General Supplies	15 000
Power 35HP installed over 3 hour/day	
19 582 KWH annually at $0.14	2 700
Administration/changehouse	
say 20 000 kwh at $0.14	2 800
	$20 500

Cost per tonne $0.21

15.19 OPERATING COSTS: Life of Mine

Costs in $000

Year	1	2	3	4	5	6	7
Annual Tonnes	100	100	110	115	120	125	125
Mining $3.66/T	366	366	403	421	439	458	458
Milling $3.97/T	397	397	437	457	476	496	496
Admin.	61	61	61	61	61	61	61
Gen. Mtce.	21	21	21	21	21	21	21
Personnel Cost	624	624	711	711	711	711	711
Freight $205/T	150	150	151	150	150	144	133
	1 619	1 619	1 784	1 821	1 858	1 891	1 880
Cost per tonne	16.19	16.19	16.22	15.83	15.48	15.13	15.13
$ Cost/tonne Sn in concentrates	3 175	3 175	3 464	3 571	3 643	3 859	4 132

16. **SALES REVENUES**

   1.   **Tin in Concentrates**

   The mill production will consist of tin in concentrates at a minimum of 70 percent tin metal content. Under the marketing agreement with Tennant Trading concentrates would be shipped to Malaysia for smelting. Indicative terms have been discussed with Tennant and potential purchasers.

   Shipments would be made in 20 tonne multiples with a minimum of 20 tonnes.

   Deductions and payments would be as follows:

   Indicative terms assuming a concentrate containing 65-70% Sn, minimal iron and wolfram and no other significant impurities.

   (Also based on current rates of exchange for A$/M$, A$/US$ and A$/SIG).

   Basis 400 tonnes per year of concentrates, shipments of 20 tonnes minimum, and in 20-tonne multiples.

   1.   **UNITAGE DEDUCTION**

   1.00 unit at 70% Sn increasing by 0.10 unit for each 1.00% by which Sn content is below 70%. Minimum deduction 1.00 unit.

   2.   **TREATMENT CHARGE**

   A$300 per dry tonne of concentrate, delivered smelters works.

   3.   **PENALTIES FOR IMPURITIES**

   Iron - up to 2.5% - free
   Over 2.5% up to 5.0% deduct 0.2% from gross tin assay.

   Wolfram - same as iron.

   (N.B. Deductions from tin assay for iron and wolfram are applied prior to calculating unitage deduction).

Sulphur	A$15 per dry tonne of concentrate for each 1% contained.
Antimony	A$200 per dry tonne of concentrate for each 1% contained.
Bismuth	Same as Antimony
Lead	A$60 per dry tonne of concentrates for each 1% contained.
Copper	Same as Lead
Arsenic	Same as Lead

   4.   **PAYMENT**

   Assuming concentrates were sold to a smelter we would expect a provisional payment of 90% of estimated value on arrival of carrying vessel at discharge port.

   Balance to be settled promptly after agreement of assays.

WEIGHING AND SAMPLING

Results at receiving works would be final. Spectrum would have the right to be represented at outturn operations (at Spectrum's cost).

ASSAYS

To be exchanged in the usual manner. Splitting limit for tin would probably be 0.5% but might be 0.3%.

There are severe penalties for certain impurities such as Antimony, Bismuth and other metals. It is essential that separation of sulphides be extremely thorough. The penalties for wolfram and iron are less severe.

The flowsheet includes flotation and magnetic separation and the treatment ought to be sufficiently rigorous to remove penalisable impurities.

2.   ## Silver in Sulphide Concentrates

The silver values in A and B Lenses are averaged at 10.5g/Tonne with values in parts of A lens in the range 25-30g/Tonne. The silver appears to be associated with copper on a statistical basis which is present at levels of 0.05%. Zinc is present at similar levels. Copper and zinc sulphides have been very responsive to flotation and high recoveries appear possible together with silver.

A silver recovery of 75 percent should result in 25 000 oz of silver in concentrates per year. Realisable value may be $6.00 oz. Copper and zinc would also be payable. Some 45-50 tonnes of each metal could be recoverable per year. This would be saleable at a discount.

Silver - 25 000 oz at $6.00 oz	$150 000
Copper and zinc content	75 000
	$225 000

3.   ## Wolfram Concentrates

Some wolfram concentrates may be available from the dry separation stage of tin concentrate cleaning. It is unlikely that this would exceed 10 tonnes per year. The realisable value would be small perhaps $10-20 000.

Royalty and Charges

1.   **Renison Royalty**

This is based on 0.9% of net proceeds after realisation of the sale of all concentrates from the property.

2.   **Annual Fee to Tennant**

The annual fee is set at $24 000 in year one with escalation at not more than 10 percent per annual based on CPI.

3.   **Commission to Tennant**

A commission based on 2 percent of the value of concentrates less refining charges is payable.

5.   **Nargun Interest**

Nargun is entitled to 2 percent of the production. Nargun is carried to Feasibility Study with Nargun share of costs being recovered from Nargun's 2 percent share of production in the proportion 70 percent to Spectrum and 30 percent to Nargun until the costs are recovered. Thereafter Nargun is entitled to its 2 percent of production. Nargun contributes to capital investment and working costs, which are recoverable from Nargun's share.

6.   **Tasmanian Government Royalty**

Where annual profits do not exceed $10 000 000 2.5% of proceeds from sale, or 5% of profits from sales whichever is the lesser are payable.

16.1 CALCULATION OF REVENUES

Assumption: A tonne of concentrate contains 70% Sn. (70 units) and has no penalisable impurities.

At a price of M$17.430 tonne of tin metal and an exchange rate of M$/A$ of 2.017 a tonne of tin metal is worth A$8642_

A tonne of tin concentrates contains	70 tin units
Smelter deducts 1 unit for losses	1 tin units
Payable units are	69 tin units
Value of a tonne of concentrates is	69 x 86.42
	A$ 5963
Less a treatment charge	300
Net smelter return	A$ 5663
Less	
Tennant Trading Commission 2%	113
Renison Limited Royalty 0.9%	51
Net Revenue	5499

BASE CASE

PROFIT AND LOSS $A'000

	YEAR	1	2	3	4	5	6	7	Total
Revenue Tin Sales		4 014	4 014	4 042	4 014	4 014	3 849	3 574	27 521
By Product		225	225	225	225	225	225	225	1 575
Salvage								600	600
Total Revenue		4 239	4 239	4 267	4 239	4 239	3 974	4 399	29 696
Operating Costs		1 619	1 619	1 784	1 821	1 858	1 891	1 880	12 472
Tennant fee		25	29	33	37	41	45	49	259
Diamond drilling		24	60	60					144
Amortisation		428	428	473	473	473	473	473	3 221
Total Costs		2 096	2 136	2 350	2 331	2 372	2 409	2 402	16 096
Operating Profit		2 143	2 103	1 917	1 908	1 867	1 565	1 997	13 600
Tas.Govt. Royalty		107	105	96	95	93	78	100	674
Profit Before Tax		2 036	1 998	1 821	1 813	1 774	1 487	1 897	12 926
Tax 39%		794	779	710	707	692	580	740	5 002
Profit after Tax		1 242	1 219	1 111	1 106	1 182	907	1 157	7 924

BASE CASE

CASH FLOW $A'000

	YEAR	1	2	3	4	5	6	7	Total
Revenue Tin Sales		4 014	4 014	4 042	4 014	4 014	3 849	3 574	27 521
By Product		225	225	225	225	225	225	225	1 575
Salvage								600	600
Total Revenue		4 239	4 239	4 267	4 239	4 239	3 974	4 399	29 696
Operating Costs		1 619	1 619	1 784	1 821	1 858	1 891	1 880	12 472
Tennant Fee		25	29	33	37	41	45	49	259
Diamond Drilling		24	60	60					144
Capital Costs		2 995		225					3 220
Royalty – Tas.Govt.		107	105	96	95	93	78	100	674
Company Tax		794	779	710	707	692	580	740	5 002
Total Costs		5 564	2 592	2 908	2 660	2 684	2 594	2 769	21 771
Surplus (Deficit)		(1 325)	1 647	1 359	1 579	1 555	1 380	1 630	7 925
NPV (DCF 90%)		(1 325)	866	376	230	120	55	33	357

CASE + 10% TIN PRICE

PROFIT AND LOSS $A'000

YEAR	1	2	3	4	5	6	7	Total
Revenue Tin Sales	4 437	4 437	4 467	4 437	4 437	4 255	3 951	30 421
By Product	225	225	225	225	225	225	225	1 575
Salvage							600	600
Total Revenues	4 662	4 662	4 692	4 662	4 662	4 480	4 776	32 596
Operating Costs	1 619	1 619	1 784	1 821	1 858	1 891	1 880	12 472
Tennant Fee	25	29	33	37	41	45	49	259
Diamond Drilling	24	60	60					144
Amortisation	428	428	473	473	473	473	473	3 221
Total Costs	2 096	2 136	2 350	2 331	2 372	2 409	2 402	16 096
Operating Profit	2 566	2 526	2 342	2 331	2 290	2 071	2 374	16 500
Tas.Govt. Royalty	128	126	117	117	115	104	119	826
Profit before Tax	2 438	2 400	2 225	2 214	2 175	1 967	2 255	15 674
Taxation 39%	951	936	868	863	852	767	879	6 116
Profit after Tax	1 487	1 464	1 357	1 351	1 323	1 200	1 376	9 558

CASE + 10% TIN PRICE

CASH FLOW $A'000

YEAR	1	2	3	4	5	6	7	Total
Revenue Tin Sales	4 437	4 437	4 467	4 437	4 437	4 255	3 951	30 421
By Product	225	225	225	225	225	225	225	1 575
Salvage							600	600
Total Revenues	4 662	4 662	4 692	4 662	4 662	4 480	4 776	32 596
Operating Costs	1 619	1 619	1 784	1 821	1 858	1 891	1 880	12 472
Tennant Fee	25	29	33	37	41	45	49	259
Diamond Drilling	24	60	60					144
Capital Costs	2 995		225					3 220
Tas.Govt. Royalty	128	126	117	117	115	104	119	826
Company Tax	951	936	868	863	852	767	879	6 116
Total Costs	5 742	2 770	3 087	2 838	2 866	2 807	2 927	23 037
Surplus (Deficit)	(1 080)	1 892	1 605	1 824	1 796	1 673	1 849	9 559
NPV (DCF 90%)		996	445	266	138	68	39	872

CASE + 10% OPERATING AND CAPITAL COSTS

PROFIT AND LOSS $A'000

	YEAR	1	2	3	4	5	6	7	Total
Revenue Tin Sales		4 014	4 014	4 042	4 014	4 014	3 849	3 574	27 521
By Product		225	225	225	225	225	225	225	1 575
Salvage								600	600
Total Revenues		4 239	4 239	4 267	4 239	4 239	4 074	4 399	29 696
Operating Costs		1 781	1 781	1 962	2 003	2 044	2 080	2 068	13 719
Tennant Fee		25	29	33	37	41	45	49	259
Diamond Drilling		27	66	66					159
Amortisation		471	471	521	520	520	520	520	3 543
Total Costs		2 304	2 347	2 582	2 560	2 605	2 645	2 637	17 680
Operating Profit		1 935	1 892	1 685	1 679	1 634	1 429	1 762	12 016
Tas.Govt. Royalty		97	95	84	84	82	74	88	604
Profit before Tax		1 838	1 797	1 601	1 594	1 551	1 354	1 673	11 412
Taxation 39%		717	701	624	622	605	528	652	4 449
Profit after Tax		1 121	1 096	977	972	946	826	1 021	6 963

CASE + 10% OPERATING AND CAPITAL COSTS

CASH FLOW $A'000

	YEAR	1	2	3	4	5	6	7	Total
Revenue Tin Sales		4 014	4 014	4 042	4 014	4 014	3 849	3 574	27 521
By Product		225	225	225	225	225	225	225	1 575
Salvage								600	600
Total Revenues		4 239	4 239	4 267	4 239	4 239	4 074	4 399	29 696
Operating Costs		1 781	1 781	1 962	2 003	2 044	2 080	2 068	13 719
Tennant Fee		25	29	33	37	41	45	49	259
Diamond Drilling		27	66	66					159
Capital Costs		3 295		248					3 543
Tas.Govt. Royalty		97	95	84	84	82	74	88	604
Company Tax		717	701	624	622	605	528	652	4 449
Total Costs		5 942	2 672	3 017	2 746	2 772	2 727	2 857	22 733
Surplus (Deficit)		(1 703)	1 567	1 250	1 493	1 467	1 347	1 542	6 963
NPV (DCF 90%)			825	346	218	113	52	32	(117)