

TNT MINES LIMITED
Aberfoyle Project
Project Reports Review



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1. EXECUTIVE SUMMARY

TNT Mines Limited (TNT) requested that GR Engineering Services (GRES) undertake a review of existing studies completed for the tin and tungsten prospects located on the east coast of Tasmania.

1.1 Exploration Targets and Recoveries

A summary of the exploration targets was provided by TNT and is included in Table 1.

Exploration Target	Approximate Tonnage	%Sn	Sn Recovery	%WO ₃
Coarse Jig	570,000	0.14	32.00	
Slimes	198,000	0.36	40.00	0.35
Great Pyramid	3,130,000	0.22	50.00	
Lutwyche Surface	500,000	0.59	72.00	0.25
Aberfoyle Open Pit	5,500,000	0.20	60.00	0.03
Lutwyche Underground	1,080,000	0.70	72.00	0.30
Kookabarra	600,000	0.59	72.00	0.25
Royal George	800,000	0.34	90.00	
Storey's Creek	70,400	0.05	60.00	0.59

Table 1 Concentrate Grade and Recovery Anticipated for Exploration Targets

There is very little information within the provided reports to support the recoveries stated in Table 1, however there are a significant number of historic reports (as noted in Appendix A), that need to be reviewed as part of the next phase to confirm processing metallurgical recoveries and likely concentrate grades. A high level overview of some of these reports has provided some details and the following must be noted:

- All assays conducted on concentrates to date appear to only include Sn and WO₃, assays for concentrates need to include for impurities also ensuring the concentrate is saleable and if any penalties / credits may apply for other minerals in the concentrate. Full assays of concentrates from testwork is required to gain a clear understanding of the flow sheet required to produce a saleable concentrate;
- Traceability of the samples tested to demonstrate that samples are representative of even a preliminary mining plan is not able to be established on the data reviewed to date. It will be important to establish testwork sample representivity for testwork to be conducted in the future and for any testwork that has been generated to date that is to be used in the PFS;
- Understanding the content of sulphur in all ore sources is important from the point of view of concentrate grades, additional processing steps required and also for sulphide tailings storage required as acid forming tailings will need to be stored in a specifically designed facility;

1.2 Operating (OPEX) and Capital Cost Estimate (CAPEX) Review

GRES has conducted a review of the capital cost estimates and the following bullets provide a summary of the findings:

- The accuracy level of costs stated in the conceptual reports was -20%, +50%, 3Q07, based on the information provided and likely flow sheet GRES would describe the costs as -0%, +40%, 1Q14 at best, based on the described battery limits, target concentrate grades and recovery for both CAPEX and OPEX;
- The conceptual study flow sheet short falls as pointed out in Devlure Pty Ltd report contribute a significant increase in capital cost for the process facilities and would consume more than the contingency allowance included in the conceptual study estimates;
- The conceptual study flow sheet does not include for separate treatment of the WO₃ concentrate as Great Pyramid does not contain WO₃, this will require additional process steps including potentially flotation, thickening and filtration of WO₃ concentrates;
- GRES estimate the capital costs required for a full process plant to be in the order of \$AUD70m (±25%) including all infrastructure required to run the plant and dispose of tailings but excluding mining infrastructure. A gravity only plant would see this capital reduce to the order of \$AUD47m (±25%) for 500ktpa plant;
- GRES estimate that a gravity only OPEX of \$AUD20 - \$AUD22 (±25%) and a full plant OPEX of \$AUD30 - \$AUD33 (±25%) is the order of magnitude for a 500ktpa plant;
- Mining CAPEX and OPEX is for open pit, not underground which needs to be updated.

1.3 Recommendations

GRES recommend the following steps to be taken:

- Conduct a review of the historic reports and summarise the findings in terms of potential recovery and treatment flow sheet required on each exploration target;
- Confirm from the review of the historic data, the sequence of exploitation of the exploration targets based on flow sheet requirements and estimated cash flows;
- Document testwork data already available including suitability and representivity for the planned sequence including traceability of samples used in testwork;
- Source representative samples to conduct new testwork on where historic information is not complete;
- Engage a mining contractor to undertake preliminary mining plans and costing of mining capital and operating costs for the different exploration targets including confirming transport rates from Great Pyramid;
- Complete recommendations from Vince Algar's report on exploration targets;
- Produce a PFS documenting the findings from the above bullet points.



2. INTRODUCTION

2.1 General

TNT Mines Limited (TNT) requested that GR Engineering Services (GRES) undertake a review of existing studies completed for the tin and tungsten prospects located on the east coast of Tasmania.

This report represents stage 1 of GRES' two stage proposal to TNT, which included a review of existing reports, information and give feedback on suitability of the process flow sheet and the accuracy level of capital cost estimates included in the studies.

The exploration targets span a number of locations in the Rossarden / Stories Creek region on the east coast of Tasmania. TNT is considering the viability of economic development of a mine and process plant that is able to accept feed from multiple sites for recovery of metal concentrates.

2.2 Previous Reports to be Reviewed

The following reports were included in the review conducted by GRES:

Description	Report No	Date	By
Review of Conceptual Studies		Nov 12	Devlure
Great Pyramid Tin Project Conceptual Study	1406-STY-001	Oct-07	Lycopodium
Anchor Tin Project Conceptual Study	1406-STY-002	Oct-07	Lycopodium
Aberfoyle Project, Exploration Potential		Jan 2014	Vincent Algar

Table 2 List of Reports to be Reviewed (Stage 1)

This report presents the following:

- Commentary on the selected process flow sheet and recommendations;
- Commentary on the level of accuracy and current value of the conceptual study capital and operating costs, including any impact from process flow sheet recommendations;
- Single page block flow diagram of the recommended flow sheet

TNT have indicated that the exploration targets as shown in the report by Vincent Algar are to be the focus of the PFS, meaning that the Conceptual Study on the Anchor Tin Project should only be noted and not commented on.

GRES have incorporated some comments associated with a high level review of some of the reports include in Appendix A, a more detailed review is required in the next stage of development.

3. PROCESS REVIEW

3.1 Introduction

The conceptual studies completed by Lycopodium were for hard granitic ore feed from the Great Pyramid and Anchor prospects located approximately 65 km from the Storeys Creek and Aberfoyle Mine sites (the proposed site of the TNT process plant and associated facilities) and were based on a study of the processing of mineralised material from the Anchor Mine undertaken in 1983 by Messieurs S Cross and W Selby of Renison Limited titled "The Metallurgical Treatment of Blue Tier Mineralisation". The Anchor Project was ultimately developed and commissioned in 1989 by the Blue Tier J.V. (Spectrum Resources Australia Pty Limited and Nargun Pty Limited) – mining activities at and around the Anchor Mine had been conducted from 1876 to 1914 and from 1939 to 1945. The project was closed down in 1991. TNT however is considering the processing of the materials outlined in Table 3 - the Anchor deposit has not been included.

The TNT strategy concept was for the initial re-treatment of the heavy media rejects and jig tailings generated from previous operations that treated the Aberfoyle and later, Storeys Creek ores and which ceased in 1981 – Storeys Creek ore was treated at the Aberfoyle plant after the Storeys Creek plant was shut down in December 1971. The jig tailing had been discharged into the creek, along the banks of the creek and ultimately, with the heavy media rejects, in a dry stack with an estimated 489,000 tonnes of plant discharge material. There is an estimated 81,000 tonnes of coarse jig tailing stacked at the Storeys Creek plant site 3 km north west of the Aberfoyle site.

Feed Source	Estimated Tonnage	Estimated Grade %Sn	Estimated Grade %WO ₃	Estimated Recovery %
Coarse Jig Tailing	570,000	0.14		32
Slimes Dams	198,000	0.36	0.35	40
Great Pyramid	3,130,000	0.22		50
Lutwyche Surface	500,000	0.59	0.25	72
Aberfoyle Open Pit	5,500,000	0.20	0.03	60
Lutwyche Underground	1,080,000	0.70	0.30	72
Kookaburra	600,000	0.59	0.25	72
Royal George	800,000	0.34		60
Storeys Creek	70,400	0.05	0.59	60

Table 3 Feed Sources Under Consideration by TNT

The remainder of the plant tailings from the Aberfoyle treatment plant was discharged into a total of six "slimes" dams, five of which have been investigated for re-treatment – early operations disposed of tailings into the creeks (Aberfoyle Creek, Side Creek and Storey's Creek) and unconfined onto adjacent land. The mineralised veins of the Lutwyche zone lie 1 km north-north-east from the main shaft of the



Aberfoyle Mine while the Kookaburra lode lies between Lutwyche and the Spiers Shaft of the Aberfoyle Mine.

The Great Pyramid deposit has a significant estimated tonnage of low grade tin mineralisation and lies approximately 75 km to the north east of the Aberfoyle Mine. Due to the low grade, historical production (1925 to 1936) was consequently small (336 tonnes ore).

The Royal George deposit was worked from 1911 until 1922 and is located approximately 25 km south east of the Aberfoyle Mine.

Given the geographical locations of the prospects, a centralised plant at the Aberfoyle Mine allows for treatment of all the potential resources although transporting low grade mineralised material may not be economic and some form of pre-concentration at specific sites or a modular, transportable plant are also options for consideration. The different mineral combinations (for both valuable and gangue – particularly sulphide - minerals) that occur in the 'satellite' deposits indicate that some process stages may not be required for all feed sources. This could be allowed for with a staged approach to treatment depending on the order of treatment and the economic sustainability for each stage – for example, initial re-treatment of tailings by screening to reject coarse material followed by several stages of gravity recovery including regrinding of middlings may not be financially sustainable which would obviate the addition of more processing stages.

The subsequent addition of flotation to reject sulphides, or heavy media separation to reject a high proportion of mass from hard rock or run of mine material may need to be implemented at an early stage which would be an interference and distraction to existing operations.

Alternatively, a heavy media module could be used at a satellite deposit such as Great Pyramid which the pre-concentrate transported to the Aberfoyle site to supplement or follow the tailings re-treatment.

Remediation and rehabilitation activities have been undertaken at the Aberfoyle and Storey Creek sites to mitigate the effects of acid and heavy metal contamination of land and water which has affected water supplies and fishing as far as the South Esk River.

Treatment of material over the past 30 years, in particular the tailings areas with limestone and other chemicals, may have effects on the processes used in any proposed concentrator that were not evident in previous test work.



Historical metallurgical test work that has been carried out on the proposed feed sources includes the following which have been used by GRES for this review, refer Table 4:

Description	Report No	Date	By
Preliminary Report on Tin – Tungsten Mineralization in ML 27M/77		Mar-83	T. G. Summons
The Shell Company of Australia Ltd Metals Division E.L. 10/80 – Great Pyramid Progress Report on Exploration During the Period 3/8/82 to 1/9/83	08.2065	14-Sep-83	P.A. Ruxton
Concentration Tests and other Studies on Tin Ore, Great Pyramid Prospect, Upper Scamander for the Shell Co of Australia	R814	3-Mar-83	H.K. Wellington & P.L. James
The Metallurgical Treatment of Blue Tier Mineralisation (Renison Limited)	RL8713	14-Feb-83	S. Cross & W. Selby
Tin & Tungsten Distribution in Coarse Tailings From The Aberfoyle Mine at Rossarden and a Recovery Test Incorporating Grinding of a Selected Size Range of the Tailings (Department of Mines – Tasmania, Launceston)	R829	27-May-83	H.K. Wellington & L.J. Rhodes
Rossarden Tin Mine: An Investigation of the Open Cut Potential of the Rossarden Tin Mine, North-Eastern Tasmania (Stackpoole Enterprises)	RL8808	Jun-Jul-89	C. Roberts & M. Teh

Table 4 Historical Reports Used in Process Review

Many of the historical reviews and reports question the representivity of the samples collected for testing either based on the method of sampling, the location of the samples, the difference between assayed and reconciled head grades with the resource estimate for grade and sometimes the difference between the calculated and assayed head grade.

The assays available have concentrated on tin with little or no assaying of other elements (sulphur, arsenic, iron, copper, lead, molybdenum) or compounds (silica, alumina).

3.2 Tailings

3.2.1 Historical Flowsheets

Bulk tin and wolfram concentrate was generated by gravity separation using jigs. Batch flotation was introduced in 1940 to 1941 to remove sulphides. Magnets were used to separate the bulk concentrate into respective tin and wolfram products. Annual production for 1940 to 1941 was 323 tons of tin concentrate and 26 tons of wolfram concentrate generated from 16,000 tons of ore grading 1.534% Sn. The mill operated 10 hours per day, 5 days per week. Mill performance for January to March of 1942 is shown in Table 5.

Stream	Mass %	Tin Grade %Sn	Tin Recovery %	Tungsten Grade %WO ₃
Feed	100	1.988	100	?
Primary Gravity Tail	89.9	0.17	7.7	
Secondary Gravity Tail	6.9	1.20	4.2	
Gravity Concentrate	3.2	54.7	88.1	
Flotation Concentrate 1	0.3	2.00	0.3	
Flotation Concentrate 2	0.2	2.40	0.2	
Magnetic Separation WO ₃ Magnetics	0.4	0.70	0.1	72.5
Magnetic Separation Tin Non-Magnetics	2.33	74.64	87.48	

Table 5 Mill Production Statistics 3 Months 1942 (from M. Mawby & P. Nye)

In 1978, the processing plant at the Aberfoyle Mine had a flowsheet (simplified) as shown in Figure 2.1. The feed was crushed to less than 25 mm and screened at 2 mm. The coarse material (approximately 84% of the feed mass was plus 2 mm) passed through a heavy media separation bath (a conventional Atkins screw classifier converted for use as a heavy media bath) operating with a medium density of 2.54 achieved using ferrosilicon. The rejects (floats) from the heavy media bath comprised the quartz gangue while the cassiterite and wolfram minerals reported into the sinks fraction. Despite the low specific gravity of the medium relative to the density of quartz, separation of the minerals was possible due to the lower settling rate of the silica particles. 60% of the feed mass reported to the floats reject stream with a loss of 3% of the cassiterite and 4% of the wolframite. The fines (minus 2 mm) contained just over 50% of both the tin and tungsten in 16% of the feed mass – the fines assayed 1.3% Sn and 1.3% WO₃ compared to the plant feed grade of 0.4% Sn and 0.41% WO₃.

Previous to the heavy media circuit installation, jigs were used to reject the quartz from the coarse fractions. Jigs were also used in 1978 after desliming, to concentrate the heavy minerals prior to upgrading on shaking tables.

The heavy media circuit floats and tailings from the jigs were discharged concurrently. Given the change in flowsheet from jig only tailing to heavy media floats and jig combined tailing later, it is likely that there



will be size distribution and grade variations in the material reclaimed from the coarse jig tailings pile and that consideration to some form of blending or mixing during the method of reclaim may be required. Alternatively, screening to remove all plus 9 mm material that may be barren. Note that the Storeys Creek pile was generated from jigs only.

3.2.2 Test Work Review

Details of testing on material from the coarse tailings were reported by H. Stackpoole and T. Summons in 1985 (Wheal Lutwyche Pty Ltd, Annual Report for the Year Ended 5th June 1985).

3.3 Coarse Jig Tailings

3.3.1 Historical Flowsheets

Bulk tin and wolfram concentrate was generated by gravity separation using jigs.

3.4 Great Pyramid

3.4.1 Introduction

Metallurgical assessment of the Great Pyramid deposit near St Helens was undertaken by several organisations and the findings summarised in a report by D. Hall and B. Carter in January 1986 titled "Great Pyramid Tin Deposit North East Tasmania Resource Estimate", report number 08.3157. It included results from the following investigations:

"Preliminary Metallurgical Investigation of Tin Ore Deposit from Tasmania – Great Pyramid", report number 16.020.1/1 for The Shell Company of Australia Limited Metals Division, by Mineral Deposits Ltd (MDL), 1984

"A Further Study on Cassiterite Ore in Middling Products from Test Work Report 16.020.1/1", report number 16.020.1/2 for The Shell Company of Australia Limited Metals Division, by MDL, July 1984

Earlier test work carried out on the Great Pyramid prospect by the Department of Mines – Tasmania (DMT) for the Shell was included in the "Progress Report on Exploration During the Period 3/8/82 to 1/9/83" compiled by P. Ruxton (report number 08.2065):

"Concentration Tests and other Studies on Tin Ore, Pyramid Prospect, Upper Scamander", report R814 for The Shell Company of Australia, by the Department of Mines – Tasmania (H. Wellington & P. James), 3rd March 1983.

The low grade nature of the deposit, low tin prices and the requirement to obtain a quota for production of tin all contributed to the lack of development of the Great Pyramid prospect.

Despite the extensive metallurgical testing completed, no flowsheet was confirmed due to the relatively fine grind size (-600 μm to - 400 μm) that was indicated as being necessary, the large losses of tin in slimes rejects and difficulty in balancing the results (as noted by Mr Kevin Foo from Aberfoyle Limited in his review dated 12 November 1985).

3.4.2 Mineralogy

Mineralisation occurs in veins and veinlets at joints and fractures in the hard host rock which comprises beds of sandstones, siltstones and shales which have been intruded by granite and dolerite. The sandstone is silicified and is equivalent to quartzite. The host rocks are therefore competent and comments were made in test reports alluding to the hard nature of the material which will require high energy input into size reduction stages. Therefore, rejection of barren material at a coarse size needs to be a major consideration in process design. The low grade (0.22% Sn), the lack of mineralisation in the host rock, and the relatively long distance from the other potential mine sites also support reduction in mass during processing as soon as practicable – ore sorting, rejection of coarse rock by screening and heavy media separation are possible process stages for a treatment facility located at the Great Pyramid site.

The mineralised fractures and veins which are fine with variable widths up to 5 mm wide contain quartz, muscovite, tourmaline, iron oxides (limonite or hematite), cassiterite and traces of wolframite, fluorite and pyrolusite. Sulphide minerals (pyrite, molybdenite) also occur at depth and have been converted to iron oxides in the upper oxidised zone. The cassiterite is often associated with the sulphides and wolframite. Contamination of concentrates with sulphide minerals may therefore be a potential problem requiring a sulphide flotation stage to remove.

The cassiterite is generally coarse grained (>100 μm) with some particles up to 600 μm in size being present. The Department of Mines - Tasmania report R814 however noted that cassiterite was present as clusters of grains which ranged in size from 5 μm to 30 μm . Size reduction stages should therefore target small reduction ratios and grinding mills closed with screens rather than cyclones to minimise overgrinding and shattering of the clusters as recovery from the fine size fractions are typically low.

The DMT report R814 also noted that there was a significant proportion (25% to 40%) of material in the samples investigated that was sericitic – fine grained muscovite and mica with particle sizes of 5 μm to 10 μm . These mineral fines tend to have high surface charges and contribute to viscosity effects that reduce the efficiency of separation in hydrosizers, heavy media circuits and flotation.

3.4.3 Test Work Review

Detailed testing was carried out by the DMT and MDL that indicated the samples were amenable to pre-concentration and tin concentrate could be produced at reasonable grade (>40% Sn) and recovery (>60%) when the head grade was high (>0.4% Sn) but that recoveries were low (50%) when the head grade was similar to the estimated reserve (0.22% Sn). Test results are summarised in Table 6.



Year	Source	Sample	Head Grade %Sn	Conc Grade %Sn	Tin Recovery %	Conc Mass %	Test Conditions / Flowsheet	Tin Distribution
1945	DMT		0.64	22	78.6		Grind to -420 µm, gravity	
			0.64	27	80.6		Grind to -250 µm, gravity	
			0.64	27	80.2		Grind to -180 µm, gravity	
			0.64	11	71.8		Flotation	
			0.64	6.4	81.8		Flotation	
1963	DMT	550 kg	2.17	61.6	79.26		Grind, gravity	
			2.17	59.7	84.55		Included regrind of +250 µm tail	
			2.17	59	87.05		Included regrind of +150 µm tail	
			0.76	50.5	73.81		Grind, gravity	
			0.76	48.9	77.25		Included regrind of +250 µm tail	
			0.76	46.9	79.76		Included regrind of +150 µm tail	
			0.47	46.9	60.59		Grind, gravity	
			0.47	43.8	66.61		Included regrind of +250 µm tail	
			0.47	41.3	71.37		Included regrind of +150 µm tail	
1971	Aberfoyle	73 kg	0.296	47.6	63.5		Gravity, magnetic separation	32% -850 µm +500 µm
		drill chips						33% -500 µm +300 µm
1979	Aberfoyle		0.63	28.9	86.0	1.9	Sinks HLS @ 2.96 sg -600 µm +45 µm	
			0.60	38.13	63.7	1.0	Gravity	
			0.19	5.6	68.7	2.3	Sinks HLS @ 2.96 sg -600 µm +45 µm	
			0.20	7.9	13.7	0.34	Gravity	
1983	DMT	50 kg Pilot	0.22	52.9	54.5			54% -300 µm +125 µm
	for Shell	546.7 kg	0.22	1.3	73		Sinks HLS @ 2.95 sg	47% -300 µm +75 µm
1984	MDL		0.414	48	43		Hydrosizer, Spiral, Table, Mag Sepn	
	for Shell						-400 µm grind	

Table 6 Summary of test results for the Great Pyramid deposit



The pre-1979 test work was carried out on very high grade samples and returned recoveries of 65% to 85% using standard gravity separation (jigs, spirals, tables) at generally reasonably fine feed sizes (P_{80} of less than 400 μm) and flowsheets that included regrind stage/s. Despite the fine grinds, concentrate grades were relatively low (less than 45% Sn) which probably reflects contamination by other high specific gravity minerals and misreporting fine gangue.

The size – assay results from samples crushed at the DMT to less than 8 mm indicates that the tin is distributed fairly well across all size fractions – no size – assay data were presented by MDL. HLS tests on individual size fractions indicate that liberation of the tin minerals is not sufficient until a grind size of about 600 μm . The DMT results (1983) for material minus 1.24 mm rejected 87% of the mass for a loss of 27% of the tin at a separation density of 2.7 t/m^3 , and 98% of the mass was rejected with 29% of the tin at a density of 2.95 t/m^3 . Aberfoyle tests (N. Moony, 1979) on a finer grind (-600 μm +45 μm) rejected 98% of the mass at a density of 2.96 t/m^3 with a loss of 14% of the tin for a high grade sample (0.63% Sn) while the low grade sample (0.19% Sn) had a higher tin loss of 31% with 97.7% of the mass. The floats assayed 0.07% Sn to 0.09% Sn – similar content to gravity tailing.

Therefore, despite the preference to pre-concentrate at as coarse a size as possible due to the inferred hard nature of the material, the results of HLS and size – assay tests indicate that a fine size is needed to effect rejection of feed material without significant loss of tin. Tin losses with the floats reject ranged from 14% to 35%.

The DMT pilot flowsheet did not use heavy medium separation and the 50 kg sample was ground to less than 170 μm . A jig was used to remove a concentrate from the grinding circuit circulating load (0.04% mass pull at a grade of 14.4% Sn and 2.6% tin recovery). A hydrosizer was used to generate two size fractions for separate gravity treatment. The hydrosizer rejected 27% of the overall feed to slimes with a loss of 23% of the tin. In conjunction with the 18% mass and 7% tin rejected by the deslime cyclone (-5 μm), the high proportion of tin lost in fines indicates over-breakage of the cassiterite during size reduction. The final concentrate assayed 53% Sn with 58% recovery from a 0.22% Sn head grade.

The MDL pilot tests used a similar flowsheet to that employed by the DMT in 1983 and again tin losses into the slimes streams (deslime cyclone overflow and hydrosizer overflow) were high – 51% mass rejection with 35% loss of tin. A high proportion of fines is either inherent in the material or has resulted from handling and size reduction methods used. The samples may have been overground however, the vast difference in the fines losses and the effect on ultimate recovery is highlighted by the test result comparisons in Table 7 and emphasises the importance of how samples are obtained (not drill chips or pulps) and how they are handled and prepared. Calculated mass balances for both the DMT and MDL pilot tests are given in Appendix C.



Fines Losses	Mass %	Grade %Sn	Tin Loss %	Head Grade %Sn
DMT 1983	45.1	0.146	30.1	0.22
MDL 1984	51.6	0.25	33.3	0.385
Moony 1979	9.5	0.13	2.1	0.63
Moony 1979	12	0.13	8.0	0.20

Table 7 Fines (nominally -45 μ) rejects from classification stage

3.5 Royal George

3.5.1 Introduction

170,000 tonnes of ore grading 0.65% Sn was mined from the Royal George deposit between 1911 and 1922 to produce over 900 tonnes of tin concentrate that assayed 65% Sn to 70% Sn. This indicates tin recoveries of 52% to 57% were achieved from the relatively high grade plant feed. The tailings dump occupies 36,000 m³ and since 2006 has been remediated as part of the Tasmanian Government programme to reduce acid contamination of water and soil generated by historical mining activity. Drains were installed and crushed limestone and fertiliser were applied to the surface of the dump.

The estimated reserve is 800,000 tonnes at a grade of 0.34% Sn. Based on the lack of test work and limited details of historical performance, a sampling and metallurgical test programme will be needed to progress the justification for including the Royal George deposit in the mine sequence.

3.5.2 Mineralisation

The deposit comprises granite lithologies of different textures but with similar mineralogy. Coarse grained granite has large grains of feldspar up to 50 mm in length and anhedral and subhedral quartz up to 25 mm in size. Fine grained, porphyritic granite contains phenocrysts of feldspar and euhedral quartz in the fine grained matrix. Greisen granite is also present. Biotite, a mica mineral, and tourmaline are present in both granite forms.

The coarse granite will result in liberation of minerals at coarse sizes while the porphyritic granite will require a finer grind size to liberate the minerals present. This indicates a flowsheet comprising several stages of size reduction and gravity separation or use of regrinding of (variable and relatively high tonnage) middlings streams. Coarse granite is likely to be readily upgraded by pre-concentration (ore sorting or heavy media separation) however a much finer feed size will be required for pre-concentration of the porphyritic granite.

To complicate the gravity separation stages, the granites have undergone alteration to sericite and talc with some kaolinisation close to the mineralised zones. Consequently there will be a high proportion of fine particles generated during size reduction stages. Any coarse micaceous particles will tend to block screens and increase circulating loads unless slotted apertures are used – note that these minerals also report as oversize in cyclone overflow giving a biased product size distribution that is coarse and can lead to overgrinding of the cassiterite if corrective actions are taken to reduce the grind size. The fine sericite



and clay will also tend to increase viscosity of pulps thereby affecting the efficiency of classification, gravity separation, flotation and heavy medium separation (by contaminating the medium).

Tourmaline is a black aluminium silicate mineral that tends to report with the gravity concentrate due to its tabular shape despite a low specific gravity (3) relative to cassiterite (specific gravity of 7).

The cassiterite tends to be implanted on the quartz and fills cracks in the quartz veins and greisen quartz. The mineralised veins are variable in width so some head grade variability is likely to occur although the descriptions of the geology state that the tin grades are fairly consistent in the different granite hosts. The wider veins tend to be more altered and silicified – the sericite and clay interferences in processing will be greater when fed to a concentrator.

Sulphide minerals are common in the tin rich veins. Pyrite and chalcopyrite are the main accessory sulphide minerals but arsenopyrite and sphalerite are also disseminated through the greisen host and along fractures. In some areas, the sulphides were reported to comprise up to 10% of the rock. Analysis of the acid mine drainage solutions gave levels of arsenic, copper, lead, manganese, sulphur and tin above 100 ppm indicating that sulphide mineral levels in the plant feed will possibly result in contamination of the tin concentrate and that a sulphide flotation stage will need to be included in the flowsheet.

Oxidised uranium minerals were identified in old workings in 1955. Subsequent drilling in the deposit intersected only two narrow bands of radioactive minerals which assayed 0.01% U_3O_8 to 0.1% U_3O_8 . The mineral was identified as torbernite, a secondary uranium copper phosphate mineral and was present as flakes or speckles on rock surfaces and in joints. Although rare, its presence will need to be monitored in the process and products.

3.5.3 Test Work

No test work on samples of Royal George material was found. Based on the mineralogy, metallurgical performance is expected to be slightly less than that achieved with the Aberfoyle hard rock material. A tin recovery of about 60% may be achieved.



4. FLOW SHEET

4.1 Conceptual Study Flow Sheet

The flow sheet selected in the conceptual study for the Great Pyramid exploration target is based on a gravity (spirals and tables) plant being able to produce a saleable concentrate.

The flow sheet selection for a conceptual study has excluded the potential issues of contaminants in concentrate and sulphides in tailings rather than addressing them. Contaminants, especially sulphides are almost certain to add process, handling and storage steps into the conceptual study flow sheet.

4.2 Proposed Changes and Stages to Investigate

The main points including the points highlighted in the Devlure report that need to be addressed in the PFS development are:

- Review existing and conduct new testwork to investigate the potential for mill feed rate reduction by using ore sorting or heavy media separation on representative samples from each exploration target;
- Ensure comminution (crushing and grinding) circuits minimise the potential for overgrinding (sliming) of minerals which will cause excessive losses to tailings, this includes:
 - Crushing to optimal sizing (tertiary crushing);
 - Rod milling as primary mill rather than ball mill;
 - Screen classification rather than cyclones;
 - Coarse gravity and fine gravity steps which will require the inclusion of a regrind stage;
- Include provision for sulphide removal in either pre-treatment stage or a concentrate cleaning stage or both;
- Include provision for WO₃ concentrate gravity, flotation, thickening and filtration;
- A staged approach is imperative from both a smooth operation perspective and optimisation of the required fines gravity and flotation circuits;
- Representivity of ore samples used in testwork of the first two to three years of operation;
- Confirmation, by new or existing testwork, of recoveries achievable at each of the following process steps for each exploration target to assist in confirmation of staged approach flow sheet:
 - Sulphide pre-treatment where required;
 - Coarse gravity;
 - Fine Gravity;
 - Tin Flotation;
 - Concentrate cleaning (sulphide and/or other impurity removal);

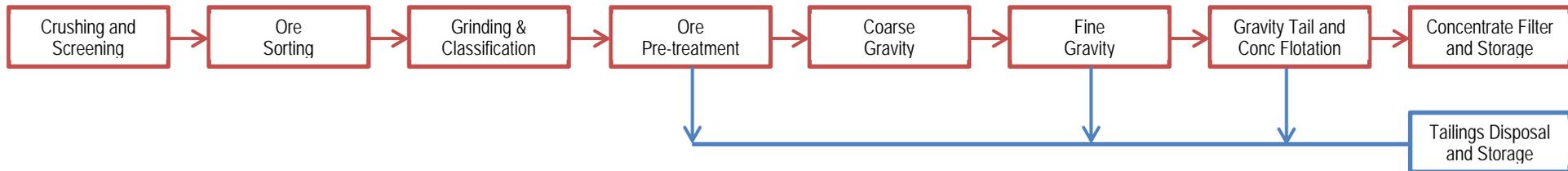


Figure 3.1 Typical Process Flow Block Diagram

Process Stage	Description
Crushing & Screening	Typically a three stage crushing circuit to reach optimum size for ore sorting and feed to grinding
Ore Sorting	Can significantly reduce process costs for the subsequent process steps if rejection of non-mineralised material is significant (>20-30% of mass) with little loss of metal. This is likely to be a significant factor for Great Pyramid and could reduce transport costs.
Grinding & Classification	Rod mill as a primary mill with vibrating or Derrick style classification screen
Ore Pre-treatment	Is potentially needed for high sulphide ores to remove sulphides prior to gravity, depending on liberation sizes regrinding of sulphide concentrates may also be required.
Coarse Gravity	Includes spirals and tables for production of coarse concentrates, some of which (middling) will require regrinding to liberate metals
Fine Gravity	Includes spirals and tables for production of fine concentrates
Gravity Tailings and Concentrate Flotation	Is used to scavenge any remaining Sn and WO ₃ from coarse gravity tails, fine gravity tails and to remove any impurities (such as sulphur) from the concentrate that may incur penalties, separate circuits are required for Sn and WO ₃
Concentrate Filter and Storage	Concentrates for both Sn and WO ₃ will need to be filtered to remove water and stored in an enclosed shed / containers ready for transport.
Tailings Disposal and Storage	Sulphide or PAF (potentially acid forming) tailings will need to be stored in a specifically design storage facility for environmental purposes, this is typically a different (smaller lined facility) to the NAF (non-acid forming) tailings to reduce overall costs of the tailings storage facility.



4.3 PFS Flow Sheet

A typical block diagram is included as Figure 3.1. This is a simplified flow sheet and each process step is not necessarily required for each exploration target.

To effectively assess a staged approach to the project, metal recovery, concentrate grade and impurities at each of the coarse gravity, fine gravity and flotation stages of the flow sheet needs to be understood for each exploration target to ensure saleable concentrate is able to be produced.

4.4 Flow Sheet Requirements by Ore Source

The following section describes the process flow sheet that GRES feel is required for each ore body based on the information reviewed to date and our process knowledge.

4.4.1 *Coarse Jig Tails*

The coarse jig tailings is ore that has been treated by gravity previously and has a low grade. To recover additional metal units the process plant will be relying on better technology or more process steps.

The coarse jig tailings will not require a crushing plant, the material will be reclaimed by front end loader (FEL) and fed into the plant by an feed hopper. Depending on the regrind requirements for this material and if it is blended into a main feed or campaigned separately the material could potentially be fed into a scrubber / repulper for removal of organic material and fed direct into a regrind mill for processing.

The coarse jig tailings material is likely to require regrinding to liberate minerals. Review of existing testwork and additional testwork on representative samples will provide definition as to which of coarse gravity, fine gravity and flotation process steps are required to achieve 32% recovery as stated in the schedule.

It is anticipated that this material as a minimum will require regrinding, fine gravity and concentrate dressing flotation to produce a saleable concentrate.

4.4.2 *Slimes Tails*

The slimes tailing is fine material and has a medium grade. Reclaiming these tailings may require a scrubber / repulper to remove organic material and produce process slurry for treatment. The crushing plant is not required for the slimes treatment.

As the material is fine it is likely that this material will require a flotation stage to achieve the 40% recovery stated in the schedule.



Tungsten minerals in the slimes will also require flotation to recover metal to concentrate; this will be an independent flotation circuit from the tin flotation and will require specialised reagents.

Removal of sulphides from the slimes is likely to require a dressing stage of flotation.

4.4.3 Great Pyramid

Great Pyramid is an open pit exploration target located approximately 75km by road from the Aberfoyle deposit and introduces a requirement to either;

- Transport ore to a plant located at the Aberfoyle deposit; or
- Relocate the process plant at some stage in the mine life to Great Pyramid;

Given the relatively high cost of infrastructure around the process plant including cost of tailings storage facilities, buildings, power and water supply unless there is a significant upgrade in the exploration target grades and/or total tonnage it will be more economical to transport the Great Pyramid material to Aberfoyle where infrastructure and storage facilities will be installed.

A comparison of the costs of crushing and ore sorting of Great Pyramid material in close proximity to the deposit versus the total transport costs of as mined ore should be conducted in the PFS. It is likely that this will be a more expensive option as duplication of the crushing and ore sorting plant will be capital intensive, however if the Great Pyramid exploration target was processed at the start or finish of the mine life this could be a feasible option.

The low grade of the Great Pyramid exploration target does mean that it is a relatively marginal resource and a detailed review of grades (by mining) and recovery (by new and review of exiting testwork) should be made to ensure this target is able to be mined and processed economically.

Due to sulphide mineralisation it is likely that Great Pyramid material will require a complete process plant (excluding the tungsten recovery stages) to achieve economic recovery, which will need to be determined by testwork, but is likely to be in the order of 60-70%. It is possible that this ore could be treated in a gravity only circuit to achieve the 50% recovery stated in the schedule and then gravity circuit tailings processed at a later date through flotation circuits to recover additional metal.

4.4.4 Lutwyche Surface

Lutwyche Surface will require a full process plant (including tungsten recovery stages) to produce a saleable concentrate at 72% recovery. As mentioned for the Great Pyramid deposit this ore body may be able to be processed on a gravity only circuit initially with a compromise on recovery and then process gravity flotation tailings at a later date when mining and processing procedures have settled and cash flow allows for additional expenditure on a flotation circuit.



4.4.5 *Aberfoyle Open Pit*

The Aberfoyle open pit exploration target envelopes existing underground workings which will introduce some significant issues in regards to mining and treatment. The old workings will mean that the crushing plant will be required to handle waste material (wire, timber etc). If this is not removed at an early stage it will end up in mills and gravity circuits and cause process interruptions.

Aberfoyle is low grade and will require a full treatment plant (excluding the tungsten recovery stages) to produce a 60% recovery as stated in the schedule. The grade of tungsten present in the Aberfoyle Open Pit is unlikely to be economically recovered at a grade of 0.03%.

4.4.6 *Lutwyche Underground*

Lutwyche underground will require a full process plant (including tungsten recovery stages) to produce a saleable concentrate at 72% recovery.

4.4.7 *Kookaburra*

Kookaburra will require a full process plant (including tungsten recovery stages) to produce a saleable concentrate at 72% recovery.

4.4.8 *Royal George*

Royal George will require a full process plant (excluding the tungsten recovery stages) to produce a saleable concentrate at 90% recovery.

4.4.9 *Storey's Creek*

Storey's Creek is a tungsten target with the grade of 0.05% tin unlikely to be economically recoverable to a saleable concentrate without blending. Storey's Creek will require a tungsten recovery plant to produce a saleable concentrate.



5. CAPEX AND OPEX REVIEW

The capital and operating costs generated in the conceptual study for the process plant and infrastructure includes for a gravity (spirals and tables) plant to treat Great Pyramid ore and produce a 50% tin concentrate at an overall recovery of 80% tin into concentrate.

The current exploration target schedule provided by TNT includes multiple potential ore sources with variable feed grades of tin and some ore sources that contain tungsten minerals; hence the capital and operating costs developed in the conceptual study are not directly transferable to the required flow sheet for the treatment of all exploration targets.

In addition to the ore source changes, the process flow sheet defined in the conceptual study does not allow for contaminate removal such as sulphides which at best would incur a penalty in concentrate sales and at worst would restrict or prevent concentrate from being sold. This is likely to introduce additional process steps which will have an impact on capital and operating costs depending on the final flow sheet requirements defined by testwork.

5.1 Capital Costs and Operating Costs

A table demonstrating the additional capital and operating costs that would be required between the conceptual study process flow sheet and the potential flow sheet required to enable all exploration targets to be exploited is included in Appendix B. As can be seen from these tables, the tin flotation, tungsten flotation and impurities removal steps have a significant impact (up to 25% on CAPEX and over 50% on OPEX) on costs of the plant.

The plant becomes more complex requiring more operators and the cost of flotation reagents for tin and tungsten minerals are very expensive.

GRES estimate the capital costs required for a full process plant to be in the order of \$AUD70m ($\pm 25\%$) including all infrastructure required to run the plant and dispose of tailings but excluding mining infrastructure. A gravity only plant would see this capital reduce to the order of \$AUD46m ($\pm 25\%$) for 500ktpa plant;

GRES estimate that a gravity only OPEX of \$AUD20 - \$AUD22 ($\pm 25\%$) and a full plant OPEX of \$AUD30 - \$AUD33 ($\pm 25\%$) is the order of magnitude for a 500ktpa plant.



6. RECOMMENDATION AND CONCLUSIONS

6.1 Recommendations

GRES recommend that a detailed review of the historic reports, as contained in Appendix A, is completed and summarise the findings in terms of potential recovery and treatment flow sheet required for each exploration target. The aim of this review is to bring all available data into one place and provide a basis for selecting the sequence of exploitation of the exploration targets based on a staged flow sheet development.

Review the requirements to be able to implement a staged process plant installation, which would include:

- Stage 1 – install crushing, grinding, gravity, concentrate handling and tailings circuits;
- Stage 2 – install flotation circuits for concentrate cleaning and fine metal recovery;

To undertake a staged approach testwork is required to confirm that saleable concentrate (at grade and free from impurities) is able to be produced at each stage which is potentially a limiting factor on some exploration targets.

Document testwork data already available including suitability and representivity for the planned exploitation sequence including traceability of samples used in testwork that has been conducted to date. Finance providers require relatively stringent traceability of testwork samples and the representivity of samples for the first two to three years of the exploitation sequence. Where there are any gaps in the testwork sample representivity or testwork results fresh representative samples will need to be produced to conduct new testwork, this is likely to require site drilling of some or all of the exploration targets.

As the mining methods will change significantly for each exploration target it will be necessary to engage a mining contractor / consultant to undertake preliminary mining plans and costing of mining capital and operating costs for the different exploration targets once the exploitation sequence is defined. There will be significant price differences for each exploration target varying from low cost reclaim of jig and slimes tails, open pit or bodies to underground operations.

Once impurities from each exploration target are understood, discussions with off takers will need to be conducted to understand the impact of penalties and/or credits if concentrate production is out of specification. These discussions would need to take place near the end of the study when a clear understanding of the testwork is achieved and will provide critical inputs to the financial model and risk models for cash flow.

Historical test work is encouraging however a fine grind size (-400 μm) is indicated to provide adequate liberation. Further HLS testing is recommended across the size range from 6.3 mm to 45 μm to quantify the potential for heavy media separation in more detail – note that sample selection



and handling need to be considered (diamond drill core, preferably NQ or PQ with staged crushing using small size reduction ratios of about 2 each stage).

There is a fairly clear indication that lower head grades result in lower recoveries however this may be related to the fine target particle size in the tests driven perhaps by the mineralogy and may warrant consideration of a tin flotation stage in the test work.

There was a significant proportion of tin reporting to middling streams further indicating that finer grinding for liberation is required – regrinding of the middlings streams should be included in the test work.

Assays of contaminant elements are needed.

Based on the supposed 'hardness' of the Great Pyramid material, some work index determinations are recommended.

For the estimated feed grade of 0.22% Sn for the Great Pyramid deposit, a concentrate grade of 55% Sn and tin recovery of 60% is recommended based on test results reviewed.

6.2 Conclusion

Although the Conceptual Study for Great Pyramid is not relevant for all of the exploration targets and was priced in 2007, the order of magnitude of capital and operating costs is within a -0%, +40%, 1Q14 range for the gravity only plant. The impact of inclusion of sulphide and other impurity removal steps and the requirement to recover tungsten minerals also introduces flow sheet complexities and will increase the capital costs



APPENDIX A HISTORIC REPORTS

The following list was made available on TNT website and lists the reports available for the Exploration Targets to be considered in stage 2 of the PFS development.

Not all of the listed reports are relevant, however this does provide a significant amount of historic data that may be used as a basis for the PFS and potentially minimise the amount of additional testwork to be conducted to generate a PFS.

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The references have been placed in chronological order which is considered to be more useful and have been grouped geographically (Anchor Tin Mine, the tin and tungsten prospects within EL27/2004 (Aberfoyle, Lutwyche, Rex Hill, Royal George, Storey's Creek,) Great Pyramid, Moina, Oonah, Waratah, Ringarooma. References for Tasmanian projects generally quote an MRT report number. These reports can be accessed or copies downloaded from the Mineral Resources Tasmania web site www.mrt.tas.gov.au then go to "Library Services" and "Document Search". The code following "MRT" should be inserted exactly as given for successful access.

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APPENDIX B CAPEX AND OPEX REVIEW SUMMARY SHEETS

The following tables are provided as very preliminary outlines to demonstrate the additional capital and operating costs associated with flotation stages and tungsten stages in the process flow sheet.

CAPITAL COST OUTLINE (+,-25%)	Concept Study	Full Plant	Gravity Only
AREA 200 - PLANT SITE BULK EARTHWORKS	333,200	400,000	300,000
AREA 310 - CRUSHING	6,375,000	4,500,000	4,500,000
AREA 320 - ORE STORAGE		1,600,000	1,600,000
AREA 325 - ORE SORTING		3,000,000	3,000,000
AREA 330 - GRINDING AND CLASSIFICATION	11,106,796	4,100,000	4,100,000
AREA 331 - SULPHIDE FLOTATION		3,000,000	3,000,000
AREA 350 - COARSE GRAVITY	4,784,659	1,100,000	1,100,000
AREA 352 - GRAVITY REGRIND & CLASSIFICATION		2,250,000	2,250,000
AREA 353 - FINE GRAVITY		900,000	900,000
AREA 354 - DRESSING FLOTATION		1,500,000	
AREA 356 - TIN & TUNGSTEN FLOTATION		6,800,000	
AREA 358 - CONCENTRATE HANDLING AND FILTRATION		2,500,000	2,000,000
AREA 360 - REAGENTS		2,600,000	900,000
AREA 370 - POWER AND RETICULATION		8,000,000	5,500,000
AREA 390 - WATER SUPPLY	828,300	650,000	400,000
AREA 400 - TAILINGS	3,287,626	1,100,000	800,000
AREA 401 - TAILINGS STORAGE	700,000	2,500,000	1,000,000
AREA 411 - FUEL FARM	264,000	200,000	200,000
AREA 420 - COMPRESSED AIR	174,000	300,000	120,000
AREA 490 - SITE BUILDINGS	1,328,400	400,000	300,000
AREA 495 - MOBILE PLANT & EQUIPMENT	655,200	1,000,000	850,000
AREA 499 - PLANT PIPING		3,500,000	2,400,000
AREA 804 - CONSTRUCTION EQUIPMENT	1,768,800	1,800,000	1,100,000
Total Direct Costs	31,605,981	53,700,000	36,320,000
Indirect Costs			
AREA 500 - ENGINEERING	6,219,523	11,000,000	7,650,000
SITE SEWAGE & MAAGEMENT HOUSING	1,248,000		
AREA 510 - COMMISSIONING		750,000	500,000
AREA 600 - PRELIMINARIES AND GENERAL	1,350,000	2,100,000	1,500,000
AREA 700 - OWNERS COSTS	1,656,000	1,200,000	1,000,000
Total Indirect Costs	10,473,523	15,050,000	10,650,000
TOTAL CAPITAL ESTIMATE	42,079,504	68,750,000	46,970,000

Table 8 Capital Cost Comparison



OPERATING COST OUTLINE (+,-25%)	Concept Study	Full Plant	Gravity Only
Expenditure [\$]			
AREA 310 - CRUSHING & SCREENING		2.50	2.50
AREA 320 - FINE ORE STORAGE		0.30	0.30
AREA 325 - ORE SORTING		1.50	1.50
AREA 330 - GRINDING AND CLASSIFICATION		4.00	4.00
AREA 331 - SULPHIDE FLOAT & REGRIND		3.50	3.50
AREA 350 - COARSE GRAVITY		0.20	0.20
AREA 351 - SULPHIDE DRESSING FLOTATION		0.25	
AREA 353 - COARSE GRAVITY REGRIND & CLASSIFICATION		1.20	1.20
AREA 354 - FINE GRAVITY		0.30	0.30
AREA 355 - FLOAT DESLIME		0.25	
AREA 356 - FLOTATION (TIN, TUNGSTEN & DRESSING)		4.10	
AREA 360 - CONCENTRATE HANDLING		1.20	1.20
AREA 380 - REAGENTS		0.50	0.20
AREA 390 - WATER SUPPLY		0.15	0.10
AREA 400 - TAILINGS		0.40	0.30
AREA 420 - AIR SERVICES		0.20	0.15
AREA 499 - PLANT PIPING		0.20	0.10
AREA 410 - POWER AND RETICULATION		0.40	0.30
SUBTOTAL	16.31	21.15	15.85
Workshop		0.30	0.20
Laboratory		0.35	0.20
Administration	1.58	1.00	0.70
Management & Services		1.20	0.90
Production Labour		4.50	2.50
Maintenance Labour	1.43	2.50	1.80
Total	19.32	31.00	22.15
\$/Tonne Treated	9,660,000	15,500,000	11,075,000

Table 9 Operating Cost Comparison



APPENDIX C MASS BALANCES FOR PILOT TESTS (GREAT PYRAMID)

Year	Source	Sample	Head Grade %Sn	Concentrate Grade %Sn	Tin Recovery %	Concentrate Mass %	Test Conditions/Flowsheet	Tin Distribution
1945	DMT		0.64	22	78.6		Grind to -420 µm, gravity	
			0.64	27	80.6		Grind to -250 µm, gravity	
			0.64	27	80.2		Grind to -180 µm, gravity	
			0.64	11	71.8		Flotation	
			0.64	6.4	81.8		Flotation	
1963	DMT	550 kg	2.17	61.6	79.26		Grind, gravity	
			2.17	59.7	84.55		Included regrind of +250 µm tail	
			2.17	59	87.05		Included regrind of +150 µm tail	
			0.76	50.5	73.81		Grind, gravity	
			0.76	48.9	77.35		Included regrind of +250 µm tail	
			0.76	46.9	79.76		Included regrind of +150 µm tail	
			0.47	46.9	60.59		Grind, gravity	
			0.47	43.8	66.01		Included regrind of +250 µm tail	
			0.47	41.3	71.37		Included regrind of +150 µm tail	
1971	Aberfoyle	73 kg drill chips	0.296	47.6	63.5		Gravity, magnetic separation	32% -850 µm +500 µm 33% -500 µm +300 µm
1979	Aberfoyle		0.63	28.9	86.0	1.9	Sinks HLS @ 2.96 sg -600 µm +45 µm	
			0.60	38.13	63.7	1.0	Gravity	
			0.19	5.6	68.7	2.3	Sinks HLS @ 2.96 sg -600 µm +45 µm	
			0.20	7.9	13.7	0.34	Gravity	
1983	DMT	50 kg Pilot	0.22	52.9	54.5		Sinks HLS @ 2.95 sg	54% -300 µm +125 µm 47% -300 µm +75 µm
1984	MDL	546.7 kg for Shell	0.22	1.3	73		Hydroziser, Spiral, Table, Mag Sepn	
			0.414	48	43		-400 µm grind	

DMT 1983 (from Ruxton)	STREAM	MASS %	ASSAY %Sn	Metal Units	Tin Distribution %
	Feed	100	0.22	22	100
	Cyclone Overflow	17.78	0.08	1.4	6.5
	Hydroziser Overflow	27.33	0.19	5.2	23.6
	Gravity Feed (combined)	54.89	0.28	15.4	69.9
	Gravity Concentrate	1.2	11.6	13.9	63.3
	Gravity Middlings	4.71	0.16	0.8	3.4
	Middlings R/G Concentrate	0.08	2.60	0.2	0.9
	Middlings Gravity Tail	4.63	0.12	0.5	2.5
	Primary Gravity Tail	48.98	0.015	0.71	3.2
	Total Gravity Tail (mids + prim)	53.61	0.023	1.26	5.7
1.28	Total Gravity Concentrate	1.28	11.04	14.1	64.2
	Supplide flotation concentrate	0.01	2.08	0.02	0.1
	Magnetic separation Magnetics	0.95	1.185	1.1	5.1
1.03	Magnetics Mids	0.08	3.57	0.29	1.3
	Total Tailing	99.76	0.09	9.30	42.3
	Final Concentrate	0.24	52.9	12.7	57.7
		0.32	42.2	13.5	61.4
		100		22.00	100.0

MDL 1984 (from Hall & Carter, 1986)

assay = 42.36 (6.5%)
assayed = 0.51 (47.3%)

assay = 47.92 (15.65%)

Assay = 49.21

Assay = 52.29

STREAM	MASS %	ASSAY %Sn	Metal Units	Tin Distribution %
Feed	100	0.385	38.5	100
Cyclone Overflow	22.78	0.286	6.5	16.9
Grind Circuit Product	77.22	0.414	32.0	83.1
Hydroziser Slimes	28.88	0.22	6.4	16.5
Medium Gravity Feed	17.93	0.31	5.6	14.4
Medium Spiral Middlings	3.4	0.22	0.7	1.9
Medium Spiral Tail	12.65	0.075	0.9	2.5
Medium Table Tail	1.6	0.18	0.3	0.7
Medium Table Middlings	0.17	20.80	3.5	9.2
Medium Table Concentrate	0.11	0.381	0.04	0.1
Coarse Gravity Feed	30.41	0.66	20.1	52.1
Coarse Spiral Middlings	4.49	0.34	1.5	4.0
Coarse Spiral Tail	23.67	0.11	2.6	6.8
Coarse Table Tail	1.707	0.28	0.5	1.2
Coarse Table Middlings	0.42	17.50	7.4	19.1
Coarse Table Concentrate	0.123	65.97	8.1	21.1
Total Table Concentrate	0.233	35.0	8.2	21.2
Assume Medium Table Con	0.11	42.36	4.66	12.1
Assume Coarse Table Con	0.123	47.92	5.89	15.3
Medium Magnetics (Tail)	0.0993	5.9	0.05	0.14
Medium Non Magnetics (CONC)	0.1137	51.36	5.84	15.17
Coarse Magnetics (Tail)	0.0116	5.5	0.064	0.17
Coarse Non Magnetics (CONC)	0.1114	52.34	5.83	15.14
Total Concentrate	0.2251	51.84	11.67	30.31
MDL (GRES Calculated from assay)	0.17	69.1	11.75	30.51
MDL Table Balance	<0.2	69.1	20.27	
Combined Middlings	8.48	1.55	13.2	34.2

