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RENISON LIMITED

INDICATIVE FEASIBILITY STUDY

ANCHOR MINE DEPOSIT

Prepared by: Operating Staff
Renison Limited

January, 1980

Submitted by: J.W. Mitchell
General Manager

CONTENTS

LIST OF ENCLOSURES

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1. OBJECTIVE

To evaluate the tin mineralisation outlined at the Anchor Mine (North-East Tasmania) in the light of current data and knowledge and reach a conclusion as to the potential viability of a mining operation.

2. SUMMARY

In October 1979, geological reserves of 2 million tonnes of 0.40% tin as cassiterite were calculated from the diamond drilling data on the Anchor Mine. A cut-off grade of 0.2% tin was used.

This study indicates that a small to medium sized operation using open pit and underground mining methods is capable of processing 200,000 t.p.a. of ore at a grade of 0.34% Sn over a life of ten years.

The open pit mining operation would produce 1,015,000 tonnes of ore at 0.35% Sn after the removal of 526,000 cu. metres of decomposed rock overburden (999,999 tonnes), overburden ratio would be 1:1.83.

The underground operation would produce 997,000 tonnes of ore at a grade of 0.32% Sn by pillar and stall, open stoping, and cut and fill stoping methods following development by a decline access.

The metallurgical process would involve crushing, grinding and gravity separation of the cassiterite into a 55% Sn concentrate at a recovery of 85%. Annual production would be 574 tonnes of Sn metal in 1,044 tonnes of dry concentrate.

The capital expenditure on the project would be $9.55 million (unescalated) with a potential salvage value of $0.67 million. The major areas of capital expenditure are Concentrator $5.27 million, Mine Equipment and Development $2.67 million, Lease Buildings and Civil Works $1.37 million. Virtually no infrastructure was included as it was assumed that employees would be drawn from St. Helens and the surrounding area. A Contingency of 20% was added to the total capital expenditure for financial calculations.

Capital investment is equivalent to $1660/tonne of Sn metal produced over the life of mine.

The operation requires a relatively small establishment involving 67 employees, working the mine on a single shift and the mill on three shifts over a five day week.

Operating costs are generally low and amount to $13.83 per tonne of ore treated with the following breakdown.

Mining	$4.03
Concentrating	$4.38
Administration & Overheads	$3.07
Contingency	$2.30

The cost of production is $4,840 per tonne of Sn metal produced in concentrates.

The study gives DCF/ROR of 13% after tax, but becomes more attractive (DCF/ROR of 21% after tax) if the production rate is increased to 250,000 t.p.a. without increase in capital expenditure. To achieve this production rate an additional 500,000 tonnes of similar grade ore are required.

The study recommends that

1. Diamond drilling be resumed to locate an additional 0.5 million tonnes of ore.

2. If the additional reserves are located, carry out a detailed diamond drilling programme including the collection of a bulk sample.

3. If the results are encouraging, proceed to a Final Feasibility Study.

3. CONCLUSIONS AND RECOMMENDATIONS

The study indicates that the project could be promising, particularly at the higher rate of mill throughput of 250,000 t.p.a. The operation has the potential to become a small to medium tonnage producer of low cost clean tin concentrates. There is also potential for by-product recovery of copper - silver concentrates which would assist revenue.

The return on capital would appear to be attractive and meet general financial guidelines.

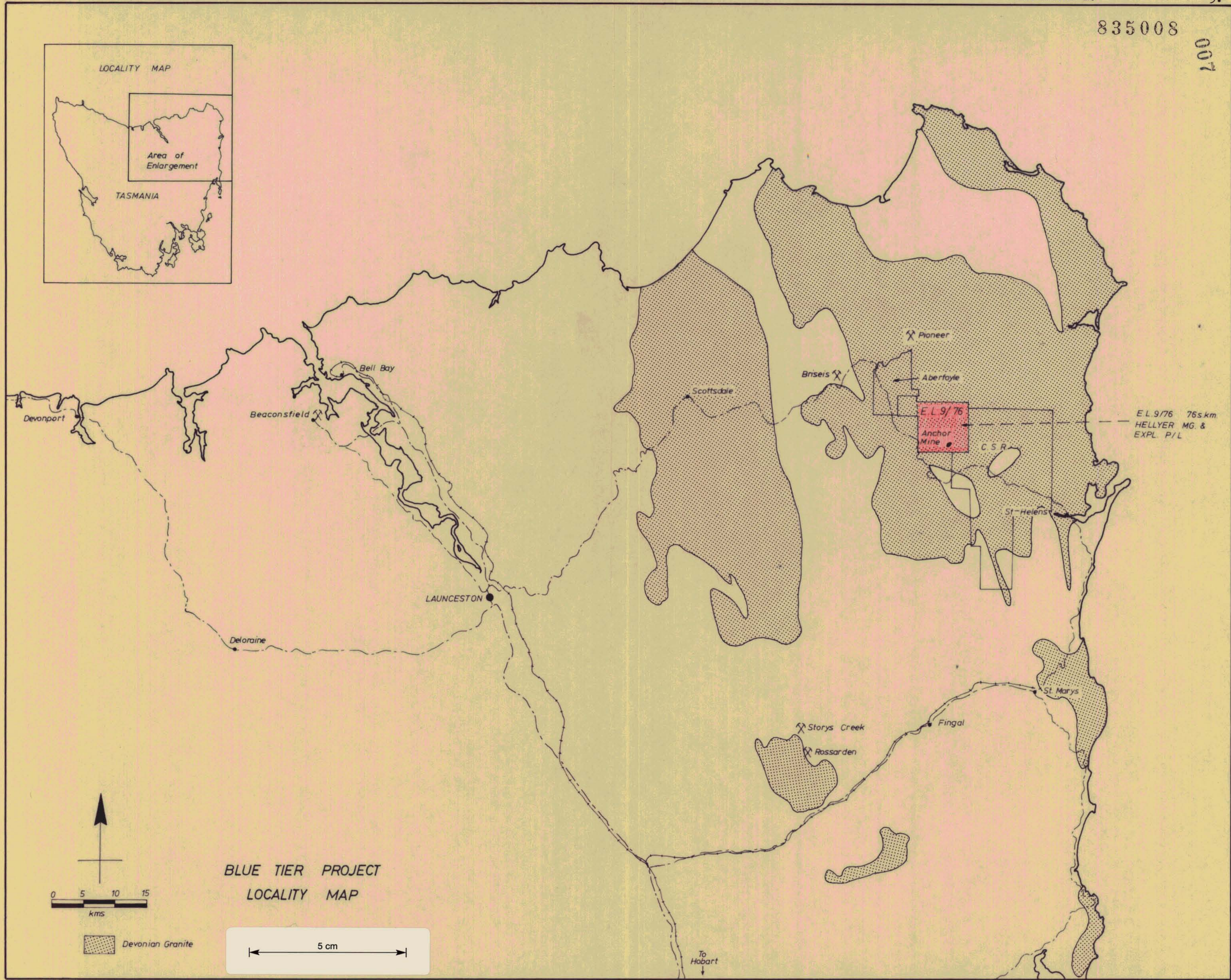
The project is volume sensitive, and the higher the throughput the better will be the risk factor. The project appears viable at 200,000 t.p.a., and attractive at 250,000 t.p.a. Any increase above this level will require additional capital input, and require additional tailings dam sites.

It is recommended that the project be increased in status and the following programme be carried out.

1. Resume drilling to locate an additional 0.5 million tonnes of minable reserves, preferably in an open cut environment.

2. If the reserves are located, carry out a detailed drilling programme on the reserves to improve the confidence level in the grade distribution of the ore and obtain a bulk sample, possibly by big-hole coring, for metallurgical testing as a basis for the development of a definitive flow sheet.

3. If the assumptions used in the Indicative Feasibility Study are confirmed then proceed to a Final Feasibility Study which would involve detailed engineering estimates.


LOCALITY MAP
Area of Enlargement
TASMANIA
Pioneer
Briseis
Aberfoyle
Bell Bay
Scottsdale
Beaconsfield
Devonport
E.L. 9/76
Anchor Mine
C.S.R.
E.L. 9/76 76s.km.
HELLYER MG. &
EXPL. P/L
St Helens
LAUNCESTON
Deloraine
St. Marys
Fingal
Storys Creek
Rossarden
BLUE TIER PROJECT
LOCALITY MAP
0 5 10 15
kms
Devonian Granite
5 cm
To Hobart

4. THE DEPOSIT

4.1. Location:

The Anchor Mine lies 20 road kilometres west of St. Helens in north-east Tasmania. There are two principal accesses to the mine, firstly by travelling 12 kilometres along the main sealed St. Helens - Launceston highway and thence 8 kilometres of unsealed road through Goulds Country to the mine, or secondly by travelling 16 kilometres along the main St. Helens - Launceston highway and then 4 kilometres of second class dirt road to the mine.

There is a third, lesser used dirt road which leaves the St. Helens - Launceston highway some 30 kilometres from St. Helens and reaches the Anchor Mine after travelling east for 5 kms.

Access is available all year, by normal two wheel drive vehicles, being approximately 30 minutes drive from St. Helens.

Climatically, the area is best described as cool, temperate, moderate rainfall. Most of the rain falls in Winter. The deposit is approximately 300 m. above sea level, and winter snow is common on the plateau immediately above the Anchor Mine.

All vegetation is regrowth subsequent to the previous open-cut mining operations. It consists of tall gums and wattles with a frequently dense bracken and fern undergrowth.

4.2. Geology:

In north-eastern Tasmania, sedimentary rocks called the Mathinna Beds have been intruded by upper Devonian - lower Carboniferous granites. Most of the tin, tungsten and gold deposits of the area are genetically related to these granites.

In the Anchor Mine area, the geology essentially consists of an early coarse grained granite phase which has been intruded by a later fine grained stanniferous granite. The deposit is composed of cassiterite bearing greisenised granite developed in the roof zone of the younger fine grained granite. Laterally the mineralisation appears to be controlled by flexures in the granite contact, whilst vertically, the best mineralisation occurs immediately beneath the contact with the coarse grained granite, which is normally characterised by a thin pegmatitic zone.

The intensity of greisenisation and mineralisation decreases with depth and the majority of the mineralisation is confined to the top 30 - 40 m. of the fine grained granite, although some deeper greisen zones do exist.

Mineralisation is somewhat erratic within the greisen zone. It normally consists of visibly identifiable cassiterite, together with trace amounts of sulphides (chalcopyrite, sphalerite and molybdenite) and silver.

The overlying coarse grained granite has weathered deeply and is frequently decomposed and in places unconsolidated.

4.3. History:

Primary tin mineralisation at Anchor was discovered in 1881 during alluvial mining operations. Recorded production (1890 - 1942) is 2,360 tonnes contained tin. This was achieved from intermittent operations. the largest of which was the Anchor Company, which produced 2,548 tonnes of tin concentrate from 1.3 million tonnes of ore. This represents an estimated recovered grade of 0.14% tin. Most of this concentrate was smelted at St. Helens and shipped from St. Helens as metallic tin.

The workings now consist of a series of abandoned open pits, known collectively as the Anchor Open Cut, occupying an area of approximately 5 hectares. Ore was transported by hand trolleys and drays to a 100 head stamp mill powered by water wheel on the nearby Groom River. As the mine declined, the size of this battery was gradually reduced to 20 heads, the remains of which still exist on site.

Mining activity appears to have declined and

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finally ceased due to a combination of low head grades and increasing overburden thicknesses.

4.4. Land Tenure:

The Anchor Mine and the majority of the other smaller tin workings which constitute the Blue Tier Tinfield are covered by Exploration Licence 9/76 of 76 sq. kilometres. The Licence is held by Hellyer Mining and Exploration Pty. Limited, and is renewable every six months at the discretion of the Director of Mines.

In December 1977, Renison Limited entered into a Joint Venture Agreement with Hellyer, whereby Renison could earn a 60% interest in the area by spending $500,000 on the Licence area prior to September 1980. By June 1979, Renison had expended $253,646 on the project and had thereby earned a 30% interest.

Several small Mining Leases are held by other parties to the north and west of the Anchor Mine, but are unlikely to interfere in any way with operations in the vicinity of the Anchor.

Several small private farming properties exist in the area, but none is closer than 0.5 kms. to the Anchor deposit.

4.5. Exploration Completed:

During its active operational life (ie.) up until 1942, it is believed that a considerable amount of exploration including core and non-core drilling was undertaken in the vicinity of the Anchor deposit. Results of some of this work are available but the quality of the work is now known and thus it has largely been disregarded.

Very little exploration work appears to have been done on the deposit between 1942 and 1964, possibly thereby reflecting several decades of low tin prices and unsure market conditions.

In 1964, Aberfoyle commenced the first of several core drilling programs and between 1964 and 1966, they completed 39 short vertical holes, mainly to the immediate north-east of the open-cut.

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Their stated aim was to define approximately one million tonnes of 1% tin ore, in an opencuttable situation. At the conclusion of their drilling programs, they had failed to achieve this.

In 1968, the Tasmanian Department of Mines drilled one hole, designed to test a theory of ore extension to the north.

During 1977 and 1978, Renison completed twenty-nine holes, aimed at exploration for both lateral and vertical extensions of the mineralisation defined by Aberfoyle.

A further pattern of holes has recently been completed but results of these holes have not been considered for the purpose of this study.

The drill core obtained by Renison has been thoroughly logged and extensively assayed for tin, tungsten, molybdenum, copper, lead, zinc and silver. The possible presence of uranium has been tested with negative results.

The accuracy of assay results from the Renison Laboratories has been periodically checked by Amdel, and agreement is considered quite adequate.

Within the mineralised zones, suites of samples were selected for petrographic studies by Central Mineralogical Services in Adelaide.

Following this assaying and petrological work, the Renison Metallurgical Department were advised of drill hole intervals which were considered to be of potential economic interest. On the basis of this advice, drill core samples from these intervals were to be bulked and subjected to appropriate metallurgical testwork designed to determine the treatment amenability of the mineralisation.

4.6. Ore Potential:

A possible ore potential of 2,000,000 tonnes of 0.40% tin as cassiterite has been calculated from

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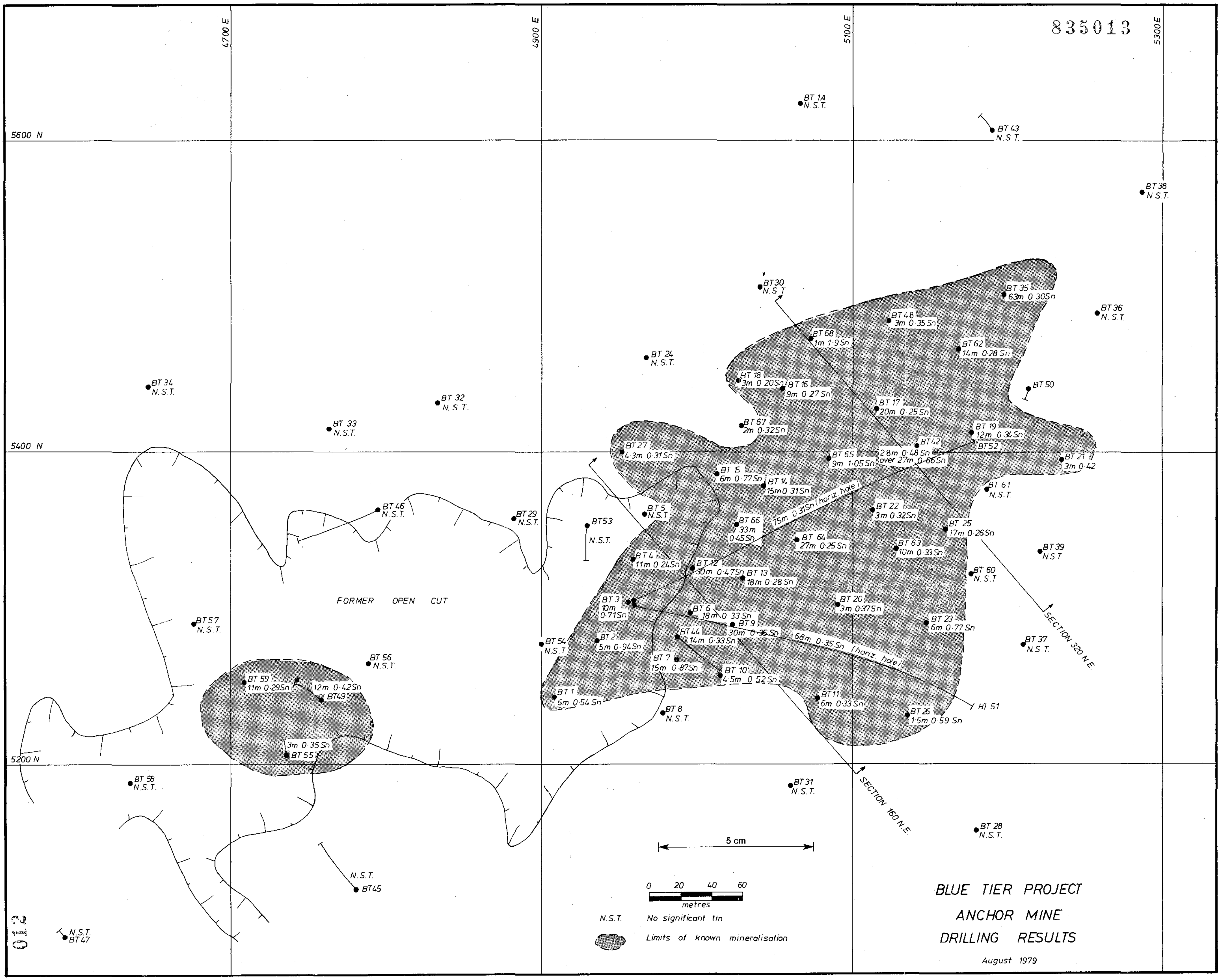

BLUE TIER PROJECT

ANCHOR MINE

DRILLING RESULTS

August 1979

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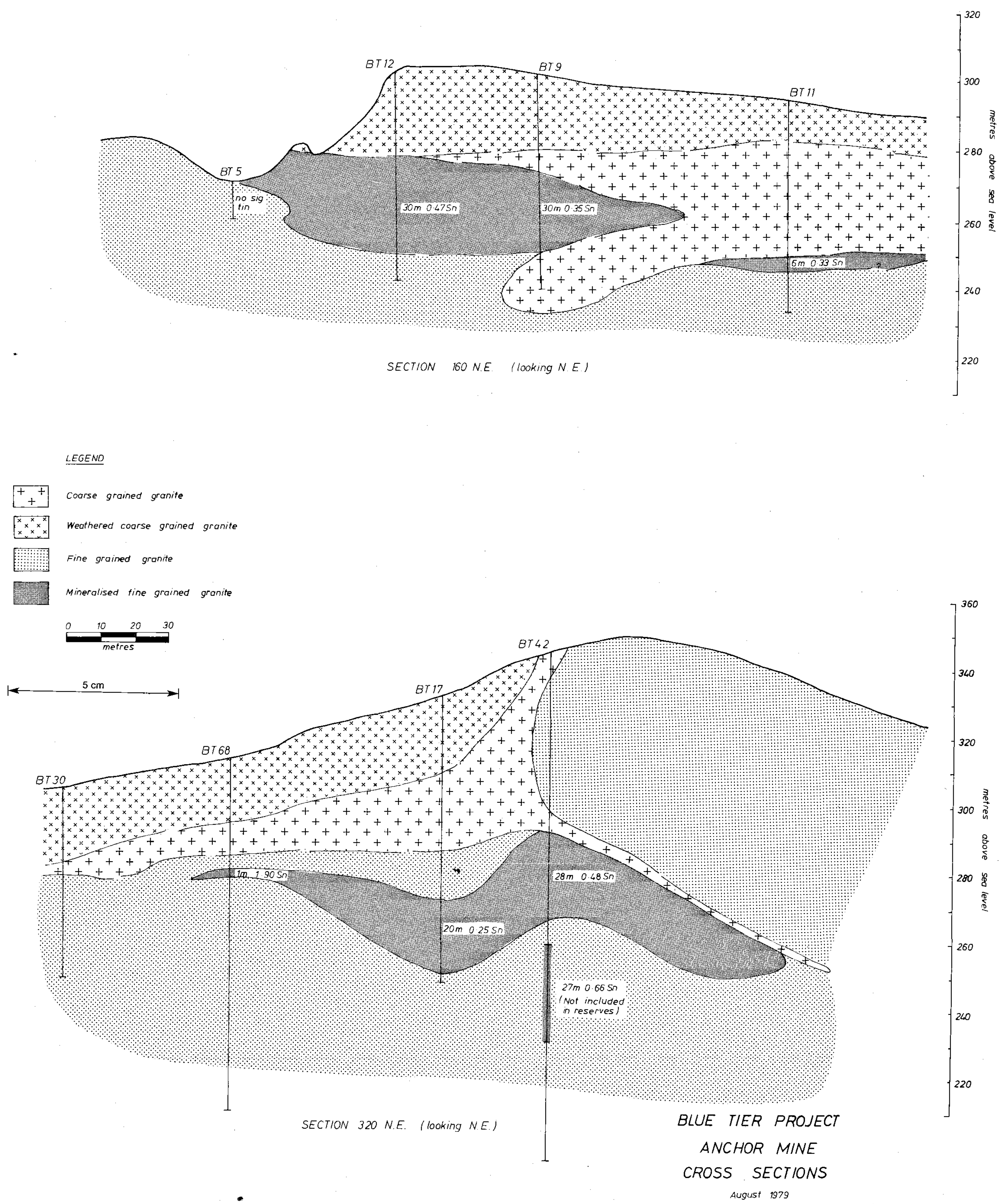
BT 12
BT 9
BT 11
BT 5
no sig tin
30m 0.47 Sn
30m 0.35 Sn
6m 0.33 Sn
320
300
280
260
240
220
metres above sea level
SECTION 160 N.E. (looking N.E.)
LEGEND
Coarse grained granite
Weathered coarse grained granite
Fine grained granite
Mineralised fine grained granite
0 10 20 30
metres
5 cm
BT 42
BT 17
BT 68
BT 30
1m 1.90 Sn
28m 0.48 Sn
20m 0.25 Sn
27m 0.66 Sn
(Not included in reserves)
360
340
320
300
280
260
240
220
metres above sea level
SECTION 320 N.E. (looking N.E.)
BLUE TIER PROJECT
ANCHOR MINE
CROSS SECTIONS
August 1979

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diamond drilling data on the Anchor Mine. A cut-off grade of 0.2% tin was used.

This reserve has largely been estimated by Geologists Wells and Ross, using several different methods on various occasions.

The mineralisation has been classified as "Possible Ore" and not a true reserve for several reasons.

(i) Ore continuity between drill holes has not been adequately demonstrated.

(ii) Drilling density is regarded as inadequate to confidently predict geological structures which control the ore.

(iii) Data from Aberfoyle drilling has been used in the estimation and its quality is not accurately known.

However, it is felt the estimates are sufficiently accurate to form the basis of an Indicative Feasibility Study.

In conducting the ore potential estimates, several features of the deposit were observed which are considered relevant.

(a) Whilst the deposit is generally low grade and thick, there were some thinner, quite high grade zones.

(b) Several zones of mineralisation have not been included in the estimates because their geological relationships to the main body are not clear.

The most significant of these is 27 m. 0.66% tin in BT42 which appears to be 7 m. beneath the main zone. The full significance of this very major intersection will only be determined by additional drilling.

(c) The ore zone so far defined is open to further extensions in several directions, and the chances of increasing the estimated tonnage are considered excellent.

Uning the geological "possible ore" figure, a mining system was designed and tonnages of ore and waste were then recalculated by Senior Geologist Wells. These tonnage estimates have been presented in detail in a separate report "Anchor Mineralisation, Possible Ore for Mining Proposal as at October 1979" by K. Wells.

A summary of this data is presented below:-

	Tonnes	Grade (% Sn)
Geological Ore	2,000,000	0.40
Open Cut Ore	863,924	0.38
Underground Ore	991,211	0.34
OPEN CUT OVERBURDEN	1,900,000	-

4.7. <u>Exploration Potential:</u>

Neither the exact geological nature nor the limits of potential economic mineralisation are fully known. Additional closer pattern drilling in the mineralised zone will be necessary if accurate tonnage and grade estimates are required.

The limits of mineralisation to the north-east, south and south-west (beneath the existing open cut) are not known and significant extensions in all these directions are considered likely.

A limited drilling program currently in progress has confirmed possible extensions south and south-west and has indicated additional drilling programs will be required to more fully delineate the extremities of the mineralisation.

This additional work is seen as highly desirable as an increase in tonnage could substantially effect the economics of a low grade deposit of this nature.

5. MINING

5.1. General

The first essential point for appreciation is that the ore reserve as outlined is relatively small, and is of low grade. The capital costs have to be minimised, and the lowest operating costs possible must be planned for in the mining methods and equipment, and milling, together with the overhead costs. If the operation is not envisaged as a tightly run marginal mine, it will fail.

The mining operations are envisaged as commencing with open pit mining and then moving to underground operations.

A small zone of approximately 60,000 tonnes geological reserve lies outside the main orezone and is a remnant remaining from cessation of past mining operations. This is immediately available by open pit mining and no stripping is required. Additional ore could be proved up in the area.

The main orezone would be opened up by stripping off the highly decomposed granite, which overlies the orezone directly in the S.W. and W. of the orezone, or has a thin layer of solid coarse or fine grained granite between the decomposed material and the ore.

The lack of sufficient solid rock cover between the ore body and the overlying decomposed material would make underground mining in this area impossible due to lack of competent roof.

The point of entry would be in the S.W. and W. from existing open pit faces. Once the decomposed material is stripped off very little solid overburden has to be stripped in the initial stages to uncover the ore. The orebody extends N.E. beneath steeply rising ground and the overburden depths begin to increase rapidly until open pit mining becomes uneconomic.

The ore made available from open pit mining would be sufficient for five years of milling operations, and a transition has to be made from open pit to underground work well before this time.

The orebody is very variable in thickness which is maximised along the N.E. - S.W. direction with thinning to the east and west.

The S.E. portion of the underground orezone lends itself to a Pillar and Stall stoping method up to 6 metre thickness. The ore dips to the S.E., but access can be obtained by cutting in on the contours from the open pit. The section immediately to the N.E. of the open pit would be mined by Post Pillar mining system. This is Pillar and Stall mining with fill so as to cope with the increase in ore thickness up to 20 m.

Fill would be derived from stripped overburden dumps. Another section of Pillar and Stall stoping would lie to the N.W. of this section.

The last major area is to the N.E. where the ore reaches major thickness of over 60 m.m but grades are low. Open stoping would be employed as the shape of the orebody and thickness ideally lends itself to this type of mining. However, this is the least known area and could change with new drilling.

Access to the underground stopes would be by means of a decline of 1 in 7 gradient which would spiral down to the bottom of the known orezone. Should the designated open stoping zone extend further N.E. then the bottom portion of the decline would have to be redesigned.

Ventilation would be provided by force fans during development and permanent airways on the N.W. side of the ore bodies.

The rock conditions in the open pit and underground should be excellent. This will permit steep open pit batters, and large supported openings underground.

Strong reservations exist concerning the rate of mining achievable from underground operations. A figure of 200,000 t.p.a., has been used for the purpose of this study, and it must be assumed that the underground development is virtually complete before stoping is commenced. To have any real expectation of sustaining 200,000 t.p.a. all the stoping blocks would have to be exploited together. This would have equipment savings as an advantage particularly in the drilling area as the drilling jumbos are specialised between open stoping and other drilling activities.

The overburden ratios to ore in the north east of the open pit appear excessive, and the pit could be redrawn to reduce the amount of overburden to be removed. This would place greater proportion of the reserve into the underground category.

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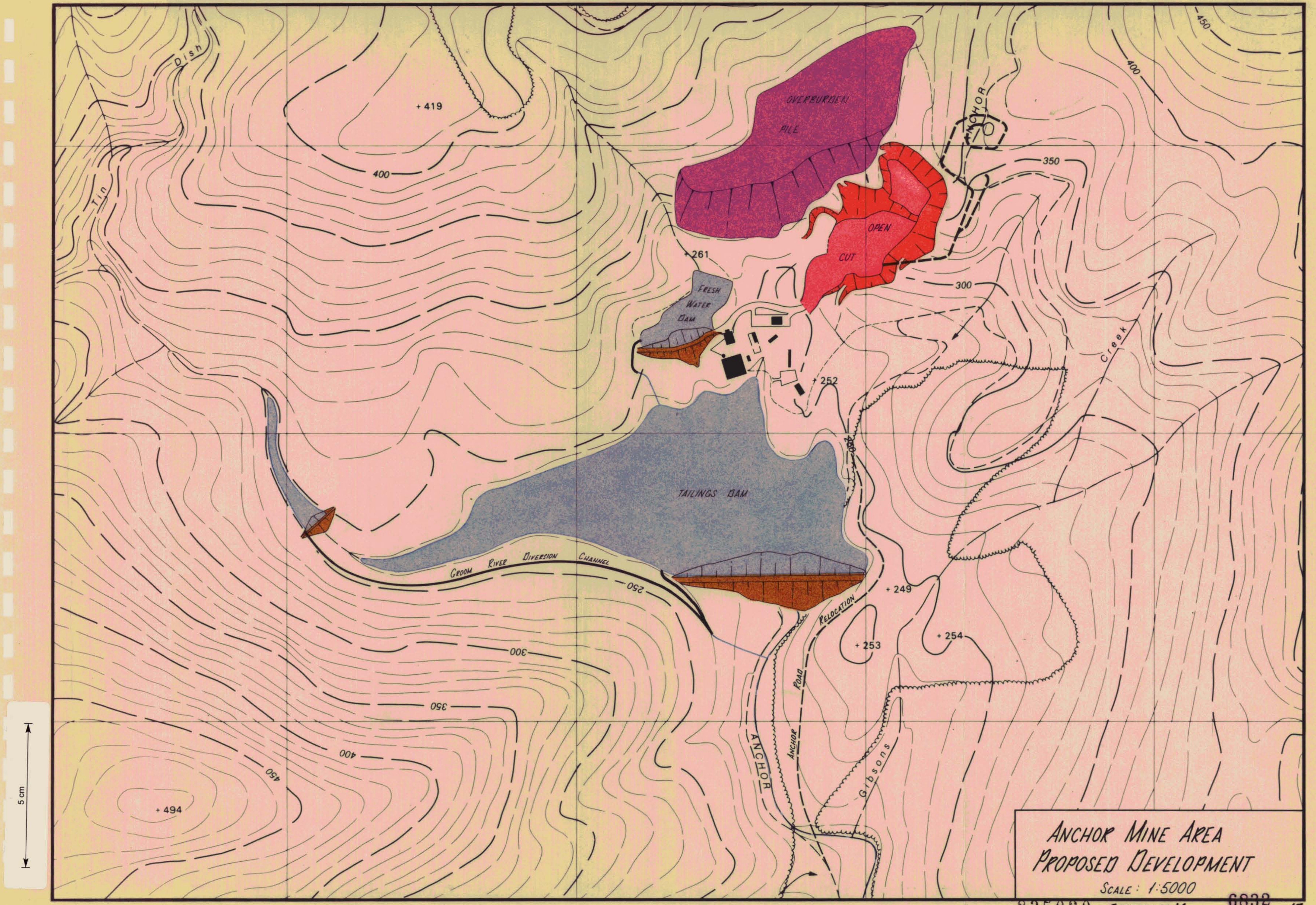
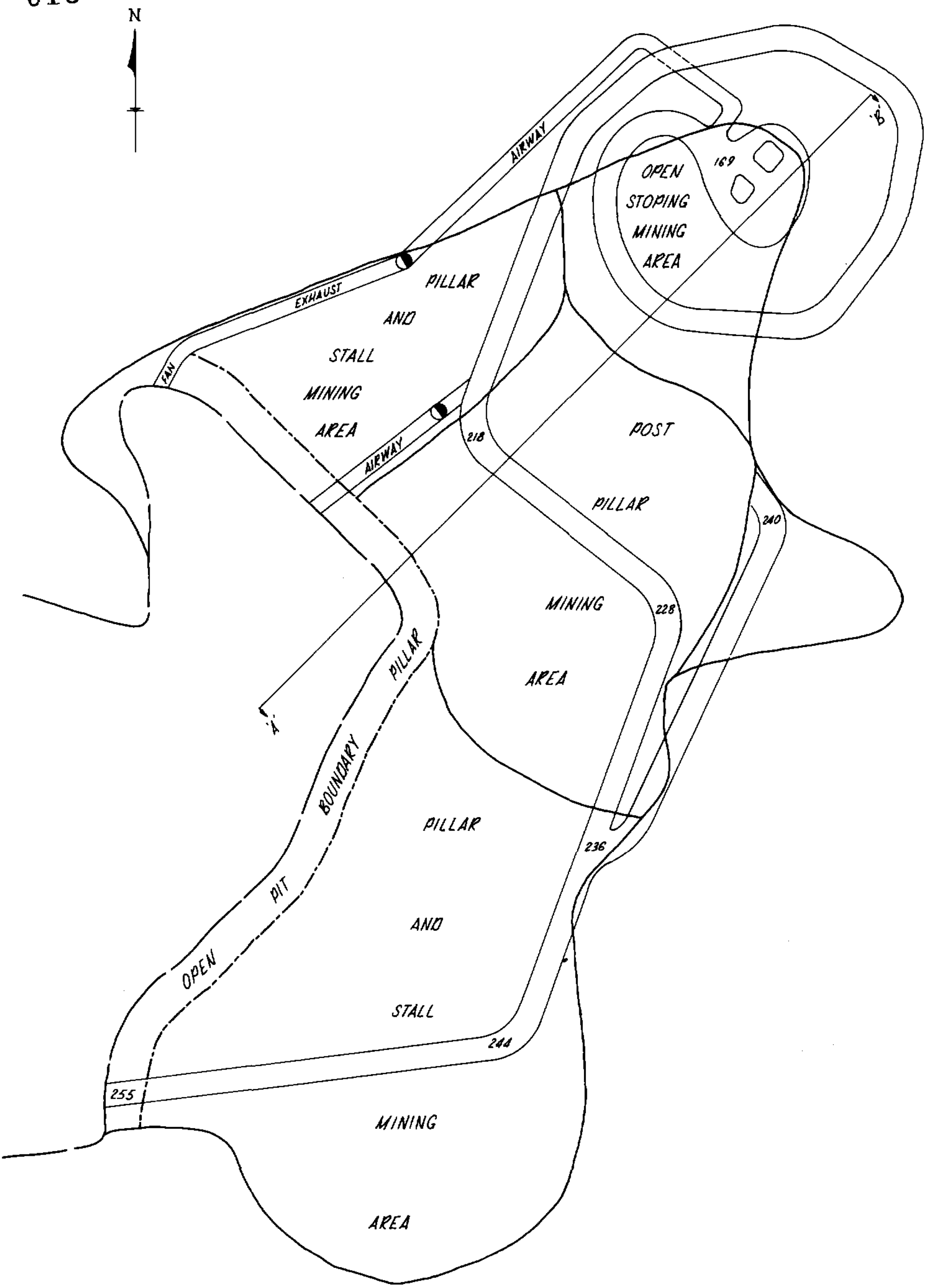
ANCHOR MINE AREA	TRACED	J.M.S
PROPOSED DEVELOPMENT	DATE	
	SCALE	1:1000
SCALE. 1:1000 METRES	DRAWING No.	

835019 ENCLOSURE IV 6831 16.



835020 ENCLOSURE V 6832 17.

018 835021



ORE RESERVE
CATEGORY PLAN

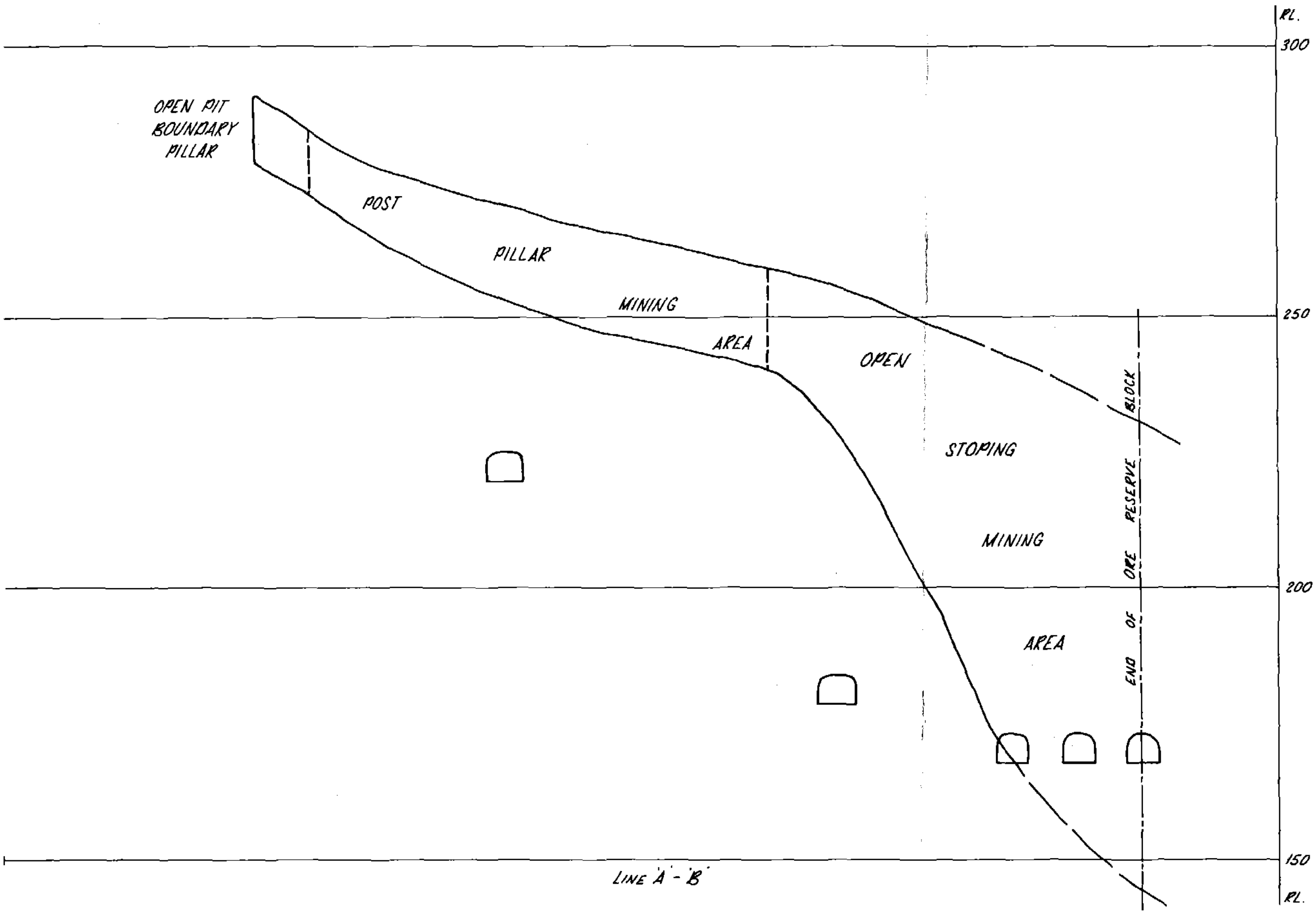
NOT TO SCALE

MINING RESERVES

ENCLOSURE VI

Category	Geological Tonnes	Geological Grade % Sn	Extraction %	Dilution %	Grade Dilution % Sn	Mined Tonnes	Mined Grade % Sn
Open Pit Main	864,000	0.38	100	10	0.05	949,000	0.35
Open Pit "Pods"	60,000	0.36	100	10	0.1	66,000	0.34
Open Stoping	313,000	0.31	100	10	0.1	344,000	0.29
Post Pillar	380,000	0.33	85	10	-	357,000	0.30
Pillar & Stall							
East Margin	20,000	0.37	80	5	0.1	17,000	0.35
North Margin	106,000	0.34	80	5	0.1	89,000	0.33
South Margin	179,000	0.42	80	5	0.1	145,000	0.40
Boundary Pillar	90,000	0.37	50	-	-	45,000	0.37
Underground						997,000	0.32
Open Pit						1,015,000	0.35

Enclosure VII



SECTION THROUGH PRINCIPAL DIP DIRECTION

NOT TO SCALE

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RENISON LIMITED - BLUE TIER

ENCLOSURE VIII:

SCHEDULE OF ACTIVITIES

	Quantity	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Clear Timber														
Strip Decomposed Overburden	526000 M^3	326000	200000											
Strip Rock Overburden	854000	100000	300000	200000	200000	54000								
River Diversion Channel														
River Diversion Dam														
Spillway														
Tailings Dam														
Water Dam														
Roads														
Prepare Mill Site														
Mill Erection														
Offices and Workshops Site														
Erect Buildings														
Power Line														
Substation														
Rehabilitation														
Close Down														
Open Pit Production	1015000T			200000 0.35	200000 0.35	200000 0.35	184000 0.35	195000 0.35	36000 0.35					
U/ground Production	997000T						Underground Developement							
							16000 0.37	5000 0.33	164000 0.32	200000 0.32	200000 0.32	200000 0.32	200000 0.32	12000 0.32

5.2. OPEN PIT MINING

5.2.1. Overburden Removal:

Decomposed Granite:

This material may be removed by scrapers, loader and trucks, or by bulldozing. It is assumed that an earth moving contractor will be retained for this work.

Overburden will be required for building impounding dams for water storage, and tailings impoundments, the latter being a significant structure for an operation of this size.

The material contains some clay, but should be easy digging material, and should not be sticky.

Loader and truck haulage has been assumed for the purpose of this exercise as this method would have to be used for hauling to dam construction sites because of this distance involved.

Final design batter angle has been taken as 50° which is slightly conservative when compared with existing on site batters. Maximum vertical height has been taken as 27 m., when a bench is designed to break the batter. Some care has been taken to avoid "perching" faces in decomposed material on top of solid rock part way up a final face batter. This should minimise danger from slippage.

In situ Granite Overburden:

Rock overburden will consist of coarse and fine grained granite, both highly competent. This strong rock type will permit high angle batters on working faces and, permit high final batters.

The slope of the ground after removal of decomposed material will be quite steep, and also tonnages of rock to be removed on the upper benches are not large. This implies that stripping of the decomposed material will need to be well advanced so that sufficient time exists for hard rock stripping on narrow benches that will have to be worked end on.

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There could be scope for contractor involvement, but the relatively restricted working areas could cause problems between mine and contractor crews.

There could be problems with grade cut off points as quite small grade changes affect large tonnages of material. A dump of marginal grade material should be stockpiled separate from barren material.

A nominal design bench height of 9 m. is proposed for working faces with a batter angle of 80°. Up to three benches would be run together for final pit batters. Provision would be made for working sub-benches of 4.5 - 4 m. for grade control at the top and bottom of the ore zone.

Rock would be removed by front end loader and truck after drilling with airtrac type units and conventional blasting.

5.2.2. ORE REMOVAL

The orebody consists of fine grained granite mineralised with cassiterite. Grade in the proposed open pit area varies from over 1% Sn to the cut-off 0.2% Sn, and below.

There appears to be an enriched upper zone varying between 6 - 10 m., and immediately underlain by material in the range 0.2 - 0.3% Sn. This lower grade material represents a significant tonnage.

A dilution factor on mining could be anticipated as about 10% of a grade of 0.1% Sn.

Ore would be removed in the same manner as rock overburden with similar bench heights and batters.

A system of grade control would be required, which could involve percussive drilling of the orezone and margins in advance of face blasting. Some rapid method of assay of cuttings, would be required, and also the possible use of in the hole logger equipment.

Grade control would be fundamental to the success of the operation as mineralisation is both erratic and variable in grade.

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5.2.3. DRAINAGE

The main area of the open pit will be free draining with the exception of two low points. Only one of these could cause a problem but is likely to be back filled early in the operation.

Use of a small face pump could be required for two short periods. No problems are foreseen.

5.3. UNDERGROUND MINING

5.3.1. ACCESS

A main haulage decline would be excavated from the open pit to gain access initially to the Pillar and Stall stoping zone in the S.E. It will then turn to head N. dropping beneath the orezones so as to turn N.W. under the Post Pillar mining zone. It then turns again N.E. to finally spiral around the open stoping area and reach the base of the orebody, and the draw points. Total distance is almost 500 m.

The decline will be 1 in 7 gradient for the majority of its length and would have finished dimensions of 7 m. wide by 5 m. high. The large dimensions are taking full advantage of the excellent ground conditions so as to allow open pit trucks to travel underground with very little modification.

The usual services will be carried in the roof of the decline.

The possibility exists for the proving up of an extra large pod, or zone of ore beneath the designated Post Pillar mining zone. The decline is designed to accommodate this zone if it is proved to exist.

A number of openings will be made off the open pit benches to gain access to the Pillar and Stall stoping areas, and to create ventilation returns.

A subsidiary ramp will be driven off the main decline to reach the bottom level of the ore in the Post Pillar stoping zone.

The exhaust rise system would be provided with ladders. Equipping the rise system will require an Alimak rise climber on site.

5.3.2. STOPING SYSTEMS

The grade of the orebody to be extracted by underground methods is relatively low and a cheap stoping cost has to be the aim. This implies high mechanisation and bulk mining methods even though the reserve tonnage is not large in absolute terms.

Mobile diesel equipment adopted from open pit mining has been adopted as a capital cost saving measure, and stoping systems dependent upon a minimum of waste development, and support has been adopted.

The orebody lends itself to three stoping systems, all of which are highly méchanised.

Pillar and Stall Stoping:

This would be the first stoping system to be initiated and would begin at an early stage by development from the open pit in the S.E. area.

Strike drives would be driven along the contour with break throughs being made on the dip.
Up to 7.5 metres height could be taken in one pass, but in practice this would more likely be 6.0 m. The dip generally is too steep for up and down dip travel. An 80 percent extraction could be anticipated with varying pillar height - width ratios depending upon the thickness of the ore. Total extraction is doubtful because of the close proximity of the open pit, and possible undermining of the open pit batters.

Minimum stoping height will be 3 m. Narrower thicknesses could be chased, but different mucking arrangements would be required. This would depend upon the ore grade.

Mining would be by multi-boom drilling jumbo, loaders and trucks.

Post Pillar Stoping:

This is a variant of Pillar and Stall stoping where the lowest level of ore is extracted leaving pillars, followed by introduction of fill to form a new working floor and surround the pillars. Another level of ore is then extracted and so on until the top of the ore is reached. This system can cope with irregular ore outlines and fluctuating grades, and can cope with thickness of ore exceeding 30 m. if needed.

Fill would be obtained from overburden dumps and trucked back into the mine, or from any waste development that may be current at the time.

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Open Stoping:

The deepest and lowest grade section of the orebody has the rough shape of an inverted cone, which lends itself to long hole stoping with extraction from draw points at the lowest level.

The thickness of the ore is up to 67 m. and it would be proposed to drill out this part of the orebody by down-the-hole hammer rigs and six 1/2 in. holes. Once a cut-off raise is opened the full height of the ore, and access gained to the top of the orebody, drill holes can be blasted to open up a slot from the cut-off raise. Successive slices of ore will be blasted off, and ore extraction from the draw points by loaders.

Mining would require multi-boom drill jumbo to develop the block, and the down-the-hole hammer rigs to drill the long holes. Loaders would load trucks in the draw point area.

Return ventilation would require an exhaust drive and connection rise to the exhaust system at higher levels.

The development of the rises would be the greatest problem. Alternatives would be to rise bore by contract (expensive), rent equipment and rise by Alimak climber, or attempt to crater rise by long hole drilling (cheapest, but requires expertise).

Another alternative drilling method would be to split the block into two vertically, and use standard long hole drilling utilising redundant drilling rigs from the open pit operation.

5.3.3. VENTILATION SYSTEMS

The main decline would form the main air intake and this would be ventilated in the development stage by 125 h.p. exhaust fan and 60 h.p. overlap fan.

The contour drives from the open pit in the Pillar and Stall stoping system would serve as exhausts as these are broken through.

A rise from the proposed Post Pillar block would link the decline development, and stoping to an exhaust drive to the open pit on the N.W. side of the orebody.

Later an exhaust drive from the open stope draw point area would come back under the initial exhaust rise system and be linked to it.

Extra forcing fans would be required for ventilating development ends. Another two 125 h.p. fans and another three 60 h.p. fans should be allowed for.

400 m. of 1.0 m. diameter ventilation ducting would be required.

Final exhaust fan would be capable of 150,000 c.f.m. and would require about 150 h.p. as mine resistance should be low.

Mine Pumping:

Mine pumping requirements are inprecise at this stage. No water flows have been reported from diamond drilling. A system capable of 100 g.p.m. has been allowed for. Stage pumping from small sumps by 25 h.p. submersible pumps would be proposed.

5.4. OVERBURDEN DUMP AREA

The sites available for dump areas are restricted to two steep valleys to the N.W. and E. of the final pit limits. The valley to the N.W. would be utilised and capacity exists for all overburden both unconsolidated and rock.

Total unconsolidated overburden is estimated as 526,000 cu. m. and rock overburden as 587,000 tonnes.

Large amounts of material would be required for construction of dams for fresh water storage, and particularly for a tailings dam.

Tailings dam requirements are 115,000 cu. m., and fresh water dam 14,700 cu. m.

The water flow in the valley would need to be diverted, or suitably channelled, so that the stability of the overburden dump is not affected.

5.5. REHABILITATION

Some rehabilitation of the open pit area, and overburden dumps could be affected by placing a layer of decomposed granite and top soil over the areas in question. Self regeneration in the old mining areas is very marked, and response on future rehabilitation is likely to be very good.

Tailings dam rehabilitation is likely to revolve around creation of a lake type setting as significant water storage will remain within the dam.

RENISON LIMITED - BLUE TIER

ENCLOSURE IX:

LIFE OF MINE PRODUCTION SCHEDULE

	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Total
Ore Production			200000	200000	200000	200000	200000	200000	200000	200000	200000	200000	12000	2012000
Grade - % Sn			0.35	0.35	0.35	0.35	0.35	0.33	0.32	0.32	0.32	0.32	0.32	0.34
Tonnes Sn to Mill			700	700	700	700	700	660	640	640	640	640	38	6758
Recovery 85%			85	85	85	85	85	85	85	85	85	85	85	85
Tonnes Sn Recovered			595	595	595	595	595	561	544	544	544	544	32	5744
Grade of Concentrate			55	55	55	55	55	55	55	55	55	55	55	55
Tonnes of Concentrate			1082	1082	1082	1082	1082	1020	989	989	989	989	58	10444
Moisture 8%			87	87	87	87	87	82	79	79	79	79	5	838
Tonnes of Concentrates Shipped			1169	1169	1169	1169	1169	1102	1168	1168	1168	1168	63	11282

6. CONCENTRATING

6.1. Metallurgical Testwork:

Metallurgical testwork on the Blue Tier prospect is limited to heavy liquid assessments on mineralised intersections from nine drill holes, some super panning separations and mineralogical work. This work has indicated the following characteristics.

(i) The cassiterite grainsize is relatively coarse and extremely variable. The variation in cassiterite liberation with particle size has been determined.

(ii) Results suggest a coarsening of cassiterite towards the north-eastern extensions of the orebody.

(iii) Significant quantities of minerals such as topaz and biotite occur, which are likely to cause problems producing clean cassiterite concentrates.

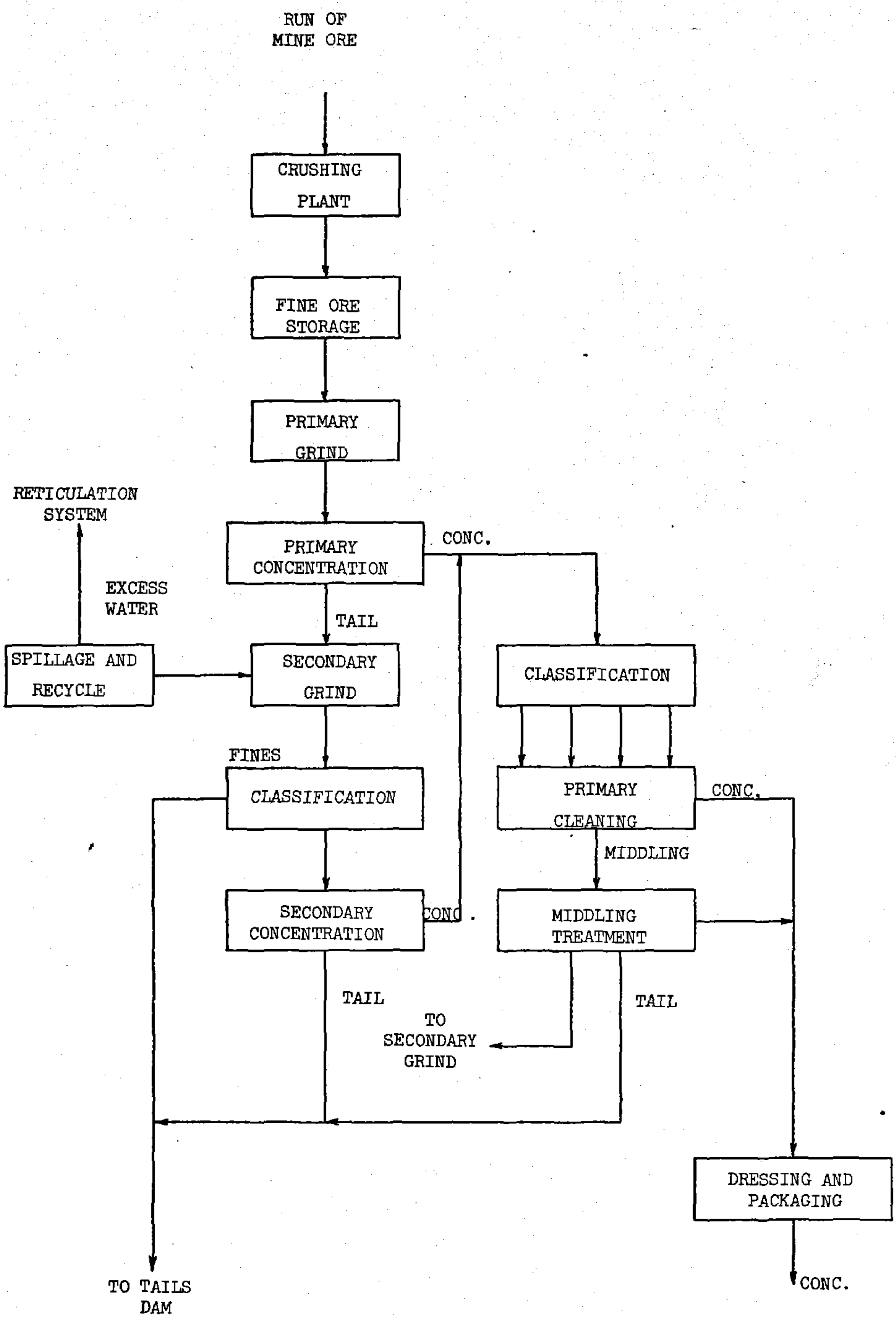
(iv) Sulphides are present in trace amounts and do not generally appear in composite with cassiterite. The sulphide assemblage is complex and variable and can include sulphides of copper, iron, arsenic, zinc, bismuth and molybdenum.

(v) Silver is present but its mode of occurrence has not been clearly defined. Evidence suggests it can occur as a sulphide and is concentrated to the cassiterite concentrate.

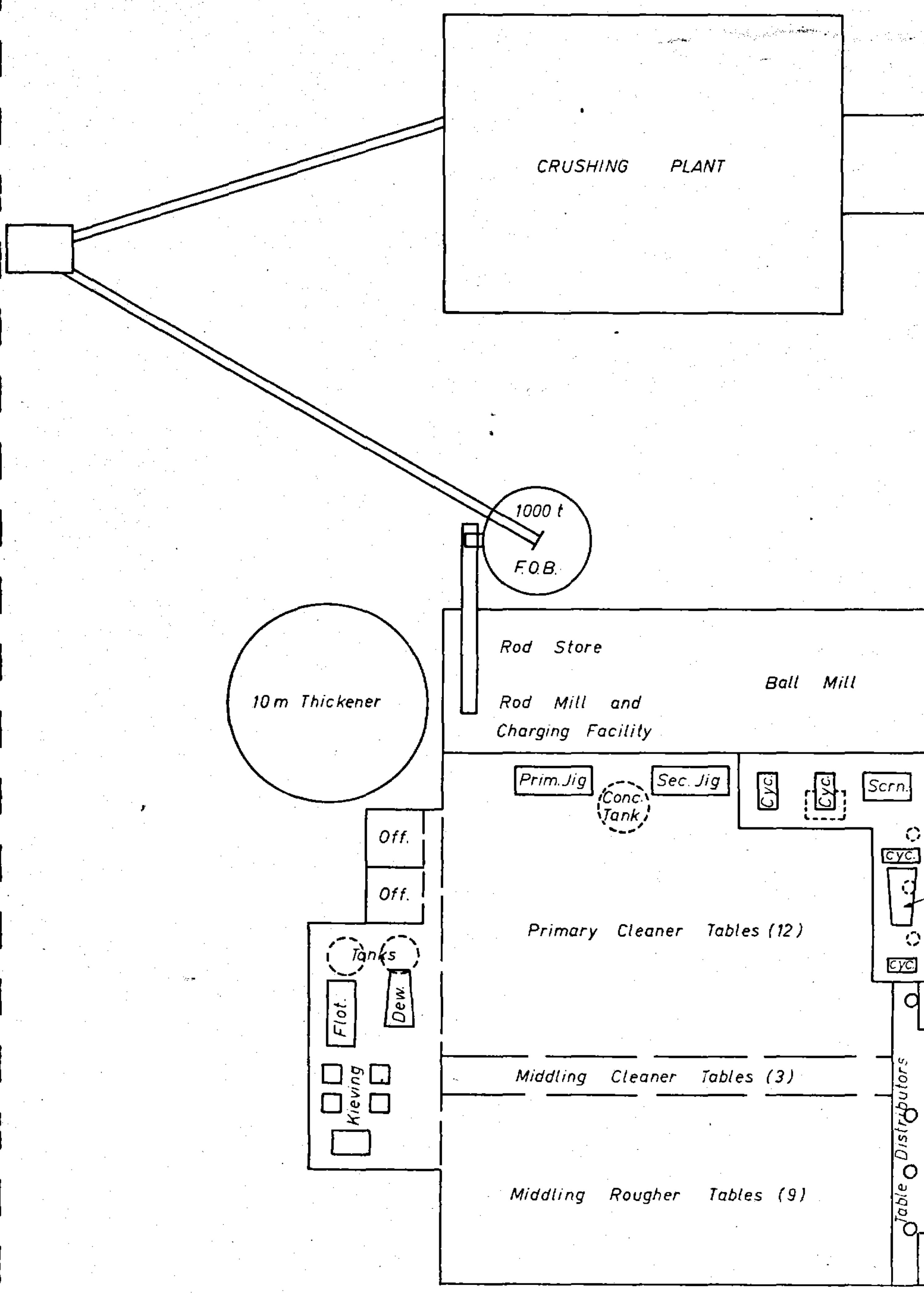
Based on these results a preliminary treatment flowsheet has been designed which aims at early recovery of coarse cassiterite to minimise overgrinding and which uses high capacity primary concentration devices to reject low specific gravity barren silicates as early as possible in the process. For the purpose of this study a constant grinding product size has been assumed for all areas of the orebody.

BLUE TIER TREATMENT FLOWSHEET

ENCLOSURE X:



032 BLUE TIER CONCENTRATOR LAYOUT 835036



Not to scale — schematic only

033 It is estimated total tin recovery will be 85% to a 55% tin concentrate.

From results available it has not been possible to determine

(i) if special separation techniques need to be included to remove topaz etc. from the final concentrates.

(ii) If regrinding of final concentrates prior to a sulphide flotation stage is needed to adequately reduce sulphur levels.

(iii) The potential value of silver in sulphide concentrates or wolfram recovery. No allowance has been made for these factors in the cash flow model.

Considerable metallurgical testwork on bulk ore samples will be necessary to adequately define treatment requirements for Blue Tier ore should the decision be made to proceed beyond this study.

6.2. METALLURGICAL FLOWSHEET

6.2.1. Crushing:

The crushing plant reduces run-of-mine ore to a particle size of less than 12.5 mm in three stages. Primary crushing is in a jaw crusher, with secondary and tertiary crushing in cone crushers. Screens prior to the secondary and tertiary stages remove fines before crushing. Fine ore is conveyed to a 1,000 tonne storage bin.

6.2.2. Grinding:

Grinding is carried out in two stages. A rod mill first reduces the fine ore to approximately 1 mm. After recovery of coarse cassiterite, a ball mill operating in closed circuit with D.M.S. screens further grinds the ore to finer than 500 microns.

6.2.3. Concentration:

Cassiterite is recovered from rod mill discharge using a jig. This early recovery minimises the possibility of overgrinding coarse valuable mineral. Jigs and spirals are used to recover cassiterite from ball mill/screen product. Tails are pumped to the tailings dam.

6.2.4. C[illegible]e Treatment:

T[illegible]entrates are dewatered in a spiral d[illegible] then treated in a flotation stage to re[illegible]lphides. Cassiterite concentrates are fu[illegible]graded and dewatered by kieving.

RENISON LIMITED - BLUE TIER STUDY

ENCLOSURE XII:

UNESCALATED CAPITAL EXPENDITURE ESTIMATES - $'000

ITEM Year	1	2	3	4	5	6	7	8	9	10	11	12	13	Total	Salvage
River Diversion	290													290	-
Tailings Dam	263													263	-
Water Supply	7													7	
Power	300													300	30
Road and Site Works	40													40	
Mine Development						271	216							487	-
Mine Equipment															
Air Trac Drills	100													100	14
Compressors	68													68	10
Drifters	28													28	-
D. Hole Hammer							60							60	7
2 Boom Jumbo						300								300	30
Cat. 980 Loader	370													370	20
Cat. 769 Trucks	280	140												420	30
Cat. 120G Grader	60													60	5
Cat. D8 Dozer	180													180	20
Service Units	114		18	14	184	100	32	90	18	14	18			602	22
Alimak Rise Climber						50								50	20
Underground Equipment						100								100	10
Crusher	490	1000												1490	100
Concentrator	1168	2317	30	30	30	30	30	30	30	30	30	30		3785	
Lease Buildings															
Administration Block	56													56	
Stores Block	29													29	
Change/1st Aid/Lamp	38					7								45	90
Workshop	115													115	
Assay Office	16													16	
Housing	240													240	240
Surface Plant	56			23			23	2		23				127	10
Fixed Surface Plant	82													82	10
TOTAL	4390	3457	48	67	214	858	361	122	48	67	48	30		9710	668

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7. CAPITAL EXPENDITURE

7.1. GROOM RIVER DIVERSION

The diversion is assumed to consist of a small dam upstream of the tailings dam area at the approximate 247 R.L. This would divert the river into a diversion channel cut around the hill side which would discharge into the existing river valley below the tailings dam. The diversion channel is 700 m. long and is assumed to be a shallow U shape with a 3 m. concrete invert at the bottom. Depth is assumed to be 1 metre on the sides and 2 metres in the centre overall width is assumed at 10 metres.

The practical translation of these assumed dimensions to on-site excavation have not been examined, and it is not known if such a channel can be created.

A cost of $10.00 cu.m. excavated is assumed	=	$150/m length
Consolidation and compaction of bed		$ 20/m length
Concrete invert		$100/m length
Other		$ 30/m
		$300/m
Over 700 m.		$210,000
Diversion Dam		40,000
Spillway		20,000
Consultancy		20,000
		$290,000

Tailings Dam - Volume of Overburden		194,000 cu.m.
Insitu S.G. 2.0, Tonnage of Overburden		388,000
Transport and Placement Cost		
388,000 x $1.35	=	$525,740
If Stripping only -		
388,000 x $0.87	=	$337,560
		$188,180

Clearing and site preparation	$ 20,000
Spillway	$ 25,000
Placement Cost (incremental over stripping)	$188,000
Consultancy Fees	$ 30,000
	$263,000

7.2. TAILINGS DAM AREA

This is the most difficult problem of the whole study. The valley finally selected for the overburden dump was examined for suitability as a tailing disposal area. However, the capacity was inadequate, and the site not well regarded from the view of dam stability.

The only possible site in the immediate locality is to the south of the mine area, where large flat areas exist. The area is flat probably because all overburden and tailings from old mining operations were sluiced down the hill into the valley.

The Groom River flows through this area. This small river is one of tributory head waters of the catchment for the St. Helens water supply.

The current proposal is to divert the Groom River around the hill side, and dam the valley for the disposal of tailings. A dam could be constructed from overburden material. Greatest height would be approximately 20 m. and a crest length of 330 m. Capacity would be 1,239,000 cu.m. to 239 R.L. The volume of material to build the wall would be 194,000 cu.m.

The tailings dam would also impound water for recirculation to the mill.

The Groom River diversion channel could be a costly exercise as 700 metres have to be rechannelled.

Rainfall figures have been examined, and it is unlikely that provision could be made to cope with a 30 year flood situation. In this case excess flood water would divert to the tailings dam and exit via the dam overflow spillway. The tailings are innoxious being crushed granite with no sulphides, and apart from milky colouration, which could probably be masked by creek run off, there should be no problem.

The various Environmental Boards would need to be convinced of this.

7.3. WATER SUPPLY

It would be proposed to utilise part of the old open pit workings together with an earth dam to impound a fresh water supply of approximately 5.4 million gallons at capacity.

The bulk of the capacity lies withing a deep pit, which was originally excavated by means of two small adits. These adits will have to be blocked with concrete dams before flooding. The pit can be filled to a level of 245 R.L. initially with the level controlled by a valve in the upper adit dam. Ore exists on the northern margin of the dam and this would have to be removed before the water level could be allowed to rise further.

The earth dam would be constructed by the contractor stripping the overburden from the open pit.

A bulldozer and a compactor would be needed for dam construction. It is anticipated that this would be included in the contract price.

The volume of overburden required for the water supply dam would be 14,700 metres. This would involve the hauling of 29,400 tonnes of overburden a distance of 300 m. The additional cost of placing this overburden is minimal. $7,000 has been allowed.

7.4. POWER AND TELEPHONE

This is a problem area as the H.E.C. line that runs from Derby to St. Helens does not have the capacity to supply the power required by mining operations. The limited amount of feed back is that the line upgrading could cost $200,000. The Government should be encouraged to absorb this cost if an operation were to commence. The 6 km. of spur line into the mine site would be to the cost of the operation. The probable cost per km. of this line is $15,000 giving a total cost of $90,000.

The tariff charge is likely to be at the top rate of the industrial scale.

A main substation would be required - probable cost is $200,000.

A total cost of $300,000 has been allowed.

7.5. ROAD AND SITE WORKS

An allowance of $40,000 has been allowed for initial creation of roads and drainage around the site area. This would require review in the final planning stage.

7.6. MINE DEVELOPMENT

	Year 6		Year 7	
Decline at $236	480 m.	113,280	15 m.	3,540
Incline at $236	120 m.	28,320	120 m.	28,320
Drives & X-cuts at $236	40 m.	9,440	100 m.	23,600
Rises at $168	50 m.	8,440	140 m.	23,520
		$159,440		$78,980
Scaling Unit		8,000		8,000
		$167,440		$ 86,980
Labour		104,000		129,000
		$271,000		$216,000

7.7. MINING EQUIPMENT

7.7.1. Drilling Equipment

Single boom airtrac type units would appear preferable to in-the-hole drilling equipment because of the low bench height and the need to have good fragmentation. Improved grade control will also be a factor. Capital costs are much lower and the smaller units are quite adequate for the duty envisaged.

It is assumed that extension steel and 3 1/2 in. bits would be used in the drill string.

Compressed air drilling is assumed. Two drilling rigs are just adequate for 200,000 t.p.a. ore, and 200,000 t.p.a. overburden. Any increases in tonnage rates would require an entire rig and operator, or 10 hour shift on single shift. Initial purchase of hydraulic drilling rigs could also be considered.

One drilling unit is assumed to have a productivity of 12 m/hr., or 78 m. per shift of 6.5 hours. On a 2.5 x 2.5 drilling pattern this results in a capacity of 1,100 t.p.s.

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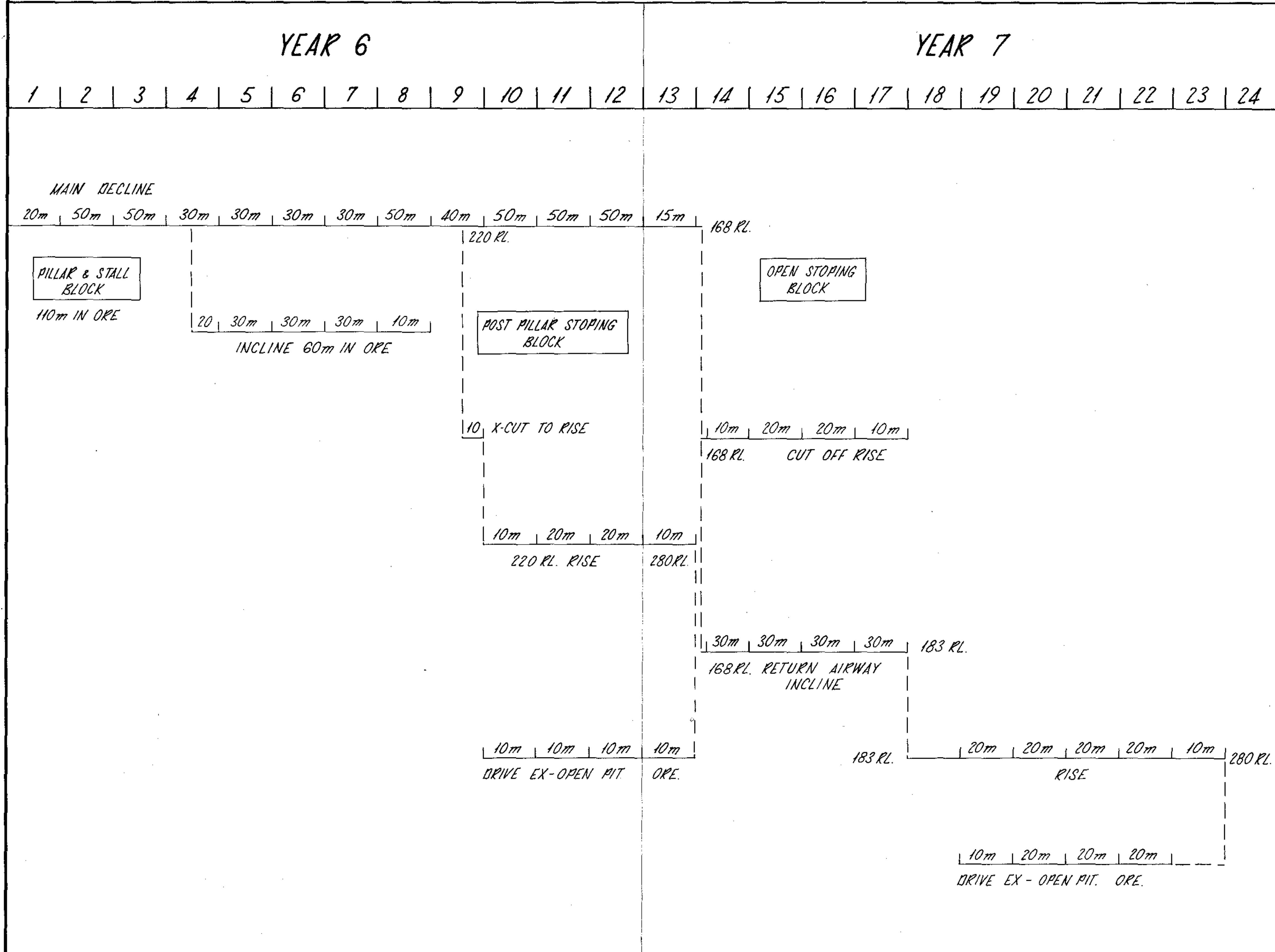
Development Schedule — Underground Capital



1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	
20	50	50	30	30	30	30	50	40	50	50	50	15												DECLINE
			20	30	30	30	10						30	30	30	30								INCLINE
								10	10	10	10	10						20	20	20	20	10		DRIVE & X-CUT
									10	20	20	10	10	20	20	10		10	20	20	20			RISE
1400	3600	3000			1300	2000	700		1000	1000	1000	1000						700	1450	1450	1400			ORE (TONNES)
0·40	0·40	0·40			0·35	0·35	0·35		0·33	0·33	0·33	0·33						0·33	0·33	0·33	0·33			GRADE (%)

Two drilling rigs would be required on site for ore and overburden drilling at the rate of 200,000 t.p.a. for each activity. As the overburden tonnage reduces, one unit would pass to standby.

For underground mining, this is the most critical item of equipment for underground work. It is very desirable to work the mine on a single shift as other equipment has the capacity to do this, and also supervision and its cost is minimised.

To produce the daily requirement of 530 t.p.d. from Pillar and Stall and Post Pillar stoping will require over 420 metres of blasted drill hole. One hydraulic drilling rig would be capable of coping with this duty. Alternatively, two 3-boom pneumatic jumbos would be required, plus extra operators plus a fixed compressed air installation of at least 2500 cfm capacity.

The hydraulic drilling rigs would have the same purchase cost of one and a half pneumatic jumbos, and the savings in compressed air capital expenditure and the extra operating cost over several years operations would be very significant.

A down-the-hole hammer rig for open stoping would be required later in the operation if this is the final choice of drilling method. One rig would be required which would be capable of coping with the tonnage of about 270 t.p.d. Compressed air would be required to operate this rig. This would best be supplied by electric powered compressor underground.

The options of lease, or rental have not been explored, and purchase has been assumed in each case.

7.7.2. Loading Equipment

The low bench heights and the need for close grade control will enforce relatively small tonnage blasts. The need for flexibility and mobility of loading equipment indicate the use of diesel powered front end loaders. Also these can later be adapted to underground mining.

The use of Caterpillar 980C loaders would be suitable for this application with an estimated loading capacity of 200 t.p.h. when matched with a Caterpillar 769C haul truck. This is well within the requirements of tonnage demand of 800 t.p.d. of ore and 800 t.p.d. waste with two units on single shift operation.

It is assumed the open pit loading equipment will be modified to work underground. This could be scheduled as the overburden stripping is completed. A very tight schedule could indicate that a second hand unit be used initially.

7.7.3. Haulage Equipment

Caterpillar 769C 35 ton capacity haul trucks will match well with Caterpillar 980C loaders, and will give the required duty. They are also very suitable for conversion to underground haulage. Two units would be required on site during the stripping phase, and a third would be advisable as back up.

A Caterpillar D8H Bulldozer will be necessary for clearing the benches for drilling and removal of toe rock. It will also be indispensible for road building at various times in the operation.

A Caterpillar 120-G grader will be required for grading of surface roads, pit haul roads, and later for underground haul road grading.

A heavy roller should also be purchased for road building. This can be towed by the grader.

The open pit trucks would be modified for underground use. There is no technical problem as this has been done many times.

7.7.4. Service Units

The tonnage involved does not justify large specialised explosive vehicles. Mixing of ANFO explosives would be carried out by a standard mixing unit and transported to site by a small explosives vehicle equipped in line with Regulations.

Blasting will be by ANFO explosives with dynamite boosters. Initiation would be detonating cord and fuse firing.

A Toyota 4 W.D. will be required for supervision of operations and also transport of supplies, particularly to drills.

The relatively large size of underground openings will require the use of a mobile unit fitted with a telescopic boom and work basket for charging of faces for blasting, barring down and installation of services.

7.7.5. New or Second Hand Equipment

The active life of the operation, both open pit and underground, would be between 8 - 10 years at a rate of 200,000 to 250,000 t.p.a. The general aim is to work a single shift mining operation at the 200,000 t.p.a. level with extension to 10 hour shift in open pit working at 250,000 t.p.a. The higher rate could only be satisfied from underground mining by two trucks operating on single shift.

The operation would be looking at 16 - 20,000 trucking hours, and 12 - 16,000 loader hours.

If three trucks are required 5 - 6,500 hours are required over the life of operation. Second hand trucks could be purchased and overhauled at a cost of about $150,000 each, and they would still have a good resale value at the end of the operation. A new truck of the same type would require the expenditure of $250,000 each.

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Second hand loaders and drills are not recommended. A good used grader would be adequate for the work envisaged. A new bulldozer would be needed.

7.8. UNDERGROUND EQUIPMENT

The following items were allowed for:-

	Quantity	Price	Total
Submersible Face Pumps 5 h.p.	3	3,000	9,000
Submersible pumps sump 25 h.p.	3	6,000	18,000
125 h.p. Exhaust Fan	3	9,000	27,000
60 h.p. Exhaust Fan	4	5,000	20,000
1,000 mm. Ventilation Ducting and Joiners	400 m.	2,500	10,000
			$ 84,000
Other			16,000
			$100,000

7.9. CONCENTRATOR

The concentrator consists essentially of a grinding aisle, an operating level for concentration equipment and a pumping/spillage collection level below the concentration equipment.

Classification equipment and table distributors are located on raised platforms as shown so products can gravitate as required.

The general arrangement is shown in Enclosure XI and details of capital estimates are itemised in Enclosure XII.

7.10. CRUSHER

Crushing equipment is contained in a separate building to the main concentrator. The three stages of crushing are laid out so the ore can gravitate from one unit to another. An area is provided in the crusher building for preparation of replacement parts etc.

The general arrangement is shown in Enclosure XI and details of capital estimates in Enclosure XII.

RENISON LIMITED - BLUE TIER

ENCLOSURE XIV;

MILL CAPITAL - CRUSHING EQUIPMENT

ITEM	DETAILS	TOTAL H.P.	BUDGET $	INSTALL $	TOTAL COST $
1. Crushers					
- Jaw	42" x 32" double toggle	100	156,000	70,000	226,000
- Secondary	7 - 36" cone	125	70,000	40,000	110,000
- Tertiary	2 - 36" cone	125	70,000	40,000	110,000
2. Screens					
- Secondary Feed	5' x 12' double deck, type XH	15	15,000	15,000	30,000
- Tertiary Feed	5' x 16' double deck, type XH	15	15,000	15,000	30,000
3. Feeders					
- Jaw	4' x 16' low head, electrical/mechanical	20	22,000	10,000	32,000
- Secondary	4' x 16' low head, electrical/mechanical	20	22,000	10,000	32,000
4. Bins					
- R.O.M. Ore	100 tonne capacity - steel, 4m x 4m x 3m		30,000	-	30,000
- Backload Hopper	Steel, on F.O.B. Feed conveyor		15,000	-	15,000
5. Conveyors					
- Tertiary Feed	12m at $1400/m	10			16,800
- F.O.B. Feed	75m at $1800/m i.e. 18° rise, 5m diff. in mill/crusher R.L.	40			135,000
- Transfer Tower					25,000
6. Miscellaneous					
- V-belt drives	All crushers				7,500
- Motors	All crushers				14,000
- Rotoclone/Ducting/Pump		60			65,000
- Cleanup	Bucket elevator, etc.	3			70,000
- Overhead Cranes	1 x 10 tonne capacity, 1 x 5 tonne capacity	20			50,000
- Metal Protection	3 magnets				9,000
7. Crusher Building	12 x 20 metres = 240m^2 at $1250/m^2				300,000
8. Electrical	14% of equipment and building costs (includes local transformers but not switchyard needs) i.e. 14% of $1,307,300				183,022
TOTAL		533			1,490,322

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RENISON LIMITED - BLUE TIER

MILL CAPITAL - CONCENTRATOR EQUIPMENT

ITEM	DETAILS						TOTAL H.P.	BUDGET $	INSTALL $	TOTAL COST $
1. Rod Mill	5' x 12', 120 H.P., all incl.						120	125,000	50,000	175,000
2. Ball Mill	7' x 8', 150 H.P.						150	150,000	50,000	200,000
3. F.O.B.	1000 tonne capacity - steel 6m Ø									150,000
4. Conveyors, Feeders										
- F.O.B. Discharge	V/S belt						10	40,000	10,000	50,000
- Rod Mill Feed	10m at $1400/m						10			14,000
5. Screens										
- Primary grinding	DSM panels, 1mm aperture, 2 x 3ft panels							15,000	15,000	30,000
- Concentrates	DSM panel, 1 x 2ft							7,000	7,000	14,000
6. Pumps	TPH Solids	% Solids	m^3/hr	Cyclone Feed ?	Pump Size	H.P.				
- Ball Mill Discharge	96	45	142	No	6/4	30	30			25,000
- Jig Feed Cyclone	40	35	85	Yes	6/4	50	50			25,000
- Spiral Feed Cyclone	12	12	57	Yes	6/4	50	50			25,000
- Rod Mill Discharge	40	60	40	No	4/3	10	10			15,000
- Jig/Spiral Concs - variable speed	5	25	15	Yes	3/2	10	10			20,000
- Fines cyclone	2	10	25	Yes	3/2	20	20			12,000
- Middling Transfer - 3 off	2	20	10	No	2/1.5	10	30			30,000
- Cleaner Middling Recycle	2	20	10	No	2/1.5	10	10			10,000
- Regrind Transfer	6	20	30	No	3/2	10	10			12,000
- Concentrate	0.5	10	5	No	2/1.5	5	5			10,000
- Middling Conc. Transfer - 3 off	0.5	25	2	No	2/1.5	5	15			30,000
- Tails	40	50	50	No	4/3	20	20			15,000
- Spillage - 6 off				No		15	90			40,000
- Batch Flotation Feed	2	50	2.5	No	2/1.5	5	5			10,000
- Kieve Feed	2	50	2.5	No	2/1.5	5	5			10,000
7. Cyclones	TPH Solids	% Solids	m^3/hr	d_{50}	No. Off	Size				
- Jig Feed	40	35	85	74	2	20"				28,000
- Spiral Feed	12	12	57	30	2	10"				14,000
- Jig/Spiral Concentrates	5	25	15	30	2	6"				8,000
- Fines cyclone	2	10	25	30	2	6"				8,000
	Sub-Total Items 1. - 7.						650			980,000

RENISON LIMITED - BLUE TIER

MILL CAPITAL - CONCENTRATOR EQUIPMENT (continued)

ITEM	DETAILS	TPH Solids	% Solids	m^3/hr	TOTAL H.P.	BUDGET $	INSTALL $	TOTAL COST $
8. Jigs								
- Primary Jig	2 cell Yuba Jig	40	60	40	4			25,000
- Secondary Jig	2 cell Yuba Jig	27	60	27	4			25,000
9. Spirals	4 twin start/7 turn, with frame					10,200	10,000	20,200
10. Shaking Tables	24 Holman Tables, each $16,000 installed				48			384,000
11. Holding Tables								
- Jig/Spiral Concentrate	6 TPH at 25% solids, 30 min. residence time $10m^3$				10			15,000
- Densifier Product	3m x 2m Ø				10			15,000
- Kieve Feed	3m x 2m Ø				10			15,000
- Fresh water head tank	$90m^3$/hr, 10 minute residence time 3m x 2.5m Ø							5,000
12. Hydrosizer	3 spigot, 4.5 TPH solids at 60% solids							35,000
13. Thickener								
- Spillage & Recycles	30ft thickener				5			75,000
14. Spiral Densifier	0.34 TPH solids at 10% solids				10			20,000
15. Flotation Cells	2 x 48 Agitair (i.e. 2 spindles) incl. blower				20			17,000
16. Kieving Installation								
- Vibrators	4 vibrators				2			16,000
- Kieves								2,000
- Hoppers, screen, bag holder	Hopper $0.5m^3$							10,000
- Scales								12,000
- Crane	2 tonne Crane				8			15,000
17. Water Reticulation								
- Tank	$220m^3$/hr with 30 minute capacity - 3m x 7m Ø							10,000
- Pumps	$220m^3$/hr with spare (incl. shed.)							20,000
18. Mill Building	$970m^2$ at $\$1000/m^2$							970,000
	Sub-Total Items 8. - 18.				131			1,708,200

RENISON LIMITED - BLUE TIER

MILL CAPITAL - CONCENTRATOR EQUIPMENT (continued)

ITEM	DETAILS	TOTAL H.P.	BUDGET $	INSTALL $	TOTAL COST $
19. Miscellaneous					
- Distributors	3 x 4 way units, 3 x 3 way units				14,000
- Sampling	Feed and Tail Sampling				20,000
- Weightometer					10,000
- Cranes	Grinding bay 20 tonne unit / Tables 1 tonne unit	40			40,000
	Sub-Total of Items 1. - 19.	821			2,772,200
20. Electrical	14% of plant & building, excl. piping, i.e. 14% of $2,772,200.				388,108
21. Piping	16% of plant only. i.e. 16% of $1,802,200.				288,352
22. Instruments	2% of plant only. i.e. 2% of $1,802,200.				36,044
	Total of Items 1. - 22.				3,484,704

7.11. LEASE BUILDINGS

Standard transportable type units will be used for all offices and standard warehouse structures for larger buildings and workshops. Good salvage value can then be realised.

7.11.1. Administration Office

This will accommodate the Manager, mining, geological, metallurgical and administration staff (12 in total) and also includes toilets and lunch/meeting room.

7.11.2. Stores Office

A large stores holding is not envisaged - particularly of Caterpillar equipment as considerable reliance should be placed on Cat. servicing. Some increase in holdings may occur as underground development gets underway, but this will be reasonably predictable.

The Stores building will house the Stores Clerk and a Storeman. An office will be fitted inside the main building for covered storage. The store yard will be fenced for heavy and bulky items. Inflammables will be stored separately.

Main Building 18 m. x 6 m. with roller door enclosed within an area of 36 m. x 12 m., remainder to be fenced with chain link netting and barbed wire topped including double gate to admit trucks.

7.11.3. Change/First Aid/Crib/Lamp Room

This will serve open pit and underground workers. Fitted with clean and dirty side with four showers and two toilets, urinal in centre. 25 lockers for dirty and clean side. Fitted with heaters, hot water heater 80 gallons. Septic tank treatment. Also contains an area for crib room, lamp room which will not be required until Year 6, and a garage for the ambulance.

7.11.4. Workshop

Maintenance Bay:

To handle mobile diesel equipment. Length 18 m., width 7 m. Roller door at each end. Equipped with travelling electric hoist - 5 tonne lift. Small benches along wall for tools. Standard warehouse building shell.

Washing Bay:

Along-side maintenance bay. 18 m. x 6 m. 20 cm. reinforced concrete with run-off drains. Equipped with steam cleaner, compressed air and water.

Electrical and Mechanical Workshop:

Only basic mechanical and electrical fitting is anticipated. Other jobs will be sent out.

18 m. x 5 m. x 3 m. to eaves. V-roof line. One roller door, plus side door. Concrete floor 10 cm. reinforced. Fitted with benches along walls with vices and tool cupboards.

Equipment to include electric welder, 10 tonne press, power hacksaw, vertical drill, bench grinder, small lathe, bit grinder, small 100 cmf. air compressor. Some could be second hand.

7.11.5. Assay Office

3 m. x 9 m. building of three rooms. One with concrete floor for sample preparation, one for assay, and one for balance and desk top analyser.

7.12. HOUSING

At this stage consideration could be given to the purchase of six houses in St. Helens for key personnel. The investment in housing would probably be sound in its own right.

However, this requires significant capital, and a long term rental would no doubt be cheaper and a charge against working costs with tax advantages.

No consideration should be given to providing any other accommodation or infrastructure as the small labour force can be drawn from the surrounding area. The operation is a small, low grade mine and cannot withstand high infrastructure costs.

7.13. SURFACE PLANT

Allowance has been made for the purchase of a 7 tonne capacity mobile crane which would be available for heavy lifts around the site, (a second hand unit would be suitable).

A stores fork lift would be required and a second hand unit would be suitable. Several surface vehicles with a three year renewal period are also included.

7.14. FIXED PLANT

The following allowances were made:-

Mill Header tank	10,000
Pumps and pipeline ex Dam, 2 x submersible + 250 mm. 800 m. line	10,000
Fuel Storage Tank, 20,000 gals.	25,000
Explosives Magazines	15,000
Ammonium Nitrate Storage	5,000
ANFO Mixing Shed	7,000
	$82,000

8. OPERATING COSTS

Average operation costs over life of mine are detailed as follows:-

Mining	$4.03 per tonne ore
Milling	$4.38
Administration	$0.66
Overheads	$0.97
Salary and Wage Fringes	$1.44
Contingency	$2.30
	$13.83

Mining Costs - details:

Open Pit	$4.21 per tonne ore
Underground	$3.86 per tonne ore
Open Pit	$1.49 per tonne excavated
Open Pit	$1.74 per cu. m. decomposed granite (Contract)
	$0.92 per tonne decomposed granite (Contract)
Open Pit	$1.80 per tonne ore plus solid rock overburden

Overburden ratio is 1:1.83 for the open pit operation

The cost of production is $4840 per tonne of tin metal in concentrates, which is within the bottom 10% of world producers.

Unescalated production costs are detailed on Enclosure XVI.

RENISON LIMITED - BLUE TIER STUDY

ENCLOSURE XVI:

OPERATING COSTS - UNESCALATED - $'000

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	Total
Mine														
Award Wages	75	153	153	153	153	153	153	233	233	233	233	233	13	2171
Salaries	89	89	89	89	89	89	89	89	89	89	89	89	22	1090
O/B Removal - Contract	567	348												915
O/B Removal	25	75	50	50	13									213
Ore Removal	-	-	50	50	50	45	48	191	222	222	222	222	13	1335
Other Mobile Units	26	32	32	32	30	24	28	30	30	30	30	30	3	357
Grade Control and General	30	30	30	30	30	50	50	60	60	60	60	60	5	555
Maintenance														
Award Wages	30	72	84	84	72	96	96	96	96	96	96	96	6	1020
Salaries	35	35	35	35	35	35	35	35	35	35	35	35	9	429
Power	6	6	6	6	6	12	12	12	12	12	12	12	12	126
Mill														
Award Wages	-	-	240	240	240	240	240	240	240	240	240	240	14	2414
Salaries	-	37	42	42	42	42	42	42	42	42	42	42	3	460
General	-	9	15	15	15	18	18	18	15	15	15	15	-	168
Grinding Media	-	12	186	186	186	186	186	186	186	186	186	186	-	1872
Maintenance Consumables	-	-	206	206	206	213	226	266	213	213	213	213	29	2164
Power	-	-	70	70	70	70	70	70	70	70	70	70	-	700
Mill Maintenance														
Award Wages	-	14	14	14	14	14	14	14	14	14	14	14	2	156
Salaries	-	-	78	78	78	91	91	91	91	91	91	91	5	876
Administration - Salaries	88	88	88	88	88	88	88	88	88	88	88	88	23	1079
- Other	20	20	20	20	20	20	20	20	20	20	20	20	5	245
Overheads	150	150	150	150	150	150	150	150	150	150	150	150	150	1950
Salaries and Wages Fringes 30%	95	146	247	247	243	254	254	278	278	278	278	278	29	2905
Contingency 20%	247	263	377	377	369	379	382	429	436	436	436	436	68	4635
TOTAL	1483	1579	2262	2262	2199	2269	2292	2598	2620	2620	2620	2620	411	27835

9. MANNING

The mining and milling activities are technically straight forward. The mining methods are simple with the emphasis on grade control, and the concentrator flow sheet is a simple gravity circuit layout. This gives low manning requirements for direct production work, and a small administration staff.

The small staff does indicate that the section heads should be reasonably well experienced and good salaries and conditions be offered.

The build up of manpower will commence with mining activities for overburden stripping, preparation of building sites, and construction work supervision. Normal manning levels will be reached upon commissioning of the mill. There will then be a small increase over the period of underground development in Years 6 and 7 followed by a fall to the normal level of 67.

The break-up of staff and award manning is detailed in Enclosures XVII and XVIII.

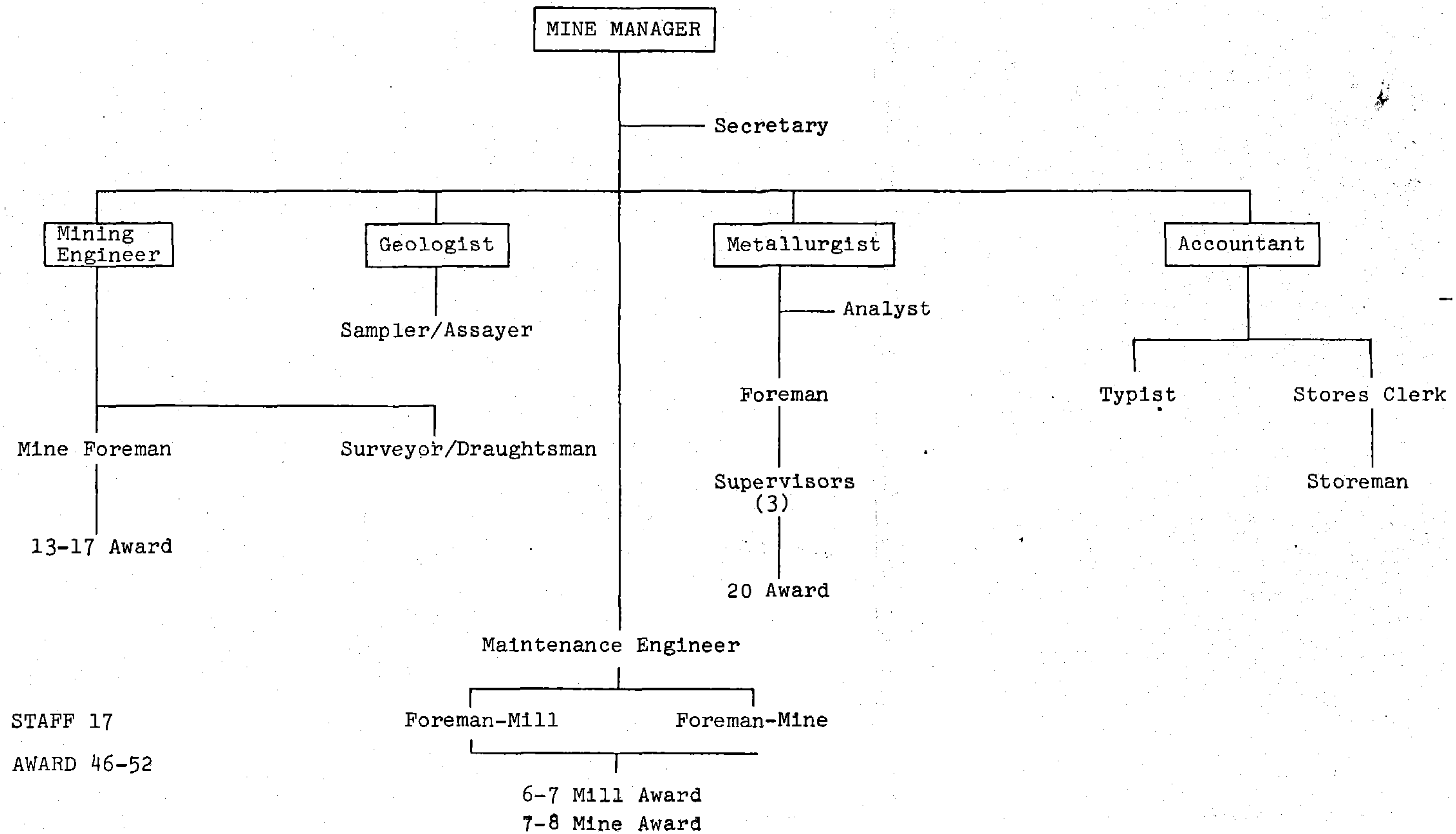
RENISON LIMITED - BLUE TIER STUDY ENCLOSURE XVII:

MANNING

	1	2	3	4	5	6	7	8	9	10	11	12	13
ADMINISTRATION													
Manager	1	1	1	1	1	1	1	1	1	1	1	1	1
Accountant	1	1	1	1	1	1	1	1	1	1	1	1	1
Secretary/Typists	1	2	2	2	2	2	2	2	2	2	2	2	2
Stores Clerk	1	1	1	1	1	1	1	1	1	1	1	1	1
Storeman	1	1	1	1	1	1	1	1	1	1	1	1	1
MINING													
Mining Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Foreman	1	1	1	1	1	1	1	1	1	1	1	1	1
Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1
Surveyor/Draughtsman	1	1	1	1	1	1	1	1	1	1	1	1	1
Sampler/Surv. Asst.	1	1	1	1	1	1	1	1	1	1	1	1	1
Drillers	2	2	2	2	2	2	2	2	2	2	2	2	2
Operators	6	6	6	6	6	6	6	6	6	6	6	6	5
General Hands	5	5	5	5	5	8	9	4	4	4	4	4	2
MAINTENANCE													
Maintenance Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Foreman	1	2	2	2	2	2	2	2	2	2	2	2	2
Tradesmen	5	6	13	13	12	15	15	15	15	15	15	15	10
CONCENTRATOR													
Metallurgist		1	1	1	1	1	1	1	1	1	1	1	1
Foreman		1	1	1	1	1	1	1	1	1	1	1	1
Shift Co-ordinators			3	3	3	3	3	3	3	3	3	3	3
Analyst			1	1	1	1	1	1	1	1	1	1	1
Operators			20	20	20	20	20	20	20	20	20	20	20
TOTAL	30	35	66	66	65	71	72	67	67	67	67	67	59

ENCLOSURE XVIII:

ORGANISATION CHART



10. FINANCIAL EVALUATION

10.1 ASSUMPTIONS

(i) All costs are in June 1979 price terms.

(ii) Escalation of 6 1/2% p.a. is allowed until the first year of revenue earning, i.e. 1982/83. There-after, constant prices are used.

(iii) Smelter Contracts are to be, in essence, similar to those currently in operation.

(iv) Tin price will be the equivalent of $A 12,500 per tonne.

(v) A contingency of 20% is allowed for both capital cost and operating cost estimates.

(vi) Working capital requirements have been assumed at $400,000.

(vii) State Royalties are assumed to be 2 1/2% of Net Sales Value.

(viii) Project commencement date is assumed to be July 1980.

(ix) Taxation and Investment Allowance rates are assumed to be those currently in force.

B) CONCLUSIONS

(i) Total net cash flows over a 14 year period ending June 1994 is $10.319 million.

(ii) Pay-back of Capital Investment takes place during Year 7 (i.e. 1986/7).

(iii) Discounted after-tax rate of return on funds employed is 13% p.a.

(iv) Profit break-even is at 76% of planned revenue levels.

C) SENSITIVITIES

(i) The project is not particularly sensitive to fluctuations in costs and prices. For example, a reduction of $500,000 in the initial capital cost increases the project return by less than 1% p.a.

(ii) Sensitivity to Tin Price is as follows:-

(a) If the price was reduced by $A1,000/tonne (i.e. to $A 11,500), the project would have a D.C.F. return of 10% p.a.

(b) For the project to have a Nil cash flow (i.e. D.C.F. return of Nil), the tin price would have to drop to $A 9,470/tonne.

(iii) However, the project is extremely volume sensitive, in that a 25% increase in annual throughput over the 14 year life (after appropriate adjustment for cost factors) increases the project return to 21% p.a.

RENISON LIMITED - BLUE TIER STUDY

ENCLOSURE XIX:

ANALYSIS OF PRODUCTION, COSTS, REVENUES AND CASH FLOW

YEAR	1	2	3	4	5	6	7	8	9	10	11	12	13	14	TOTAL
TIN METAL IN CONCENTRATES TONNES	-	-	595	595	595	595	595	561	544	544	544	544	32		5,744
TIN PRICE - $A PER TONNE	-	-	12,500	12,500	12,500	12,500	12,500	12,500	12,500	12,500	12,500	12,500	12,500		-
REALISATION ON PRODUCTION - $000	-	-	6,578	6,578	6,578	6,578	6,578	6,202	6,014	6,014	6,014	6,014	354		63,502
OPERATING COSTS	(1,579)	(1,791)	(2,732)	(2,732)	(2,675)	(2,746)	(2,769)	(3,110)	(3,162)	(3,162)	(3,162)	(3,162)	496		(33,278)
OPERATING SURPLUS	(1,579)	(1,791)	3,846	3,846	3,903	3,832	3,809	3,092	2,852	2,852	2,852	2,852	(142)		30,224
CAPITAL EXPENDITURE	(5,664)	(4,818)	(372)	(97)	(310)	(1,244)	(523)	(176)	(70)	(97)	(70)	(43)	645		(12,839)
NET CASH FLOW BEFORE TAX	(7,243)	(6,609)	3,474	3,749	3,593	2,588	3,286	2,916	2,782	2,755	2,782	2,809	503		17,385
DEPRECIATION FOR TAX (5 YEARS)	1,133	2,097	2,171	2,190	2,254	1,366	511	472	465	421	185	92	(589)		12,768
INVESTMENT ALLOWANCE	1,133	964													2,097
TAXABLE INCOME	(3,845)	(4,852)	1,675	1,656	1,649	2,466	3,298	2,620	2,387	2,431	2,667	2,760	447		15,359
TAX PAYMENTS AT 46%	-	-	-	-	-	-	-	(942)	(1,205)	(1,098)	(1,118)	(1,227)	(1,270)	(206)	(7,066)
NET CASH FLOW	(7,243)	(6,609)	3,474	3,749	3,593	2,588	3,286	1,974	1,577	1,657	1,664	1,582	(767)	(206)	10,319
CUMULATIVE NET CASH FLOW	(7,243)	(13,852)	(10,378)	(6,629)	(3,036)	(448)	2,838	4,812	6,389	8,046	9,710	11,292	10,525	10,319	-
CUM. NPV - CASH FLOW DISC. 15%	(7,243)	(12,990)	(10,363)	(7,898)	(5,844)	(4,557)	(3,136)	(2,394)	(1,878)	(1,407)	(996)	(656)	(799)	(832)	-
DCF RATE OF RETURN = 13%															

ENCLOSURE XX:

RENISON - BLUE TIER STUDY

REVENUE CALCULATIONS/CAPITAL COSTS/OPERATING COSTS WORKSHEET

YEAR	1	2	3	4	5	6	7	8	9	10	11	12	13	14	TOTAL
TONNES TIN METAL SOLD	-	-	595	595	595	595	595	561	544	544	544	544	32	-	5,744
TIN PRICE $A/TONNE			12,500	12,500	12,500	12,500	12,500	12,500	12,500	12,500	12,500	12,500	12,500		
SMELTER CHARGES			(800)	(800)	(800)	(800)	(800)	(800)	(800)	(800)	(800)	(800)	(800)		
SELLING EXPENSES*			(362)	(362)	(362)	(362)	(362)	(362)	(362)	(362)	(362)	(362)	(362)		
N.R. VALUE			11,338	11,338	11,338	11,338	11,338	11,338	11,338	11,338	11,338	11,338	11,338		
LESS: ROYALTY @ 2½%			(283)	(283)	(283)	(283)	(283)	(283)	(283)	(283)	(283)	(283)	(283)		
NET SALES - PER TONNE			11,055	11,055	11,055	11,055	11,055	11,055	11,055	11,055	11,055	11,055	11,055		
$000			6,578	6,578	6,578	6,578	6,578	6,202	6,014	6,014	6,014	6,014	354		63,502
CAPITAL COSTS - PER M.B. ($000)	4,390	3,457	48	67	214	858	361	122	48	67	48	30	(668)		9,042
ADD CONTINGENCY 20%	878	691	10	13	43	172	72	24	10	13	10	6	134		2,076
WORKING CAPITAL	50	100	250												400
TOTAL UNESCALATED	5,318	4,248	308	80	257	1,030	433	146	58	80	58	36	(534)		11,518
TOTAL ESCALATED	5,664	4,818	372	97	310	1,244	523	176	70	97	70	43	(645)		12,839
OPERATING COSTS - PER M.B. ($000)	1,483	1,579	2,262	2,262	2,215	2,273	2,292	2,575	2,618	2,618	2,618	2,618	411		27,824
- ESCALATED	1,579	1,791	2,732	2,732	2,675	2,746	2,769	3,110	3,162	3,162	3,162	3,162	496		33,278
* $300/tonne as at June 1979															
ESCALATED AT 6.5% PER ANNUM															
ESC. FACTORS @ 6.5%	1.0650	1.1342	1.2079	1.2079	1.2079	1.2079	1.2079	1.2079	1.2079	1.2079	1.2079	1.2079	1.2079		

RENISON LIMITED - BLUE TIER STUDY

ENCLOSURE XXI:

REVISION I
(Increased Tonnage) to
250,000 T.P.A.

CASH FLOWS

YEAR	1	2	3	4	5	6	7	8	9	10	11	12	13	14	Total
TIN METAL IN CONCENTRATE - TONNES	-	-	744	744	744	744	744	744	744	744	744	744	32	-	7,472
TIN PRICE $A PER TONNE	-	-	12,500	12,500	12,500	12,500	12,500	12,500	12,500	12,500	12,500	12,500	12,500	-	-
REALISATION ON PRODUCTION - $000	-	-	8,225	8,225	8,225	8,225	8,225	8,225	8,225	8,225	8,225	8,225	354	-	82,604
OPERATING COSTS	(1,579)	(1,791)	(3,168)	(3,168)	(3,109)	(3,125)	(3,154)	(3,538)	(3,579)	(3,579)	(3,579)	(3,579)	(558)	-	(37,506)
OPERATING SURPLUS	(1,579)	(1,791)	5,057	5,057	5,116	5,100	5,071	4,687	4,646	4,646	4,646	4,646	(204)	-	45,098
CAPITAL EXPENDITURE	(5,664)	(4,818)	(372)	(97)	(310)	(1,244)	(523)	(176)	(70)	(97)	(70)	(43)	645	-	(12,839)
NET CASH FLOW BEFORE TAX	(7,243)	(6,609)	4,685	4,960	4,806	3,856	4,548	4,511	4,576	4,549	4,576	4,603	441	-	32,259
DEPN. FOR TAX (5 YEARS)	1,133	2,097	2,171	2,190	2,254	1,366	511	472	465	421	185	92	(589)	-	12,768
INVESTMENT ALLOWANCE	1,133	964	-	-	-	-	-	-	-	-	-	-	-	-	2,097
TAXABLE INCOME	(3,845)	(4,852)	2,886	2,867	2,862	3,734	4,560	4,215	4,181	4,225	4,461	4,554	385	-	30,233
TAX PAYMENTS AT 46%	-	-	-	-	-	-	(1,680)	(2,098)	(1,939)	(1,923)	(1,943)	(2,052)	(2,095)	(177)	(13,907)
NET CASH FLOW	(7,243)	(6,609)	4,685	4,960	4,806	3,856	2,868	2,413	2,637	2,626	2,633	2,551	(1,654)	(177)	18,352
CUMULATIVE NET CASH FLOW	(7,243)	(13,852)	(9,167)	(4,207)	599	4,455	7,323	9,736	12,373	14,999	17,632	20,183	18,529	18,352	-
CUM. NPV - CASH FLOW DISC. @ 15%	(7,243)	(12,990)	(9,447)	(6,185)	(3,437)	(1,520)	(280)	627	1,489	2,236	2,887	3,435	3,126	3,097	-
DCF RATE OF RETURN = 21%															

RENISON LIMITED - BLUE TIER STUDY

ENCLOSURE XXII:

REVENUE AND COST WORKSHEETS (ALL COSTS IN JUNE 1979 PRICE TERMS) REVISION I: INCREASED TONNAGE TO 250,000 T.P.A.

	1	2	3	4	5	6	7	8	9	10	11	12	13	Total
Tonnes Tin Metal Sold	-	-	744	744	744	744	744	744	744	744	744	744	32	7472
Tin Price - $A/tonne			12500	→									12500	
- Smelter Charges			(800)										(800)	
- Selling Charges*			(362)										(362)	
- N.R. Value			11338										11338	
- Loss & Royalty @ 2½%			(283)										(283)	
- Net Sales - per tonne			11055										11055	
$'000			8225	8225	8225	8225	8225	8225	8225	8225	8225	8225	354	82604
Capital Cost - per M.B. ($'000)	As per Original Study													
Add: Contingency (20%)														
Working Capital														
Total - Unescalated														
- Escalated	5664	4818	372	97	310	1244	523	176	70	97	70	43	(645)	12839
Operating Costs - Mine & Rehab.	991	948	771	771	734	711	718	972	1003	1003	1003	1003	151	10779
- Mill		72	1018	1018	1018	1041	1054	1041	1038	1038	1038	1038	55	10469
- Labour Fringe 30%	95	146	247	247	243	254	254	278	278	278	278	278	29	2905
- Overheads	150	150	150	150	150	150	150	150	150	150	150	150	150	1950
- Contingency 20%	247	263	437	437	429	431	435	488	494	494	494	494	77	5220
Total - Unescalated	1483	1579	2623	2623	2574	2587	2611	2929	2963	2963	2963	2963	462	31323
- Escalated	1579	1791	3168	3168	3109	3125	3154	3538	3579	3579	3579	3579	558	37506
*$300/tonne as at June '79 escalated at 6.5% p.a.														
Escalation Factors (@ 6.5% p.a.)	1.0650	1.1342	1.2079	→										