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ZEEHAN PROJECT

PRE FEASIBILITY STUDY REPORT

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SUMMARY

- It is probable that the Zeehan tin project can be turned into a profitable underground mine, producing between 1,480 and 1,900t of Sn per year at an average cost, including depreciationof A$7,900/t. To reduce the risk to acceptable levels a further A$1.5 million needs to be spent on diamond drilling to establish continuity of the higher grade zones, and on further metallurgical work. It is recommended that this work be started immediately. If the results come up to expectation, the property would be ready for production in 4 years.

Key Assumptions

Ore reserves - The global resource estimate is 7.3M tonnes @ 0.69% Sn, in which there may be up to 3.5M tonnes @ 1.2% Sn in continuous higher grade zones.

Production rate - 150,000 tpy @ 1.37% Sn.

Metallurgical recoveries - average 80% into concentrates containing greater than 50% Sn.

Metal price - A$12,430/t (85% of 1983 price). Realisable price 90.0% for fume concentrate, 91.8% for gravity concentrate. Royalty $2.90/t.

Design

Mining - Modern trackless mining methods, accessible by a single decline; stopes average 8m width, varying from 3m to 30m; mining methods include longhole open stoping and mechanised cut and fill; selective mining.

Mill - The concentrator includes gravity, flotation and matte fuming. Flotation is a primary concentrating tool rather than a scavenger.

Costs (1983A$)

Capital - $36M initial, $49M project life.

Operating - $89/t including depreciation, $7,900/t Sn.

1 INTRODUCTION

Aberfoyle Exploration conducts the Zeehan Project in accordance with the Aberfoyle/Gippsland Oil and Minerals joint venture agreement. Aberfoyle may earn up to 70% equity in the project by completing an acceptable feasibility study. A written commitment by an acceptable institution, to lend not less than one half of all costs to be met by Gippsland for the said development is essential to acceptance. Costs during development will be born according to equity. On commencement of production, 75% of Aberfoyle expenditure on behalf of Gippsland after project expenditure of $153,000 and up to completion of the feasibility study, will be reimbursed from proceeds of the mining operation, provided that Gippsland borrowings are repaid first. Simple interest on the above expenditure will also be credited to Aberfoyle.

In mid 1982 a drilling campaign was completed at Zeehan and it was recognised that a substantial tin resource could be inferred. Three separate deposits were confirmed, each with different configuration, mineralogy and metallurgy. The largest deposit, Severn, was interpreted to be open, but opportunities for extension are at depths from the surface of greater than 400 metres. Tin was known to occur primarily as cassiterite. Preliminary evaluation of metallurgical characteristics of the various styles of mineralisation indicated that acceptable recoveries in conventional concentrators would be at least difficult. Feasibility of gravity recovery was in doubt.

The overall dimensions and tenor of the resource were throught to be adequate for an Ardlethan style mining operation (large tonneage - low grade). Higher grade sections were interpreted indicating possibilities for selective mining. Near surface high grade mineralisation was known in Queen Hill Lens. Opportunities for low capital cost mining and milling at Cleveland, required evaluation.

It was decided to initiate a study with immediate objectives of:

- providing advice to exploration on the best use of funds
- determining minimum acceptable tonneage and grade targets for a profitable operation

The overall objective was to define the scope of a profitable mine at Zeehan. Order of magnitude prefeasibility studies were initiated incorporating:

- a review of all the exploration results, compiling them into suitable formats, interpreting the geology and estimating the resource dimensions with suitable qualification
- a review of the resource distribution and by conceptual mine planning and economic analysis, determining if and how the deposits may be economically exploited

- using the most likely mining and mineral processing scenario, analysing the assumptions to determine which on refinement had the best chance of benefiting the project viability

- on definition of the scope of the likely production scenario, deciding how and with what expenditure it could be best achieved and recommending a program for development.

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2 GEOLOGY & RESOURCES

- At Queen Hill significant tin is reported in volcanics, clastic sediments and dolomite. Severn mineralisation is tabular but is located close to or on the apparent angular unconformity between the Oonah beds and the Crimson Creek sequence.

Montana lens is confined to a particular dolomite sequence. It is essentially a massive sulphide lens with little tin occurring outside the sulphide zone.

The total mineralised envelope in all three lenses was best described by using an assay cut off of 0.1% Sn. While this is very low in economic terms it serves to limit the area of intersest. In Queen Hill lens it seems to be coincident with the observed limit of sulphide mineralisation. Using the assays it is easy to pick, with few occasions where veining or erratic tin concentrations occur outside the chosen cut off. In Severn and Montana lenses the assay cut off is readily identified although the low level of veining associated with low tenor tin mineralisation makes visual identification difficult.

Higher grade zones are present in the three lenses. In Queen Hill lens high assays are definitely clustered. From hole to hole there seems to be a relationship between high grade sections, but lack of lithological control makes correlation difficult and leaves the interpretation in doubt. Severn lens is characterised by a centrally located high sulphide "lode" zone with sulphides decreasing in concentration away from the lode. Tin is distributed in a similar way with the highest tin assays occurring within the high sulphide zone. The high sulphide lode zone is identified visually and normally by the 0.5% Sn assay cut off. The high sulphide zone in Montana lens is normally identified by a 0.2% Sn cut off.

The resources were quantified by Kriging. The estimation error for Queen Hill lens qualifies the resource as being in the indicated category. Both Severn and Montana lens resources are qualified as inferred.

The resource details are summarised in the table below drawn from a detailed report (Palmer) prepared as part of the study.

Total Mineralised Envelope (0.1% Sn cut off)

LENS	CATEGORY	TONNES x 10^6	%Sn	%Cu	%Zn	%Pb	g/t Ag
Queen Hill	Category	1.8	0.82	0.08	0.45	0.77	33
Severn	Inferred	5.1	0.60	-	-	-	-
Montana	Inferred	0.4	1.22	0.02	2.00	1.41	51
Total		7.3	0.69				10.9

Within the total mineralised envelope higher grade zones are identified. These resources are:

Queen Hill	Indicated	0.93	1.39	0.10	0.47	0.55	28.9
Severn	Inferred	2.37	1.11	-	-	-	-
Montana	Inferred	0.31	1.45	0.02	2.61	1.59	58
Total		3.61	1.21				

3 CONCEPTUAL MINE PLANNING

3.1 MINE DESIGN

On completion of the resource assessment sections and plans of both geology and geological resources were made available for mining study. Appraisal of the resource distribution lead to the following observations:

(i) Queen Hill Lens

Queen Hill lens can be divided into two parts. The upper section, above RL1110, is essentially massive sulphide, relatively narrow (3 to 8 metres) but high grade and dips at 50 to 80 degrees. The hanging wall is adjacent to a fault zone coincident with Clarkes lode, and is likely to need substantial ground support in places.

The lower section of Queen Hill (RL1110 to RL1010) is a wide zone of mineralisation with relatively narrow high grade zones within the envelope. The southern end is sufficiently wide that long hole stoping may be possible whether the bulk or the high grade is to be mined. It is probable that all the high grade in the southern two thirds of the lens can be mined by bulking the high grade sections and the low grade in between.

This bulk mining option is considered the more attractive because metal recovery is good and techniques (eg. Ardwest Deeps) can be applied which allow final delineation of stope boundaries with blast holes. North of 3100N the bulked grade deteriorates and there appears to be greater advantage in attempting selective mining. Widths of selectively mined high grade would be in the range 3 to 8 metres. The overall shape and attitude is such that a "hanging wall" does not exist. The east side of the mineralisation is adjacent to the contact with the QS sediment. The contact zone is almost always sheared or faulted and ground support will need to be considered when mining approaches it.

It is unlikely that ground conditions in the stopes will be very good and stoping methods, dilution and mining recovery estimates will need to take this factor into account. The bottom of the lower section of Queen Hill is not well defined. Small cut and fill stopes may be necessary to optimise extraction at the north and south ends below RL1000 where some continuation of mineralisation is indicated by drilling.

(ii) Severn Lens

At 0.5% cut off, the upper part of Severn lens is narrow and has a short strike length, but is high grade. Both thickness and strike length increase with depth and while grades appear to decrease the yield of tin per vertical metre increases. At RL1000 the yield is about 2600 MTU's per vertical metre and at RL800 about 6700 MTU's.

At 0.3% Sn cut off the grade of the material added to the resource is less than 0.5% and unlikely to pay its own way. (Cleveland mine cut off is about 0.5%.) The overall dimensions and shape of the 0.3% cut off resource are compatible with bulk mining but not big enough to achieve economies of scale.

It is significant that the incidence of faulting observed in drill core is significantly less in the region of Severn mineralisation than it is in and around Queen Hill lens. The 0.5% cut off zone of mineralisation is surrounded by less mineralised rock which is netted with stringer sulphide mineralisation. It is likely to stand quite well in stoping.

(iii) Montana Lens

Montana Lens is narrow (2.5 to 5 metres) and short in strike length. It will need to be accessed by long crosscuts from the Queen Hill decline and will be of marginal economic benefit to the project for this reason. The old mine workings in the upper levels will be full of water posing an additional problem. In contrast, the mineralisation is high grade and massive sulphide, lending itself to visually controlled selective mining.

The most significant factor evident in all lenses and relating to mine production is the concentration of tin in zones at high cut off. In particular the benefits of mining selectively the mineralisation defined by the 0.5% cut off outweigh any advantages gained by reducing the cut off and bulk mining. The relatively narrow widths of mineralisation and the fact that in only one location is there a suggestion of parallel nearby mineralistion (Upper Severn), dictate that production rates will be modest. For a production of 1500 tonnes of tin recovered, approximately 40 vertical metres per year will need to be mined. To commence at that production level development will need to be completed to at least RL1000 during the construction period. It will be difficult to increase mine production beyond 150,000 tonnes of high grade ore per annum. To do so would require more capital and little reduction in operating cost as more development, working places and equipment would be required. Increase in volume from the same resource is almost certainly accompanied by reduction in grade or reduction in mine life. Most of the mining widths and attitudes are suited to trackless mining. The major portion of production will come from ore where assay grade control, (e.g. down hole probes) will be essential and where some ground support is needed. Since access will amost certainly be by decline, production is unlikely to commence from the bottom.

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It was decided after reviewing costs, timing and other benefits of various access alternatives that with Queen Hill and Severn as the major production sources and with a production level of 100,000 to 200,000 tonnes per year that mine access would be by decline from the western side of Queen Hill and in the footwall basic volcanics to Queen Hill lens. The volcanics appear to be the most competent rocks and the environmental problems will be minimised. The road to Trial Harbour will probably need to be relocated. The decline will remain in the footwall volcanics to about 1000 RL where it is proposed that it will cut across strike to Severn lens and continue downwards in quartz rich sediments in the footwall of Severn. The decline is nominally 4.5 metres by 3.5 metres and has slope of 1 in 7. It is considered unlikely that a haulage shaft will be justified for the production levels considered.

Queen Hill lens will be mined by cut and fill and by long hole stoping. It is likely that two rib pillars will be required for the lower part of Queen Hill and that above a sill pillar at RL1110, mining will be by cut and fill, but later in the life of the mine.

Severn lens will be mined by cut and fill in three major lifts with pillars separating each main stope. Montana will contribute a low annual production, probably by shrink stoping. Its location in relocation to the other deposits indicates high access capital and consequently less favourable economics.

Fill, in all cases will be uncemented development waste. In the long hole stoping in Queen Hill lens, fill is not essential but will depend on the need to support and the need to avoid waste haulage to the surface.

3.2 DILUTION AND MINING RECOVERY

Dilution, extraction loss and mining recovery have been estimated for individual stopes:

Ore Source	Dilution % of G.O.R.	Dilution Grade % Sn	Extraction Loss % of Diluted GOR	Mining Loss % of GOR
Q.H. Cut and Fill above RL1150	10%	0.15%	5%	
Q.H. Sill Pillar 1140-1150	-	-	-	100%
Q.H. Cut and Fill 1110-1140	30%	0.15%	10%	
Q.H. Sill Pillar 1100-1110	-	-	-	100%

Ore Source	Dilution % of G.O.R.	Dilution Grade % Sn	Extraction Loss % of Diluted GOR	Mining Loss % of GOR
Q.H. Long Hole Stope 1040-1100 RL, 3010 to 3040 N	15%	0.20%	10%	-
Q.H. Pillar 3040 to 3060 N	30%	0.20%	10%	50%
Q.H. Long Hole Stope 1020-1100 RL, 3060-3090 N	15%	0.20%	10%	
Q.H. Rib Pillar 3090-3100 N				100%
Q.H. Cut and Fill below 1100	15%	0.35%	5%	
Severn Cut and Fill 910 to 1040	15%	0.35%	5%	
Severn Sill Pillar 900 to 910	-	-	-	100%
Severn Cut and Fill 800 to 900	15%	0.35%	5%	-
Severn Sill Pillar 790 to 800	-	-	-	100%
Severn Cut and Fill 740 to 790	15%	0.35%	5%	
Montana Shrink Stope 980-1100	10%	0.05%	5%	
Montana Sill Pillar 970-980	-	-	-	100%
Montana Shrink Stope 880-970	10%	0.05%	5%	

In the mining recovery and extraction loss assessments, only those sections of the mineralisation planned to be mined are allowed for. Any mineralisation located such that anything other than simple stopes can be considered, is for the purposes of this assessment considered to be deferred production. In several sections 'ore' below a "desirable" head grade is arbitrarily excluded from production plans and judged to be sub ore. No allowance is made for additions to reserves outside the arbitrary ore boundaries.

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3.3 PRODUCTION SCHEDULE

Production planning took into account metallurgical differences, relative disposition of the ore bodies and the need to maintain a high head grade to keep the proposed operation competitive.

Queen Hill lens outcrops but achieves its greatest dimensions 100 metres or more beneath the surface. The near surface material is very high grade but the most metallurgically difficult of all the known ore types. Production from this ore source is left until late in the schedule and a high cut off grade is maintained except for one development drive which is planned for training and metallurgical orientation.

The lower section of Queen Hill is divided into two long hole stopes (each of about 100,000 tonnes), two rib pillars and a section where cut and fill mining is planned to selectively mine the narrower high grade mineralisation. The long hole stopes provide a source of ore which can be extracted quickly and stockpiled providing a buffer for the more erratic production from cut and fill mining. Ability to produce at design rates in the early years is dependent on success in these stopes and on getting the decline to the bottom levels for mucking.

Severn lens because of its size and apparently favourable metallurgy represents the foundation of the long term production schedule.

Montana lens is high grade but the small yield per vertical metre determines that production will be at a low rate and probably erratic. This together with high development costs makes Montana less attractive than Queen Hill or Severn lenses.

The proposed production schedule is detailed in Appendix 1.

An alternative scheme was reviewed (Coombe TK) where Severn was treated as the core of the mine. Low cost mining with no pillar recovery and using a cut off grade of 0.3% was studied. Shaft access direct from the surface was planned. After conservative allowances for dilution (10%) a "mineable ore reserve" was estimated to be 1.85 million tonnes at 0.99% Sn. No pillar recovery was assumed. The "ore reserve" is inadequate for either 200,000 or 250,000 T.P.A. production and would need to find at least 150,000 tonnes from outside Severn at the lower production level. The study showed that a reduction in grade (from the preferred scheme) of about 30% was likely for an increase in tonnes of 30% or more. The lower production of 200,000 T.P.A. would produce less tin at a higher cost. Rather than confirming that production rates greater than 150,000 T.P.A. are not possible, the study shows that it will be difficult and probably will require expansion of the ore reserves. It does show that reducing the cut off grade is unlikely to be attractive.

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4 METALLURGY

Since discovery, metallurgical testwork incorporated gravity, flotation, pressure leaching and fuming. Before 1980 all the testwork concentrated on samples from near surface in Queen Hill lens. The objective in all these tests was to maximise recovery into a saleable concentrate. Modest recoveries were achieved with cassiterite flotation to a concentrate containing 20% to 30% Sn. Fuming indicated that high recoveries could be achieved to concentrate grade dependant on Pb and Zn content of the feed.

In 1980 with recognition of the significance of Severn lens, ore characterisation was commenced on exploration drill core. Initially drill intersections in Severn were tested but later mineralisation in holes from both Queen Hill and Montana was characterised. The material tested varied as the treatment concept varied. Initially the whole intersection (at about 0.1% cut off) was processed when bulk mining and low grade concentrate production was envisaged. Later it was realised that selective mining of the high grade zone may be more attractive. Accordingly emphasis changed from a scheme where heavy media separation was essential to one where it was not necessary. On recognition of apparent differences in ore characteristics from previous Queen Hill results, it was decided to reinvestigate near surface Queen Hill mineralisation.

Significant differences in metallurgical characteristics were confirmed. These differences revolve around two major factors, grain size and sulphide concentration.

ORE SOURCE	D50µm	SULPHIDE CONCENTRATION
Queen Hill Upper	14	50% (80% Syngenetic)
Queen Hill Lower	21	30% (50% Zero Syngenetic)
Montana	31	50% (Zero Syngenetic)
Severn	65	20-50% (Zero Syngenetic)
Cleveland (Halls)	74	5-30% (Zero Syngenetic)

D50µm is the size that 50% of the grains are less than. Syngenetic sulphide is mostly pyrite and is barren of Sn.

The results of metallurgical testwork are reported in detail by S S Meik.

In general the cassiterite grains are very fine and they are commonly intergrown with iron sulphides. Separation of the sulphide is essential to attaining concentrate grades acceptable to any of the proposed processes. Fine grinding is necessary to liberate the cassiterite and significant losses are possible in slimes. Gravity concentration is possible but only 10-40% of the tin can be recovered to good grade concentrates. Good recoveries are possible in cassiterite flotation but only to low or moderate concentrate grades. As the concentrate grade is increased, significant losses occur. No work has yet been done to follow the path of lead, zinc, silver, arsenic or stannite in the ore characterisation.

It is proposed to use matte fuming to achieve saleable concentrate from the cassiterite flotation product. This will necessitate removing the lead and zinc from the fume feed.

Three process options were considered:

4.1 QUEEN HILL UPPER - CLEVELAND CONCENTRATOR

The near surface Queen Hill Upper mineralisdation presents an option for early low capital production. If the high grade ore could be sent direct to Cleveland concentrator, significant economic advantages were likely as long as good recoveries and good concentrate grades could be achieved. After simulation in the laboratory of Cleveland Mill performance on Queen Hill Upper mineralisation it was concluded that:

i) Concentrate produced would be 5-7% Sn and recovery about 25%.

ii) The ore is not suitable for feed to the Cleveland mill without considerable circuit rearrangement.

iii) Transport costs to Cleveland would be $12/tonne and milling charges of the order of $25/tonne.

iv) Low recoveries and concentrate grades, additional transport costs and probable additional capital costs combined to make this alternative unattractive even if capacity was available.

4.2 ZEEHAN CONCENTRATOR

Since a maximum of 40% recovery can be achieved by gravity concentration to a good grade (>50% Sn) product, at least flotation is necessary to attain the levels of recovery necessary for a satisfactory economic outcome. Adequate recovery by flotation has been demonstrated but it is expected that concentrate grades will be 6-15% Sn and the product unsaleable. Matte fuming is essential to upgrade this product. Two alternatives are presented for concentration. One process incorporates gravity, flotation and fuming, the other has no gravity and relies on fuming to upgrade the whole preconcentrate. From ore characterisation it is evident that overall recoveries are similar in each process. Whether or not to include gravity is a function of cost of recovery of each unit of tin.

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Preliminary assessment of costs indicates that gravity recovery is at least $5 per recovered MTU lower cost than the alternative process.

It is concluded that recovering a proportion of the tin by gravity is beneficial.

The selected process is:

i) Crush and grind
ii) Sulphide flotation
iii) Gravity concentration
iv) Cassiterite flotation
v) Fuming

Saleable high grade concentrates will be produced from the gravity and fuming plants.

Laboratory tests show that using this processing route overall recoveries of greater than 75% are probable.

Assessment of the impact of finer grinding on sulphide and cassiterite liberation, sliming of cassiterite, and recovery is inadequate. Investigation of the distribution of cassiterite on tables and of separation of base metal sulphide from fume feed is yet to be completed.

4.3 LOCATION OF MATTE FUMER

It is established that Zeehan processing must include fuming to get both good grade concentrate and good recoveries. The fumer may be either integrated into the concentrator or offsite. In the latter case various alternative locations are possible. Location is dependent on several factors.

Transport

Road transport of concentrates is equivalent to an extra $1.60 to $2.67 per tonne of ore mined, or $12 to $20 per tonne of concentrate fumed.

Capital

An off site fumer would need independent assay laboratories, administration and maintenance facilities, security, personnel and safety facilities, and ore handling facilities, for bought or toll treated concentrates. Compared with an integrated fumer (and particularly a small one) much of these facilities translate into significantly increased capital cost. The integrated fumer would derive most of these services by sharing similar facilities installed for other parts of the operation. Some capital disadvantage may occur at Zeehan since housing may be an additional consequence.

Operating Cost

- The operating cost of a fumer is separated into those costs directly related to fuming such as energy and materials, and labour and associated costs. It is unlikely that the former will vary with location, however labour and associated costs may be reduced if the integrated on site option is chosen. The labour costs in the area of management and supervision are likely to be reduced.

These factors combine to make an integrated on site fumer the most likely option if exhausting direct to the atmosphere is acceptable.

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5 INFRASTRUCTURE

- Queen Hill (topography) divides the resources into two groups; Queen Hill lens is west of Queen Hill and the surface expression is out of site of Zeehan and in unoccupied land. It is very close to the Zeehan - Trial Harbour road and within 1 km of the (old) Zeehan cemetry. *Severn Lens and Montana lens are on the* Zeehan side of Queen Hill and underly sparsely occupied land. Major construction in the vicinity is probably unacceptable. In contrast construction on the western side of Queen Hill, away from Zeehan is likely to be environmentally acceptable. The logical place to commence declining is on the western flanks of Queen Hill. At some stage ventilation rises will be necessary on the Zeehan side of the hill but it is unlikely that they will pose any problem. Water will be collected from nearby creeks and potable water will be obtained from town supply. The prevailing wind is from the west and this will need to be considered in locating any facility exhausting undesirable fumes to the atmosphere. Excess housing is not available (for mine staff) in Zeehan, but Queenstown may have some already housed unemployed. Tailings and mine waste will need to be pumped a distance of one or more kilometres from the concentrator since drainage in the immediate mine area runs to the east through Zeehan.

  Electricity may be obtained from the state grid, but planning will be necessary to ensure adequate capacity.

  Security will be a problem because of the vicinity of the township of Zeehan and accessibility from all directions.

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6 CAPITAL COST ESTIMATES

6.1 MINE

All waste development is capitalised including declining, crosscuts to ore horizons and rising for alternative egress and ventilation. Unit rates assume contractor does all development and are derived from order of magnitude estimates for equivalent works at Ardlethan and Que River.

		Metres	$/M	$ x 10^6
1)	Decline RL1200 to RL740	3,220	2,200	7.08
2)	Crosscuts 35% of 1	1,127	2,200	2.48
3)	Contingencies 20% of 1	644	2,200	1.42
4)	Major Rises	440	1,500	0.66
5)	Contingencies 20% of 6	88	1,500	0.13
	TOTALS	5,519		11.77

Mine equipment, (relating only to ore production) includes allowances for two jumbos, three scooptrams and three haulage trucks.

	$/unit	$1 x 106
2 Jumbos hydraulic	250,000	0.5
3 Scooptrams	180,000	0.54
3 Haulage trucks	200,000	0.60
Miscellaneous Equipment	850,000	0.85
TOTAL		2.98

6.2 CONCENTRATOR

The concentrator capital costs are based on the assumptions that:

- Crush and Grind 150,000 TPA to -200µm, some regrind to -105µm
- 150,000 TPA sulphide flotation circuit
- Single pass gravity plant
- 100,000 TPA cassiterite flotation plant

The following estimates are for installed costs and include allowances for omissions and unknowns but do not include construction overheads.

Crusher	950,000
Primary Grind	925,000
Thickeners	400,000
Sulphidiser Flotation	1,000,000
Classification	350,000
Tables	480,000
Regrind	925,000
Cassiterite Flotation	1,500,000
Filters	450,000

A matte fumer is incorporated in the concentrator. The cost is based on updated detailed design for a 4 tonne per hour fumer installed at Cleveland mine. Exhaust gases will be passed into the atmosphere through a stack.

6.3 OFFICES, SITE WORKS AND BUILDINGS

Construction of offices, maintenance facilities, security and safety and site works including provision of power and water is estimated to cost $2.3 million (including a contingency of $0.55 million).

6.4 HOUSING

It was assumed that 80% of employees would require project owned houses in Zeehan. The number of houses and total cost was calculated by assessing that the workforce would be 145 -

Workforce 145
80% require individual houses
No. of houses is 116
One house in 4 is 2 bedroom triplex
21 Triplex units at $30,000
87 Three and four bedroom houses $70,000
8 Duplex Units $40,000
Total Cost $7.04 million

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7 OPERATING COSTS

- All operating costs are developed from cost per tonne estimates. They are derived by comparisons with similar operations.

Mining

All mining costs relate to ore production. Five different types of mining are individually estimated:

. Queen Hill Cut and Fill fill with development waste	$15/tonne
. Severn Cut and Fill fill with development waste, increased cost reflects greater depth	$17-20/tonne
. Queen Hill Open Stope Large hole mining, fill optional but not planned	$10/tonne
. Jumbo development in Ore	$17/tonne

Depreciation is accounted for elsewhere

Milling

A cost of $12/tonne of ore milled is assumed for crushing, grinding, gravity and cassiterite flotation concentration. This is an average approximately in line with that of other operations. It is expected that the savings of a simplified circuit will be balanced by increases due to smaller scale.

Fuming

A cost of $125/tonne of fume feed is assumed. It is based on calculations derived by the matte fuming task force and assumes the fumer is integrated with a concentrator.

Overheads and Administration (Other)

An arbitrary $12/tonne of ore milled was assumed. This includes items -

General Services	$ 3.00/tonne
Mine Overheads	4.80/tonne
Administration	2.60/tonne
Technical Services	0.60/tonne
Exploration	1.00/tonne
	$12.00/tonne

Metal Price

In all revenue calculations the realisable price is calculated by:

Metal Price = A$12,430 per tonne (1983$, 85% of 1983 price

Realisable price = 90% for fume concentrate
91.8% for gravity concentrate

Royalty = $2.90/tonne

8 CASH FLOW ANALYSIS

- Table 1
  Appendix 1

TABLE 1 ZEEHAN PROJECT CASH FLOW ANALYSIS

YEAR	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Tonnes Milled 000's				125	150	150	150	150	150	150	150	150	150	150	
Head Grade % Sn				1.52	1.55	1.55	1.50	1.37	1.37	1.31	1.19	1.21	1.21	1.33	
Tonnes Tin in Concentrates				1403.9	1881.6	1900.2	1859.0	1671.3	1673.0	1613.8	1482.4	1507.3	1507.3	1393.7	
Gross Rev $Mill				15.78	21.20	21.43	20.99	18.84	18.86	18.21	16.74	17.03	17.03	15.68	
Capital Exp $Mill	2.42	14.14	19.37	3.84	2.21	2.21	0.50	0.50	0.50	0.75	0.75	0.75	0.75	-	
Operating Costs $Mill	0	0	1.06	6.30	8.64	8.77	8.69	8.44	8.24	8.32	8.45	8.48	8.48	8.46	
Deductions For Tax $Mill				5.83	3.86	4.11	4.17	4.24	4.33	4.51	4.69	4.94	5.32	5.23	
Tax Payable					1.06	3.80	3.74	3.55	2.69	2.75	2.35	1.57	1.57	1.41	0.87
Net Cash Flow $Mill	-2.42	-14.14	-20.43	5.36	8.86	6.22	7.65	6.04	7.12	6.12	5.01	6.04	6.06	5.71	-0.87

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9 ANALYSIS OF OUTCOME AND OPPORTUNITIES

- The economic analysis presented in section 7 shows a payback period (time taken to repay startup capital) of 4.5 years and at a 10% discount rate a Net Present Value of $2.29 million dollars after tax. The analysis is made using 1983 dollars and is independent of inflation. It compares with a risk free after tax return of 2-5%. The project as presented is attractive. It is recognised though, that this is an economic analysis of a prefeasibility study. No detailed specifications were completed in the study and the assumptions should be evaluated to determine how robust they are and if there is opportunity to enhance the outcome and consequently the investment potential. The project investment rating is also dependent on the cost of production of tin - its competitiveness.

9.1 REVENUE ASSUMPTIONS

Revenue is based on the assumptions of metal price, metallurgical recovery production rates and ore grades.

The metal price used in the analysis is that estimated by E G Russell.

Once a decision is made to go ahead with the project there will be a delay of two to three years while construction is underway prior to production commencing. This means that with two years of lead time (project development) and three years of construction, we are five years away from production. To assume the metal price will be significantly less in real terms between 1989 (five years from now) and 1998 than it is now is conservative and arbitrary.

Metallurgical recovery estimates are derived from forecasts by CMS (S S Meik) for recoveries from eight drill cores. They are based on limited testwork.

	Queen Hill	Severn
Gravity	15.05%	41.25%
Direct Fume Feed	25.45	16.31
Float Fume Feed	18.90	12.11
Sulp Cleaner Tail	22.50	16.13
TOTAL	81.90%	85.80%

Since these recoveries are high in comparison with Cleveland and Renison, it is unlikely that significant increases will be experienced.

Ore grades are based on production scheduling from the kriged block grades after allowing for mining recovery and dilution. Two basic assumptions warrant review. Dilution is arbitrary but judgement is made that ground conditions in Severn will be quite reasonable and that in Queen Hill while ground conditions will be worse the mining scheme aims at low dilution rather than high recovery.

There is little justification for altering these assumptions. Geological Resource Grade is quite well defined in Queen Hill lens but in Severn and Montana the predictions are based on very sparse drilling.

There is a suggestion that grade decreases in Severn lens with increased depth, but since the diamond drill holes are 150 to 200 metres apart (3 holes) at this level, it is not unreasonable to expect changes to take place with further drilling. There is a good chance that the Severn lens high grade resource grade will increase as definition improves. The production rate of 150,000 tonnes was chosen with the thickness, complexity and depths and dimensions of the deposits in mind. The production schedule assumes some modest extensions to resources in Severn lens but does not use those in Montana. Depth extensions in either of the lenses were ignored. Increased production rates are possible but rely on increased capital for mine haulage and mill capacity. Modest increases of the order of 15% should be possible without significant additional capital requirements.

9.2 COST ASSUMPTIONS

Capital Cost

All capital costs are order of magnitude estimates. Any changes should be based on change of scope rather than arbitrary cutting except where it can be shown that new evidence is to hand indicating substantially different unit rates.

Proposals to establish a fumer elsewhere may benefit Zeehan. Savings in capital (fumer and housing) and possible savings in operating costs may result.

Opportunities to acquire used, major capital equipment have occurred and will occur again. Shutdown of Ardlethan tin mine is a distinct possibility and should that happen, Zeehan mine could derive capital savings of up to $3 million, particularly in the concentrator.

All mine waste development is capitalised and estimated using contract rates. In house development will reduce capital. Housing capital assumes services will be available. All houses are project owned. Various alternatives involving State Government assistance will reduce capital and improve economic outcome.

Concentrator capital is estimated with a low level of scope definition. The high metal recovery cannot be sacrificed if capital is to be kept down. As scope definition improves it is more likely that capital will increase rather than decease.

Operating Cost

Operating costs are based on assumptions and comparison with other mines. Many of the unit rates are relatively high and reductions are probable on most, after further detailing of specification.

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Mining Cost -	Average assumed Zeehan	$17.14/tonne
	Interpreted equivalent Que River	$16.81 (with cemented fill)
	Interpreted equivalent Cleveland	$17.70
	Interpreted equivalent Ardlethan	$ 6.08

While Zeehan will be a new operation and costs relatively high it is unlikely that they will exceed those for Que River. In particular, production from the Queen Hill open stopes will be at a similar rate and scale to that in Ardwest Deeps. It is reasonable to expect that mining costs may be reduced by $1.0/tonne.

Milling Cost -	Average assumed Zeehan cost	$12/tonne
	Cleveland	$13.91/tonne milled
	Ardlethan	$12.24/tonne milled

It is expected that the simplified processing route together with fuming to achieve high concentrate grade will be accompanied by a relatively low milling cost. This low cost will be offset to a degree by economies of scale apparent at Cleveland and Ardlethan.

This encourages us to believe that $12/tonne of mill feed can be achieved with some opportunity for reduction.

Fuming Cost

The fuming cost is assumed to be $125/tonne of feed not including administration or capital charges. Commercial operation of a fumer is yet to be demonstrated. The assumption must carry significant risks.

Overheads and Administration (other) covers costs not specifically identified with mining, concentration or fuming. It is assumed to be $12/tonne. It is possible that this number is too high. Zeehan is a commercially independent centre and local services will reduce mine costs.

9.3 ECONOMIC ANALYSIS

The financial analysis uses a depreciation allowance based on the rule that the capital may be depreciated over the mine life or ten years whichever is the lesser. Even so the scheme pays $8.6 million tax in the first four years of production.

Any opportunity to depreciate at a higher rate in the early years of production will benefit the project.

All the waste development is capitalised and subsequently depreciated over the mine life. It is likely that a proportion of these costs will be justifiable operating costs. All waste development is completed in the first four years of production. Detailed scheduling is expected to result in some of the development being delayed. This will contribute to an improved economic outcome.

9.4 COST OF TIN PRODUCTION

- With the outlook on growth of the world market for tin at least clouded, the investment rating of the project will be partly dependent on its ability to produce tin at a low cost, particularly in relation to other well placed tin producers.

The cost of production of tin at Zeehan is summarised as follows:-

	Tonne Sn Produced	Cost/Tonne Cash	Cost/Tonne Incl Capital
Good Year (6)	1,900	$4,615/t	$6,747/t
Poor Year (11)	1,482	$5,701/t	$8,806/t
Project Life Av	1,626	$5,162/t	$7,884/t
		$M10.58/kg	$M16.16/kg

($M/$A = 2.05)

The cash cost is equivalent to the revenue which must be received when only those costs directly related to current production are incurred. The cost of production including capital is equivalent to the revenue which must be received to allow repayment of capital.

The price of tin could fall to 63% of its forecast level or 54% of the forecast 1983 level before the project would show a loss after depreciation. The total cost of production is in line with that for low cost Asian tin producers and very low compared with Australian tin producers.

The project as presented initially, was judged to have a moderate investment potential. Analysis of the opportunities changes the assessment. With better than normal opportunities for resource increase, grade increase, operating cost reduction, rescheduling of capital expenditure and reduction in taxation, the project is significantly enhanced. The low cost of production renders the operation highly competitive with other world producers and places it in a strong position to make a significant profit contribution to the Aberfoyle group. The location of the resources, within the town limits of Zeehan and adjacent to road transport and other services contributes to the attraction of the project.

In a pre feasibility study the objective is to determine whether or not the project has a reasonable chance of going on to become a profitable operation and if so how that position should be reached. The Zeehan project as studied has a moderate chance of becoming an operation but has many opportunities for enhancement. It should be developed to a point where a decision can be made on the basis of more detailed definition such as that achieved in a final feasibility. The direction, rate and cost of development are dependent on the risks and how they affect the final outcome.

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10 ANALYSIS OF RISK

By comparison of the resultant NPV at 10% discount rate, the relative significance of changes in grade, recovery, metal price, production rate, operating costs and capital costs can be compared. Table 2.

Probability of occurrence must also be taken into account since the chances of a 10% increase in recovery are not equal to the chances of a 10% decrease in recovery. The Parameter Method for Risk Analysis (Cooper and Davidson) presents a method of combining probability and variance, particularly where the distribution of variance is skewed such as in this Zeehan example.

Table 3 illustrates the results of the analysis. The weighted average net present value is significantly below that which is the "project expected value". This outcome is due to the degree of definition rather than statistical variance.

Both the mean NPV and 10% loss exceeding NPV" are largely dependant on the low and high variances for Revenue and Capital at 80% confidence level. Both can be changed and are a function of definition of the ore reserves and more metallurgical testwork. Further metallurgy is dependant on more sample and consequently can only be advanced as drilling proceeds. The ore reserves are variably defined. Queen Hill lens is drilled to an indicated resource category where the grade estimation error is +30% at a 95% confidence level. Severn lens is an inferred resource and very much underdrilled. It is Severn lens which is the major contributor to the uncertainty. No more than 10 intersections delineate a high grade resource in excess of 1.0 million tonnes at drill spacing up to 200 metres. All evidence to date suggests a drill spacing of 50 metres is the maximum acceptable for the level of definition required.

Capital costs are made up essentially of decline, mill, fumer and town costs. For the planned production level there are unlikely to be significant changes in the decline or fumer capital costs since the estimates are based on reasonably current information. The concentrator capital estimate is based on limited definition of scope and is the most likely major item to vary. Design of the concentrator is a consequence of further ore characterisation and circuit optimisation.

The proposed mine is capital intensive and consequently increases in production will have benefit as will earlier commencement of production. Access alternatives and production schedule variations are possible but will have little added credence as the resources on which they will be based are poorly defined.

To reduce the risk of project failure it is evident that the following programme should be initiated:-

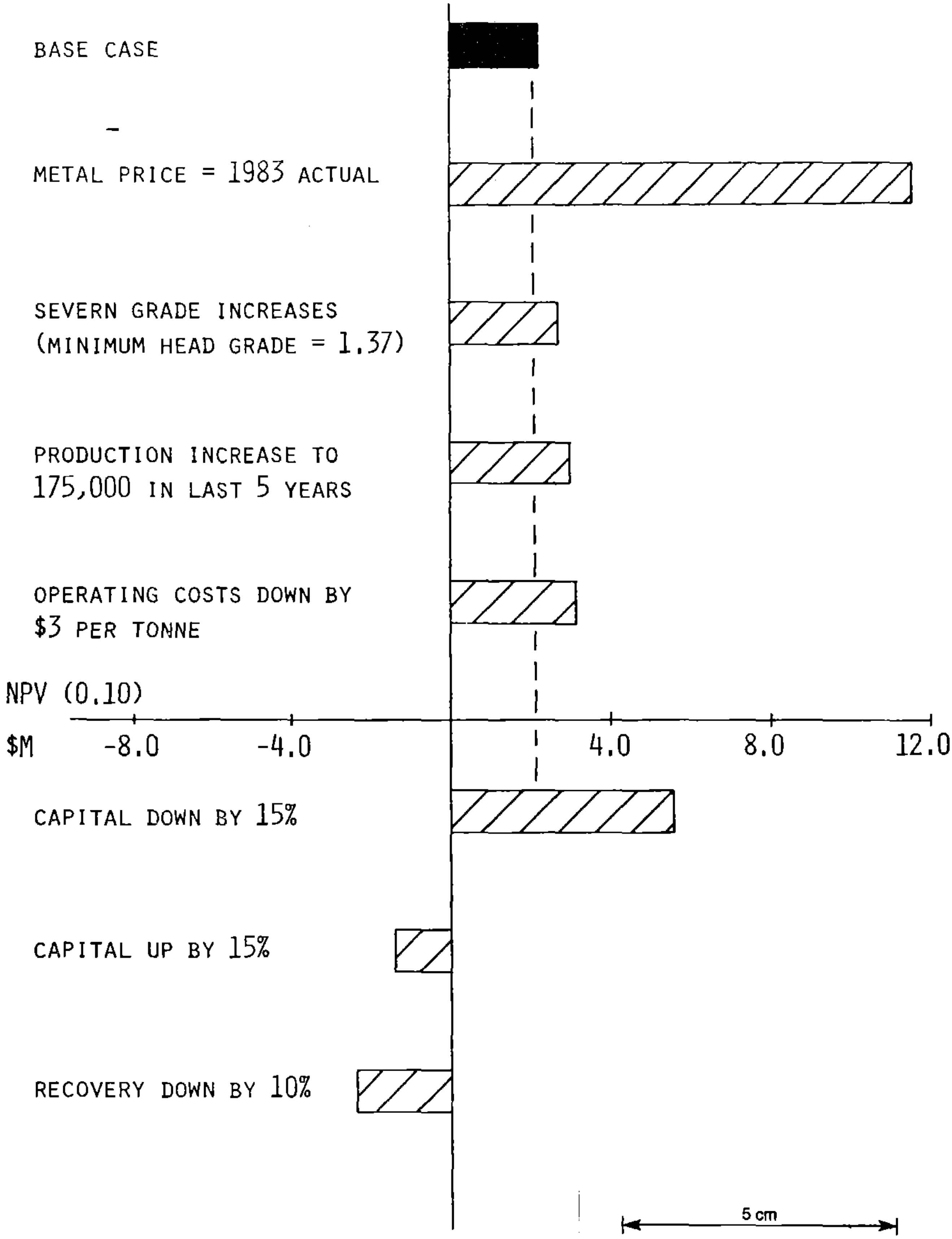

TABLE 2 - SENSITIVITY DIAGRAM

TABLE 3 WEIGHTED NPV CALCULATIONS

	LINE ITEM PRESENT VALUE	AFTER TAX IMPACT FACTOR	PRESENT VALUE AFTER TAX	80% CONFIDENCE LEVEL		IMPACT OF PROJECT PRESENT VALUE	
				LOW -%	HIGH +%	MEAN	STD DEV.
Column No.							
REVENUE 90.98	90.98	0.54	49.13	40%	25%	47.29	12.05
EXPENSES 40.86	40.86	-0.54	-22.06	10%	25%	-22.89	2.92
CAPITAL 35.39	35.39	-0.71	-24.78	7.5%	30%	-26.17	3.51
NET CASH FLOW AFTER TAX	2.29	1.0	2.29			-1.77	12.89

PROJECT VIEW	$2.29 million	10% chance that loss could exceed $18.27 million
MEAN	-$1.77 million equivalent to 8.5% DCF	
STANDARD DEV.	$12.89 million	

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i) Reduce the uncertainty of the estimation of concentrator recovery.

ii) Reduce the grade estimation error.

iii) Refine the capital cost estimates.

iv) Refine the operating cost estimates.

The question to be answered prior to commencing on this program is "how much of this work should we do and what risk should we be prepared to accept?"

Acceptable risk is a function of company policy and partly a function of upside potential not incorporated in the risk analysis. With the opportunities previously identified it is judged that if a project can be defined where the weighted NPV is positive at 8 or 9% discount rate and where there is a 90% chance that the project will return better than an 5% DCF (or an NPV of -$5.00 at 10%) (the risk free after tax return on investment) without including the potential benefits of the opportunities, then it should be considered to have passed the investment hurdles.

Recommended Program

The results presented above show that the most likely outcome without further project development is less than the predicted NPV. If the project proceeded without deviation from the plan and without further definition there is a 10% chance that the loss could exceed $18.27 million dollars. Such a loss would be unacceptable.

The relationship between further work (and expenditure), reduction of risk and change in NPV was analysed using the Paramater Method, by taking the financial model used previously and adding additional staged early capital. The results were incorporated in the parameter method tables.

The following assumptions were made:

i) Diamond Drilling

Approximately 18 new holes	7500 metres
10 wedged intersections	500 metres
10 wedging operations	($50,000)
Total Cost	$1,210,000

All resources to $\pm$30% at 95% confidence level. Generates 23 intersections for ore characterisation and approximately 500 kilograms of core for compositing and laboratory testing.

ii) Laboratory Metallurgy

Ore characterisation 23 at $5,000	$115,000
Process testing (on composite) and circuit design	$100,000
Total cost including Contingencies	$250,000

Recovery estimates upgraded to a point where it is judged that there is no more than a 10% chance that the recovery will be less than 55%.

iii) Pilot Plant

It is assumed that $1.5 million of mine waste development capital is advanced two years to provide a decline for bulk sample of Queen Hill Lower.

The extra cost to cover the penalty for small contract, establishment and mining of 2,000 tonnes is estimated to be of the order of $300,000. Pilot plant construction and operation is estimated to be $700,000. A further allowance of 21% is added to cover design, overheads and administration.

Capital cost estimation will be refined and confidence in recovery estimates improved. Operating cost estimates will be much more detailed and confidence improved.

iv) Final Feasibility is estimated to cost $500,000, but unless based on additional drilling and laboratory wet metallurgy, will do little to improve confidence in the relationship between final outcome and estimates or design.

Tables 4 and 5 illustrate the change in NPV (0.10) with exploration and development expenditure and consequent reduction in risk. The significant result is that NPV is maintained or improved to completion of the laboratory metallurgy with reduction in risk from a 10% probability of making a loss greater than $15.09 million dollars to a 10% probability of making a loss greater than $4.95 million. The outcome is equivalent to about a 95% DCF with a 90% probability of making greater than a 5% DCF. Further capital expenditure on pilot plant and bulk sampling marginally reduced the expected NPV, but has little effect on the risk and it is concluded that a decision to go to construction should be made without a pilot plant programme.

TABLE 4 ANALYSIS OF EXPENDITURE AND REDUCTION OF RISK

	LINE ITEM PRESENT VALUE (10% DISC RATE)	AFTER TAX IMPACT FACTOR	PRESENT VALUE AFTER TAX	80% CONFIDENCE LEVEL		IMPACT OF PROJECT PRESENT VALUE	
				LOW -%	HIGH +%	MEAN	STD DEV.
REVENUE							
(1) Base Case	75.18	0.54	40.60	40%	25%	39.08	9.96
(2) After DD and Final Feas.				18%	7%	39.48	3.83
(3) After DD and Met & Final Feas.				12%	6%	39.99	2.76
(4) After DD Wet Met, Pilot Plant				10%	5%	40.09	2.30
(5) Final Feas. Only				40%	25%		
EXPENSES							
(1) Base Case	33.77	-0.54	-18.24	10%	25%	-18.92	2.41
(4) After Final Feas.				10%	10%	-18.24	1.38
CAPITAL							
(1) Base Case	29.25	-0.70	-20.47	7.5%	30%	-21.62	2.90
(2)	30.76		-21.53	7.5%	20.0%	-22.20	2.23
(3)	30.99		-21.69	7.5%	15%	-22.10	1.84
(4)	31.92		-22.34	7.5%	10%	-22.48	1.48
(5) Final Feas.	29.66		-20.76	7.5%	25.0%	-21.67	2.55
NCF AFTER TAX	1.89	1.0	1.89				

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TABLE 5 WEIGHTED NPV CALCULATIONS AFTER DEVELOPMENT EXPENDITURE

	BASE CASE	B/C PLUS FINAL FEAS.	B/C, F/F, PLUS DIAMOND DRILLING	B/C, F/F, D/D, PLUS WET METALLURGY	B/C, F/F, D/D, W/M, PLUS PILOT PLANT
WEIGHTED NPV	-1.46	-0.83	-0.96	-0.35	-0.63
STANDARD DEV.	10.65	10.37	4.64	3.59	3.06
10% PROB LOSS	-15.09	-14.10	-6.90	-4.95	-4.55

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The recommended programme is:

- DRILL SEVERN TO ±30% AT 95% CONFIDENCE LEVEL
- PROCESS TESTING AND DESIGN
- REASSESS CONCEPTUAL ECONOMICS
- FINAL FEASIBILITY

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11 PROGRAMME TIMING

It is likely that the pre construction and pre production programmes will take the following times:

Diamond Drilling	9 months
Metallurgy	+ 4 months
Reassess	+ 2 months
Final Feasibility	+ 6 months
Pre Production Construction	24-36 months

The minimum period to get to production is 3.75 years and more likely is five years. The time required to get to a point where a decision can be made to go to final feasibility is at least 15 months. It is clear that drilling should commence immediately if it is independant of outside factors. However the following factors are relevant:

i) Tin market

ii) Finance

iii) Availability of Major Equipment eg. from Ardlethan closure

iv) Compatibility with development of commercial fuming

The tin market is probably the least important of all the factors. Production is five years away and final feasibility at least 15 months from commencement. The condition of the tin market can be reviewed regularly over the next two years and commitment to construction can wait until such time as conditions are favourable. But when conditions do appear to be right construction must commence without delay.

Finance should not be a major hurdle since a good investment should attract funds.

Availability of major Capital Equipment is a significant factor since acquisition of items such as used Crusher, Cassiterite flotation plant or any other suitable equipment will have an economic benefit to the project.

Should a fumer be placed in production (at Cleveland), demonstration will have benefits in reducing risk and confirming capital and operating costs.

Since it is judged to be an attractive project, it should be placed in a position to take advantage of Ardlethan shut down in 1 to 2 years time and tin market stabilisation, the program of diamond drilling and laboratory metallurgy should commence immediately.

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12 REFERENCES

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2 D O Cooper & L B Davidson, "The Parameter Method for Risk Analysis", CEP November 1976.

3 S S Meik, "Zeehan Deposits - Progress Report No 1", Aberfoyle Ltd, Central Metallurgical Services Report, November 1982.

4 K G Palmer, "Zeehan Project Geological Resource Assessment", Aberfoyle Exploration Pty Ltd, Melbourne, August 1982.

APPENDIX 1

ZEEHAN PROJECT : PRE-FEASIBILITY STUDY FINANCIAL ANALYSIS

Years	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
PRODUCTION SCHEDULE															
Devt. 00'S Tn			365	120	260		250	125						67	
OH c& f 00'S Tn				250	250	250	170		50	50	50	50	50	833	
OH op.stp. 00'S Tn				550	400	170		505	460	260					
Sev up c&f 00'S Tn					600	630	600	70	900	630					
Sev lr c&f 00'S Tn						450	480	800	90	560	1450	1450	1450	600	
PRODUCTION															
Tonnes Mill 000's				125.00	150.00	150.00	150.00	150.00	150.00	150.00	150.00	150.00	150.00	150.00	
%Sn				1.52	1.55	1.55	1.50	1.37	1.37	1.31	1.19	1.21	1.21	1.33	
Grvty rec. %				16.38	29.97	33.91	38.28	31.99	32.34	35.84	40.38	40.38	40.38	19.69	
Rec. to Fum con %				63.90	54.31	50.87	17.17	52.49	52.20	49.24	45.39	45.39	45.39	53.37	
Fumer rec. %				90.00	94.00	94.00	94.00	94.00	94.00	94.00	94.00	94.00	94.00	94.00	94.00
Fumed con. TNES TIN				1092.7	1184.8	1111.8	997.6	1013.9	1008.3	909.5	761.6	774.4	774.4	1000.8	
Grvty con TNES TIN				311.2	696.8	788.4	861.3	657.4	664.6	704.3	720.8	732.9	732.9	392.8	
TOTAL CON. TNES TIN				1403.9	1881.6	1900.2	1858.9	1671.3	1672.9	1613.8	1482.4	1507.3	1507.3	1393.7	
GROSS REVENUE $MILL. (GRV=90~92% of $12430/tonne)				15.79	21.20	21.43	20.99	18.84	18.86	18.21	16.74	17.03	17.03	15.68	0.00
Tonnes Fmd. 000's				17.34	18.01	16.90	15.16	15.41	15.32	13.82	11.57	11.77	11.77	15.21	
CAPITAL COST $MILL.															
MINE															
Waste Dev.	2.00	2.50	2.10	2.35	1.41	1.41									
Mine Equip.		.50	2.50												
CONCENTRATOR															
Crush&Grind		1.00	1.30												
Grav. Sulph.Ox flt.		1.00	1.30												
Thick.Filt.Class		.30	.60												
Fumer		1.05	2.20												
Tail Disp.		.10	.70												
Off.Site Wks.Build.		1.00	.75												
Contingencies		1.50	1.60												
TOWN		3.00	2.82												
STORE WORK & REPL.			.50	1.00	.50	.50	.50	.50	.50	.75	.75	.75	.75		
CONSTRUCTION O'HEADS	.42	2.19	3.00	.49	.30	.30	0.00	0.00	0.00	0.00	0.00	0.00	0.00		
TOTAL CAPITAL	2.42	14.14	19.37	3.84	2.21	2.21	.50	.50	.50	.75	.75	.75	.75		

APPENDIX 1

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OPERATING COSTS

Mine	$MILLION														
QH Cut & Fi $15/T	0.00	0.00	0.00	.38	.38	.38	.25	0.00	.08	.08	.08	.08	.08	1.25	0.00
Sev up C & F $17/t	0.00	0.00	0.00	0.00	1.02	1.07	1.02	.12	1.53	1.07	0.00	0.00	0.00	0.00	0.00
Sev 1r C & F $20/t	0.00	0.00	0.00	0.00	0.00	.90	.96	1.60	.18	1.12	2.90	2.90	2.90	1.20	0.00
Ore Dev. $17/t	0.00	0.00	.62	.20	.41	0.00	.13	.21	0.00	0.00	0.00	0.00	0.00	.11	0.00
QH Op Stope $10/t	0.00	0.00	0.00	.55	.40	.17	0.00	.50	.46	.26	0.00	0.00	0.00	0.00	0.00
Concentrator															
Conventiona $12/t	0.00	0.00	0.00	1.50	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.80	0.00
Fumer $125/t	0.00	0.00	0.00	2.17	2.25	2.11	1.90	1.93	1.92	1.73	1.45	1.47	1.47	1.90	0.00
Realisatn. $2.9/T	0.00	0.00	0.00	.40	.54	.54	.53	.48	.48	.46	.43	.43	.43	.40	0.00
O'hds&Admn. $12/t	0.00	0.00	.44	1.10	1.81	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.80	1.80	0.00
TOTAL OPERATING COSTS	0.00	0.00	1.06	6.30	8.64	8.77	8.69	8.44	8.24	8.32	8.45	8.48	9.48	8.46	

DEPRECIATION INVESTMENT ALLOWANCE & TAX

1) Depreciation															
Brt. frwd	0.00	2.42	16.56	35.93	35.80	34.20	32.36	28.76	25.08	21.31	17.65	13.80	9.70	5.23	
Capex	2.42	14.14	19.37	3.81	2.21	2.21	.50	.50	.50	.75	.75	.75	.75	0.00	
Mine life	10	10	10	10	10	9	8	7	6	5	4	3	2	1	
Depreciation	0.00	0.00	0.00	3.98	3.80	4.05	4.11	4.18	4.26	4.41	4.60	4.85	5.23	5.23	
Carry forward	2.42	16.56	35.93	35.80	34.20	32.36	28.76	25.08	21.31	17.65	13.80	9.70	5.23	0.00	
2) Investment All															
a) Eq mill fum rep	0.00	3.85	8.40	1.00	.50	.50	.50	.50	.50	.75	.75	.75	.75	0.00	
b)Others	0.00	1.10	1.45	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
c)70%* a) +40%*b)	0.00	3.14	9.60	10.30	.35	.35	.35	.35	.35	.53	.53	.53	.53	0.00	
Invest all	0.00	0.00	0.00	1.85	.06	.06	.06	.06	.06	.09	.09	.09	.09	0.00	
Carry fwd.	0.00	3.14	9.60	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
TOTAL DEDUCTIONS	0.00	0.00	0.00	5.83	3.86	4.11	4.17	4.24	4.33	4.51	4.69	4.94	5.32	5.23	
OTURN BEF ROYALTY	0.00	0.00	-1.06	5.50	8.77	8.62	8.19	6.22	6.36	5.48	3.70	3.70	3.32	1.99	
ROYALTY	0.00	0.00	0.00	.27	.44	.43	.41	.31	.32	.27	.18	.18	.17	.10	
PRE TAX SURPLUS	0.00	0.00	-1.06	2.31	8.27	8.12	7.72	5.85	5.98	5.11	3.42	3.42	3.06	1.89	
Carry fwd.	0.00	0.00	1.06	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
TAX DUE	0.00	0.00	0.00	1.06	3.80	3.74	3.55	2.69	2.75	2.35	1.57	1.57	1.41	.87	
NET ANN. CASH FLOW	-2.42	-14.14	-20.43	5.36	8.86	6.22	7.65	6.04	7.12	6.12	5.01	6.04	6.06	5.71	-.87

INTERNAL RATE OF RETU .12 OR 11.57%

N.P.V. AT 10% 2.29 MILLION DOLLARS

ZEEHAN FINANCIAL ANALYSIS Author.K.G.P. Date 29.04.83

APPENDIX 1

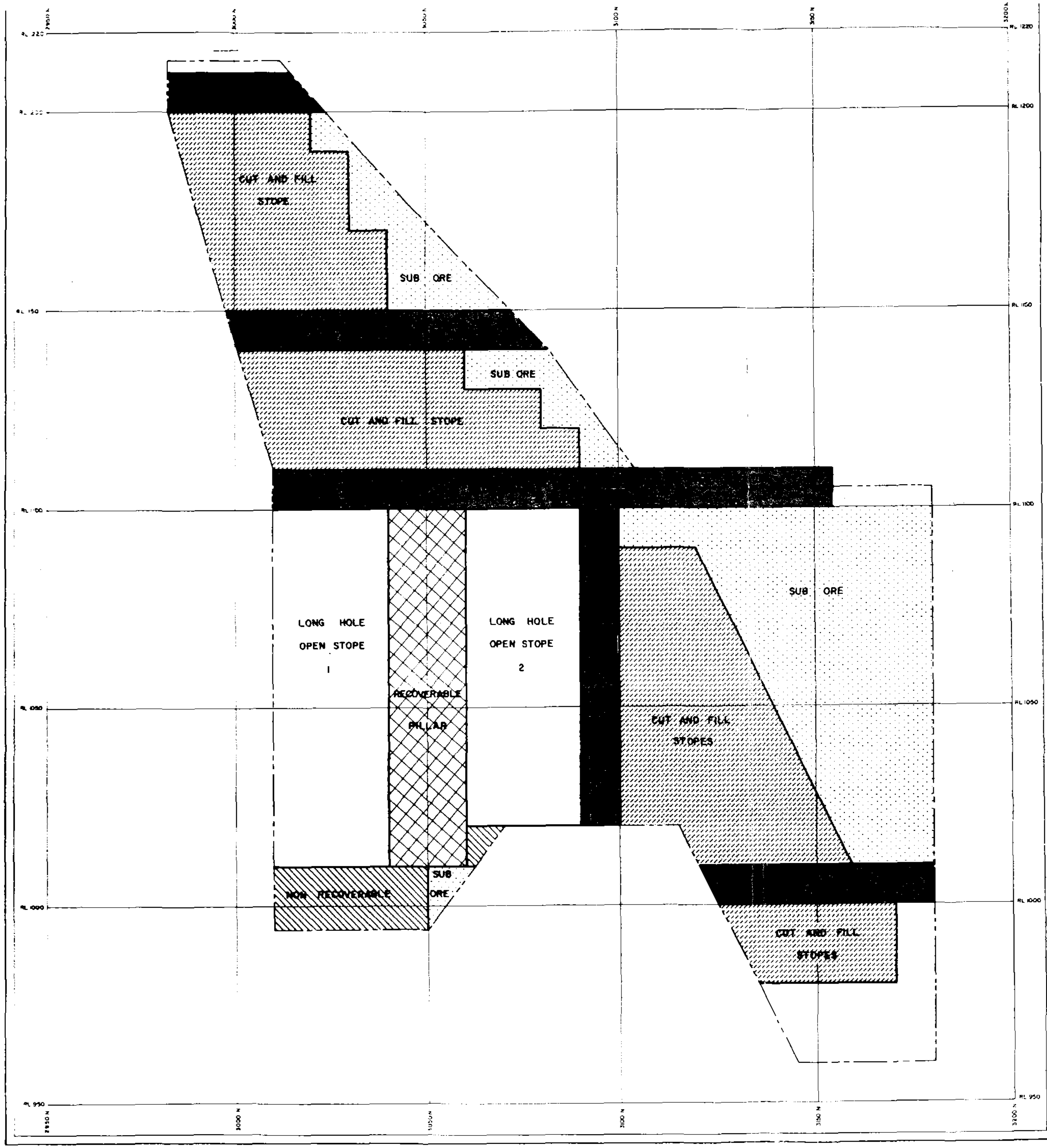
269040

A2

APPENDIX 2

A3

269041



ABERFCYLE EXPLORATION PTY. LTD.

ZEEHAN PROJECT

QUEEN HILL LENS
LONGITUDINAL PROJECTION

CONCEPTUAL STOPE DESIGN

0 5 10 20 30 40 50 METRES

5 cm

REFERENCE

- Cut and fill stope
- Long hole open stope
- Recoverable pillar
- Non recoverable pillar
- Sub ore
- Non recoverable

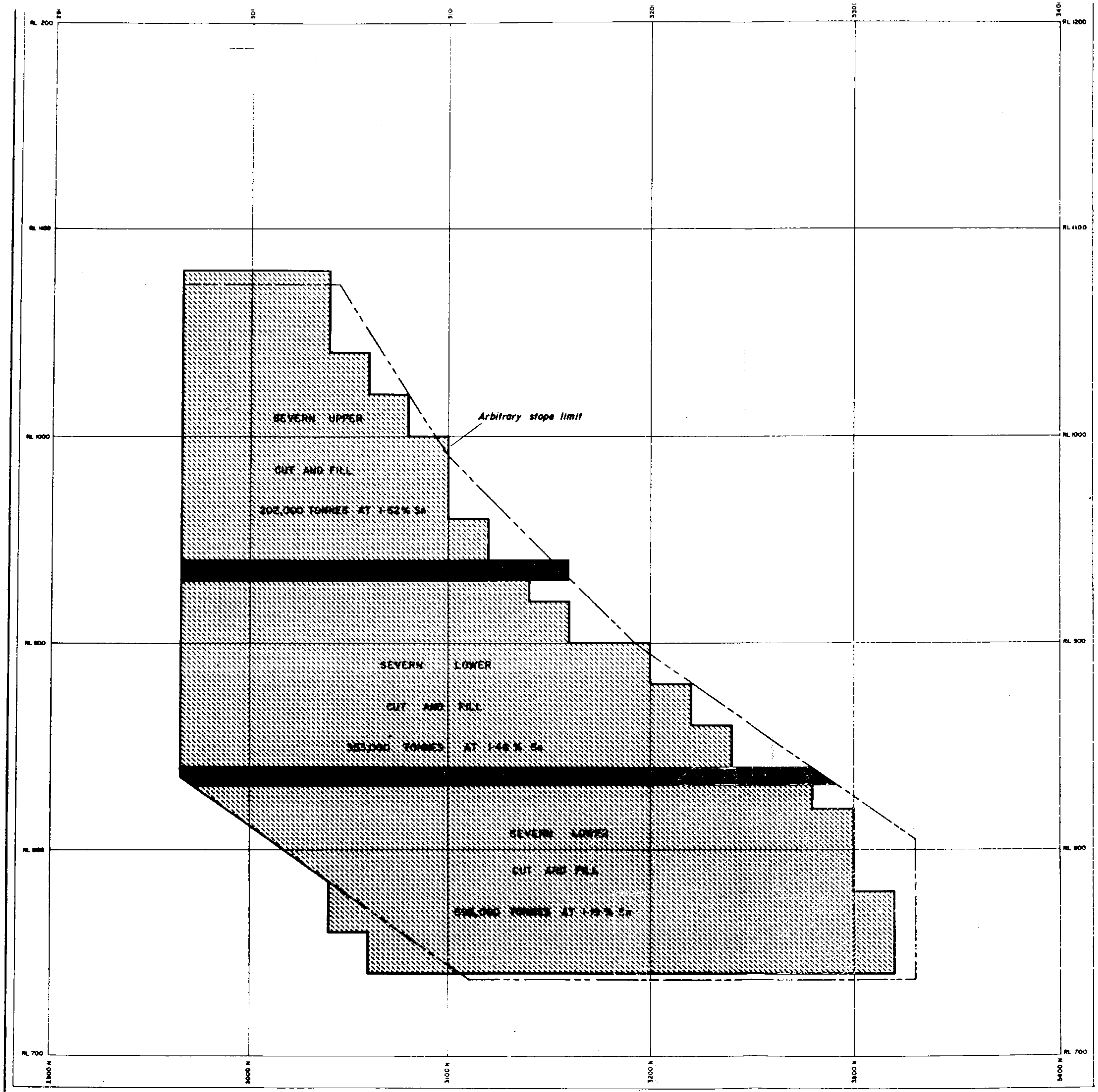
Stope design by K G Palmer
Drawn by K. G. Potter
May 1983

PLATE No. QH-221

APPENDIX 3

A4

269042



ABERFOYLE EXPLORATION PTY. LTD.

ZEEHAN PROJECT

SEVERN LENS
LONGITUDINAL PROJECTION

CONCEPTUAL STOPE DESIGN

0 10 20 40 60 80 100 METRES

5 cm

REFERENCE

Cut and fill stope

Non recoverable pillar

Stope design by K G Palmer
Drawn by K. G Potter
May 1983

PLATE No. QH-222