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AUSTRALIAN ANGLO AMERICAN LIMITED

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RINGAROOMA JOINT VENTURE PROJECT

PRE-FEASIBILITY STUDY

SUMMARY

This is a pre-feasibility report presented to the partners of the Ringarooma Joint Venture Project, as required by the agreements between them. This evaluation of the project follows completion of several drilling programmes over the period 1981-84.

Geological examination and data indicate a total geological reserve of 18.2 Mm^3 averaging 256 g cass conc/m^3 @ 70% Sn in the selected Pioneer/Endurance leads, at a cut-off grade of 140 g cass conc/m^3. Translated into a mineable (dredgeable) reserve, this amounts to 23.5 Mm^3 containing 4754 t cass conc and grading 202 g cass conc/m^3 @ 70% Sn, after due allowance being given for side slopes and some bedrock excavation. (Both these need to be confirmed by check drilling.) A break-down of the mineable reserve is presented in the following tabulation.

	Volume of Overburden and Ore (Mm^3)	Grade Overburden and Ore (g cass conc/m^3) @ 70% Sn)	Cass Conc @ 70% Sn (t)
Pioneer:			
- Proven reserve west of mining pit	3.12	257	801
- Probable reserve southwest and south of pit	11.98	170	2035
Endurance:			
- Probable reserve east of 'kink'	4.07	225	915
- Probable reserve west of 'kink'	4.33	232	1003
	23.50	202	4754

In addition, a possible reserve of 6.2 Mm^3 grading 165 g cass conc/m^3 @ 70% Sn has been indicated at Pioneer in the Gilhams Creek deviation area. Some of the old Pioneer Mine tailings have been disposed of in this area and its effect on the grade is yet to be determined.

As regards to valuation of boreholes used in the ore reserve evaluation, the method follows Amdex's procedure of applying the Radford Factor (80%) to the theoretical volume in deriving the borehole sectional grade where the actual volume recovered is less than the 'factored' theoretical volume; for cases in which actual volume exceeds the 'factored' theoretical volume, the grade is related to the former volume.

Three possible mining systems for exploiting the ore reserves have been considered. These are:

- dredging with dry stripping of overburden (preferably by contractor);
- open-pitting by dry earthmoving (preferably by contractor);
- hydraulic monitoring and gravel pumping.

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The dredging method is suitable and has the distinct advantage of a lower operating unit cost. Either an underwater bucket wheel or a small capacity bucket ladder dredge appears suited to exploiting the Pioneer/Endurance deposits, provided that future check drilling confirms that the stanniferous wash materials do not contain appreciable quantities of stiff, sticky clay, large cobbles and boulders and buried timber.

The type of dredge envisaged for the scale of operation has a 15m digging depth below pond water level and a normal annual throughput of 1.0 Mm^3 for treatment. Dry stripping of the overburden will be carried out ahead of the dredging face by earthmoving equipment. The stripping depth is 18-26m from surface and the normal rate is 1.6-1.8 Mm^3 per year

Dry, open-pit mining for both the overburden and paydirt, using earthmoving equipment, is flexible and has an appeal for the Pioneer/Endurance reserves which occur in two separate deposits. If carried out under contract, it would also have the obvious economic advantage of reduced capital outlay.

The overburden stripping depth/rate and the paydirt mining depth/rate are both similar to the dredging operation.

Hydraulic mining appears to have economic merit because of the available water supply and the natural hydraulic head from the Frome Dam, particularly for exploiting the Pioneer reserve. The unfavourable factors are the stiff clayey overburden and coarse gravelly/cobbly wash which would have an adverse effect on mining rate; a requirement of large slime storage/retention dams and higher manpower intensity than dredging.

In this pre-feasibility study, the hydraulic operating unit cost is estimated to be higher than the dredging method and therefore the operation has to be more selective in mining the ore reserves. Consequently, the lower grade reserve block at the 'southwest and south of pit' at Pioneer has been excluded from the operation. The total mineable reserve therefore is 11.52 Mm^3 of alluvium averaging 236 g cass conc/m^3 @ 70% Sn (compared with 23.5 Mm^3 grading 202 g cass conc/m^3 @ 70% Sn for the dredging method).

Hydraulic overburden stripping rate is assumed at 0.72-0.97 Mm^3 for a normal operating year. Mining and treating the underlying paydirt is 0.50 Mm^3 per year.

The preliminary estimates of the pre-production capital and operating costs for the mining methods are summarised as follows:-

	Case	Pre-production Capital Cost ($M)	Av Operating Cost ($/$m^3$ of overburden and paydirt)
Dredging:			
- Underwater bucket wheel operation, including contract stripping of overburden	A	7.4	1.47
- Bucket ladder dredge operation, including contract stripping of overburden	B	8.0	1.53
Open-pitting, including contract stripping/ore removal	C	4.8	1.71
Hydraulic monitoring and gravel pumping	D	6.6	2.35

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On the basis of the foregoing capital/operating costs and production commencement in 1989, a preliminary economic evaluation gives the following indications at an adopted tin price of $A16.50/kg:

- Dredging is probably preferred to dry open-pit and hydraulic mining, but further detailed work would be required for definitive selection.

- Exploitation by either an underwater bucket wheel or a bucket ladder dredge or hydraulic method shows poor economic returns (DCF Yields less than 1% in each case).

- Contract mining both the overburden and ore at the expected costs of $1.20-1.30/$m^3$ shows a negative DCF Yield, despite reduced pre-production capital outlay, mainly because of higher overall mining cost (ie. $1.71/$m^3$). For an optimistic contract cost at $1.00/$m^3$ for both overburden and ore removal, a 6% DCF Yield is indicated which is still below the acceptable requirement.

For the dredging project to return a 10-12% DCF Yield, it needs one of the following:

- an improvement of about 20-25% in grade of the reserve (ie. a mineable reserve in excess of 240 g cass conc/m @ 70% Sn), or

- a further 20 Mm^3 of mineable reserves grading 200 g cass conc/m^3 @ 70% Sn (ie. a total mineable reserve in excess of 40 Mm^3), or

- a tin price 25% higher than the level used in the current evaluation (ie. above $A 20/kg).

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RINGAROOMA JOINT VENTURE PROJECT
PRE-FEASIBILITY STUDY

1. INTRODUCTION

1.1 Australian Anglo American Prospecting Pty Ltd ("Anglo") and Triako Mines NL/Buka Minerals NL/Kibuka Mines Pty Ltd/Dusty Mac Ltd ("Triako") and Amdex Mining Ltd ("Amdex") have joint venture agreements to explore for alluvial tin in the Triako/Amdex tenements in the Ringarooma Valley, Tasmania. In the agreements between Anglo and Triako/Amdex, there are three stages of development work prior to production mining which are essentially outlined as follows :

Stage I - Anglo conducts all exploration, is responsible for maintenance of tenements, reports, expenditure requirements, programmes and budgets, etc; completes a Pre-Feasibility Report and determines with Triako/Amdex a programme and budget to complete Stage II;

Stage II - Commences when Anglo and Triako/Amdex agree to the programme and budget for Stage II. During Stage II, the Operator presents a feasibility study of the Mining Area Stage II leases. The stage concludes upon a recommendation to mine by either Anglo or Triako/Amdex.

Stage III - If agreement is reached to mine the Mining Area, a new agreement ensues for mining and production of tin ores and concentrates from the Mining Area.

1.2 Since the execution of the first agreement in October 1981, Anglo has been conducting exploration drilling in the joint venture areas as required in Stage I. All the drilling programmes completed to date have now culminated in the selection of possible mining reserves for evaluation purposes. A Pre-Feasibility Report is submitted herewith to the joint venture partners for consideration and approval.

1.3 Essentially, this report is prepared to present the following aspects:

- evaluation of the ore reserves from drilling data to date;
- exploitation of the selected reserves by conceptual mining methods;
- preliminary assessment of the project's economics;
- conclusions.

1.4 In addition to the above, the following important information is appended herewith which has influence in deciding the future course of action for the project:

- Tin supply/demand and price forecast study by Anglo American Corporation. (Appendix I)
- International Tin Council tin export control and buffer stock operations and Australia's tin quota allocation system. (Appendix II).

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2. LOCATION

The tenements covered by the joint venture agreements are situated mainly in the Ringarooma Valley in north-eastern Tasmania, about 80-100 km directly north-east of Launceston. Access to them from Launceston, via Scottsdale, is by a good bitumen road. (Ref Plan TAS-100-11/1 for location).

3. GEOLOGY

3.1 General

3.1.1 The Ringarooma Basin appears to have developed following a period of tectonism during the Palaeocene to early Eocene. Concentrations of alluvial cassiterite in the basin are present in Tertiary sediments derived from erosion of the tin-bearing Upper Devonian to Lower Carboniferous Blue Tier Batholith. Primary tin mineralisation occurs within flat-lying greisen sheets in granite, steeply-dipping greisen veins or pipes in granite and cassiterite stockworks within the adjacent Ordovician to Devonian Mathinna Beds. The major source of detrital cassiterite is the flat-lying greisen sheets which occur close to the upper contact of bodies of muscovite-biotite granite within the batholith.

3.1.2 Minor quantities of alluvial gold are present within the Quaternary sediments of the Ringarooma Valley and these have been derived from discordant quartz veins within the Mathinna Beds to the west of the Blue Tier Batholith. These quartz veins are confined to a north northwest trending zone.

3.1.3 The Cainozoic sediments of the Ringarooma Valley rest on deeply weathered Palaeozoic sedimentary units of the Mathinna Beds or acid intrusive rocks of the Blue Tier Batholith. In the Great Northern Plain area Mesozoic sediments and basalt are also present. The depth of weathering is variable with the greatest thickness recorded being 49m which was encountered on a basement ridge south of the Endurance Lead.

3.1.4 The Cainozoic sediments are flat-lying and attain a maximum thickness of 123m recorded in bore holes from the central part of the Ringarooma Basin. The thickness exposed at the Briseis Mine is 94m but where exposed at other localities is generally 40m to 60m. The sedimentary sequence can be divided into a lower, generally tin-bearing unit and an upper, generally barren unit. The lower part of the sequence consists predominantly of cross-bedded sands and gravels with minor peat units and the upper part consists of clays and sands with minor gravel units.

The lower part of the sedimentary sequence is regarded as being deposited in a high energy braided stream environment with the upper part of the sequence being deposited in a lacustrine to deltaic environment.

3.1.5 The late Oligocene to early Miocene sediments are locally overlain by basaltic lava flows which have been dated at 16 m.y. (early to middle Miocene). The basalts attain a maximum thickness of about 100m and three discrete flows can be recognised at the Briseis Mine with local intercalations of sediment between the flows. The two lowest flows are deeply weathered but the upper flow is generally quite fresh.

Post basaltic deposits are restricted to Quaternary alluvium associated with the present Ringarooma River and its attendant flood plain. North of Mount Cameron these deposits have probably been reworked by marine influences during the Pleistocene and are overlain locally by aeolian sands.

3.1.6 Four types of alluvial tin deposits are recognised and these are described below.

Type 1 - Deeply buried deposits of relatively fine-grained black and brown cassiterite with abundant ilmenite and monazite, in a sedimentary sequence dominated by stratified gravels, through cross-bedded gravels, planar cross-bedded sands and peat units. This type of deposit occurs at a major break in basement slope and is commonly referred to as a deep lead.

Type 2 - Tributary gutters to deep leads, elevated in respect to the deep lead and consisting of coarse black cassiterite in poorly sorted boulder-rich sediments depleted in accessory heavy minerals.

Type 3 - Shallow surficial deposits of red, yellow and black cassiterite with accessory spinel and gold in a fining-up sequence of pebbles, sands and clays. This type is restricted to the present Ringarooma River and preserved remnants of elongated zones which run essentially parallel to the river.

Type 4 - A fourth type of deposit may be identified in the Great Northern Plain area where marine influences may have modified Types 1 and 3.

3.1.7 The general geology of the area is in the accompanying Geological Maps under Sheets 24 and 32 published by the Tasmanian Department of Mines.

3.2 Geology of the Pioneer and Endurance Leads

3.2.1 Geomorphologically the two leads are rather different. The Pioneer Lead occupies a broad, low gradient, U-shaped valley which towards its upper, mined-out reaches is more sinuous and V-shaped. The palaeo-valley is undulating to terraced with moderate flanking slopes. The Endurance Lead however occupies an asymmetrical, narrow V-shaped valley. The steeper side is against Mt Cameron to the north. The tin concentration occupies a clearly traceable central gutter with a medium to low overall gradient. In the middle reaches of the presently defined reserves is a steeper, "cascade" section where a fall of some 15m occurs over a distance of less than 200m.

3.2.2 Geologically the rock types and their configuration within the valleys are similar in both leads. They are infilled by Tertiary sediments which can be classified into two styles:-

(a) A lower zone, possibly deposited in a braided stream environment which consists of:

(i) impersistent lenses of differentially reduced and oxidised quartz gravels with a silty matrix;

(ii) iron cement areas patchily occur at the base of the zone and are more frequent at Endurance than at Pioneer;

(iii) scattered through the sequence tabular and wedge cross bedded impure silts, conglomerate layers, white clay lenses and organic accumulations and stanniferous wash are encountered.

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(b) An upper sequence, probably deposited in a less active environment which consists of:

(i) thin skeletal gritty soil;

(ii) trough cross-bedded silty fine quartz gravels with lenses of kaolin and silt;

(iii) light and dark brown clays with plant matter which occurs scattered through (b).

3.2.3 The tenacity of the above lithotypes can be subjectively ranked as follows with the approximate percentage occurrence indicated:-

Low - moderate	- fine. granite derived quartz gravels of (b)(i) 30-50%
	- similar fine grained granite derived slightly ferruginous gravels of (a)(i) 20%
Moderate	- stanniferous wash of (a)(iii) 4%
	- decomposed granite basement
	- impure silts of (a)(iii) 5%
Moderate - high	- conglomerate layers of (a)(iii) 0-25%
	- silty organic accumulations of (a)(iii) and (b)(iii) 5%
	- Kaolin lenses - predominantly from (b)(ii) 10-15%
High	- pyrite or limonite cemented zones (a)(iii) and (b)(ii) 2%

3.2.4 Problematical contents of the sedimentary sequence include the cemented sands. They range from 5cm to 1.5m in thickness and compared to other rock types are relatively quite extensive. Buried logs occur, with an approximate frequency of one per 50,000 cu m, and may reach several metres in length and 75cm diameter. Boulders are uncommon and when encountered are usually less than 40cm across.

3.2.5 Bedrock is uniformly decomposed species of granite. 40% of the surface is gently undulating. 40% flat, and the remaining 20% contains more severe undulation or crevices of about 1m amplitude. The latter may contain pay wash.

Field observable demarcation of the upper and lower limit of the 'ore' horizon is generally poor or impossible unless basement is overlain by conglomerate wash in which case the onset of footwall decomposed granite is obvious.

3.2.6 A simplified geological section of the Pioneer and Endurance deposits is shown in Plan TAS-100-15/1.

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4. EXPLORATION DRILLING

4.1 General

4.1.1 Prospecting for alluvial tin in the Ringarooma Valley began in the late 1800s. Since then many companies, at different times, have undertaken drilling investigations and as a result several deposits were discovered and exploited.

4.1.2 During the period from 1935 until 1982 there were some fairly active drilling activities particularly in the Anglo/Triako joint venture area. These concentrated mainly in the known mineralised areas. Apparently little work was ever directed to locating new deposits or deep leads for which geological potential existed in the large Ringarooma Basin.

4.1.3 Following the joint venture agreement with Triako in October 1981, Anglo has been pursuing an active geological study and drilling programme with the following objectives:

- selecting geologically favourable areas for deposition of alluvial cassiterite/gold;
- reconnaissance drilling of the selected areas hopefully to discover new orebodies;
- drilling for extension of the currently known leads.

4.1.4 During the period of October 1981- July 1984, Anglo has drilled 260 boreholes totalling 11,093m. (Refer Schedule I). Of these, 153 boreholes (7,913m) were drilled by a Jetstream reverse circulation machine which aimed to expeditiously test for extensions to the mineralisation at Scotia, Gillibrand Plain, Trout Creek, Davids Creek, Arba, Pioneer and Endurance leads as well as various parts of the Ringarooma Basin.

Later, churn drills were used and these completed 88 boreholes of 3,180m mainly at Endurance and Pioneer.

The drilling results were generally disappointing, but they contributed to a better understanding of the geology and mineralisation of the Ringarooma Basin which is valuable for planning future exploration programmes. An exception to this is at Pioneer where some significant extensions of ore reserves have been delineated.

For location of the areas drilled, refer Plan TAS-100-11/1.

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4.2 Drilling Method and Sample Treatment

4.2.1 The drilling practice outlined here relates to the programmes undertaken for the Pioneer and Endurance deposits which contain sizeable ore reserves and therefore are relevant to the purpose of this report. Other deposits with limited reserves are excluded from this evaluation study and the drilling methods used for them are accordingly omitted.

4.2.2 Amdex took over the Pioneer tin operation in 1976 and since then it has carried out extensive drilling programmes to prove the reserves ahead of the mine working face. Anglo in 1982/83 carried out some drilling in the area, adopting the same technique used by Amdex. The drilling machines used were the Keystone churn type with cutting shoes of 160/178 mm external diameter. Drill samples were normally taken at 2.0m intervals down a hole.

Borehole sample treatment comprises the following stages:

- Puddling to disintegrate the clayey materials and desliming.
- Screening the deslimed materials at 20mm to discard the coarse pebbles.
- Primary recovery of a rough concentrate using a rocker box.
- Upgrading the rocker box concentrate by hand panning.
- Analysis for the tin content in the hand pan concentrate by an XRF Analyser.

As regards to borehole valuation the method applies the Radford Factor (80%) to the theoretical volume in deriving the borehole sectional grade where the actual volume recovered is less than the 'factored' theoretical volume; for cases in which actual volume exceeds the theoretical volume, the grade is related to the former volume.

In virtually all the boreholes which bottomed in decomposed granite, cassiterite values were present for some metres in the bedrock. It is very unlikely that the granite drilled carries persistently those values and the explanation for this phenomenon is 'dragging' of the cassiterite values from the overlying mineralised alluvium into the decomposed granite during the percussion drilling action. Amdex has accordingly credited these 'dragged' values to the alluvium.

There are approximately 65 boreholes in the Pioneer selected ore reserve area as at September 1984. Of these, 54 were drilled by Amdex, 5 by Anglo, 4 by Austral Malay Tin Mining and 2 by Storeys Creek Tin Mining (refer Plan TAS-10-176). In view of Amdex/Anglo's large number of boreholes and acceptable procedure of drilling and borehole evaluation, their results are considered suitable for evaluation in this report.

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4.2.3 Prospecting of the Endurance area had been carried out prior to 1935 but the boring results were found unreliable by the Endurance Tin Mining Company during its mining operations in that year. The company found it necessary to undertake a check drilling programme in 1943 and to reselect payable areas.

During 1958-69 the Tasmanian Department of Mines conducted churn drilling over the Endurance lead. Later in 1971-72, BMI Mining having acquired the mining operations, implemented a detailed drilling programme. In 1982-83, Anglo drilled the area west of the lead to determine its extension.

Endurance Tin Mining, Tasmanian Department of Mines and BMI Mining, all used a Keystone type churn drill and similar sample treatment as Amdex. The only variances are as follows:

- The valuation of the borehole sectional grades is the actual weight of the cassiterite concentrate related to the sample volume recovered;

- The majority of the boreholes did not sample the top alluvium overlying the basal cassiterite-bearing wash. Apparently only three boreholes (viz P/119, P/133 and P/184) sampled the top 18.3-30.5m to give an average grade of 42g cass conc/m^3.

Anglo however has recalculated the borehole grades of all the available bore logs based on the Amdex valuation method which is adopted in this report. The rationale is consistency in ore reserve evaluation.

5. GEOLOGICAL ORE RESERVES

5.1 At this stage the selected reserves for this report are at Pioneer and Endurance. Other deposits at Monarch and Scotia have been excluded because their limited size would not permit economic mining at prevailing costs and metal prices.

5.2 At Pioneer the ore reserve boundary is generally defined on the basis of an in-situ cut-off grade of 140g cass conc/m^3 @ 70% Sn. The latter is derived from a tin metal price of A$16.5/kg, mining/annual capital costs for a conceptual dredging operation and overall 90% ore recovery factor. The selected reserves are classified into the following categories:-

Proven reserve - borehole density closer than 100m centres;

Probable reserve - borehole density equal to 100m centres equivalent;

Possible reserve - borehole spaced at greater than 100m centre equivalent.

The evaluation method is based on the polygonal area of influence around each drill hole.

5.3 The reserve selection at Endurance uses the grade of 140g cass conc/m^3 as a guideline for demarcation but also takes into consideration some practical aspects of mining, (eg sufficient mining face width, continuity of a dredging path). Here the reserves are calculated using the cross-section method, based on ten bore lines of 30-240m (averaging 140m) spacing and with boreholes mostly at 15m intervals.

5.4 The estimated geological reserves, with no account for mining slope, at Pioneer and Endurance are given in the following tabulation.

Area	Location	Category	Av Depth (m)	Volume of overburden and ore (Mm^3)	Grade (g cass conc/m^3 @70% Sn)	Cassiterite Concentrate @ 70% Sn (t)
Pioneer	West of mining pit	Proven	42	2.6	302	785
Pioneer	Southwest and south of pit	Probable	42	9.2	215	1978
Endurance	Endurance lead	Probable	37	6.4	298	1905
			39	18.2	256	4668

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In addition, a possible reserve of 6.2 Mm^3 grading 165g cass conc/m^3 @ 70% Sn has been indicated at Pioneer in the Gilhams Creek deviation area. Some of the old Pioneer Mine tailings have been disposed of in this area and its effect on the grade is yet to be determined.

For details of ore reserve areas and borehole locations refer Plans TAS-10-176, TAS-10-49 and TAS-10-163.

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6. CONCEPTUAL MINING METHODS

6.1 General

6.1.1 In planning a mining system to exploit the Pioneer/Endurance ore reserves, consideration has been given to the following features of the deposits:

- They occur as two separate leads about 7km apart;
- They are characterised by a thick clayey overburden of approximately 30m overlying 10m of rich cassiterite-bearing gravels;
- Bedrock in each is decomposed granite and its configuration is not highly irregular;
- Each has generally flat surface terrain;
- Potential for substantial sub-surface water seepage;

6.1.2 Reliable water supply is essential for dredging operations. The sources are as follows:

- The Frome Dam which has a maximum storage capacity of 1910 Ml (420 M gal). It overflows during the heavy rains in winter;
- The Monarch Dam with a maximum holding capacity of approximately 2180 Ml (480 M gal);
- The rainfall averaging 915 mm per annum during 1971-81. Recorded yearly evaporation rate was 878 mm from 1975 to 1978. Therefore the theoretical net amount of rain water available is 37 mm per annum;
- Underground water which has been observed during the operations at Pioneer Mine to seep constantly into the open pit. The ex-mine manager (S. Everett) said seepage rate was about 2-4 kl/min (i.e. 500-1000 gal/min);
- Possibly the Ringarooma River which flows perennially.

6.2 Mining Methods

6.2.1 Taking the foregoing factors into consideration, the possible mining systems are:

- Dredging and dry stripping;
- Open-pit dry earthmoving.
- Hydraulic monitoring and gravel pumping;

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6.2.2 The dredging method is suitable and has the distinct advantage of a lower unit operating cost. An underwater bucketwheel (e.g. Neumann and Ellicot's) or a bucket ladder dredge appears suited to exploiting the Pioneer/Endurance deposits provided that future check drilling confirms that the wash materials do not contain appreciable quantities of stiff, sticky clay, large cobbles and boulders and buried timber.

Neumann Equipment Pty Ltd designs and constructs a series of underwater bucketwheel dredges (refer Appendix III for details). The capacities of their two top range Models 350-14LD and 350-16LD are rated at 600 and 1050 tph (at 35% w/w) respectively for a free flowing sand situation. These dredges have installed engine power of 687 and 872 kW and the same digging depth of 15m below pond water level. For an operation of 24 hr/day, 350 days/yr, 80% running time and 80% bucket efficiency, the theoretically rated capacities are approximately 1.6 and 2.8 Mm^3 per annum. Adjustments must however be made to these rates for the more difficult materials and pulp density variations expected in actual operation at Pioneer/Endurance. An in-depth study requiring visits to operations using the same type of dredge is necessary in order to provide a reliable rate estimate. However, for the purpose of this report, a rate of 1.0 Mm^3 per annum is used.

A bucket ladder dredge is also suitable for mining the deposits. It is more capable than the bucketwheel unit in excavating stiff clay and cobbly ground. A primary processing plant is normally installed aboard, thus having a distinct economic advantage in ore/tailings slurry pumping.

For the Pioneer/Endurance reserves, a large capacity dredge with +0.56 cu m (20 cu ft) bucket and +4.0 Mm^3 annual throughput so widely used in Malaysia is not justified. A small dredge of low capital cost and capable of handling 1.0-1.5 Mm^3 per annum may be appropriate.

The PP100 dredge developed by Osborne Chappel & Payne appears suitable. It has a theoretical dredging capacity of 135 m^3 per hour or 1.0 Mm^3 per annum, based on 75% bucket fill efficiency, 620 hours per month and a full 12-month year. The capacity of each of the 61 buckets is 100 litres (i.e. $3\frac{1}{2}$ cu ft). The standard bucket speed is 30 per minute, but this is variable. The digging range is 3m above and 12m below pond water level. Total power requirement is 275 kW.

The mineral treatment plant onboard the PP100 dredge consists of one 6 x 1.6m trommel, and eight 2-cell Pan American jigs. The standard dredge can be upgraded to accept a larger plant.

It should be noted that in this dredging method using either a bucketwheel or a small size bucket ladder dredge, pre-stripping of the thick overburden by earthmoving equipment is essential. The average pre-stripping depth ahead of dredging is about 29m at Pioneer and 18m at Endurance.

It is of course assumed that stripping by contractor at a competitive cost is available.

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6.2.3 Another alternative mining method may be open pitting involving heavy mobile earthmoving equipment. Contract mining has been assumed at this stage since the size of the available reserves probably does not warrant the large capital outlay for the joint venture company to purchase its own fleet of loaders and trucks. Under the assumed arrangement, a reliable earthmoving contractor would excavate and haul both the overburden and ore and be responsible for his own equipment ownership and maintenance. If this method of mining appeared more viable the option of plant ownership would need to be further reviewed.

6.2.4 Hydraulic mining appears to have economic merit because of the available water supply and the natural hydraulic head from the Frome Dam. The unfavourable factors are the stiff clayey overburden and coarse gravelly/cobbly wash which have an adverse effect on mining rate; requirement of large slime storage/retention dams and higher manpower intensity than dredging.

Amdex used the hydraulic monitoring and gravel pump method to mine the Pioneer deposit. The operation ceased in April 1982 and subsequently a lot of the mining equipment and machinery had been sold off. To recommence this type of operation would obviously require capital for purchase of new equipment. It is assumed in this pre-feasibility study that all the necessary equipment and machinery will be purchased as new.

Prior to the closure of the mine, Amdex mine manager planned to upgrade its operation to 804,000 m^3 per annum, comprising 504,000 m^3 of ore and 300,000 m^3 of overburden. In the present study the overburden stripping depth is increased and therefore the volume assumed at a higher 900,000 m^3 per year whilst maintaining the same ore mining rate. A detailed investigation of the mining capacity however is necessary if the hydraulic method is considered viable.

6.2.5 In the aforementioned mining methods necessitating dry earthmoving, the success of the project is vulnerable to the contract work, as it hinges strongly on its cost and reliability.

6.3 Mining Sequence

6.3.1 Mining operation employing dredging or dry open pitting method is envisaged to commence at the higher grade Endurance deposit, starting on the eastern end of the lead and proceeding westerly following the well-defined lead. On exhaustion of the ore in this area, the operation will be shifted to Pioneer. At the latter, mining will start from the pit area and then advance southerly to the lower grade "football field" area.

The existing large water hole at Endurance and the water-filled mine pit at Pioneer are both an aid to the starting-up of a dredging operation. With them the unproductive period of the dredge having to gain depth to reach the basal ore is obviated; that is, commercial production can commence at an earlier date.

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6.3.2 For the hydraulic operation, this starts at the Pioneer deposit, in the pit area where Amdex ceased operation in April 1982. The pit has since been flooded and necessitates de-watering. Having mined out the selected reserve here, the operation will then be transferred to Endurance.

6.3.3 As the mining face advances, the mined-out area will be back-filled with tailings from the ore treatment plant and the overburden materials. Dams will be constructed as required for retention of slimes.

6.3.4 A programme of tailings rehabilitation will be implemented in mined-out areas to conform with the mining lease requirements.

6.4 Ore Treatment

Primary processing of the ore would be carried out in an onshore plant for each of the afore-mentioned mining methods except the bucket ladder dredge. A gravitational circuit using jigs with possibly some spirals has been proven suitable. Upgrading of the primary concentrates to a saleable product can be accomplished in a separate tin-shed, using mainly shaking tables and Willoughby/sluice boxes.

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7. MINEABLE ORE RESERVES

7.1 In evaluating the mineable ore reserves for the conceptual mining methods described in Section 6, the following considerations have been accounted for:

- Side batters at an assumed slope of 1 in 1 for stripping/ore mining are included and are outside of the selected geological reserve blocks' outlines. Geotechnical investigations will be necessary to determine the optimum batter slopes to be consistent with safety. The grade of the side batters is estimated at 17g cass conc/m^3 @ 70% Sn.

- Excavation into the decomposed bedrock of 0.3m depth which is necessary for full recovery of the rich ore values resting on the bedrock. The bedrock is assumed to be tin-barren.

7.2 The resulting mineable reserves at Pioneer and Endurance for the dredging and dry open pitting operations are summarised as follows:-

Location	Volume of overburden and ore (Mm^3)	Grade overburden and ore (g cass conc/m^3 @ 70% Sn)	Cass Conc @ 70% Sn (t)
Pioneer:			
- Proven reserve west of mining pit	3.12	257	801
- Probable reserve southwest and south of pit	11.98	170	2035
Endurance:			
- Probable reserve east of 'kink'	4.07	225	915
- Probable reserve west of 'kink'	4.33	232	1003
	23.50	202	4754

For details of the above, refer Schedules II and III.

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7.3 In this pre-feasiblity study, the hydraulic operating unit cost is estimated to be higher than the dredging method and therefore the hydraulic operation has to be more selective in mining the ore reserves. Consequently, the lower grade reserve block at the 'southwest and south of pit' at Pioneer has been excluded from the operation. The total mineable reserve therefore is 11.52 Mm^3 of alluvium averaging 236 g cass conc/m^3 @ 70% Sn (compared with 23.5 Mm^3 grading 202 g cass conc/m^3 @ 70% Sn for the dredging method).

8. CAPITAL COSTS ESTIMATES

8.1 The preliminary estimates of the total pre-production capital for the project employing an underwater bucket wheel dredge, a bucket ladder dredge (with contract stripping of overburden in each instance) an operation with contract mining of both the overburden and ore and hydraulic monitoring/gravel pump are summarised below. (Refer Schedule IV for details)

Case A : Dredging - underwater bucket wheel operation : $7.4M

Case B : Dredging - bucket ladder operation : $8.0M

Case C : Contract dry mining operation : $4.8M

Case D : Hydraulic monitoring and gravel pumping : $6.6M

The above estimates assume construction of a new mineral processing plant and tin-shed. If at the time of implementation of this mining project the Amdex plant, currently at Pioneer, is still available in good condition and for purchase at second-hand value a saving of a few hundred thousand dollars may result.

9. OPERATING COST ESTIMATES

The approximate operating costs for the conceptual operations are as follows:

	$M/yr	$/m^3 of ore treated	$/m^3 of over-burden and ore
Case A : Underwater bucket wheel dredge operation, including contract dry stripping.	3.71	4.01	1.47
Case B : Bucket ladder dredge operation, including contract dry stripping	3.87	4.19	1.53
Case C : Dry open pit mining employing contract overburden stripping/ore removal.	4.33	4.68	1.71
Case D : Hydraulic monitoring and gravel pumping.	2.94	6.58	2.35

For details of Cases A, B and C, refer to Schedule V.

The costs for the Case D are based primarily on Amdex ex-mine Manager's estimate of approximately $1.80 per m^3 in 1982. This cost covered hydraulic ore mining and overburden stripping rates of 42,000 m^3 and 25,000 m^3 per month respectively. For this pre-feasibility study, the cost is adjusted for a higher stripping rate of about 75,000 m^3 per month whilst maintaining the same ore mining rate and also for yearly inflation to 1985. Should the hydraulic method be acceptable for future economic evaluation, its operating cost would need to be examined in detail.

Telephone enquiries to some earthmoving contractors in Tasmania and Victoria indicate that contract dry stripping will cost about $1.00 - 1.20 per bank m^3. These rates apply to a removal rate of about 1.8 M m^3/yr for 0.5 km single way haulage distance and 10 years' contract duration. For contract excavation and haulage of ore to the treatment plant an additional 5-10 cents may be incurred. However, for the purpose of this pre-feasibility report, contract overburden stripping and ore haulage costs of $1.20 and $1.30 per m^3 are used.

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10. ECONOMIC EVALUATION

10.1 In this pre-feasibility study, the following cases are evaluated:-

Case A - Mining the indicated ore reserves at Pioneer/Endurance using an underwater bucket wheel dredge and contract dry stripping.

Case B - Mining the indicated ore reserves at Pioneer/Endurance using a bucket ladder dredge and contract dry stripping.

Case C - Contract mining both the overburden and ore; ore treatment and mine management by the project owners.

Case D - Hydraulic monitoring and gravel pumping.

10.2 The parameters used for the evaluation are summarised in the tabulation overleaf. There are three parameters which need some explanation.

The first is the ore recovery factor which is a relationship between recoverable grade and borehole value. This is forecast at 90% as indicated from some production statistics during the mining operation at Pioneer by Amdex. For future evaluation, particularly the feasibility study, a detailed investigation should be undertaken to determine this factor more accurately for both the operations at Pioneer and Endurance.

The second parameter is the tin metal price of $A16.5 per kg (in 1985) which is taken from a detailed study in 1983 by Anglo American Corporation (South Africa) on the world tin supply/demand and market analysis. The latter predicted that over the 17 years' period of 1983-2000, the price with the International Tin Council control is likely to average $US14.0 per kg in 1982 money terms. This forecast, though higher than the currently traded $A14.8 (i.e. $US12.6) per kg, is viewed to be representative of a normal long term trend in which cyclical economic 'peaks' and 'troughs' are evened out.

An exchange rate of $A1.00 = $US 0.85 has been used throughout the project life.

The third parameter relates to the lead time of 4 years before production commences. The reason is the expectation of a continued government imposed tin quota for at least a few years because of the current world stockpile. In the present circumstances, it is unlikely the Federal Government Department of Trade, (the authority in regard to quota allocation), has available a surplus of quota for distribution to new operating mines. However, it is hoped that by 1989, which is this project's forecast production commencement year, the present tin export quota will have been lifted. (Refer Appendix II regarding quota allocation system).

SUMMARY OF EVALUATION PARAMETERS

		Case A	Case B	Case C	Case D
Mining :					
- Contract dry stripping	(Mm^3/yr)	1.6/1.8	1.6/1/8	1.6/1.8	-
- Contract ore removal	(")	-	-	1.0	-
- Dredge throughput	(")	1.0	1.0	-	-
- Hydraulic method	(")	-	-	-	1.2/1.7
Mineable ore reserve :					
- Total volume	(Mm^3)	23.5	23.5	23.5	11.52
- Average grade (g cass conc/m^3 @ 70% Sn)		202	202	202	236
Recovery factor	(%)	90	90	90	90
Tin metal price	(A$/kg)	16.5	16.5	16.5	16.5
Mining costs :					
- Contract dry stripping	(A$/$m^3$)	1.20	1.20	1.20	-
- Contract ore removal	(")	-	-	1.30	-
- Dredging, treating, incl. overheads	(")	1.80	1.96	1.20	-
- Hydraulic mining, treating, incl. Overheads	(")	-	-	-	2.35
Capital costs :					
- Pre-production	(A$M)	7.4	8.0	4.8	6.6
- Annual replacement	(")	0.1	0.1	0.1	0.1
Salvage value	(")	0.5	0.5	0.2	0.5
Tax	(%)	46	46	46	46
Commencement of production		1989	1989	1989	1989
Exchange rate for A$1.00	(US$)	0.85	0.85	0.85	0.85

The above monetary values are all in 1985 dollar terms.

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10.3 Cash flows were run for the four study Cases A, B, C and D in 1985 constant money terms. Details of them are presented in Schedules VI - IX and summarised in the tabulation overleaf.

10.4 The indications from evaluation results are briefly as follows:

- The conceptual exploitation of the Pioneer/Endurance ore reserves by either an underwater bucket wheel or a bucket ladder dredge shows poor economic returns of 0-1%DCF Yield.

- Contract mining both the overburden and ore also shows negligible DCF Yield, mainly because of overall higher operating cost. For an optimistic contract cost of $\$1.00/m^3$ for both overburden and ore removal, a 6% DCF Yield is indicated.

- Hydraulic mining gives a negative rate of investment return.

For the project to return a 10-12% DCF Yield for the outlined dredging methods, it needs one of the following:

- An improvement of about 20-25% in grade of the reserve;
- Further discovery of 20 Mm^3 mineable reserve grading 200 g cass conc/m^3 @ 70% Sn;
- A higher tin price of 25% above the level used in the current evaluation.

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SUMMARY OF EVALUATION RESULTS

		Case A	Case B	Case C	Case D
Stripping vol	(Mm^3)	14.83	14.83	14.83	7.41
Treating vol	(")	8.67	8.67	8.67	4.11
Total mining vol	(")	23.50	23.50	23.50	11.52
Treating grade (g cass conc/m^3 @ 70% Sn)		531	531	531	644
Recovery factor	(%)	90	90	90	90
Production (t cass conc @ 70% Sn)		4 146	4 146	4 146	2384
Tin metal price	($/kg)	16.50	16.50	16.50	16.50
Realisable price ($/kg cass conc % 70% Sn)		11.22	11.22	11.22	11.22
Gross revenue	($M)	46.51	46.51	46.51	26.75
Royalty :					
- 2% to Govt	($M)	0.93	0.93	0.93	0.53
- 5% to BMI	(")	0.97	0.97	0.97	0.95
Net revenue	(")	44.61	44.61	44.61	25.26
Total mining cost	(")	34.80	36.34	40.57	27.04
Operating surplus	(")	9.82	8.28	4.05	(1.78)
Depreciation/amortisation	(")	7.40	8.00	4.80	6.60
Profit before tax	(")	2.42	0.28	(0.75)	(8.38)
Taxable Profit	(")	3.24	2.14	1.82	-
Tax @ 46%	(")	1.49	0.98	0.84	-
Profit after tax	(")	0.93	(0.71)	(1.59)	(8.38)
Pre-production capital	(")	7.40	8.00	4.80	6.60
Annual capital	(")	1.00	1.00	1.0	1.30
Salvage value	(")	0.50	0.50	0.50	0.50
Cash adjustment - depreciation	(")	7.40	8.00	4.80	6.60
Net cash flow	(")	0.43	-1.21	-2.09	-9.18
DCF Yield	(%)	1	Negative	Negative	Negative
Capital payback period	(yr)	10	+10	+10	+ 9
Operating life	(yr)	10	10	10	9

11. CONCLUSIONS

11.1 The Pioneer and Endurance alluvial tin leads are Tertiary deposits infilling paleo-valleys that have been developed during the Paleocene to early Eocene. Geomorphologically, the two leads are rather different. The Pioneer lead occupies a broad, low gradient U-shaped valley, whereas the Endurance lead occupies an asymmetrical narrow V-shaped valley.

Geologically the rock types and their configuration within the valleys are similar in both leads. The sediments can be classified into two styles, being as follows :

- A lower zone comprising mainly loose gravels with clay and silt lenses, some organic accumulations, patches of iron-cemented sands and some buried timber;

- An upper zone of silts and stiff clay.

Bedrock is uniformly decompossed granite which is mostly gently undulating to flat.

11.2 The geological reserves (i.e. vertical blocks with no account for mining considerations) of the two leads, and at an in-situ borehole cut-off grade of 140g cass conc/m^3 @ 70% Sn, are summarised in the following tabulation :

Area	Location	Category	Av Depth (m)	Volume of overbuden and ore (Mm^3)	Grade (g cass conc/m^3 @ 70% Sn)	Cassiterite Concentrate @ 70% Sn (t)
Pioneer	West of mining pit	Proven	42	2.6	302	785
Pioneer	Southwest and south of pit	Probable	42	9.2	215	1978
Endurance	Endurance lead	Probable	37	6.4	298	1905
			39	18.2	256	4668

In addition, a possible reserve of 6.2 Mm^3 grading 165g cass conc/m^3 @ 70% Sn has been indicated at Pioneer in the Gilhams Creek deviation area. Some of the old Pioneer Mine tailings have been disposed of in this area and its effect on the grade is yet to be determined.

11.3 Three possible mining systems are considered for exploiting the ore reserves, which are :

- dredging with contract dry stripping;

- open-pit dry earthmoving by contract.

- hydraulic monitoring and gravel pumping;

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The dredging method is suitable and offers the distinct advantage of a lower unit operating cost. Either an underwater bucket wheel or a small capacity bucket ladder dredge appears suited to exploiting the Pioneer/Endurance deposits, provided that future check drilling confirms that the stanniferous wash materials do not contain appreciable quantities of stiff, sticky clay, large cobbles and boulders and buried timber.

Contract mining both the overburden and paydirt, by open pitting involving heavy earthmoving equipment, has the obvious economic advantage of reduced capital outlay. Also it has an appeal with the Pioneer/Endurance reserves occurring in two separate deposits.

Hydraulic mining appears to have economic merit because of the available water supply and the natural hydraulic head from the Frome Dam and was used at Pioneer up until 1982. The stiff clayey overburden and coarse gravelly/cobbly wash have an adverse effect on mining rate; a large slime retention dam would be required and higher manpower intensity would be experienced.

In the aforementioned mining methods necessitating dry earthmoving, the success of the project is vulnerable to the contract work, as it hinges strongly on its cost and reliability.

11.4 In Ore treatment of the ore, primary processing would be carried out in an onshore plant for each of the mining methods in the above Section 11.3, except the bucket ladder dredge which has its own onboard primary plant. A gravitational circuit using jigs with possibly some spirals has been proven suitable. Upgrading of the primary concentrates to a saleable product can be accomplished in a separate tin-shed, using mainly shaking tables and Willoughby/sluice boxes.

11.5 The mineable ore reserves estimated for the conceptual dredging/contract mining methods are given in the following tabulation. These take into account side batters and some excavation into the decomposed bedrock which is necessary for full recovery of the rich ore values resting on the bedrock

	Volume overburden and ore (Mm^3)	Grade (g Cass conc/m^3 @ 705 Sn) for total overburden and ore	Cass Conc @ 70% Sn (t)
Pioneer :			
- Proven reserve west of mining pit	3.12	257	801
- Probable reserve southwest and south of pit	11.98	170	2035
Endurance :			
- Probable reserve east of 'kink	4.07	225	915
- Probable reserve west of 'kink'	4.33	232	1003
	23.50	202	4754

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11.6 The preliminary estimates of the total pre-production capital for the project employing an underwater bucket wheel, a bucket ladder dredge and an operation with contract mining both the overburden and ore are summarised below.

Dredging - underwater bucket wheel operation = $7.4 m
Dredging - bucket ladder operation = $8.0 m
Open Pit - Contract mining operation = $4.8 m
Hydraulic monitoring and gravel pumping = $6.6 m

The above estimates assume construction of a new mineral processing plant and tin-shed. If the Amdex plant, currently at Pioneer, is still available in good condition and for purchase at second-hand value at the time of implementation of this mining project, a saving of a few hundred thousands of dollars may result.

11.7 The approximate operating costs are as follows:

	$M/yr	$m^3 of ore treated	$/m^3 of over-burden and ore
Underwater bucket wheel dredge operation, including contract dry stripping	3.71	4.01	1.47
Bucket ladder dredge operation, including contract dry stripping	3.87	4.19	1.53
Dry open pit mining employing contract overburden stripping/ ore removal	4.33	4.68	1.71
Hydraulic monitoring and gravel pumping	2.94	6.58	2.35

11.8 Preliminary economic evaluation of mining the ore reserves at Pioneer/Endurance gives the following indications:

- The conceptual exploitation by either an underwater bucket wheel or a bucket ladder dredge shows poor economic returns of 1% and negative DCF Yield respectively.

- Contract mining both the overburden and ore shows a negative DCF Yield despite reduced pre-production capital outlay, mainly because of overall higher mining cost. For an optimistic contract cost at $1.00/m^3 for both overburden and ore removal, a 6% DCF Yield is indicated which is still below the acceptable requirement.

- Hydraulic monitoring and gravel pumping method gives a negative DCF Yield.

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For the project to return a 10-12% DCF Yield for the outlined dredging methods, it needs one of the following:

- An improvement of about 20-25% in grade of the reserve;
- Further discovery of 20 Mm^3 mineable reserve grading 200 g cass conc/m^3 @ 70% Sn;
- A higher tin price of 25% above the level used in the current evaluation.

11.9 The present government tin quota allocation system, which restricts the existing mines' output, is expected to continue through the next few years in view of both the substantial overhang of world tin stockpile today and the current economic climate.

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Schedule I

SUMMARY TABLE OF AAA EXPLORATION DRILLING IN 1981-84

AREA (ref Plan)	NO OF HOLES	TOTAL METERAGE	MEAN DEPTH (m)	MEAN GRADE ($g_3$cass conc/ m^3 @ 70% Sn)	RANGE OF GRADES ($g_3$cass conc/ m^3 @ 70% Sn)	REMARKS
(1) Davids Ck	9	378.2	42.0	14	4 - 31	By reverse circulation
(2) Gellibrand Plains	10	246.5	24.7	5	1 - 8	"
(3) Trout Creek	7	203.0	29.0	3	2 - 5	"
(4) Scotia Lead	6	189.8	31.7	63	1 - 122	"
(5) Arba Flats	13	771.2	59.3	20	5 - 91	"
(6) Eastern Leads	29	1 076.6	37.1	17	2 - 54	By churn drilling
(7) Pioneer - SW Margins	19	637.0	33.5	25	1 - 185	By reverse circulation
(8) " - NW Margins	46	2 458.0	53.4	47	2 - 263	"
(9) Endurance Lead	40	1 686.6	42.2	46	6 - 493	"
(10) Ringarooma Basin	22	1 342.5	61.0	11	1 - 43	"
(11) Pioneer Dam Reserve	6	288	48.0	40	11 - 72	By churn drilling
(12) Picketts Plain	7	303.0	42.3	16	4 - 23	"
(13) Endurance "fault"	8	184.9	23.1	40	7 - 76	"
(14) Banca	1	21.0	21.0	2	2	"
(15) Boobyalla Basin	4	161.0	40.3	5	2 - 10	"
(16) Arba Lead	5	223.1	44.6	57	12 - 88	"
(17) Pioneer - Football Ground	12	403.0	33.6	115	2 - 630	"
(18) " - Poverty Point	10	346.5	34.6	29	2 - 57	"
(19) " - Gillhams Creek	2	56.5	28.3	501	57 - 944	"
(20) Scotia Lead	4	116.0	29.0	54	31 - 64	"
TOTAL/AVERAGE	260	11 092.5	42.6	38	1 - 944	

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Schedule II

SUMMARY OF GEOLOGICAL/MINEABLE RESERVES

Location	Geological Reserves (a)				Side Batter Reserves (b)			0.3m Bedrock Excavation (c) ^			Mineable Reserves (d)=(a)+(b)+(c)		
	Av Depth (m)	Volume (Mm^3)	Grade ($g_3$cass conc/ m^3 @ 70% Sn)	Cass Conc @ 70% Sn (t)	Volume (Mm^3)	Grade ($g_3$cass conc/ m^3 @ 70% Sn)	Cass Conc @ 70% Sn (t)	Volume (Mm^3)	Grade ($g_3$cass conc/ m^3 @ 70% Sn)	Cass Conc @ 70% Sn (t)	Volume (Mm^3)	Grade ($g_3$cass conc/ m^3 @ 70% Sn)	Cass Conc @ 70% Sn (t)
Pioneer:													
- Proven reserve west of mining pit	42	2.6	302	785	0.5	32	16	0.02	Nil	Nil	3.12	257	801
- Probable reserve southwest and south of pit	42	9.2	215	1978	2.7	21	57	0.08	Nil	Nil	11.98	170	2035
Endurance:													
- Probable reserve east of 'kink'	28	3.4	268	911	0.6	7	4	0.07	Nil	Nil	4.07	225	915
- Probable reserve west of 'kink'	47	3.0	331	994	1.3	7	9	0.03	Nil	Nil	4.33	232	1003
	39	18.2	256	4668	5.1	17	86	0.20	Nil	Nil	23.5	202	4754

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Schedule 111

SUMMARY OF OVERBURDEN/ORE RESERVES (MINEABLE)

	Overburden			Ore			Total		
	Vol (Mm^3)	Grade (g cass conc/m^3 @ 70% Sn)	Cass Conc @ 70% Sn (t)	Vol (Mm^3)	Grade (g cass conc/m^3) @ 70% Sn)	Cass Conc @ 70% Sn (t)	Vol (Mm^3)	Grade (g cass conc/m^3 @ 70% Sn)	Cass Conc @ 70% Sn (t)
Pioneer:									
- Proven reserve west of mining pit	1.93	24	46	1.19	634	755	3.12	257	801
- Probable reserve south west and south of pit	7.42	13	97	4.56	425	1938	11.98	170	2035
Endurance:									
- Probable reserve east of 'kink'	2.62	3	8	1.45	626	907	4.07	225	915
- Probable reserve west of "kink'	2.86	6	17	1.47	671	986	4.33	232	1003
	14.83	11	168	8.67	529	4586	23.50	202	4754

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Schedule IV

PRELIMINARY PRE-PRODUCTION CAPITAL ESTIMATE

	Case A. Under water Bucket Wheel Dredging Operation ($M)	Case B. Bucket Ladder Dredging Operation ($M)	Case C. Contract Mining Overburden/Ore ($M)	Case D. Hydraulic Monitoring and gravel pumping ($M)
Drilling	0.4	0.4	0.4	0.4
Dredge/engineering/feasibility studies	0.6	0.6	0.6	0.6
Dredge and ancillary equipment	1.5	2.0	-	-
Hydraulic Mining equipment	-	-	-	0.9
Onshore treatment plant/tin-shed	1.0	0.8	1.0	0.7
Mobile equipment	1.5	1.5	0.8	1.5
Office/workshop/misc equipment	0.5	0.5	0.4	0.5
Residential buildings	0.4	0.5	0.4	0.5
Power	0.2	0.2	0.2	0.2
Spares	0.3	0.4	0.2	0.2
Administration/management	0.4	0.4	0.4	0.4
Misc and Contigencies	0.6	0.7	0.4	0.7
	7.4	8.0	4.8	6.6

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Schedule V

PRELIMINARY OPERATING COST ESTIMATES

	Case A. Underwater Bucket Wheel Operation		Case B. Bucket Ladder Operation		Case C. Open Pit Mining	
	Av $M/yr	Av $/m^3 of ore	Av $M/yr	Av $/m^3 of ore	Av $M/yr	Av $/m^3 of ore
Contract dry stripping of overburden	1.91	2.05	1.91	2.05	1.91	2.05
Contract ore removal	-	-	-	-	1.22	1.30
Dredge direct operation	0.58	0.62	0.95	1.02	-	-
Onshore treatment plant/tin-shed	0.39	0.42	0.20	0.22	0.43	0.46
Slimes retention	0.10	0.11	0.08	0.09	0.11	0.12
Tailings rehabilitation	0.06	0.06	0.06	0.06	0.06	0.06
Provision for acid corrosion	0.05	0.05	0.05	0.05	0.05	0.05
Mechnical/electrical/maintenance workshops	0.25	0.27	0.25	0.27	0.18	0.20
Mine office/staff/general expenditure	0.27	0.32	0.27	0.32	0.26	0.32
Overheads	0.10	0.11	0.10	0.11	0.11	0.12
	3.71	4.01	3.87	4.19	4.33	4.68
Average vol mined (overburden + ore) per year, Mm^3	2.53		2.53		2.53	
Average mining cost per m^3, $	1.47		1.53		1.71	

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SCHEDULE VI 036

PROJECT : RINGAROOMA JOINT VENTURE, TASMANIA
CASE : A

YEAR ENDING DECEMBER		1985	1986	1987	1988	1989	1990
DRY STRIPPING VOL	(M cu m)				0.90	1.80	1.80
DREDGING/TREATING VOL	(M cu m)					0.70	1.00
TOTAL MINING VOL	(M cu m)				0.90	2.50	2.80
DREDGING/TREATING GRADE	(g cass conc/cu m,70%Sn)					615	635
RECOVERY FACTOR	(%)					90	90
PRODUCTION	(t cass conc,70% Sn)					387	572
TIN METAL PRICE	($/kg)					16.50	16.50
REALISABLE PRICE	($/kg cass conc,70% Sn)					11.22	11.22
GROSS REVENUE	($M)					4.35	6.41
ROYALTY:	($M)						
- 2% TO GOVT	($M)					0.09	0.13
- 5%TO BMI	($M)					0.22	0.32
NET REVENUE	($M)					4.04	5.96
DRY STRIPPING COST	($M)				1.08	2.16	2.16
DREDGING/TREATING/O'H COST	($M)				0.30	1.80	1.80
TOTAL MINING COST	($M)				1.38	3.96	3.96
OPERATING SURPLUS	($M)				-1.38	0.08	2.00
DEPRECIATION	($M)					0.74	0.74
PROFIT BEFORE TAX	($M)				-1.38	-0.66	1.26
'TAX COMPUTING STEP'	($M)				-1.38	-2.04	-0.78
TAXABLE PROFIT	($M)				0	0	0
TAX AT 46%	($M)				0	0	0
PROFIT AFTER TAX	($M)				-1.38	-0.66	1.26
CAPITAL/DEVELOPMENT	($M)	0.05	0.15	2.60	4.60		
ANNUAL CAPITAL	($M)					0.10	0.10
SALVAGE VALUE	($M)						
CASH ADJUSTMENT	($M)					0.74	0.74
NET CASH FLOW	($M)	-0.05	-0.15	-2.60	-5.98	-0.02	1.90
CUM NET CASH FLOW	($M)	-0.05	-0.20	-2.80	-8.78	-8.80	-6.90

NPV AT 10% = -2.37
NPV AT 12% = -2.60
DCF YIELD = 1%

2 of 3

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	1991	1992	1993	1994	1995	1996	1997	1998	TOTAL/AV
	0.98	1.60	1.60	1.60	1.60	1.60	1.35		14.83
	1.00	0.70	1.00	1.00	1.00	1.00	1.00	0.27	8.67
	1.98	2.30	2.60	2.60	2.60	2.60	2.35	0.27	23.50
	694	653	575	425	425	425	425	425	531
	90	90	90	90	90	90	90	90	90
	625	411	518	383	383	383	383	103	4146
	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50
	11.22	11.22	11.22	11.22	11.22	11.22	11.22	11.22	11.22
	7.01	4.62	5.81	4.29	4.29	4.29	4.29	1.16	46.51
	0.14	0.09	0.12	0.09	0.09	0.09	0.09	0.02	0.93
	0.35	0.08							0.97
	6.52	4.44	5.69	4.21	4.21	4.21	4.21	1.14	44.61
	1.18	1.92	1.92	1.92	1.92	1.92	1.62	0.00	17.80
	1.80	1.80	1.80	1.80	1.80	1.80	1.80	0.50	17.00
	2.98	3.72	3.72	3.72	3.72	3.72	3.42	0.50	34.80
	3.54	0.72	1.97	0.49	0.49	0.49	0.79	0.64	9.82
	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	7.40
	2.80	-0.02	1.23	-0.25	-0.25	-0.25	0.05	-0.10	2.42
	2.03	-0.02	1.21	-0.25	-0.51	-0.76	-0.72	-0.82	
	2.03	0	1.21	0	0	0	0	0	3.24
	0.93	0	0.56	0	0	0	0	0	1.49
	1.87	-0.02	0.67	-0.25	-0.25	-0.25	0.05	-0.10	0.93
									7.40
	0.10	0.20	0.10	0.10	0.10	0.10	0.10		1.00
								0.50	0.50
	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	7.40
	2.51	0.52	1.31	0.39	0.39	0.39	0.69	1.14	0.43
	-4.39	-3.86	-2.55	-2.16	-1.78	-1.39	-0.71	0.43	0.43

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PROJECT : RINGAROOMA JOINT VENTURE, TASMANIA
CASE : B

YEAR ENDING DECEMBER		1985	1986	1987	1988	1989	1990
DRY STRIPPING VOL	(M cu m)				0.90	1.80	1.80
DREDGING/TREATING VOL	(M cu m)					0.70	1.00
TOTAL MINING VOL	(M cu m)				0.90	2.50	2.80
DREDGING/TREATING GRADE	(g cass conc/cu m,70%Sn)					615	635
RECOVERY FACTOR	(%)					90	90
PRODUCTION	(t cass conc,70% Sn)					387	572
TIN METAL PRICE	($/kg)					16.50	16.50
REALISABLE PRICE	($/kg cass conc,70% Sn)					11.22	11.22
GROSS REVENUE	($M)					4.35	6.41
ROYALTY:	($M)						
- 2% TO GOVT	($M)					0.09	0.13
- 5%TO BMI	($M)					0.22	0.32
NET REVENUE	($M)					4.04	5.96
DRY STRIPPING COST	($M)				1.08	2.16	2.16
DREDGING/TREATING/O'H COST	($M)				0.30	1.96	1.96
TOTAL MINING COST	($M)				1.38	4.12	4.12
OPERATING SURPLUS	($M)				-1.38	-0.08	1.84
DEPRECIATION	($M)					0.80	0.80
PROFIT BEFORE TAX	($M)				-1.38	-0.88	1.04
'TAX COMPUTING STEP'	($M)				-1.38	-2.26	-1.22
TAXABLE PROFIT	($M)				0	0	0
TAX AT 46%	($M)				0	0	0
PROFIT AFTER TAX	($M)				-1.38	-0.88	1.04
CAPITAL/DEVELOPMENT	($M)	0.05	0.15	2.60	5.20		
ANNUAL CAPITAL	($M)					0.10	0.10
SALVAGE VALUE	($M)						
CASH ADJUSTMENT	($M)					0.80	0.80
NET CASH FLOW	($M)	-0.05	-0.15	-2.60	-6.58	-0.18	1.74
CUM NET CASH FLOW	($M)	-0.05	-0.20	-2.80	-9.38	-9.56	-7.82

NPV AT 10% = -3.27
NPV AT 12% = -3.42
DCF YIELD = **Negative**

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1991	1992	1993	1994	1995	1996	1997	1998	TOTAL/AV
0.98	1.60	1.60	1.60	1.60	1.60	1.35		14.83
1.00	0.70	1.00	1.00	1.00	1.00	1.00	0.27	8.67
1.98	2.30	2.60	2.60	2.60	2.60	2.35	0.27	23.50
694	653	575	425	425	425	425	425	531
90	90	90	90	90	90	90	90	90
625	411	518	383	383	383	383	103	4146
16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50
11.22	11.22	11.22	11.22	11.22	11.22	11.22	11.22	11.22
7.01	4.62	5.81	4.29	4.29	4.29	4.29	1.16	46.51
0.14	0.09	0.12	0.09	0.09	0.09	0.09	0.02	0.93
0.35	0.08							0.97
6.52	4.44	5.69	4.21	4.21	4.21	4.21	1.14	44.61
1.18	1.92	1.92	1.92	1.92	1.92	1.62	0.00	17.80
1.96	1.96	1.96	1.96	1.96	1.96	1.96	0.60	18.54
3.14	3.88	3.88	3.88	3.88	3.88	3.58	0.60	36.34
3.38	0.56	1.81	0.33	0.33	0.33	0.63	0.54	8.28
0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	8.00
2.58	-0.24	1.01	-0.47	-0.47	-0.47	-0.17	-0.26	0.28
1.37	-0.24	0.77	-0.47	-0.95	-1.42	-1.60	-1.86	
1.37	0	0.77	0	0	0	0	0	2.14
0.63	0	0.36	0	0	0	0	0	0.98
1.95	-0.24	0.65	-0.47	-0.47	-0.47	-0.17	-0.26	-0.71
								8.00
0.10	0.20	0.10	0.10	0.10	0.10	0.10		1.00
							0.50	0.50
0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	8.00
2.65	0.36	1.35	0.23	0.23	0.23	0.53	1.04	-1.21
-5.16	-4.80	-3.44	-3.22	-2.99	-2.77	-2.24	-1.21	-1.21

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SCHEDULE VIII 010

PROJECT : RINGAROOMA JOINT VENTURE, TASMANIA
CASE : C

YEAR ENDING DECEMBER		1985	1986	1987	1988	1989	1990
DRY STRIPPING VOL	(M cu m)				0.90	1.80	1.80
DREDGING/TREATING VOL	(M cu m)					0.70	1.00
TOTAL MINING VOL	(M cu m)				0.90	2.50	2.80
DREDGING/TREATING GRADE	(g cass conc/cu m,70%Sn)					615	635
RECOVERY FACTOR	(%)					90	90
PRODUCTION	(t cass conc,70% Sn)					387	572
TIN METAL PRICE	($/kg)					16.50	16.50
REALISABLE PRICE	($/kg cass conc,70% Sn)					11.22	11.22
GROSS REVENUE	($M)					4.35	6.41
ROYALTY:	($M)						
- 2% TO GOVT	($M)					0.09	0.13
- 5%TO BMI	($M)					0.22	0.32
NET REVENUE	($M)					4.04	5.96
DRY STRIPPING COST	($M)				1.08	3.07	3.46
DREDGING/TREATING/O'H COST	($M)				0.30	1.20	1.20
TOTAL MINING COST	($M)				1.38	4.27	4.66
OPERATING SURPLUS	($M)				-1.38	-0.23	1.30
DEPRECIATION	($M)					0.48	0.48
PROFIT BEFORE TAX	($M)				-1.38	-0.71	0.82
'TAX COMPUTING STEP'	($M)				-1.38	-2.09	-1.27
TAXABLE PROFIT	($M)				0	0	0
TAX AT 46%	($M)				0	0	0
PROFIT AFTER TAX	($M)				-1.38	-0.71	0.82
CAPITAL/DEVELOPMENT	($M)	0.05	0.15	1.10	3.50		
ANNUAL CAPITAL	($M)					0.10	0.10
SALVAGE VALUE	($M)						
CASH ADJUSTMENT	($M)					0.48	0.48
NET CASH FLOW	($M)	-0.05	-0.15	-1.10	-4.88	-0.33	1.20
CUM NET CASH FLOW	($M)	-0.05	-0.20	-1.30	-6.18	-6.51	-5.31

NPV AT 10% = -2.59
NPV AT 12% = -2.60
DCF YIELD = **Negative**

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1991	1992	1993	1994	1995	1996	1997	1998	TOTAL/AV
0.98	1.60	1.60	1.60	1.60	1.60	1.35		14.83
1.00	0.70	1.00	1.00	1.00	1.00	1.00	0.27	8.67
1.98	2.30	2.60	2.60	2.60	2.60	2.35	0.27	23.50
694	653	575	425	425	425	425	425	531
90	90	90	90	90	90	90	90	90
625	411	518	383	383	383	383	103	4146
16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50
11.22	11.22	11.22	11.22	11.22	11.22	11.22	11.22	11.22
7.01	4.62	5.81	4.29	4.29	4.29	4.29	1.16	46.51
0.14	0.09	0.12	0.09	0.09	0.09	0.09	0.02	0.93
0.35	0.08							0.97
6.52	4.44	5.69	4.21	4.21	4.21	4.21	1.14	44.61
2.48	2.83	3.22	3.22	3.22	3.22	2.92	0.35	29.07
1.20	1.20	1.20	1.20	1.20	1.20	1.20	0.40	11.50
3.68	4.03	4.42	4.42	4.42	4.42	4.12	0.75	40.57
2.84	0.41	1.27	-0.21	-0.21	-0.21	0.09	0.38	4.05
0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.48	4.80
2.36	-0.07	0.79	-0.69	-0.69	-0.69	-0.39	-0.10	-0.75
1.10	-0.07	0.72	-0.69	-1.39	-2.08	-2.48	-2.57	
1.10	0	0.72	0	0	0	0	0	1.82
0.50	0	0.33	0	0	0	0	0	0.84
1.86	-0.07	0.46	-0.69	-0.69	-0.69	-0.39	-0.10	-1.59
								4.80
0.10	0.20	0.10	0.10	0.10	0.10	0.10		1.00
							0.50	0.50
0.48	0.48	0.48	0.48	0.48	0.48	0.48	0.48	4.80
2.24	0.21	0.84	-0.31	-0.31	-0.31	-0.01	0.88	-2.09
-3.07	-2.85	-2.02	-2.33	-2.65	-2.96	-2.97	-2.09	-2.09

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PROJECT : RINGAROOMA JOINT VENTURE, TASMANIA
CASE : D

YEAR ENDING DECEMBER		1985	1986	1987	1988	1989	1990
DRY STRIPPING VOL	(M cu m)				0.40	0.81	0.72
DREDGING/TREATING VOL	(M cu m)					0.38	0.50
TOTAL MINING VOL	(M cu m)				0.40	1.19	1.22
DREDGING/TREATING GRADE	(g cass conc/cu m,70%Sn)					634	634
RECOVERY FACTOR	(%)					90	90
PRODUCTION	(t cass conc,70% Sn)					217	285
TIN METAL PRICE	($/kg)					16.50	16.50
REALISABLE PRICE	($/kg cass conc,70% Sn)					11.22	11.22
GROSS REVENUE	($M)					2.43	3.20
ROYALTY:	($M)						
- 2% TO GOVT	($M)					0.05	0.06
- 5%TO BMI	($M)					0.00	0.00
NET REVENUE	($M)					2.38	3.14
DRY STRIPPING COST	($M)				0.00	0.00	0.00
HYDRAULIC MINING/TREATING	($M)				1.00	3.10	3.10
TOTAL MINING COST	($M)				1.00	3.10	3.10
OPERATING SURPLUS	($M)				-1.00	-0.72	0.04
DEPRECIATION	($M)					0.73	0.73
PROFIT BEFORE TAX	($M)				-1.00	-1.45	-0.69
'TAX COMPUTING STEP'	($M)				-1.00	-2.45	-3.14
TAXABLE PROFIT	($M)				0.00	0.00	0.00
TAX AT 46%	($M)				0.00	0.00	0.00
PROFIT AFTER TAX	($M)				-1.00	-1.45	-0.69
CAPITAL/DEVELOPMENT	($M)	0.05	0.15	1.10	5.30		
ANNUAL CAPITAL	($M)					0.10	0.10
SALVAGE VALUE	($M)						
CASH ADJUSTMENT	($M)					0.73	0.73
NET CASH FLOW	($M)	-0.05	-0.15	-1.10	-6.30	-0.82	-0.06
CUM NET CASH FLOW	($M)	-0.05	-0.20	-1.30	-7.60	-8.42	-8.48

NPV AT 10% = -7.07
NPV AT 12% = -6.71
DCF YIELD = **Negative**

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1991	1992	1993	1994	1995	1996	1997	TOTAL/AV
0.60	0.90	0.90	0.90	0.97	0.97	0.24	7.41
0.38	0.50	0.50	0.50	0.50	0.50	0.35	4.11
0.98	1.40	1.40	1.40	1.47	1.47	0.59	11.52
634	626	626	636	671	671	671	644
90	90	90	90	90	90	90	90
217	282	282	286	302	302	211	2384
16.50	16.50	16.50	16.50	16.50	16.50	16.50	16.50
11.22	11.22	11.22	11.22	11.22	11.22	11.22	11.22
2.43	3.16	3.16	3.21	3.39	3.39	2.37	26.75
0.05	0.06	0.06	0.06	0.07	0.07	0.05	0.53
0.02	0.16	0.16	0.16	0.17	0.17	0.12	0.95
2.36	2.94	2.94	2.99	3.15	3.15	2.21	25.26
0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
3.10	3.10	3.10	3.10	3.10	3.10	1.24	27.04
3.10	3.10	3.10	3.10	3.10	3.10	1.24	27.04
-0.74	-0.16	-0.16	-0.11	0.05	0.05	0.97	-1.78
0.73	0.73	0.73	0.73	0.73	0.73	0.76	6.60
-1.47	-0.89	-0.89	-0.84	-0.68	-0.68	0.21	-8.38
-4.60	-5.50	-6.39	-7.23	-7.91	-8.59	-8.38	
0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
-1.47	-0.89	-0.89	-0.84	-0.68	-0.68	0.21	-8.38
							6.60
0.50	0.10	0.10	0.10	0.10	0.10	0.10	1.30
						0.50	0.50
0.73	0.73	0.73	0.73	0.73	0.73	0.76	6.60
-1.24	-0.26	-0.26	-0.21	-0.05	-0.05	1.37	-9.18
-9.71	-9.98	-10.24	-10.45	-10.50	-10.55	-9.18	-9.18

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DEPT. OF MINES
REF. No 3336/85

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PROJECT NAME: RINGAROOMA JOINT VENTURE

TITLE: PRE-FEASIBILITY STUDY

VOLUME 11 - MAPS AND APPENDICES

AREA NAME/S, STATE 1:250,000 SHEET NO/S & COORDINATES: North-eastern Tasmania

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TABLE NOS: 9 Schedules

APPENDICES: 3 Appendices

AUTHOR/S: W K Lee, R W Shaw

DATE: February 1985.

AUSTRALIAN ANGLO AMERICAN LIMITED

Incorporated in the State of Victoria

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(Vol 2 of 2)

Appendix 1

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ANGLO AMERICAN CORPORATION OF SOUTH AFRICA LIMITED

TECHNICAL DIRECTOR'S OFFICE

TIN - EXECUTIVE REVIEW

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INTRODUCTION

The initial group tin study was completed in 1980. One of the principal conclusions at that time was that resources of the metal, in the Western World at least, appeared to be limited, and that a shortage might become apparent in 15 - 20 years time. Subsequently, a detailed study of long term tin consumption, and its relationship to the price of the metal was commissioned, in order to determine what effect the impending shortage might have on the market. This tin consumption study was completed late in 1982; it showed that demand was likely to remain flat for the foreseeable future; moreover, any increase in the price of the metal was likely to result in significantly increased economisation and substitution; consumption would rapidly trend downwards, and the rise in price would probably be rather short-lived.

A full update of the tin study was carried out in 1983. This made use of the 1982 demand analysis, it also included a description of the International Tin Council, its function in setting and controlling the price of the metal, and consideration of its future effectiveness. The update was discussed at the 1983 Metal Studies conference in November last year; this paper considers both the study, and the discussion at the conference.

In the past, a major problem area with all metal studies has been the difficulty in predicting Western World economic activity as a whole. This year, AAC, assisted by the Research and Economic Services department of Charter Consolidated, and E V Newland, has developed short and long term global scenarios, for which possible courses of economic growth in the Western World were derived. The upper and lower limits of probable economic activity which were derived for these scenarios have been used in this year's studies in the final derivation of consumption.

This review starts with a description of the outlook for tin, including an assessment of the Tin Council and factors affecting the tin price. This is followed by an assessment of the various AAC group interests in the metal, and by recommendations for tin exploration policy.

It is appropriate at this point to redefine some of the terms used in these commodity analyses:-

Firstly, all prices and operating costs are given in terms of the international purchasing power of the United States dollar in 1982. Since 1982, the internal purchasing power of the dollar has fallen by some 3,5 per cent due to cost inflation within the United States, but its average international purchasing power has only declined by 1,4 per cent (as of February 1984), as the dollar has firmed against other currencies.

"Operating costs", unless specifically defined otherwise, is taken to be cash operating cost, excluding any element of finance charges, interest payments, depreciation, or capital redemption, but including price or cost based royalties, and also "stay-in-business" capital, i.e. capital expenditure which is necessary to maintain the present operating rate.

"Demand", is better defined as requirement for supply; it differs from "consumption" in that it takes account of other factors, e.g. East/West trade and stockpile provements. "Demand" is normally directly comparable with "Supply Capability". In this context, capability is the assessed ability to produce in a normal year, unaffected by strikes or other hazards. It differs from "capacity", in that account is taken of factors such as grade variations, abnormal operating efficiency, problems in scheduling through complex plants, etc.

TIN - THE COMMODITY

a) Principal Forecasts

		1983	1985	1987	1989	1991
Western World Supply Capability '000 tonnes per annum						
	high	200	202	203	204	206
	low	200	181	180	178	176
Demand for Western World Tin '000 tonnes per annum						
i) Restructed Growth Scenario:-						
- tin price $10 per kg		171	182	184	190	196
- tin price $15 per kg		171	177	176	180	183
ii) Protracted Hard Times Scenario:-						
- tin price $10 per kg		171	173	169	170	171
- tin price $15 per kg		171	170	164	162	160
1980 Tin Study Demand Estimate		194	199	206	211	-

The estimates of future tin consumption were revised down significantly following the detailed study of tin consumption in 1982. In 1980, we were still predicting a much more rapid growth of the Western World economy; moreover we had not appreciated the extreme dependence of consumption on the price of the metal.

The principal factor affecting tin prices over the next ten years will be the effectiveness or otherwise of the International Tin Council. The problems of the Tin Council, and its effect on prices are explained in more detail in the next section.

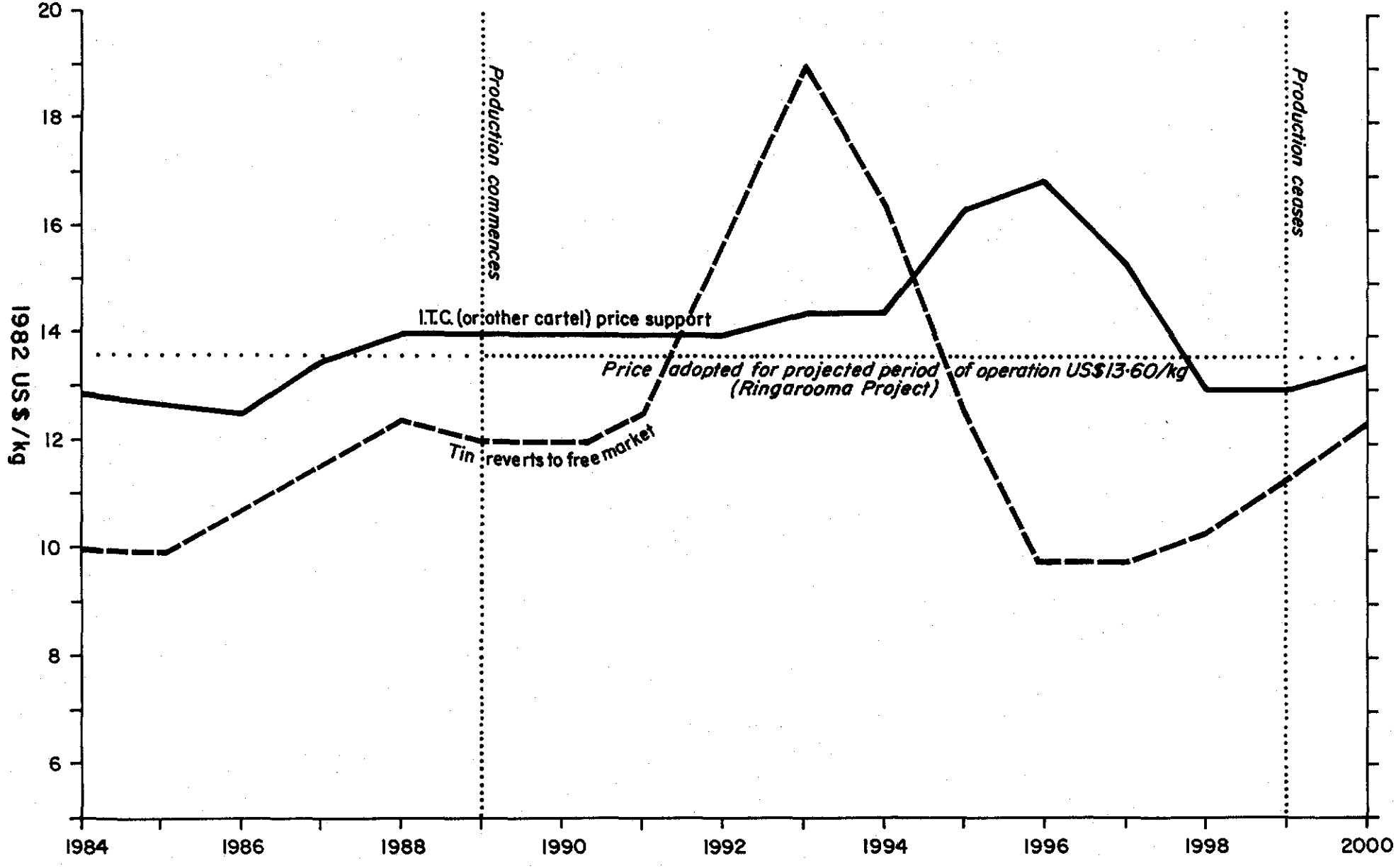
b) The Tin Market

The tin market has, for many years, been essentially different from all other base metals. The International Tin Council (ITC), by virtue of successive tin agreements, has acted in many ways as a cartel, to control production in such a way as to support the price of the metal. Theoretically these agreements were intended to limit both upward and downward price movements. In practice they have only served to defend floor prices; periodic shortages of the metal have resulted in prices exceeding the International Tin Council ceiling.

The present economic downturn has placed severe strains on the ITC and its producer members. Recent production surpluses have caused the ITC to impose cutbacks of up to 40 per cent on producing countries, and to finance tin stocks valued in excess of 800 million US dollars. Amongst the producer countries, Malaysia is becoming openly dubious about the value of retaining membership of the ITC. There seems therefore to be a very real chance that the organisation will break up, either during the next 6 - 12 months as a result of the strain of financing stocks and the difficulties of operating under the arduous export quotas, or more probably in 1987 when the present agreement comes up for renewal.

Should tin revert from its present market condition, which is akin to a producer cartel, to a free market, the surplus of production capacity, excess stocks, and poor outlook for consumption growth would result in significantly lower average prices for the metal. Figure 1 overleaf shows our evaluation of the tin price under the present ITC controlled market, and under free market conditions for the period to the year 2000 in constant 1982 dollar terms. (The ITC break-up scenario assumes collapse in the current year). We believe that the effect on tin prices of the different economic scenarios is likely to be insignificant compared to the effect of a change to free market conditions. The curves shown in Figure 1 may therefore be taken to apply to either of the Restructed Growth or Protracted Hard Times scenarios.

The following explanations for the two price curves should be read in conjunction with the graph:-



FUTURE TIN PRICE SCENARIOS

TAS/10/101

20705 51

1984-1986

Stocks continue to overhang the market. With ITC control, price remains in low range, but the ITC price range weakens slightly. Under free market conditions price drops to $10; many producers go out of business, production gradually increases, as stocks are liquidated, and the price slowly recovers until it reaches a level sufficient to justify 85 - 90 per cent of production capability remaining in operation.

1987-1990

Under both conditions the market is in rear balance. Consumption is slightly higher under free market conditions due to lower prices. Under ITC control the price moves up slightly as the ITC takes the opportunity to recover the loss due to the weakening price range from 1984-1986. Under free market conditions demand remains around 85 - 90 per cent of mine capability.

1991-1994

With ITC control, consumption remains somewhat lower due to previous higher prices, market remains nearly in balance, ITC stocks tend to reduce.

Under a free market, shortage develops due to increased consumption generated by prices in $10-12 per kg range. Price rises rapidly, and falls equally rapidly as consumption is choked off.

1995-2000

With ITC control, a brief shortage of supply arises, and the ITC is powerless to prevent the price rising to a little over the ITC ceiling, consumption falls as a result of the high price, and a limited high stock position develops.

Under free market conditions, the price is initially very depressed due to excess production during previous high price surge, but recovers later as stocks become depleted.

Over the 17 year period 1983-1990, the price with ITC control is likely to average $14,0 per kg, compared with only $12,2 per kg if there is no price support.

c) The Supply and Cost of Production of Tin

Western World tin supply capability is at present very close to 200 000 tonnes per annum; much as was forecast in the 1980 study. It is likely to fall gradually over the next ten years, primarily due to closures enforced by low demand for the metal. Mine supply capability in 1990 is likely to be in the region 185 - 190 000 tonnes. Producer countries of the ITC are responsible for some 70 per cent of mine supply capability; the important South-East-Asia countries, Malaysia, Indonesia and Thailand, for about 64 per cent.

Due to the imposition of production quotas on ITC producer countries, primary tin supply is currently running at around 160 - 165 000 tonnes per annum. However, the non ITC countries principally Bolivia, Brazil and the United Kingdom are all producing to the maximum of their ability, taking advantage of the price support afforded by the ITC. In consequence, the ITC countries proportion of actual production, as compared to capability, has fallen to about 60 per cent. This is a major cause of concern to the ITC.

Our previous analysis of tin reserves and resources has been confirmed. It still seems probable that some tin shortages will become apparent during the last few years of this century; however the analyses of consumption, and its dependence on price show quite clearly that any price rise caused by the shortage would be relatively short-lived.

The available cost data on tin producers is complicated by the imposition by many countries of large, almost punitive, royalties. In 1982, if royalties are excluded, over seventy five per cent of Western World tin was produced at an operating cost of less than $13 000 per tonne, sixty per cent for less than $10 000 per tonne. If royalties are included, these figures rise to $14 000 and $12 000 respectively. Royalties of course, are one of the means by which government both exert a measure of control over the industry, and attach a major part of any profits.

When the industry is going through a lean time as at present, royalties tend to be reduced, as they have been during the past year in Malaysia, and are likely to be next year in Thailand. The above costs include royalties commensurate with an average tin price in 1982 of $13 400 per tonne. Although the metal price has declined to around $12 000 per tonne in current money terms, mainly due to the strengthening of the dollar, most producers are still able to make a profit on operating costs, excluding the royalty element.

d) Tin Demand

Consumption of tin in the Western World has fallen from some 186 000 tonnes in 1970 to 161 000 tonnes in 1982. However, as is shown in figure 2 overleaf, this fall has partly been shielded from the Western World producers as the United States GSA sales have also fallen, and as the Socialist Bloc has switched from being net exporters to net importers. For much of the period, tin prices have been well over $15 per kg in constant 1982 terms, and these high prices have been partly responsible for the drop in Western World consumption.

Figure 3 shows the result of our analysis of the relationship between the metal price, and demand for tin (making due allowance for East/West trade, and GSA sales) for the period up to 1992. If the metal price remains below $15 per kg in 1982 money terms, we expect some recovery in demand from the low figures of the past three years; if the price were to drop as low as $10 per kg, the recovery would be marked, but probably not sufficient to put pressure on supply capability before 1992. If the metal price were to revert to the $15 - 17 range in 1982 money that characterised the years 1977 - 1981, we expect that consumption would continue to fall. However, with the present gross over-supply position there seems no possibility that the ITC will seek to raise the metal price for some years from its present level, in the low ITC range of $12,5 - 14,4 per kg. Moreover, if the ITC were to lose control of the market, the metal price would probably drop well below this range.

Tinplate remains the most important end-use sector in the Western World, accounting for some 37 per cent of total tin consumption. This sector is particularly subject to economisation and substitution. Tin cans are being replaced by plastic/paper packaging, by aluminium cans, and by tin-free-steel. Tin consumption in what remains of the tin can market is declining due to lower coating thicknesses and other economisations. Continued erosion of the tinplate market is the principal reason why tin consumption is likely to remain more or less static, depending on the price of the metal, whilst the Western World economy shows significant growth during the next ten years.

e) Tin Stocks

At the end of 1982, the overall level of stocks surplus to normal working levels was estimated as:-

Mines/Smelters/Refiners	12 700	tonnes
Official ITC stocks	52 700	tonnes
Unofficial ITC financed stocks in the commercial sector	25 000	tonnes
Total	90 400	

By the end of 1983 the overall stock position was probably slightly less; however, there has been a significant increase in the mine/smelter/refiners stocks, financed by the producers, of 6 000 - 9 000 tonnes, and a fall in the ITC financed stocks of some 11 000 tonnes. In 1984 there should be some recovery in tin consumption, and the 40 per cent export quotas will become fully effective. (These "40 per cent quotas" impose a reduction of 40 per cent, i.e. limit exports to approximately 60 per cent of the rate in 1981, before the quotas were imposed). However it will inevitably be some years before the excess stock position is fully liquidated; depending on the rate of economic growth, we expect that a large overhang of stocks, necessitating continued export quotas, will continue for at least the next three years.

FIG. 2 - HISTORIC DEMAND FOR WESTERN WORLD TIN 1970 - 1982

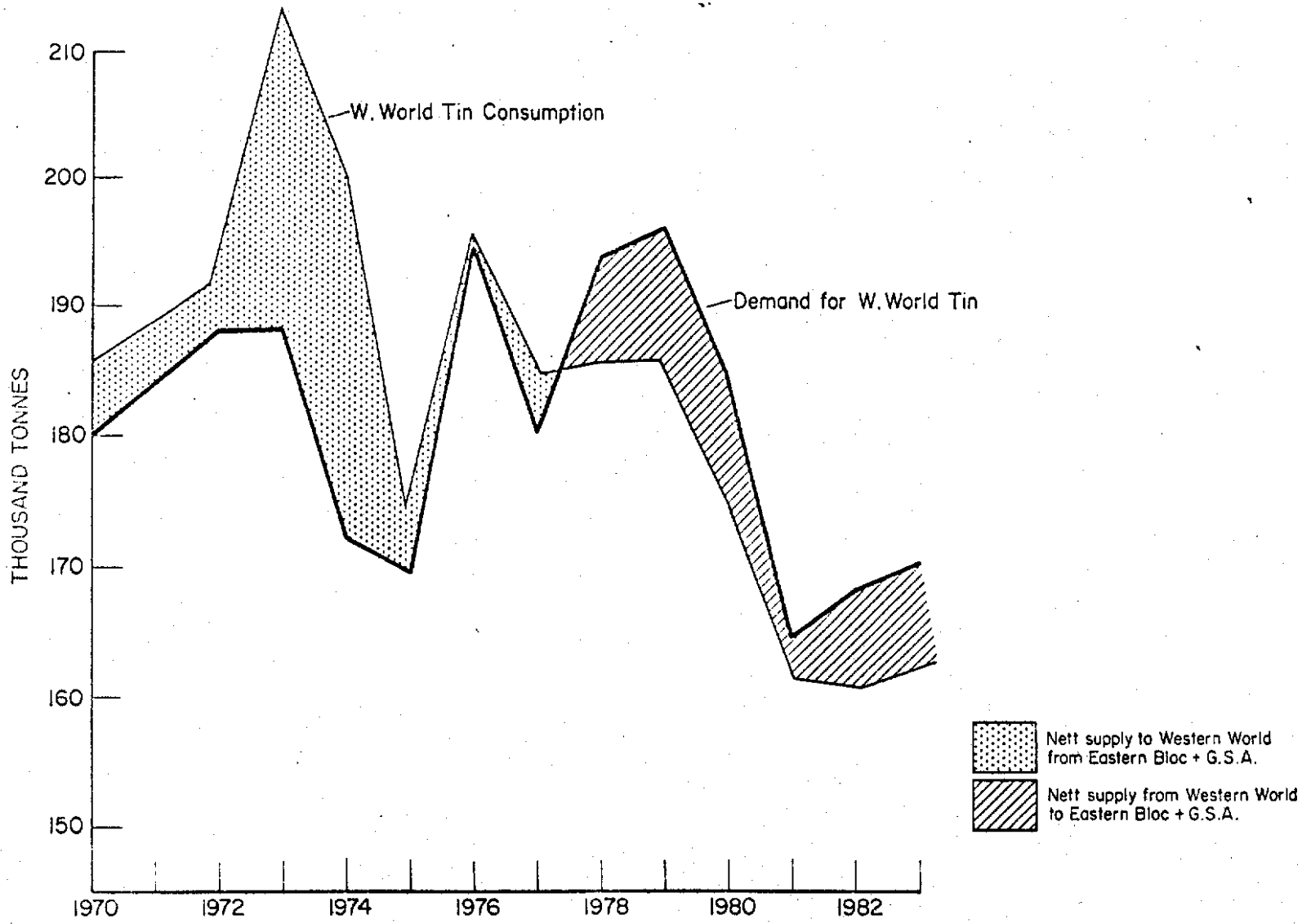


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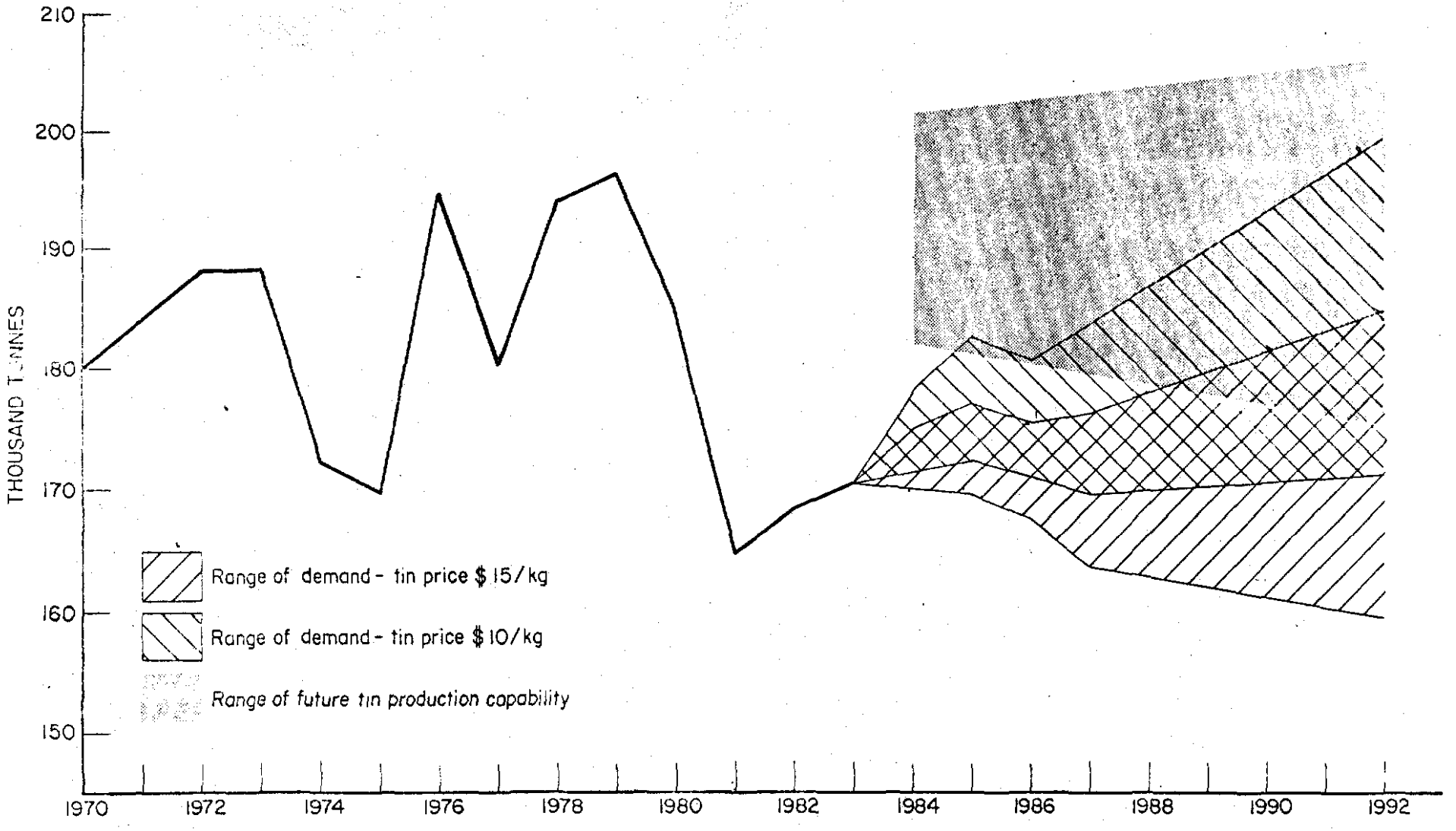
FIG. 3 - HISTORIC AND EXPECTED DEMAND FOR WESTERN WORLD TIN 1970 - 1992



AAC INTERESTS AND POLICIES TOWARDS TIN

The major uncertainty in the tin market is the continuation of the ITC. Since breakdown of the ITC in the next three years seems to us to be at least a 30 per cent possibility, it is necessary to examine what effect such a breakdown might have on the various AAC interests in the metal.

The first effect the breakdown of the ITC would have, if it occurred during the next three years, would be the collapse of the tin price. We expect the price would stabilise at around $10 per kg for a year or two before gradually rising to $12 per kg, all in 1982 dollar terms. If producers managed to institute major savings in operating costs, thus reducing the average cost of production below present levels, the price might be expected to fall even further; this type of secondary reaction has recently characterised the copper market.

South Crofty

It is clear from our analyses of operating costs that the tin industry in the United Kingdom with its average cost close to $12,5 per kg in 1982 would, as a whole, be rendered chronically unprofitable. At present the United Kingdom industry appears to be operating at close to maximum efficiency, just to keep costs below the present tin price; there is little or no fat that could be trimmed off if the tin price did fall to around $10, and we believe that most if not all of the Cornish tin industry would inevitably be forced to close. We have no information that would suggest that Charter Consolidated's South Crofty is better placed than the average Cornish tin mine.

Malaysia Mining Corporation (MMC)

The bulk of MMC tin is produced at an operating cost of less than $10. If the price were to fall to this level, royalty payments would become zero, but since the fall would be accompanied by removal of the export quotas, the company would be free to produce up to capacity, as would other Malaysian producers. There should still be some contribution to profits, which might in fact be as great as that accruing at present whilst operations are restricted by the export quotas. MMC would be able to recover the market share that it stands to lose if the export quotas remain in force much longer.

There is obviously no easy way to assess the effect of a change in the ITC on operations in Malaysia. Whichever way things go it will be some time before MMC returns to the sort of profitability that characterised the late 1970's.

Zaaiplaats

The Zaaiplaats tin mine in South Africa is controlled and managed by Zimro. Being outside the ITC, the mine is unaffected by production or export quotas, and is producing to the maximum of its capacity. With the weakening rand:dollar exchange rate, Zaaiplaats operating costs in dollar terms have tended to fall in recent years. The mine is currently producing at an operating cost just under $10 per kg (including allowances for stay-in-business capital). Unless the gold price rises rapidly allowing the rand to firm against the dollar, it seems likely that Zaaiplaats will be able to stay in business even if the ITC collapses. However, the accumulation of cash reserves against such a possibility would seem to be a wise precaution.

Tin Prospecting

Three years ago we made positive recommendations on tin prospecting in countries outside the Far East Asia tin belt. The recommendations were made against the background of a probable shortage of the metal which would lead to higher prices by the end of this century. We now believe such higher prices are unsustainable, and any recommendations for prospecting must be modified accordingly.

We now recommend that, unless there are good reasons for believing that cheap-to-mine deposits can be found, which could be justified at what we believe to be the long-term equilibrium price under free market conditions, of $12,0 per kg, prospecting for tin should be abandoned. Such cheap-to-mine tin may not be impossible to find. However, Brazil is virtually the only country that we believe to be sufficiently prospective to warrant an exploration programme for tin in the forseeable future. Even in Brazil, the favourable fiscal regime, and highly prospective nature of the ground in the Rondonia province are largely off-set by problems of infrastructure. Any plans for tin prospecting in Brazil might be better postponed until after 1987, when the future of the ITC, and of the tin market becomes a little clearer.

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85-2358 (Vol 2 of 2) APPENDIX 2

207060

A. SIXTH INTERNATIONAL TIN AGREEMENT

THE BUFFER STOCK AND TIN EXPORT CONTROL

The International Tin Agreement is established by the world's tin producing and consuming countries and is binding on the participating members. The main objectives of the Agreement, in so far as relating to the subject of this report, are :

(a) To provide for adjustment between world production and consumption of tin and to alleviate serious difficulties arising from surplus or shortage of tin, whether anticipated or real;

(b) To prevent excessive fluctuations in the price of tin and in export earnings from tin;

(c) To ensure conditions which will help to achieve a dynamic and rising rate of production on the basis of a remunerative return to producers, which will help to secure an adequate supply at prices fair to consumers and to provide a long-term equilibrium between production and consumption;

(d) In the event of a surplus of supplies of tin occurring or being expected to occur, to take steps to mitigate serious difficulties which producing countries might encounter.

In order to achieve those objectives, the Agreement has provisions of articles which can be broadly grouped under the following :

- The Buffer Stock Operation
- The Tin Export Control

BUFFER STOCK OPERATION

A Manager, appointed by the Council and responsible to the Executive Chairman of the Council, is in charge of the Buffer Stock Operation. His primary function is to attempt to control and maintain the world tin price between the ceiling and floor price levels set up by the Council, with the means of an available buffer stock and operating under conditions in the ceiling/floor price spectrum.

The buffer stock consists of a normal stock of 30 000 tonnes of tin metal financed from government contributions, and an additional stock of 20 000 tonnes to be financed from borrowing.

The Council decide the floor and ceiling tin prices, on the basis of studies conducted by the Economic and Price Review Panel, a subsidiary body. In conducting its reviews of the prices, the Council consider the short-term developments, and the various levels and trends of tin production and consumption, the production costs of tin, the existing capacity for mine production, the adequacy of the current price to maintain sufficient future mine production capacity and other relevant factors affecting movements in the price of tin.

The range between the floor and ceiling prices is 30% of the floor price and is divided into three equal sectors, being the upper, middle and lower sectors. The prices are expressed in Malaysian ringgit or in any other currency which the Council may decide.

The conditions under which the Buffer Stock Manager operates in the price sectors are illustrated in the attached Diagram I.

TIN EXPORT CONTROL

The Council may, by a two-thirds distributed majority, declare a control period when at least 70% of the maximum volume of the buffer stock is held in tin metal in the buffer stock. It may declare a control period when that reaches at least 80%.

In declaring a control period, generally corresponding to quarters, the Council shall fix a total export tonnage for Producing Members, taking into account the estimates of production and consumption, the quantity of tin metal and cash held in the buffer stock, the quantity, availability and probable trend of other stocks of tin, the trade-in tin, the current price of tin metal and any other factors.

The total permissible export tonnage for any control period is divided among Producer Members in proportion to their production or export figures for the last four consecutive quarters which preceded the control period and which were not declared control periods.

If the net exports of tin from a Producing Member in any control period exceed its permissible export tonnage for that control period by more than 5%, the Council may require the Member concerned to make an additional contribution to the buffer stock not exceeding the tonnage by which such exports exceed its permissible export tonnage.

At any time when it has declared a control period the Council may, by a two-thirds distributed majority, permit a 'special export' of a specified quantity of tin in addition to the permissible export tonnage, on the condition that :

- The 'special export' is destined to form part of a government stockpile;
- The 'special export' is unlikely to be used for any commercial or industrial purpose during the currency of the Agreement.

If, during a control period, the 15-day moving average of the market price of tin remains at or above the upper limit of the lower price sector for 12 consecutive market days, the permissible export tonnage shall be increased to :

- The level of exports for the corresponding period calculated on the basis of the quarterly average level of exports during the last four consecutive quarters which preceded the control period and which were not declared control periods; or
- 110% of the total permissible export tonnage fixed for that period; whichever is the greater.

If, at any time when the price is in or above the upper sector, the Council concludes that a serious shortage of supplies of tin has developed or is likely to develop, the Council :

(i) Shall terminate the export control and recommend the level of stocks which should not be exceeded;

(ii) Shall recommend to Members that they take all possible steps to ensure as rapid an increase as possible in the amount of tin which they are able to make available.

B. AUSTRALIA - TIN QUOTA ALLOCATION TO MINES

In Australia the Department of Trade at Canberra is responsible for controlling and monitoring its domestic tin exports during a tin export control period and setting up local rules for distribution of mine quotas to the tin producers in the country. During the control period the country's tin producers are notified quarterly the procedure of the distribution and the quotas allocated to them.

The method of allocation is detailed in the attached copy of the notice "Tin Export Controls : Arrangements for the April-June 1984 Quarter" issued by the Department of Trade. This is illustrated in Diagram II.

It is important to note that :

- quotas are established in the name of the current legal owner of a mine or, with the owner's agreement, in the name of the operator of a mine in the case of a tributing arrangement;

- quotas are not transferable except that persons or companies that operate more than one mine may aggregate their individual mine quotas and that persons or companies purchasing an existing mine which satisfied the conditions for the establishment of a mine despatch quota may on completion of the purchase and subject to the approval of the Department of Trade, have the quota established in their name(s) where this is requested by the buyer and seller;

DIAGRAM I

BUY / SELL CONDITIONS FOR BUFFER STOCK MANAGER TO OPERATE

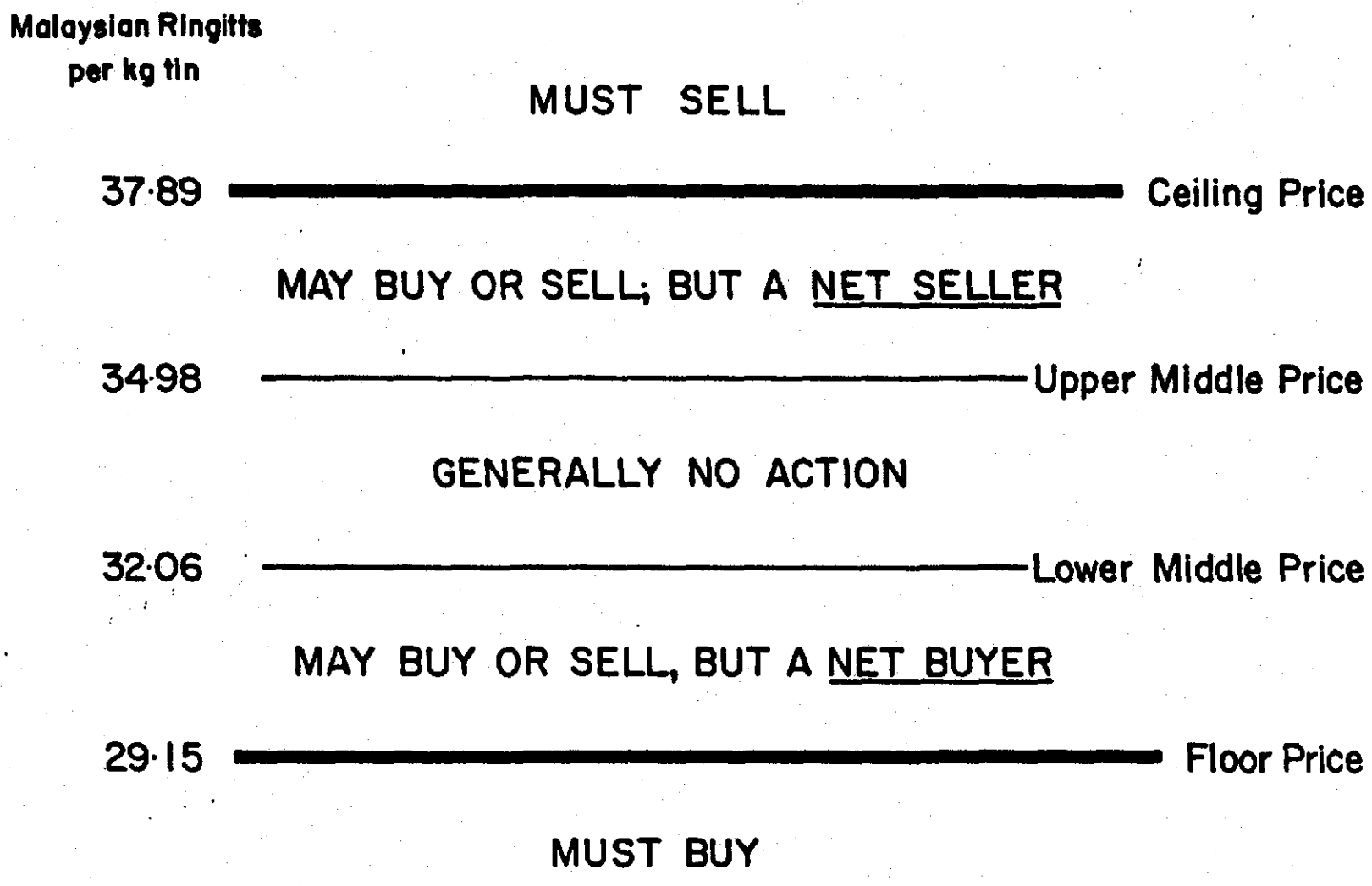

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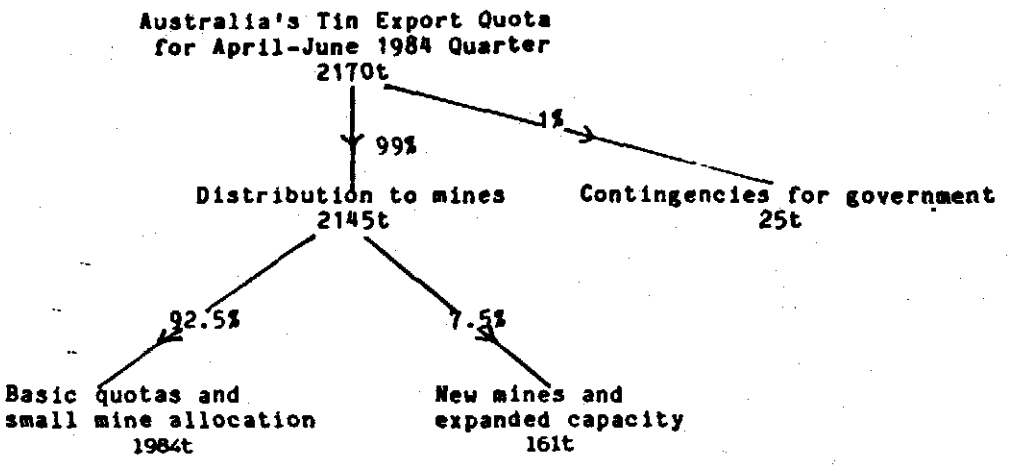
Diagram II



- Basic quotas are for each mine that produced 2 tonnes or more of contained tin in 1981 and is, in the current quarter, an operating mine with plant and labour in place. These quotas are distributed on the basis of percentage shares of 1981 production in respect of each mine;

- Small mine allocation is a collective allocation for mines that are not eligible for a quota (e.g. for tin buyers). This is calculated on the relevant percentage share of the overall Australian quota related to 1981 production of mines in this category for which individual quotas have not already been established.

- For new mines opened in 1982 and have been in actual production or ready for production before 1 October 1982.
  Base figure for the calculation of the share of the available new mine allocation shall be the actual additional capacity on an annual basis or 150 tonnes contained tin whichever is the lesser.
  No quota for mines with a claimed new capacity of less than 20t contained tin per year.

- For expanded capacity at an existing mine, the increased capacity must have been in actual production or ready for production before 1 October 1982.
  Base figure for the calculation of the share of the avialable expanded capacity allocation shall be the actual additional capacity on an annual basis or 50% of 1981 production which ever is the lesser.
  No quota for mines that produced less than 10 tonnes contained tin in 1981.

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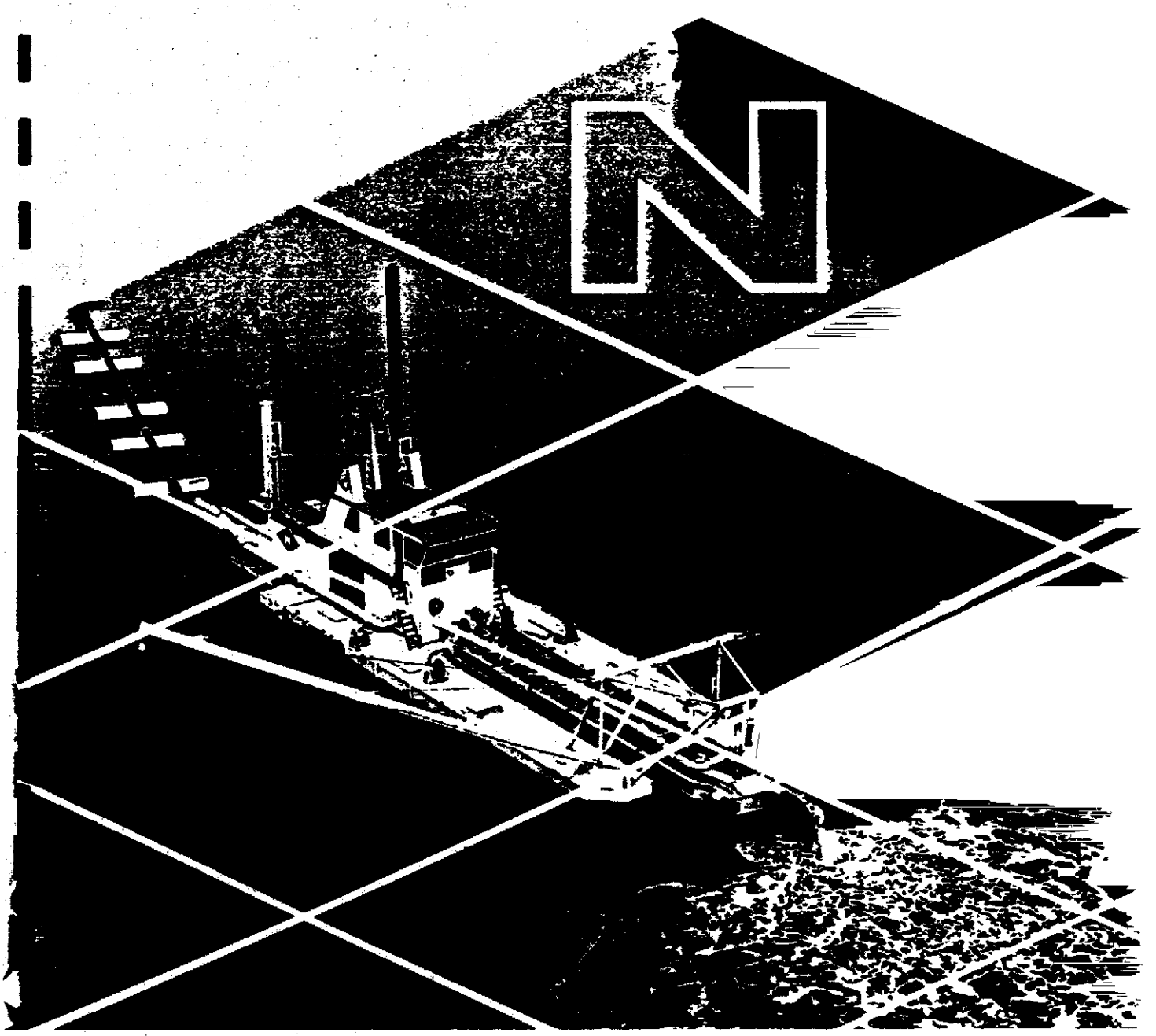
Vol 2

Neumann Equipment Pty Ltd

207065

The Neumann



064 : AA6

207066

Ready reference data sheet

Neumann Series 250-10L, 250-12L, 350-14LD, 350-16LD

Measurements shown in metric and imperial standard

DREDGE SERIES	250-10L	250-12L	350-14LD	350-16LD
Ladder pump size	10/8	12/10	14/12	16/14
Deck pump size			14/12	16/14
Suction diameter	250mm (10in.)	300mm (12in.)	350mm (14in.)	406mm (16in.)
Delivery diameter	250mm (10in.)	300mm (12in.)	350mm (14in.)	406mm (16in.)
Main diesel engine	3408T	3408TA	3412TA	3512 V12
Auxiliary diesel engine			3406TA	3408TA
Main engine power continuous	225KW (300HP)	272KW (365 HP)	447KW (600HP)	600KW (800HP)
Auxiliary engine power continuous			240KW (325HP)	272KW (365HP)
Flow rate	220 l/s (2900 G.P.M.)	300 l/s (4000G.P.M.)	500 l/s (6600 G.P.M.)	650 l/s (8500 G.P.M.)
T.P.H. at 35% w/w	320T (325t)	480T (488t)	600T (610t)	1050T (1070t)
Total developed head	43m (141ft)	50m (164ft)	60m (197ft)	60m (197ft)
Maximum digging depth	8.5m (27ft 6in.)	8.5m (27ft 6in.)	15m (49ft)	15m (49ft)
Width of dredge cut	28m (92ft)	28m (92ft)	50m (164ft)	50m (164ft)
Maximum particle size	178mm (7in.)	220 mm (8.7 in.)	241 mm (9.5 in.)	254mm (10ins.)

Above data is calculated on line velocity of 4 m/s (13.1ft/sec.)
Diesel engine power are Caterpillar approved ratings for these applications.
T.P.H. rate is for free flowing sand situation — adjust rate for more difficult materials and pulp density variations.
The specifications on this sheet are current as at 1st October, 1983, and supersede all previous issues.

Neumann Equipment Pty Ltd

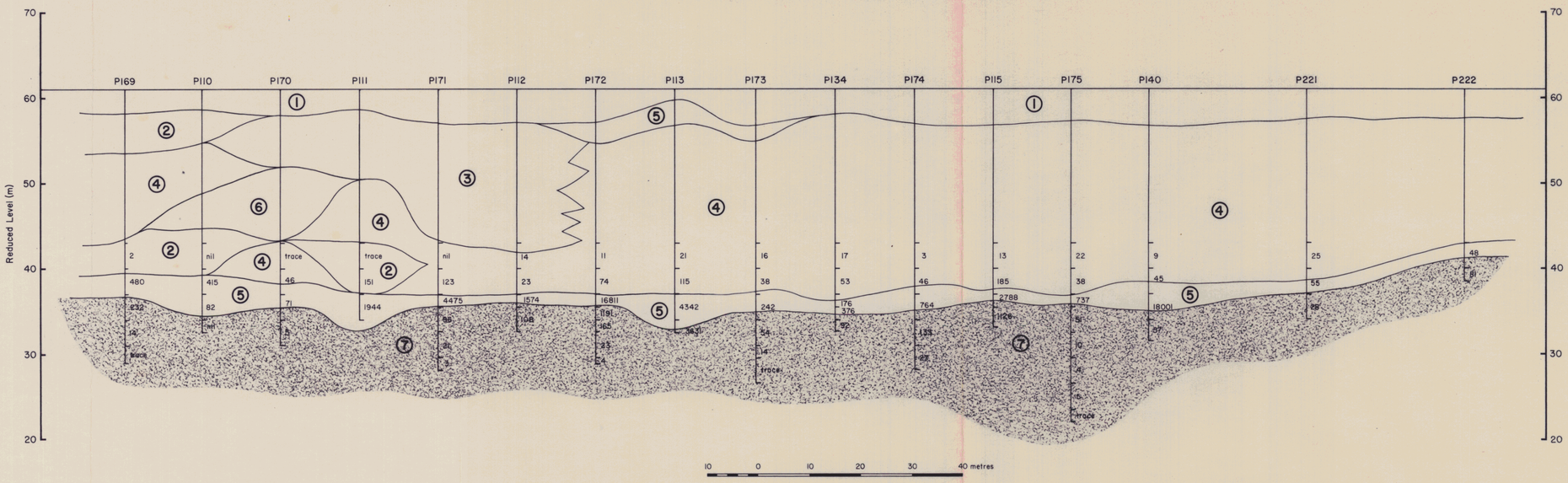
Nuban Street, Currumbin, Gold Coast, Queensland, Australia. 4223.
Telephone: (075) 342700 Telex: AA 42314 NEUMN
Cable: NEUMINAUST. Brisbane

147°50'
148°00'
41°00'
41°10'
New Dorset Lease Block
Monarch Lease Block
Endurance Lease Block
Amber Hill Lease Block
East Banca Lease Block
South Mt. Cameron
Pioneer Lease Block
Pioneer
Gladstone
Herrick
Fromes Dam
Branxholm
EL 21/77
EL 28/76
Boobyalla River
Ringarooma River
Gladstone Road
Wyndford River
Tasman Highway
400
300
200
100
600
500
1200
1000
800
5 cm
0 2 4 6Km.
20706?
Quaternary and Tertiary sediments.
Quaternary sediment.
Tertiary basalt.
Tertiary sediment.
Blue Tier Granite Batholith
Mathinna Beds.
A.A.A./Triako joint venture area.
Water race.
Exploration drilling in 1981-84
1 Davids Ck.
2 Gellibrand Plains
3 Trout Ck.
4 Scotia Lead
5 Arba Flats
6 Eastern Leads
7 Pioneer - S.W. Margin
8 Pioneer - N.W. Margin
9 Endurance Lead
10 Ringarooma Basin
11 Pioneer Dam Reserve
12 Pickets Plain
13 Endurance "Fault"
14 Banca
15 Boobyalla Basin
16 Arba Lead
17 Pioneer - Football Ground
18 Pioneer - Poverty Point
19 Pioneer - Gillhams Ck.
20 Scotia Lead.
Amdex mining leases included in joint venture, October 1982.
AUSTRALIAN ANGLO AMERICAN LIMITED
NORTH EAST TASMANIA
LOCATION AND GEOLOGY OF RINGAROOMA VALLEY
001
COMPILED W.K. Lee
DRAWN August, 1984
SCALE 1:100,000
TAS-100-114
85-2358 (Vol 2 of 2)

ENDURANCE LEAD

GEOLOGY OF SECTION 2

HORIZONTAL SCALE = 1:500 VERTICAL SCALE = 1:300



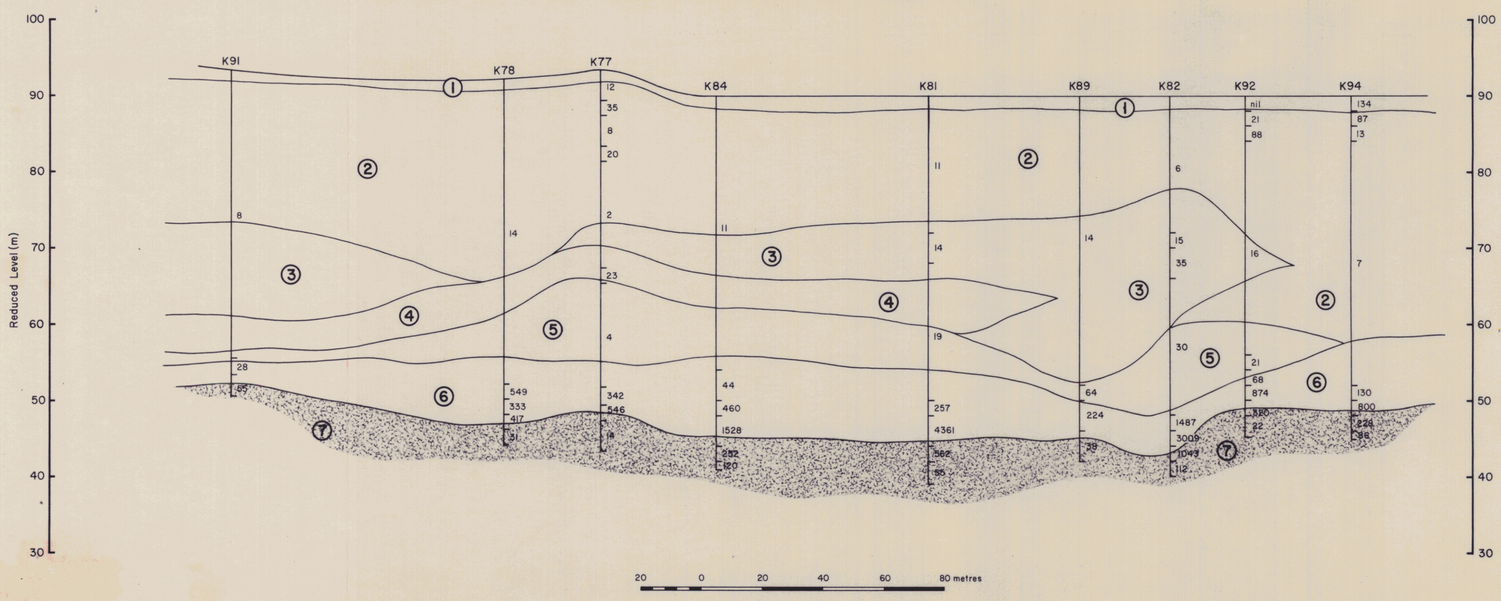
1 Silt
2 Clayey sands
3 Fine sands
4 Coarse sands
5 Coarse sands/wash
6 Hard sands with pyrite lumps
7 Decomposed granite
P222 Borehole number
48 Grade (g cassiterite concentrate /m^3 at 70% Sn)

NOTE
1. The top 18m (60ft) of samples of the boreholes were not processed, therefore no cassiterite values recorded in this section.
2. Borehole collar elevations are not available. Therefore arbitary datum of 61m has been used

PIONEER LEAD

GEOLOGY OF SECTION A–B

HORIZONTAL SCALE = 1:1000 VERTICAL SCALE = 1:400



1 Black top soil
2 White clayey fine & coarse sands
3 Brown clayey fine & coarse sands
4 Yellow clayey fine & coarse sands
5 White/grey coarse sands with wash
6 Wash
7 Decomposed granite
K94 Borehole number
87 Grade (g cassiterite concentrate/m^3 at 70% Sn)

20706S

5 cm

AUSTRALIAN ANGLO AMERICAN LTD			
PROJECT	003 85-2355 Vol 2/2		
AREA	NORTH EAST TASMANIA		
DATA	PIONEER & ENDURANCE LEADS GEOLOGICAL CROSS-SECTIONS		
COMPILED	W.K.Lee	SCALE	See sections
DRAWN	August 1984	REF No	TAS-100-15/1
AMENDED			

575000mE 575500mE 576000mE 576500mE 577000mE

5460000mN 5459500mN 5459000mN

wet area

gravel pit

OLD PORT ROAD

MD 9

100 150 155 200

water race

ORE BLOCK

wet area

RP7/14 RP6/14 RP5/22 RRC 16/18

207069

LEGEND

Area selected for economic evaluation

/231 GRADE - g/cu.m. cassiterite conc. (70% Sn)

*1 Dept of Mines-Percussion 1969

oA145 BMI Mining - Auger 1971-72 Non-sample; unreliable drilling

•P124 BMI Mining-Percussion 1971-72

•14C BMI Mining-Percussion 1973-74

⊕ERC1 Amdex Mining-Reverse Circulation 1982

□DDH11 Dept of Mines-Diamond 1982

⊕ERC3 Aust. Anglo American-Reverse Circulation 1982

■RP5 Aust. Anglo American - Percussion 1983

44.2 Depth to Basement (m)

5 cm

OLD PORT ROAD TAS-10-38 | BLUE LAKE TAS-10-35 | TAS-10-159 | PICKETTS PLAIN TAS-10-39 | RUBY CREEK TAS-10-36 | HILL 168 TAS-10-40 | ABC TAS-10-37

AUSTRALIAN ANGLO AMERICAN LTD			
PROJECT	AAA-TRIAKO RINGAROOMA JV		
AREA	OLD PORT ROAD		
DATA	DRILL HOLES 004		
	85-2358 CVol 2(2)		
COMPILED	RAAM 2.83	SCALE	1:2500
DRAWN	JAH 9.82	REF No	TAS-10-49
AMENDED	HR 9.84		

FIG. 5

005

577500mE 578000mE 578500mE 579000mE 579500mE 580000mE

5460000mN 5459500mN 5459000mN

Clifton Creek

150 100 100

62 M/76

Granite outcrop

Granite outcrop

CLIFTON 1980 MINE

CLIFTON 1975 MINE

pump

pipeline

Drain

CLIFTON

Blue Lake

WEST ENDURANCE

WOODS HOLE

LITTLE JIG

TAILINGS IN THE WORKED ENDURANCE DEEP LEAD

sluice

pipeline

ENDURANCE No.1

SHALLOW WORKED GROUND

SHALLOW GROUND

workings

ORE BLOCK

ORE BLOCK

SWAMP

SWAMP

Approx. positions

MINE TAILINGS

SHALLOW GROUND

SHALLOW GROUND

SHALLOW GROUND

Granite outcrop

Granite outcrop

6M/76

58M/73

Transformer remains

CONCRETE DAM

'WATTS'

wet area

70

LEGEND

DRILL HOLES

MAP SHOWS HOLE NUMBER / CODE & GRADES (if sampled)

- Endurance Tin Mining Co. hand plant = sample = 1940's
- Department of Mines cable tool = 1969
- W.L. Sides & Co. cable tool = 1967
- B.M.I. Mining Pty. Ltd. cable tool = 1971 - 74
- B.M.I. Mining Pty. Ltd. auger = 1971 - 72
- Amdex Mining Ltd. cable tool = 1979 - 81
- Amdex Mining Ltd. auger = 1980 (Non-sample - unreliable drilling)
- Australian Anglo American Ltd. cable tool = 1982 - 83

502.350 Basement R.L., Depth to Basement (m)

60 BASEMENT CONTOURS 5m Interval

BOUNDARY OF GRANITE OUTCROP

MINERAL LEASES

Area selected for economic evaluation.

GRADE - Colour Coding - g/cu.m. cassiterite conc. (70% Sn)

BELOW	50
50	99
100	199
200	399
ABOVE	399

OLD PORT RD TAS-10-38 | BLUE LAKE TAS-10-35 | TAS-10-159 | PICKETTS PLAIN TAS-10-39 | RUBY CREEK TAS-10-36 | HILL 158 TAS-10-40 | ABC TAS-10-37

20707

5 cm

AUSTRALIAN ANGLO AMERICAN LTD			
PROJECT	AAA-TRIAKO RINGAROOMA JV		
AREA	BLUE LAKE SHEET - N.E. TASMANIA		
DATA	DRILL HOLE LOCATION MAP & GRADE MAP (g/m³ SnO_2) 005		
	85-2358 CV01 2(2)		
COMPILED	R. Munro	SCALE	1:2500
DRAWN	R. Munro 8/83	REF No	TAS-10-163
AMENDED	H R 9/84		

FIG. 8

577 000mE
577 500mE
578 000mE
578 500mE
453 000mN
452 500mN
452 000mN
DAM
DAM RESERVE
PIONEER MINE RESERVE
OVERBURDEN DUMP
AMDEX MINING WORKINGS
GILHAM CREEK DEVIATION
Bradshaws Creek Deviation Race
TAILINGS
PIONEER TIN POND
MINING MINE WORKINGS
ELEVATOR STOCKPILE
CYCLONE TAILS
OVERBURDEN DUMP
V. WOODS WORKINGS
ELEVATOR STOCKPILES
FOOTBALL GROUND
LIMIT OF PROVEN RESERVES
PROVEN RESERVES
PLANT(2)
PLANTS (1, 1A)
BOOSTER PUMP (1)
PUMP (3)
PUMP (4)
PUMP (5)
S/S PUMP
TROMMEL
BOOSTER PUMP LINE
POWER LINE
TELEPHONE LINE TO ELFIN
TELEPHONE LINE TO MT. CAMERON
Boundary of Pioneer Tin Mining Co. Workings
Boundary of Old Pioneer Tin Mining Co. Workings

SURFACE TO BASEMENT GRADES

0–49g 70%Sn/m³	50–99g 70%Sn/m³	100–199g 70%Sn/m³	200–399g 70%Sn/m³	400+g 70%Sn/m³

Surface of pit
Overburden/Mine boundary
Base of pit

Limit of proven reserves
Limit of Pioneer Deposit
Resource marginal to the Pioneer Deposit
Kibuka-AAA percussion, reverse circ. drilling
Pioneer Tin Mining Company drilling
Austral Malay drilling
Storey Creek Tin Mining Co. drilling

K100 — Drill hole number
56 — Overall grade g. cass. conc./cu.m. (70%Sn)
61-76 — Basement R. L. (m)
(27·4) — Depth to basement (m)

5 cm

Area selected for economic evaluation

SHEET ONE
SHEET TWO

207071

AUSTRALIAN ANGLO AMERICAN LTD			
PROJECT	RINGAROOMA JOINT VENTURE		
AREA	N. E. TASMANIA		
DATA	PIONEER ALLUVIAL TIN DEPOSIT RESERVE BLOCKS 85·2358 (Vol 2 (2)) 002		
COMPILED	R. Munro	SCALE	1:2500
DRAWN	H. Rowley 6/84	REF No	TAS-10-176
AMENDED	8/84		

BOOBYALLA

GEOLOGICAL SURVEY OF TASMANIA – DEPARTMENT OF MINES – HOBART

FIRST EDITION 1978

GEOLOGICAL ATLAS 1:50 000 SERIES SHEET 8416 S.

REFERENCE

CAINOZOIC

QUATERNARY

- Alluvium, swamp and marsh deposits.
- Younger aeolian dune sand with some dune crests indicated, beach sand and gravel.
- Dolerite talus.
- Older aeolian sand with dune crests indicated where present on Qpo and Qpu.
- Clay, sand, minor peat and gravel, sometimes containing marine shells and transported fragments. An occurrence of non–marine marl indicated (Qpcm at EQ 559778).
- Erosional surface.

TERTIARY

- Sand, gravel and clay (Tz); ferruginous fine sand dominant (vertical line overprint Tzf) – probably overlies basalt; basalt (Tb). Lag and outcrop of silicified quartz sandstone and conglomerate indicated (diagonal line overprint Tzs). Late Tertiary marine limestone occurrence shown (Tsl at EQ 829878). Stippled overprint indicates man disturbed.
- Low angle unconformity.

PALAEOZOIC MESOZOIC

TRIASSIC

- Fresh–water cross–bedded quartz sandstone (Upper Parmeener Super–Group).

PERMIAN

- Sparsely fossiliferous marine sandstone and siltstone with drapstones (Lower glacio–marine sequence). Undifferentiated Permian rocks at EQ 530520 indicated (Pu).
- Angular unconformity.

DEVONIAN–SILURIAN–ORDOVICIAN (?)

- Quartzwacke turbidite sequence (SDs). Contact metamorphic psammite, spotted pelite and minor schist (SDsm).

PARMEENER SUPER–GROUP

MATHINNA BEDS

IGNEOUS ROCKS

TERTIARY

- Alkali basalt (Tba), tholeiite (Tbt), olivine nephelinite (Tbn).

CRETACEOUS

- Appinite (Ca) with andesitic lava flow (Cal) and Lamprophyre dykes (Cal) indicated.

JURASSIC

- Dolerite.

MINOR GRANITE INTRUSIONS

- Quartz–feldspar porphyry (Dmp), aplite (Dma), quartz–muscovite rocks (Dmq), fine–grained granite/ adamellite (Dmg), coarse–grained granite (Dmd), pegmatitoid quartz–feldspar porphyry with minor medium–grained dolerite of unknown age (Dmgp), quartz–orthoclase– tourmaline pegmatite (Dmc).

MAJOR GRANITE INTRUSIONS

DEVONIAN–CARBONIFEROUS

SCOTTSDALE BATHOLITH

- Biotite–hornblende granodiorite.

BLUE TIER BATHOLITH

- Porphyritic, fine – to coarse – grained garnet – bearing biotite – muscovite granite / adamellite (Dbag).
- Dominantly fine – to medium – grained granite/adamellite (Dbau); equigranular, dominantly medium – grained, biotite – variable muscovite granite/adamellite (Dbaem); porphyritic fine – to medium grained biotite–muscovite (Dbaeg, Dbagf) and biotite – minor muscovite (Dbagc? Dbagf?) granite/adamellite (() with phenocrysts of feldspar and rounded quartz (Dbapg, Dbapg') (()) with phenocrysts of feldspar (Dbagf), sparse phenocrysts (Dbagf')
- Porphyritic, coarse–grained biotite and biotite – minor muscovite granite/adamellite (Dbapc), sparsely porphyritic to equigranular (Dbapc).
- Biotite – hornblende granodiorite (Dbg–near EQ890735), biotite granodiorite (Dbb). Areas of float and small outcrops of granodiorite and hornfelsed Mathinna Beds indicated by decoration on Dbb, Dbg.

- Geological boundary – position approximate.
- Geological boundary – inferred.
- Fault – inferred.
- Strike and dip of beds; right way up, overturned, facing unknown.
- Vertical bedding; facing known, facing unknown.
- Strike and dip of early cleavage, Vertical.
- Strike and dip of late crenulation cleavage, Vertical.
- Direction and plunge of minor synclinal fold hinge – line with direction and dip of axial surface.
- Strike and dip of foliation due to mineral alignment in igneous rocks, vertical foliation, horizontal foliation.
- Strike of apparent lineation in outcrop surface of igneous rocks
- Strike and dip of joint, vertical joint.
- Field station for adjacent readings on map.
- Bore–hole with depth of rock – type encountered and final depth.
- Palaeocurrent directions.
- Alluvial workings.
- Abandoned alluvial workings.

- Road
- Vehicular track
- Trigonometric station

AP 1/80

EL 2/77

UNIVERSAL GRID REFERENCE

GRID ZONE DESIGNATION: 55G

100,000 METRE SQUARE IDENTIFICATION: EQ

TO GIVE A STANDARD REFERENCE ON THIS SHEET TO NEAREST 1,000 METRES

SAMPLE POINT: At MT CAMERTON

1. Read letters identifying 100,000 metre square in which point lies: EQ
2. Locate first VERTICAL grid line to LEFT of point and read LARGE figure labelling the line in either the top or bottom margin: 79
3. Estimate tenths from grid line to point: 5
4. Locate first HORIZONTAL grid line BELOW point and read LARGE figure labelling the line in either the left or right margin: 82
5. Estimate tenths from grid line to point: 6

SAMPLE REFERENCE: EQ795826

If reporting beyond 10° in any direction prefix grid zone designation, ex: 55GEQ795826

IGNORE the SMALLER figures of any grid number; these are for finding the full co–ordinate, Use ONLY the LARGER figures of the grid number; Example: 120mE

HORIZONTAL DATUM: Australian Geodetic Datum 1966.
GRID: Black numbered lines are 10 000 metre intervals of the Australian Map Grid, Zone 55.
NOMENCLATURE: Topographic names on this sheet have been approved by the Nomenclature Board of Tasmania.
PROJECTION: Transverse Mercator Projection.
VERTICAL DATUM: Australian Height Datum.
MAGNETIC VARIATION: For the centre of this sheet approximately 13°30'E. Annual change +3'E.

SCALE 1:50 000

CONTOUR INTERVAL 20 METRES

5 cm

KILOMETRES 1 0 1 2 3 4 5 6 7 KILOMETRES

MILES 1 0 1 2 3 MILES

RESPONSIBILITY DIAGRAM

Geology by P. W. Baillie, B.Sc.; N. J. Turner, B.Sc. (Hons); S. F. Cox, B.Sc. (Hons); with acknowledgements to D. J. Jennings, B.Sc. (Hons) and F. L. Sutherland M.Sc.
Base map redrawn from Cape Portland 1 : 100 000 map, produced by Lands Department, Hobart.
Geological map production by Regional Mapping Section of the Cartographic Drawing Office, Department of Mines, Hobart.
Cartography by C. Meach, H. Mackinnon.
E. Williams, B.Sc. (Hons), Ph.D., F.G.S., Supervising Geologist in charge of Regional Mapping.
I. S. Jennings, B.Sc. (Hons), Chief Geologist.
Compiled under the direction of A. G. Symons, B.E., Director of Mines.
Issued under the authority of the Minister for Resources and Energy.
Published 1979

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LOCATION MAP

ADJOINING SHEETS

		BANKS STRAIT
NOLAND BAY	BOOBYALLA	EDDYSTONE
PIPERS RIVER	RINGAROOMA	BLUE TIER

BOOBYALLA
SHEET 8416S (24)

464

207072

SHEET 32 FIRST EDITION 1977

RINGAROOMA

GEOLOGICAL SURVEY OF TASMANIA DEPARTMENT OF MINES — HOBART

GEOLOGICAL ATLAS 1:50,000 SERIES 8415 N

EL 2/77

EL 28/76

REFERENCE

CAINOZOIC — QUATERNARY

- Qa — River alluvium, swamp and marsh deposits; river terrace deposits (Qar).
- Qw — Windblown sand.
- Qt — Talus.
- Qm — Marine terrace: sand with shells, clay, organic material.

CAINOZOIC — TERTIARY

- Tb — Eroded surface. Basalt (Tb).
- Tv — Gravel sand, clay and mud (Tv); carbonaceous and micaceous clay with fossil tree fragments (Tvf).
- Tg — Agglomerate and tuff (Tg).
- Tf — Ferricrete (Tf).
- Ty — Grey-billy and silcrete (Ty).

(Ferricrete,silcrete and gravel tags indicated by symbol with asterisk (*).)

- Tm — Conglomerate with clasts of Mathinna Beds (Tm).

Low angle unconformity.

PALAEOZOIC MESOZOIC — DEVONIAN – SILURIAN – PERMO ORDOVICIAN(?) TRIASSIC

- PL — Conglomerate and sandstone. } PARMEENER SUPER – GROUP
- Unconformity.
- SDs — Quartzwacke turbidite sequence (SDs).
- Dmm — Contact metamorphic pommmite, spotted pelite and minor schist (SDsm). } MATHINNA BEDS

Black stippled overprint indicates rock disturbed.

IGNEOUS ROCKS

(Very coarse grain > 30 mm; coarse grain 5 - 30 mm; medium grain 1 - 5 mm; fine grain < 1 mm).

TERTIARY

- Tb — Basalt, alkali basalt (Tba); olivine nephelinite (Tbn); limburgite (Tbl); hawaiite (Tbh); basinite (Tbb); nodular basalt (Tbx); Quaternary basalt – derived lag deposits (Tb').
- Tbp — Porphyritic (pyroxene – rosettes) basalt.

LATE DEVONIAN(?) – CRETACEOUS(?)

- DGd — Dolerite dykes (DGd).

UPPER DEVONIAN – LOWER CARBONIFEROUS

MINOR GRANITIC INTRUSIONS

Quartz-feldspar porphyry (Dmp); aplite (Dma); fine-grained granite/adamellite,equigranular (Dmgr), porphyritic (Dmg); garnetiferous quartz-feldspar porphyry (Dmgg); quartz-orthoclase pegmatite (Dmc).

MAJOR GRANITIC INTRUSIONS

SCOTTSDALE BATHOLITH

UPPER DEVONIAN – LOWER CARBONIFEROUS

- Dsa — Biotite adamellite (Dsau).
- Dsc — Coarse-grained, pink, biotite adamellite (Dsac).
- Dsm — Medium-grained, pink, biotite adamellite (Dsam).
- Dsh — Coarse-grained, pink, hornblende-biotite adamellite (Dsah).
- Dsa — Medium - to coarse - grained, sparsely porphyritic, white biotite adamellite (Dsaw).
- Dsg — Biotite-hornblende granodiorite (Dsg).

BLUE TIER BATHOLITH

UPPER DEVONIAN – LOWER CARBONIFEROUS

- Dbaa — Equigranular, fine - to coarse - grained, biotite-muscovite granite/adamellite (Dbaa).
- Dbap — Porphyritic, fine-to medium - grained, biotite-muscovite granite/adamellite (Dbap); (i) with phenocrysts of feldspar and rounded quartz (Dbapp);
- Dbapf — (ii) with phenocrysts of feldspar (Dbapf) – locally coarse-grained porphyritic (Dbapp).
- Dbapm — Porphyritic, coarse-grained biotite and biotite-minor muscovite granite/ adamellite (Dbapc); coarse - to very coarse - grained variety (Dbapcc); variety with >2% muscovite (Dbapcm).
- Dbag — Biotite-hornblende granodiorite (Dbg).

Areas with poor outcrop indicated by the addition of 's' to the symbol (eg. Dbapcs).

Geological boundary — observed, dip shown by tick where measured.

Geological boundary — transitional.

Geological boundary — position approximate.

Geological boundary — inferred.

Strike and Dip of beds – right way up. Facing known from sedimentary features.

Strike and dip of overturned beds. Facing known from sedimentary features.

Strike and dip of beds–facing unknown.

Vertical bedding.

Horizontal bedding.

Strike and dip of sandstone cleavage and slaty cleavage; crenulation cleavage.

Trace of axial surface of major anticline hinge.

Direction and plunge of minor fold hinge with direction and dip of axial surface.

Direction and plunge of minor early fold hinge with direction and dip of axial surface.

Fold vergence dextral, none, sinistral viewing down plunge.

Trend of apparent mineral lineation in horizontal surface in igneous rocks.

Strike and dip of foliation due to mineral alignment in igneous rocks.

Vertical foliation.

Horizontal foliation.

Igneous compositional layering.

Strike and dip of joint.

Vertical joint.

Strike and dip of dyke: / Vertical dyke: / Strike and dip of vein: / Vertical vein: / Dyke dip indeterminate: — quartz (q), aplite (Dma), quartz-feldspar porphyry (Dmp), fine-grained granite/adamellite (Dmg).

Field station for adjacent readings on map.

208 m Dg 210 m — Borehole with depth of rock type encountered and final depth.

Palaeocurrent directions.

Mine.

Abandoned mine.

Quarry prospect.

Abandoned quarry.

Alluvial workings.

Abandoned alluvial workings. — Kaolin (Ck), Gold (Au), Gravel (Gr), Tin (Sn).

Road.

Vehicular track.

Railway.

Trigonometric station.

UNIVERSAL GRID REFERENCE

GRID ZONE DESIGNATION 55G

100,000 METRE SQUARE IDENTIFICATION: EQ

IGNORE the SMALLER figures of any grid number: these are for finding the full co-ordinate. Use ONLY the LARGER figures of the grid number. Example : 450000

TO GIVE A STANDARD REFERENCE ON THIS SHEET TO NEAREST 1,000 METRES

SAMPLE POINT: 497 E, 497 N — 7720497

1. Read letters identifying 100,000 metre square in which point lies
2. Locate first VERTICAL grid line to LEFT of point and read LARGE figure labelling the line in either the top or bottom margin.
3. Estimate tenths from grid line to point.
4. Locate first HORIZONTAL grid line BELOW point and read LARGE figure labelling the line in either the left or right margin.
5. Estimate tenths from grid line to point.

SAMPLE REFERENCE: EQ497N4

If reporting beyond 10° in any direction prefix grid zone designation, ie: 55EQ497N4

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NOMENCLATURE: Topographic names on this sheet have been approved by the Nomenclature Board of Tasmania.
PROJECTION: Transverse Mercator Projection.
VERTICAL DATUM: Australian Height Datum.
MAGNETIC VARIATION: For the centre of this sheet approximately 12°30'E. Annual change +2'E.

5 cm

KILOMETRES 1 0 1 2 3 4 5 6 7 KILOMETRES

MILES 1 0 1 2 3 MILES

SCALE 1:50 000

CONTOUR INTERVAL 20 METRES

RESPONSIBILITY DIAGRAM

Geology by A. V. Brown, B.Sc. (Hons); M. P. McClenaghen, B.Sc. (Hons), Ph.D.; W. R. Moore, B.A., M.Sc.; N. J. Turner, B.Sc. (Hons).; J. McClenaghen, B.Sc. (Hons), Ph.D.; P. R. Williams, B.Sc. (Hons); P. W. Baillie, B.Sc.; K. D. Corbett, B.Sc. (Hons), Ph.D.; E. B. Corbett, B.Sc. (Hons); E. F. Cox, B.Sc. (Hons); D. I. Groves, B.Sc. (Hons), Ph.D.; S. P. Pike, B.Sc. (Hons).
Base map enlarged and redrawn from Ringarooma 1:63 360 map, produced by Lands Department, Hobart.
Geological map production by Drawing Office, Department of Mines, Hobart.
Cartography by R. Mackinnon.
E. Williams, B.Sc. (Hons), Ph.D., F.G.S., Supervising Geologist in charge of Regional Mapping.
I. B. Jennings, B.Sc. (Hons), Chief Geologist.
Compiled under the direction of J. G. Symons, B.E., Director of Mines.
Issued under the authority of The Honourable E. D. Chisholm, Minister for Resources and Energy.
Published 1977.

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LOCATION MAP

ADJOINING SHEETS

WOLAND BAY	BOOBYALLA	EDDYSTONE
PIPERS RIVER	RINGAROOMA	BLUE TIER
LAUNCESTON	ALBERTON	ST. HELENS

RINGAROOMA
SHEET N°32

689

207073