

Chronological and Metallurgical Steps in the Development of Processing of the Mt Garnet Tin Ore Deposits

A. Historical Workings pre-1974

No reports are available for workings prior to RGC involvement from 1974 onwards. However the Heberton area was subjected to dredging operations which recovered the cassiterite from weathered ore deposits by the use of conventional gravity jiggling methods. Consolidated Tin Mines are currently sampling areas of the dredge tailings minerals and testing samples using conventional gravity equipment in an attempt to ascertain whether these waste deposits can provide cassiterite concentrates in sufficient commercial quantities and grade. The re-mining of these minerals and recovery of cassiterite may provide revenue to the relocation of these waste materials to an alternative location within the mine lease. This in turn will allow the exposure of additional tin mineral deposits from areas below the dredging operation wastes.

B. 1974-1998

RGC and Otter Exploration were involved with these deposits until 1998. Otter held the leases until 1985 and RGC continued to hold them until 1998.

Renison Goldfields Consolidated (RGC) at that time were the premier tin producing company in Australia, with their Renison Bell Mine operation being their main tin production mine. It was considered that the Mt Garnet area with Gillian, Windemere and Smith's Creek deposits would provide supplementary feed to the Rentup facility (Renison Thermal Upgrading) planned at the Renison Mine. The production of a low grade tin concentrate feed to the future tin fuming facility planned for the Renison area was at that time under consideration. It was felt that a Concentrator set up in the Mt Garnet area could produce both marketable high grade tin concentrate and a low grade tin concentrate for the tin fuming plant.

RGC completed diamond drilling and trenching of the area in order to define the volumes and grades of the resources within the area. These samples provided both an estimate of the resource and the minerals to complete a comprehensive mineralogical examination of the ores. Thereafter these samples provided a resource for a variety of metallurgical tests carried out during the proceeding years.

C. Early Mineralogical Tests and Identification of Two Tin Minerals

A series of reports were submitted to RGC management by H.W.Fander of Central Mineralogical Services (CMS) during 1975 and into 1979 (Ref 1). A wide range of drill core samples were examined in order to ascertain the tin mineral sizing and mode of tin minerals within each ore sample. Initially Wally Fander's reports identified the tin mineral as cassiterite with size ranges from 10microns to 1.6 mm within the ores. The majority of cassiterite particles were found to be fine throughout the ore samples and probably were at average of 30 microns and below. However as his examinations progressed he realized that there was more tin assaying in samples than could be identified as only

cassiterite. He eventually arrived at the conclusion that there was another source of tin within the deposits which could not be identified as cassiterite. This was first described in a CMS report dated 14/10/75. He found that this form of tin was closely associated with the goethite minerals within the ores and described the mineral as Stanniferous Goethite, which was later termed Gillianite, from the area where this mineral was first located. This mineral was also found to dissolve in a concentrated hydrochloric acid leach which initially was being used to digest the bulk of iron oxides from ore samples. This led to the tin in the ore being described as soluble or insoluble. Cassiterite content being insoluble and the goethitic based mineral termed soluble. The solubility test became standard practice for testing of each drill core sample, it consisted of subjecting the 20 gram ground mineral sample to 2 hour leach at constant boiling point in a 55%HCl water mixture. The samples had previously been assayed by XRF for total tin content and the difference between total and soluble tin assumed the remainder as cassiterite. Mineral examination of the leach residues determined the remaining tin as cassiterite. The range of tin solubility throughout the samples differed from 8% to 88% with the majority of the iron dissolved and the leach residues consisting of quartz, precipitated silica and cassiterite.

D. Identification of Acid Soluble Tin Mineral

Three mineral samples with tin grades of 2.28-2.7%Sn and their leach residues from tin solubility tests, giving minor to total tin solubility, were submitted by Renison to Amdel laboratories in South Australia. The samples were subjected to XRD analysis, mineralogical examination and electron probe point analysis. The conclusion drawn was that all of the acid soluble tin was in iron oxides, with the majority present in goethite, but with a minor amount in martite (hematite produced after magnetite). They stated that there was no evidence that the tin in goethite was not in high concentrations, suggesting that it was in the crystal lattice, or if as cassiterite it would be very fine as sub micron inclusions. This factor determines that conventional gravity or flotation techniques would not allow recovery of tin minerals to succeed from this type of ores.

E. Gravity Tests on Minerals Samples

A series of gravity tests were carried out at the Renison Mine laboratory facilities. Crushing and grinding to fine mineral size below 75 microns before subjecting the minerals to superpanning. A series of superpanning tests at screened mineral size intervals did not produce consistent recoveries and a wide range of concentrate grades. This indicated that the goethitic minerals were not responding to gravity recovery and another treatment route was required in order to generate a reasonable grade of tin concentrate.

F. Leach Testing of Acid Soluble Ores

Renison realized that the use of concentrated hydrochloric acid would not provide a viable treatment route to produce tin metal from the goethitic ores. They decided to trial a 30% sulphuric acid leach on samples of ores with a high soluble tin content (Ref 2). The leach tests were run at 95°C for times ranging from 1-8 hours. The samples had previously been ground to <75 microns. The results revealed that although sulphuric acid leach did not achieve the same total tin dissolution as

the boiling hydrochloric acid test, an 8 hour leach dissolved 63.6% of the acid soluble tin and 79% of the Fe₂O₃ content of the ore sample. This may provide a recovery route for tin with a probable ferric sulphate bi-product. Tin recovery by electrowinning or solvent extraction from the leach liquor may provide a viable tin metal product route. However the advice from Electrometals Technology Ltd is that the high iron content of the leach filtrate would prevent the recovery of tin metal.

G. Heavy Media Separation Testing

CMS also ran some heavy media separation tests on a range of ore samples. The practice was in progress at Renison Mine at that time which reduced the feed of very low grade tin content minerals to their milling circuit. The tests attempted to upgrade ground minerals of 5 size fractions examined. Although the floats gave lower tin grades at around 0.5%Sn, middlings ranged from 1.3-1.7%Sn, and sinks assayed at 1.57-2.35%Sn. There were minimal differences in the acid soluble and insoluble tin contents of the three products and the method did not achieve a significant removal of a low tin tailing as a float product. The process method was not considered viable for these ores.

H. Thermal Testing on Acid Soluble Ore

Renison contacted Amdel in Adelaide to run testing on acid soluble containing tin ores (Ref 3). This was carried out on the same minerals which had been examined by Amdel to identify the nature of the acid soluble constituents in the ores as described in section D. The samples containing the highest goethite content were subjected to thermal treatment and analysis. One sample was initially heated to 350°C and exhibited a weight loss due to decomposition of the goethite to hematite, indicating a goethite content of 88%. All heating tests were carried out in air. Tests proceeded to heat the mineral samples to temperatures ranging from 910-1300°C. The samples produced were then examined by optical microscopy and electron-probe microanalysis. The results demonstrated after the initial decomposition of the goethite to hematite was completed progressively higher temperatures hematite particles of low tin content were formed, indicating that the tin was being expelled from solution in the hematite as the hematite crystals were being formed. At 1300°C the magnetite formation from hematite allowed only 1.4%Sn tin to remain in the magnetite crystals, however at that temperature the hematite crystals were then able to contain 4.0%Sn. This indicated that if all of the iron oxides were converted to magnetite then around 60% of the tin content may be segregated to enable it to be removed by leaching. This work was not followed up by Renison, but has indicated that reduction roasting may reduce the temperature required to convert the goethite through to magnetite and expel tin metal or tin/iron alloy from this type of ore in future tests. This method may be modified to enable the cassiterite to be reduced to stannous oxide, and then sulphidised to stannous sulphide, in order to allow the tin to be fumed off from the high iron concentrates extracted from conventional mineral dressing techniques.

I. Otter- London Fuming Testwork 1982

Otter commissioned Warren-Springs, based in London, to solve the metallurgical treatment difficulties of the Gillian ores. They put forward a variation on the fuming process to get a recovery of tin of up to 95% from a previously prepared concentrate. However the recovery of a fuming feed

concentrate needed to be established before the benefit of the fuming stage could be realized. It is not reported that any fuming testwork, or concentrate was produced for this test.

J. Otter Gold 1992

A detailed review of the deposits and metallurgy of extraction from these ores was completed for Otter by Thornton in 1992. He concluded that Kelsey Jig and Sirosmelt processing should be used to treat these ores. Kelsey Jigs had recently been introduced and developed at the Renison Concentrator at that time and were being considered as a success for the recovery of fine cassiterite. Griffiths continued with this review and also suggested that Kelsey Jigs should be used to upgrade a fumer feed for the Sirosmelt unit. The use of Kelsey jigs at Renison has proved to be beneficial to the recovery of cassiterite particles down to 15 microns but their efficiency diminishes significantly below 10 microns. There is no report of metallurgical testwork being carried out to confirm these conclusions, but Griffiths qualified them by adding that at the current tin price the processing was uneconomic and any preconcentration of Gillian ores would not enhance the Sirosmelt economics.

K. Tin Australia 1998

Tin Australia bought the leases and many other tin projects in North Queensland with the aim of setting up a central plant at Mt Veteran and carting ore from these deposits. Esker Milling and Processing (Nick Moony) carried out testing on 8 drill core samples from the Gillian deposit where insoluble tin (cassiterite) predominated. The ore grades varied between 0.85%Sn and 2.15%Sn, with an average as 1.75%Sn content. 20-35% of the tin in the samples occurred as fine cassiterite which was only fully liberated at <50 microns. However even at this size the cassiterite appeared to be coated with goethite and would only be liberated fully after acid leaching. Superpanning of these samples produced concentrates ranging from 2.93-35.32%Sn content, with recovery values of tin into concentrates from 1.1-21.1%. Estimates and assumptions were made on recoveries after centrifuge and leaching would have been applied to the concentrates produced. It was concluded that a 17%Sn flotation concentrate could be produced from the Gillian ores and a combined gravity (10%) and flotation (45%) recovery would be 55% overall for these ores.

L. Tin Australia 1999

No testwork was completed to verify the conclusions reached on the previous tests carried out by Esker Milling and Processing. However the source of sulphuric acid produced from the Mt Garnet zinc sulphide ores was included in the development of a conceptual flowsheet to include leaching in the tin recovery circuit. This concept was abandoned when Kagara Zinc began shipping zinc sulphide concentrates from the Mt Garnet area.

M. Virotec International 1999-2005

Virotec purchased the leases but again no metallurgical testwork is reported for this period.

N. Bluestone Tin 2005

Bluestone Tin acquired the Gillian and Windemere leases from Virotec in May 2005. Bluestone had acquired the Collingwood tin deposits and were developing a processing concentrator to recover cassiterite minerals from this site and had also purchased the Renison Mine and had the Renison Concentrator commissioned and in production at that time. Their Rentails recovery project had also been started and was at the prefeasibility stage with the target to ultimately set up a Tin Fuming facility at the Renison site. The Mt Garnet ores were considered to be able to provide both a fuming feed and tin concentrate sales when developed. However no further testwork was undertaken on the metallurgical treatment of these ores during this period. Bob Watchorn, the Group Geology Manager prepared a report during November 2005 on the Gillian and Windemere Deposits (Ref 4). Ore sorting to remove barren feed from the ore supply was considered for these ores but no testing was undertaken to confirm the viability of 6 methods known. Probably the most promising of these would be magnetic due to the high iron content of the ores, however the association of tin with the iron oxides would reduce selectivity.

O. Summary of Metallurgical Testwork prior to Consolidated Tin Mines 2007

The bulk of metallurgical testwork up to 2007 on these ore deposits was undertaken during the ownership of the leases by RGC who had extensive laboratory facilities at their Renison Mine. The testing however tended to concentrate on the recovery of tin from the acid soluble component of the ores, but no final conclusions as to the viability of recovery of the tin from the goethitic tin were published. Later testwork using superpanning to recover the free cassiterite components of these ores was undertaken by Esker indicated that up to a 38%Sn concentrate could be achieved. However no testing of the ores containing free cassiterite minerals was carried out on commercially available gravity equipment, such as spirals or tables. In essence no assessment of a viable tin recovery as cassiterite concentrates, tin oxide fume or tin metal had been evaluated to this point in time. Consolidated Tin Mines acquired the leases for the area in 2007 and have proceeded to evaluate the area for recovery of the tin by a commercially viable route and their efforts are outlined in the following paragraphs. The efforts by Consolidated Tin have focused on the development of a viable flowsheet and process in order to recover both marketable cassiterite at a grade suitable for conventional tin smelter operations and a low grade tin concentrate of 10-20%Sn grade suitable for sale to operations of a tin fuming facility. The mineral deposits are likely also to be able to produce a high grade magnetite mineral for sale to iron ore pellet plant operation, and this would contribute to revenue for the overall project.

The project flowsheet development is being carried out in stages targeted to produce outcomes for each process expected to be required within the production of saleable minerals. The various operations required within the process are listed below and the outcomes from each aspect of the testwork are detailed under each heading.

P. Crushing of Ores

Minimal detailed work has been carried out on this aspect of the flowsheet. The development of this process can only be carried out accurately with testing on diamond drilled cores. The drilling to date has only developed RC core material, which is unsuitable to generate crushing and grinding parameters accurately for equipment design purposes. At this stage the ore supplies do not indicate high hardness values requiring heavy duty crushing and grinding equipment. Also the emphasis at this stage of testing has been to evaluate the processes required after the crushing and grinding stages. The RC mineral core products available are considered suitable for testing of magnetic, gravity and other processes expected within a conventional metallurgical treatment plant. The crushing equipment considered for these ores will follow current iron ore processing equipment with conventional single toggle jaw crushing followed by, either secondary and tertiary cone crushers, or high pressure grinding roll (HPGR) equipment. The HPGR unit provides a benefit to the ball mill as it imparts micro cracks into the mineral particles which reduces the power requirements for the ball mill grinding stage.

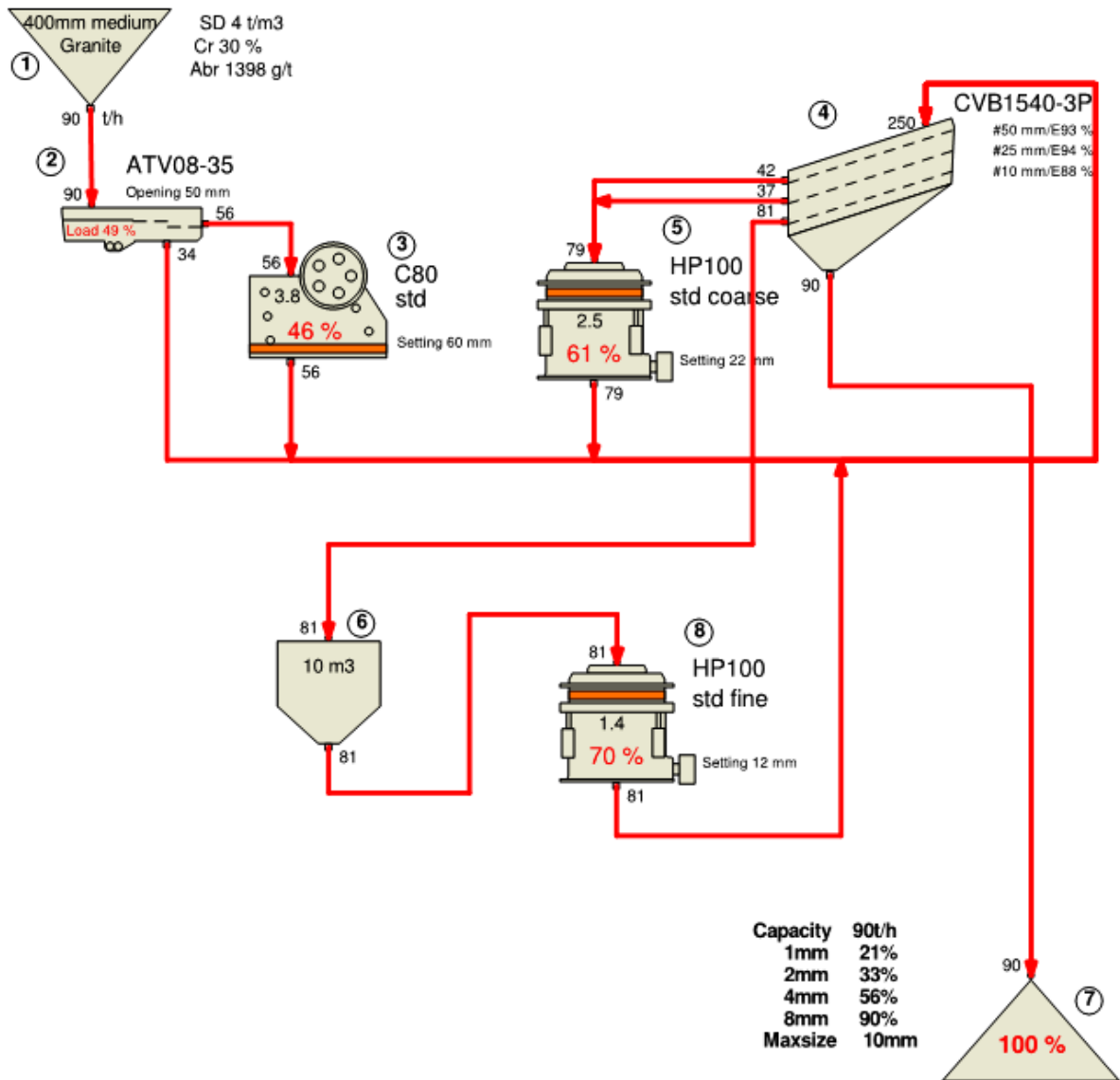
Consolidated Tin Mines have now started to process a bulk sample of ores assembled from the Mount Garnet area. This has included areas covered by Hole 7, Hole 83/84 and extra drill areas not previously tested but considered to form the future ROM pad feed blend. Initial crushing and grinding tests on this mixture have revealed that the crushing stage will produce a significant proportion of <75 micron product prior to subjecting the crushed product to grinding. Therefore in this case it is expected that after crushing, prescreening using a 3mm trommel and 75 micron stacksize screen would remove this undersize material and bypass the Ball milling circuit. This will have the benefit of reducing the tonnage to the Ball Mill thereby reducing both the Ball Mill capacity and power requirements. There would also be an added benefit of a minimizing of overgrinding, which contributes to cassiterite losses to tailings.

Screen Size µm	Screen Weight g	Weight Distribution %	Weight Passing %
250	141.7	42.5	57.5
212	9.30	2.79	54.7
180	7.20	2.16	52.5
150	8.70	2.61	49.9
106	14.9	4.47	45.4
90	8.30	2.49	42.9
75	7.50	2.25	40.7
63	7.30	2.19	38.5
53	6.50	1.95	36.5
45	5.30	1.59	34.9
38	5.00	1.50	33.4
-38	111.4	33.4	
Total	333.1	100.0	

This table shows size fractions produced from crushing of a mixed bulk ore sample to 3.35mm. High proportion of <75 micron size has already been generated in the product.

Metso provided a budget quote for a 90 tonne/hr crushing circuit and the detailed flowsheet for this layout is shown below.

Proposed Metso 3 stage Crushing and Screening Circuit



Q. Esker Milling and Processing Pty Ltd

Nick Moony of Esker Milling and Processing Pty Ltd carried out a series of tests on an ore sample during 2009 which was considered to contain a significant proportion of tin as free cassiterite (Ref 5). The ore assay was 1.3%Sn, 0.08% soluble Sn and 37.6%Fe. The ore sample was subjected to fine grinding to a p80 of 53microns, magnetic separation and tin flotation. He reported that a tin recovery of 30% was achieved when producing a 50%Sn concentrate, and at a 17-22%Sn concentrate the tin recovery rose to 50-55% and another 15-20% recovery could be achieved into a

4.5%Sn low grade concentrate. The recovery of iron from this ore was estimated at 50% at an iron concentrate grade of 65%.

The testing used LIMS and WHIMS magnetic separation at a range of magnetic flux densities, with LIMS at 1000 gauss and WHIMS at 4000, 8000 and 13400 gauss. The LIMS separation showed that iron grades in the magnetic product could exceed 65%Fe with only low silica, alumina and phosphorus contamination, and that the tin losses were low at <0.3%Sn. The gravity separation method used a superpanner at a series of mineral sizings on the magnetic and non-magnetic products, with sizings ranged from +38 to +14 microns. Results on the superpanning were reported to achieve tin grades exceeding 60% Sn grade on non-magnetic minerals but at low recovery values. Mineralogical examination of products off the superpanner showed that not all of the free cassiterite was recovered to concentrate. Finally a preliminary tin flotation test was run on a sample of 8000 gauss non-magnetic product, using styrene phosphonic acid as a collector. The results were poor and only 49% of the tin was recovered to a tin grade of 5.06%Sn from a feed grade of 2.14%Sn. The grade of the rougher tails was 0.65%Sn. The styrene phosphonic acid collector is believed to be the most selective cassiterite collector available at this time and flotation conditions are known for maximum recovery from long term Renison Bell Mine experience. However the flotation has shown that a high proportion of the iron oxides had floated with the cassiterite.

In summary the Esker work showed promise in extraction of a saleable tin concentrate exceeding 50%Sn content but at only 30% recovery. Nick Moony summarised that a further 30-40% of the tin could be recovered to a 6-12%Sn grade. However these tests were carried out on an ore sample which was known to contain mostly free cassiterite but the liberation size for the cassiterite was not accurately known. He recommended that tin flotation testing needed to be optimized. The use of enhanced gravity methods also needed to be investigated in order to recover the free cassiterite. Finally he suggested that testwork needed to be pursued on the ores which would provide the majority of the feed to a future Concentrator.

R. Mineralogical Examination of Ore Supplies

Initially two ore samples were examined using QEMSCAN. One ore sample, Hole 7, was known to contain high quantities of free cassiterite from conventional mineralogical examination. This was subjected to QEMSCAN analysis, after the core had been crushed and ground to a p80 of 106 microns (Ref 6). A second sample, from Holes 83/84, was again examined by QEMSCAN analysis. This ore sample was considered to be typical of the majority of the ore supplies within the Mt Garnet area. Again this ore sample had been subjected to the same sample preparation and grinding to p80 of 106 microns using only steel media grinding.

Hole 7 QEMSCAN analysis results

The XRD analysis of the sample tested is shown in the table below. The average grain size of the cassiterite particles found in the sample was 13.27 microns and the distribution of the cassiterite mineral particle sizes is shown in Fig 1 attached, which indicates that 73% of cassiterite particles were >10 microns with the remainder 27% at >7-<10 microns. At this grind size minimal quantities of

cassiterite particles were fully liberated, but the majority of cassiterite was agglomerated with goethite with minor numbers locked to other iron oxides, hematite being the main oxide. Only cassiterite was identified in the sample with no stanniferous goethite detected.

Hole 83/84 QEMSCAN analysis results

The XRD analysis of the sample tested is shown in the table below. The average grain size of the cassiterite particles found in the sample was 10.09 microns. In this case 50% was >10microns, 29% at >7-<10 microns, 13% at >5-<7 microns and 8% at <5 microns, this signifying that 20% was less than 7 microns in this Hole 83/84 sample, and only cassiterite was identified in this sample.

XRD results on Hole 7 and Hole 83/84 Minerals

Mineral ID	Chemical Composition	MASS PERCENT (%)	
		Hole 7	Hole 83/84
Hematite	Fe ₂ O ₃	28	46
Goethite	FeO(OH)	6	7
Quartz	SiO ₂	33	20
Clinochlore	(Mg,Fe) ₆ (Si,Al) ₄ O ₁₀ (OH) ₈	3	N/A
Magnetite	Fe ₃ O ₄	4	19
Calcite	CaCO ₃	0	N/A
Kaolinite	Al ₂ Si ₂ O ₅ OH ₄	5	2
Cassiterite	SnO ₂	8	6
Orthoclase	KAlSi ₃ O ₈	13	N/A

Hole 7 QEMSCAN analysis after magnetic separation

It was considered that, after the first QEMSCAN and mineralogical examinations, the first process step was to arrive at a grind size which would liberate more cassiterite particles for recovery by conventional gravity methods. The grind size at 106 microns left the majority of cassiterite locked as binaries and there was a problem of agglomeration with the iron oxides shown particularly in Hole 7 minerals. It was decided to use inert media grinding in order to minimize the risk of surface iron contamination on liberated cassiterite particles. The p80 target grind for the Hole 7 minerals was set at 50 microns. This sample was then subjected to magnetic separation at 3 gauss field strengths. LIMS (low intensity magnetic separation) at 1000 gauss and two WHIMS (Wet high intensity magnetic separation) at 4000 and 8000 gauss field strengths. The four magnetic separation products, LIMS magnetic, WHIMS 4000 gauss magnetics, WHIMS 8000 gauss magnetics and WHIMS 8000 gauss non-magnetics, were examined by QEMSCAN analysis (Ref 7).

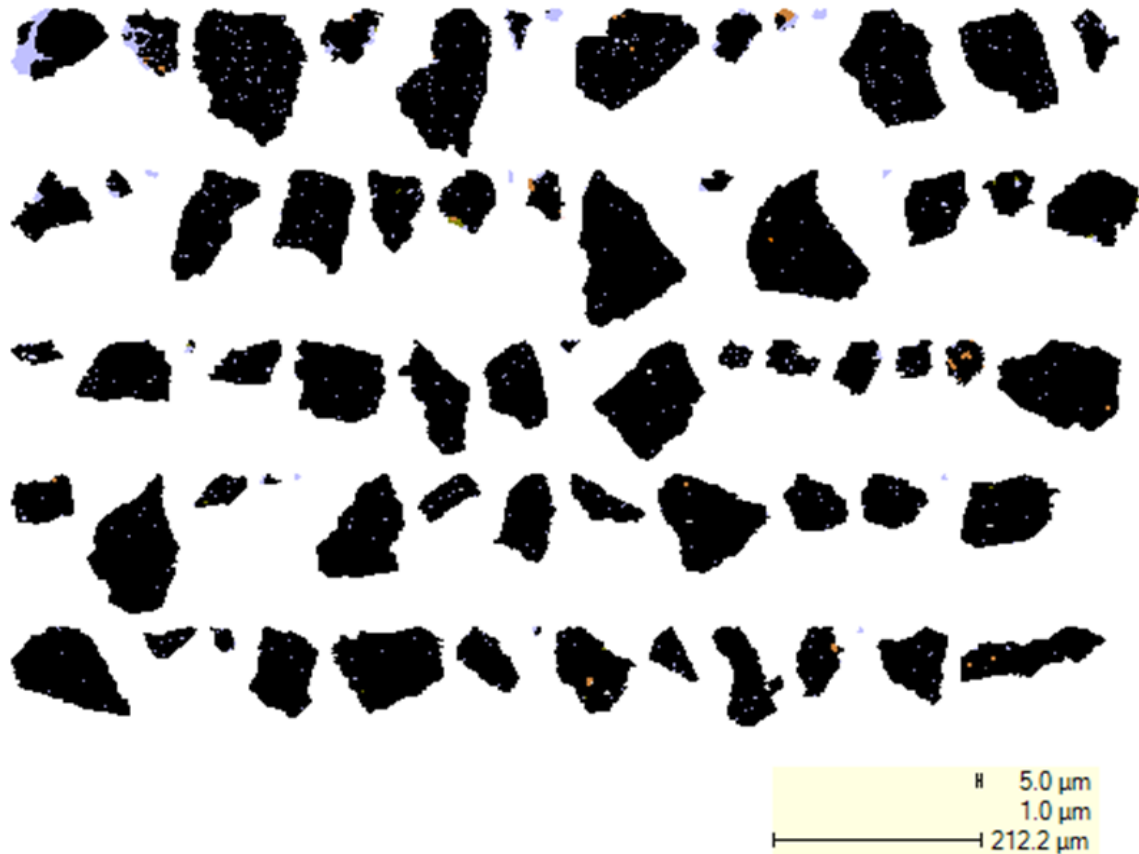
The LIMS magnetic product showed high iron oxide grade of 67.4%Fe content. The assay and test conditions for this mineral product were as follows:

XRF Assay Values LIMS Product from Hole 7 - Downer Laboratory

Hole	p80	Grind	LIMS	% of total	Total	Total	Total	Total
No.	microns	Media	gauss	weight	Fe	SiO2	Al2O3	Sn
7	53	Sand	1000	1.0%	67.40%	1.57%	0.56%	0.24%

The high iron grade of this product confirms previous LIMS separation can give a product that can be marketed as a pelletising plant feed for iron ore products. There are very low deleterious contaminants of silica, alumina and phosphorus and minimal tin losses at 0.24%. Also the LIMS magnetics weight for this mineral was only estimated at 1% of the feed weight so tin losses overall to this product were minimal off this mineral. The QEMSCAN analysis of the LIMS minerals and particle images show that all of the cassiterite is as very fine inclusions of 5 microns and less in size, which indicates that further grinding of this product and attempts at cassiterite recovery are not viable. There were a few liberated grains of cassiterite detected in the sample but this was considered to be due to only a single pass used for the LIMS separation and a cleaning stage would probably have eliminated any trapped non-magnetic contamination. There were a few coarser particles of iron oxides which did contain veins of cassiterite which would be released at a finer grind but this was considered as marginal in value, controlled grinding would minimize the coarse binary particles. The XRD of this product indicated that there was significantly more iron as hematite than as magnetite in the sample.

Particle Images of LIMS product from Hole 7 Minerals



These images of LIMS particles show the magnetite represented as the large black particles with the light blue inclusions are the cassiterite. The cassiterite inclusions are usually less than 5 microns with the occasional vein of cassiterite through the magnetite particle as shown by the first two particles in the top left hand corner images.

The two magnetic products from the WHIMS magnetic separation at 4000 and 8000 gauss field strengths showed similar mineral suites, with more hematite present in the 4000 gauss sample which had been extracted before the 4000 gauss non-magnetics had been subjected to the WHIMS 8000 gauss treatment. Free cassiterite grains were detected in both of these samples. However the iron content of these cassiterite particles was found to be low at less than 2%Fe content so they would not appear to be paramagnetic to the point where they would be pulled into the magnetic stream. Observation of the WHIMS treatment of the sample suggested that it could be caused by the strength of water flow washing non-magnetic mineral contamination into the magnetic stream. WHIMS separation will probably not possess sufficient selectivity for the very fine particles generated within this process and an alternative high intensity magnetic wet separation process was sought. The presence of free cassiterite in all of these magnetic products indicates a requirement to use a rougher/cleaner method in an effort to pull all of the free cassiterite into the non magnetic portion of the processing. The iron minerals present in the WHIMS magnetics were mostly hematite but goethite was present in significant proportions of each product. The magnetic products from

both field strengths did not show sufficient separation of iron oxides from the non-magnetics. There were significant amounts of liberated cassiterite particles and composited particles containing cassiterite were associated with iron oxides and goethite within the sample. This suggested that in general the grind size was insufficient to ensure maximum cassiterite liberation and that the WHIMS equipment did not appear to provide adequate selectivity.

The 8000 gauss non-magnetic sample showed that there was less cassiterite liberation and the predominant iron mineral was goethite. Free cassiterite particles were present and also cassiterite as coarse grains were locked with goethite.

The results from this Hole 7 mineral magnetic separation indicated that the LIMS separation was probably close to optimal but a cleaning stage after a roughing stage would minimize any free cassiterite being caught in the LIMS magnetic product. The WHIMS separation at both field strengths did not produce a satisfactory separation result, with free cassiterite particles present in all products. There was coarse cassiterite in all products which negates the use of WHIMS separation at this grind size. Finer grinding is required to liberate more cassiterite and the WHIMS selectivity will not be sufficient at this particle size. The minerals should be reground to <10 microns after a LIMS separation has removed the ferromagnetic iron oxides, and then another high intensity wet magnetic separation method was sought to improve iron mineral separation away from fine liberated cassiterite. The recovery of coarse cassiterite after an initial grind to 50 microns may liberate cassiterite particles, and particles with a majority of cassiterite, and recovery of these would be possible in conventional or enhanced gravity equipment. However the remainder of the very fine cassiterite can only be recovered by further fine grinding to <10 microns before a significant proportion of the cassiterite will be liberated. Once this is achieved then the separation of iron oxides and goethite away from the cassiterite becomes possible if the efficiency of the high intensity magnetic separation at this fine sizing is sufficient.

Hole 83/84 QEMSCAN analysis after magnetic separation

Hole 83/84 mineral sample is considered to represent a significant proportion of the ores available for processing at Mt Garnet. The sample was subjected to a finer initial grind at a p80 of 26 microns in order to hopefully liberate more cassiterite as the initial QEMSCAN analysis, on the 106 micron mineral grind, gave a lower average of 10 microns for the cassiterite particle size, with a high proportion in the 3-7 micron range.

The issues with the WHIMS separation method on the Hole 7 mineral sample prompted the magnetic separation test to be carried out on the Outotec SLon high intensity magnetic separation equipment after an initial LIMS magnetic separation. Inert media grinding in silica sand media was used to generate the sample prior to the LIMS followed by SLon magnetic separation. The LIMS separation was carried out at 700 gauss field strength and the SLon separation at 4000 and 8000 gauss field strengths. The SLon separation was run on LIMS non-magnetic products and not in series as had been carried out in the WHIMS separation. This method would be practiced in a normal plant operation with the decision on fixed field strength being decided at 4000 or 8000 gauss. A

single pass was used through the SLon equipment with half of the sample run at 4000 gauss and the remaining half processed at 8000 gauss.

The QEMSCAN analysis on the LIMS separated product showed similar results to those exhibited for the Hole 7 minerals, with magnetite particles containing <5 micron inclusions of cassiterite (Ref 8). The SLon magnetic products showed that the cassiterite in both the 4000 and 8000 gauss magnetics was associated with the goethite and not the iron oxides (hematite). The SLon non-magnetics also showed the cassiterite to be associated with goethite but also some minor association with silicates. Although there was evidence of liberated cassiterite in all of the SLon products, the cassiterite grain size was estimated to be <5 microns indicating that more liberation would be required to recover a clean cassiterite concentrate. The UF Falcon concentrate, which had previously been ground to <10 microns showed the maximum quantity of liberated cassiterite particles.

S. Inert Media Grinding

The results of QEMSCAN analysis and other mineralogical tests had indicated that there was a tendency for iron minerals to agglomerate during grinding and the fine sizing of the cassiterite would require ultrafine grinding to be included in the flowsheet. There are a choice of several fine grinding methods and inert media, either sand or ceramic media provides the optimum conditions. Vertimill or Tower Mill will not provide a <10 micron grind product necessary for full cassiterite liberation. The IsaMill is probably the most advanced in development with units of 3mW capacity operating in commercial plants. This unit has been used for laboratory testing and production of fine mineral samples for flotation, magnetic and enhanced gravity tests. A recently developed alternative to the IsaMill is the Deswik Mill, it claims lower power input requirements to achieve the comparative sizings. There have been recent sales of 100 tonne/hr capacity units to processing plants. The drive system is hydraulic and this allows the vertical mill to be restarted under full load conditions. It is proposed to do comparative tests on this unit during later tests to evaluate these claims in comparison to the IsaMill unit. An attritioner type vertical sand mill has been used to produce small samples for laboratory flotation tests but the tendency with this unit is to overgrind and the control of p80 sizing was limited. The IsaMill unit provides an accurate grind size, and overgrinding is minimized, as the residence time within the horizontal mill is very low and the mill retains the coarse particles until they have been reduced to the correct sizing. Initial tests, using inert media ground minerals compared to steel media ground minerals for tin flotation tests, showed a measurable improvement to tin flotation. These tests were carried out on a LIMS/WHIMS non- magnetic Hole 7 mineral sample which contained free cassiterite, and used a standard tin flotation method using the cassiterite collector styrene phosphonic acid (SPA). The results are shown below:

Tin Flotation Result on Steel Media Grind Hole 7 Minerals

PRODUCTS	WT g	WT %	Sn %	DIST	Fe %	DIST	SiO2 %	DIST
T2 SnCl1Con1	33.4	4.5	2.69	11.7	55.0	7.0	6.75	1.0
SnCl1Con2	74.7	10.0	2.37	23.0	54.7	15.5	6.47	2.2
SnCl1Con3	130.1	17.4	1.66	28.0	53.8	26.6	8.08	4.8
Cl1Tail	59.6	8.0	0.70	5.41	32.7	7.4	26.6	7.3
RoTail	448.1	60.1	0.55	32.0	25.5	43.4	41.0	84.6
CALC	745.9	100.00	1.03	100.0	35.2	100.0	29.1	100.0
ASSAY HEAD			1.08		36.3		29.6	

CUM PRODUCTS	CUM Wt	WT %	Sn %	CUM	Fe %	CUM	SiO2 %	CUM
T2 SnCl1Con1	33.4	4.5	2.69	11.7	55.0	7.0	6.75	1.0
SnCl1Con2	108.1	14.5	2.47	34.6	54.7	22.5	6.56	3.3
SnCl1Con3	238.2	31.9	2.03	62.6	54.2	49.2	7.39	8.1
Cl1Tail	297.8	39.9	1.76	68.0	49.9	56.6	11.2	15.4
FEED	745.9	100.00	1.03	100.0	35.2	100.0	29.1	100.0

The combined cleaner concentrate was only 2.03%Sn content and more pertinently the rougher tail assay was 0.55%Sn with the overall recovery to concentrate of 62.6% of the tin and 49.2% of the iron in the overall cleaner float.

Tin Flotation Result on Inert Media Grind Hole 7 Minerals

PRODUCTS	WT g	WT %	Sn %	DIST	Fe %	DIST	SiO2 %	DIST
T4 SnCl1Con1	34.7	4.6	3.55	14.9	56.6	6.8	5.45	0.9
SnCl1Con2	81.6	10.8	3.15	31.1	58.2	16.5	5.93	2.2
SnCl1Con3	124.6	16.5	1.66	25.0	53.6	23.3	10.3	5.9
Cl1Tail	40.1	5.3	0.66	3.20	31.0	4.3	30.7	5.7
RoTail	472.8	62.7	0.45	25.7	29.8	49.0	39.2	85.3
CALC	753.8	100.0	1.10	100.0	38.1	100.0	28.8	100.0
ASSAY HEAD			1.08		38.7		29.4	

CUM PRODUCTS	CUM Wt	WT %	Sn %	CUM	Fe %	CUM	SiO2 %	CUM
T4 SnCl1Con1	34.7	4.6	3.55	14.9	56.6	6.8	5.45	0.9
SnCl1Con2	116.3	15.4	3.27	46.0	57.7	23.4	5.79	3.1
SnCl1Con3	240.9	32.0	2.44	71.0	55.6	46.6	8.11	9.0
Cl1Tail	281.0	37.3	2.18	74.3	52.1	51.0	11.3	14.7
FEED	753.8	100.0	1.10	100.0	38.1	100.0	28.8	100.0

The improvement in the cleaner concentrate from the inert media ground minerals is significant with a combined cleaner concentrate up to 2.44%Sn content and a rougher tail assay at 0.45%Sn with an overall recovery to concentrate of 71.0% of the tin and only 46.6% of the iron in the cleaner float.

Inert media grinding is now considered as the optimum method for producing a feed mineral for tin flotation but the choice of mill to perform the ultrafine grind of <10 micron is required to ensure maximum cassiterite liberation is achieved and is currently under review. The IsaMill and Deswik Mill are the probable choices, a signature plot has been carried out for the IsaMill on a main ore feed sample and a Deswik test will be carried out at a later date. The budget capital cost of an IsaMill of suitable capacity has been obtained and the power requirements for the grind down to <10 microns are known. The Deswik Mill claims lower power input requirements and capital cost but performance will be ascertained from a comparative test and then a budget price will be requested for a suitably sized mill. Recent commercial units of 100 tonne/hr capacity have been installed so the Deswik Mill can now be regarded as a viable option to the IsaMill. The project will require a pilot plant test to confirm a flowsheet performance and it is known that Xstrata can provide an M20 (20 litre capacity) unit which can input up to 18 kW grinding power, which would be suitable for pilot plant processing of Mt Garnet ores. This would be able to grind at a rate of up to 1 tonne/hr for a pilot plant process prior to SLon magnetic separation and UF Falcon and tin flotation testing.

The laboratory scale IsaMill M4 unit is pictured below. The horizontal stainless steel drum of 4 litre capacity is fed with slurry by a variable speed peristaltic pump into the left hand end of the drum. The ground mineral product is discharged through a wedge wire screen mounted inside the drum and out of the right hand end of the drum. There is a pressure gauge mounted at the entry to the drum to detect pressure buildup in the drum caused by blockages or drum overloading. There is a temperature gauge mounted at the discharge end. Both gauges have maximum set points which trip the mill drive. There are 5 rubber lined discs inside the drum with radial holes in each disc. The discs are mounted on a horizontal shaft which is driven by a variable speed motor from the left hand end of the mill. The media partially fills the drum and the media and mineral slurry is lifted by the radial holes in each disc. The addition of slurry pushes through to the discharge end of the mill and the slurry particles are ground when mixed with the media. There is a solid disc at the end of the mill shaft which has horizontal fingers mounted radially and as they rotate they push the oversize slurry particles back into the mill for regrinding. The ground fines are allowed to discharge through the centrally mounted wedge wire screen. The mill body and central shaft is lined with polyurethane in order to reduce wear and damage to the drum body and central drive shaft. There is a pressure seal on the drive shaft at the drive end which is cooled and maintained at high pressure to prevent slurry entry to the drive system and main bearing on the shaft.

Laboratory Sized M4 IsaMill Testing Unit



Currently it is anticipated that conventional ball milling using steel media can be used to grind to the first stage of a p80 of 50 microns, prior to the use of conventional gravity methods to recover liberated cassiterite. The target grade for this sizing is >40%Sn concentrate product. Thereafter inert media grinding will be utilized to grind gravity tailings to p80 of <10 micron, in order to ensure maximum cassiterite liberation, before high intensity magnetic separation, acid leaching, enhanced gravity separation or a combination of these methods are used to upgrade a tin flotation feed.

T. Magnetic Separation

Low Intensity Magnetic Separation (LIMS)

It became obvious that the initial stage of mineral dressing, after the grinding of the minerals, must be to subject them to low intensity magnetic separation (LIMS). Both mineral samples exhibited a differing proportion of iron oxides being drawn into a LIMS ferromagnetic product when tested. However both LIMS products exhibit similar iron, silica, alumina, phosphorus and tin assay contents. The QEMSCAN examination of this LIMS product, from both Hole samples, shows that the tin content is present as fine particle inclusions of cassiterite, usually at sizes <5 microns. Further processing of the LIMS magnetics would probably not recover cassiterite to any degree and the only

consideration for LIMS treatment would be to optimize the grind size, maximize the field strength for LIMS separation and to operate the LIMS in a rougher and cleaning mode, in order to minimize the entrapment of non-magnetics in this ferromagnetic product. The grade of LIMS product from the results achieved so far should exceed 68%Fe content , with silica at <1.5%SiO₂, alumina at <1.0% and phosphorus at <0.1%. The tin content will probably assay at 0.2-0.25%Sn, and depending on the proportions of Hole 7 to Hole 83/84 mineral within the ore feed, then the tin loss to this product is estimated to be 4%- 20% of tin in feed to the process. The grind size for LIMS testing has been run at p80 ranging from 90 microns down to <4 microns and in general the tin assay in the LIMS magnetic has remained in the 0.30-0.20%Sn range.

XRF Assay Values for Various LIMS Products from Hole 7 and Hole 83/84 Minerals

Hole No.	p80 or size microns	Grind Media	LIMS gauss	% of total weight	Total Fe	Total SiO ₂	Total Al ₂ O ₃	Total P	Total Sn
7	53	Steel	1000	0.4%	65.66%	2.33%	1.07%	0.03%	0.29%
7	53	Sand	1000	1.0%	67.40%	1.57%	0.56%		0.24%
83/84	26	Sand	700	14.3%	68.87%	0.88%	0.34%	0.008%	0.21%
83/84	38	Sand	1000	63.6%	68.80%	0.86%	0.29%		0.26%
83/84	p80=90	Sand	1000	66.7%	69.0%	0.82%	0.29%	0.01%	0.31%
83/84	p80=90	Sand	1000	61.3%	69.4%	0.67%	0.26%	0.01%	0.25%
83/84	<10 of Cyc/OF	MT1	1000	11.8%	67.3%	1.94%	0.87%	0.012%	0.25%
83/84	<10 of Cyc/OF	MT1	1500		68.7%	1.03%	0.48%	0.009%	0.22%
83/84	<10 of Cyc/OF	MT1	1500	23.7%	69.8%	0.55%	0.29%	0.006%	0.165%
83/84	Cyclone/UF 64	MT1	1000		69.02%	0.88%	0.30%	0.010%	0.27%
83/84	Cyclone/UF 90	MT1	1000	20.2%	68.4%	1.67%	0.29%		0.23%
83/84	Cyclone/UF 63	MT1	1000	21.8%	68.8%	1.28%	0.28%		0.23%
83/84	Cyclone/UF 45	MT1	1000	28.2%	68.7%	0.97%	0.29%		0.24%
83/84	Cyclone/UF 38	MT1	1000	3.3%	68.5%	1.07%	0.33%		0.30%
83/84	Cyclone/UF 14	MT1	1000	8.7%	68.9%	0.58%	0.29%		0.29%
83/84	Cyclone/UF 11	MT1	1000	12.6%	68.6%	0.94%	0.37%		0.34%
83/84	Cyclone/OF >38	MT1	1000	23.0%	69.1%	0.94%	0.28%		0.22%
83/84	Cyclone/OF 14	MT1	1000	18.7%	69.7%	0.61%	0.28%		0.21%
83/84	Cyclone/OF 11	MT1	1000	10.9%	69.7%	0.81%	0.30%		0.22%
83/84	Cyclone/OF 7	MT1	1000	11.5%	69.3%	0.79%	0.31%		0.22%
83/84	Cyclone/OF 5	MT1	1000	10.3%	68.9%	0.78%	0.33%		0.22%
83/84	Cyclone/OF 4	MT1	1000	6.5%	69.3%	0.72%	0.32%		0.20%
83/84	Cyclone/OF <4	MT1	1000	19.0%	68.7%	0.84%	0.38%		0.19%

SLon magnetic separation at 10000 gauss on Hole 83/84 minerals at p80 of <10 microns

Assays

Stream	% WT	Fe	SiO2	Al2O3	CaO	Mn	S	K2O	Sn	Cu	P	Zn
SLon Non mags	60.7	38	23	8.35	0.4	0.87	0.02	0.104	1.56	0.55	0.08	0.18
SLon mags	39.3	41.3	19.9	7.43	0.44	0.84	0.02	0.095	1.44	0.51	0.08	0.17
CALC HD FD	100.0	39.3	21.8	7.99	0.42	0.86	0.02	0.100	1.51	0.5	0.1	0.2

Distribution												
Stream	% WT	Fe	SiO2	Al2O3	CaO	Mn	S	K2O	Sn	Cu	P	Zn
SLon Non mags	60.7	58.7	64.1	63.4	58.4	61.5	60.7	62.8	62.6	62.7	62.5	61.7
SLon mags	39.3	41.3	35.9	36.6	41.6	38.5	39.3	37.2	37.4	37.3	37.5	38.3
	100	100	100	100	100	100	100	100	100	100	100	100

The tin flotation results on the SLon non-magnetic products separated at 10000 gauss are shown in section **W**.

An XRD examination of the SLon non magnetic product from Hole 83/84 minerals is shown below.

Phase	Weight%
Goethite	57.4
Kaolin	19.6
Quartz	12.1
Hematite	7.8
Cassiterite/Garnet?	1.4
Smectite clay	1.8
Magnetite	0.0

The SLon magnetic products contained tin levels which were too high to regard as a tailings discard grade, and alternative treatments are required in an attempt to recover the tin from these minerals. It is also considered that a magnetic cleaning stage at a field strength of 8000 gauss in a second SLon unit will remove any residual non magnetic minerals missed in the roughing stage. This should encourage lower tin, silica and alumina levels in the magnetic product.

The treatment of the SLon magnetic minerals should then include the use of reduction roasting which is discussed in sections **X and Y** of this report.

U. Conventional Gravity Tin Mineral Separation

The earlier testwork carried out by Esker (Nick Moony) on the ground mineral samples, which had been subjected to magnetic separation and superpanning, demonstrated that it was possible to recover a >40%Sn content concentrate by gravity methods. The testing undertaken by Consolidated Tin Mines has used commercially available gravity equipment in order produce this grade of concentrate. The QEMSCAN examination of two ore sources has indicated that a portion of the cassiterite can be liberated at >10 microns. The strategy to recover this size of cassiterite particles has to be developed to fit to the considered limit of the conventional equipment.

The conventional gravity equipment considered for use in the process has been fine spirals, Holman tables, Kelsey jigs and Falcon concentrators. The final route for gravity separation can employ a mix of these units and each unit is limited by its ability to separate cassiterite at differing size limits. Therefore the processing strategy must ensure that the mineral sizing presented to each unit falls within the capability of these machines. The experience operation with each of these units dictates the following mineral sizing must be presented in the feed:

1. **Fine spirals**- They are considered to be able to separate efficiently down to a mineral sizing of 30 microns. The unit tested was an FM1 spiral manufactured by the Mineral Technologies Division of Downer EDI. Mineral Technologies manufacture a wide range of spiral designs used throughout the mineral sands and iron ore processing industries worldwide. Their FM1 spirals have been used in the iron ore processing industry for recovery and upgrade of the fines generated in the processing of iron ores. They consider that 30 microns is the lower limit of capability for their FM1 spiral. Spiral operation is very simple and only requires a mineral pulp feed of 30-35% pulp density at a feed rate of 0.8-1.0 tonnes/hr to each spiral. The spirals are usually constructed with 3 individual spirals on a single central mounting tube, fabricated in fiberglass and the spiral trough is lined with polyurethane for long life wear. The spirals are assembled in banks of usually 8 columns per row in two rows mounted in a steel frame. Concentrate, middlings and tailings launders are mounted underneath the rows of spirals to collect these products from each individual spiral. The individual feed of pulp to each spiral is controlled by a central circular feed hopper fitted with polyurethane spigots, each spigot with the same diameter central discharge hole, mounted around the outer edge of the feed hopper. Each spigot discharges a similar pulp flow into a manifold of polyethylene pipes feeding each individual spiral unit. As the pulp flows down the spiral in a circular motion the heavier particles are drawn to the centre of the spiral and the lighter particles are forced by the centripetal force of the circular spinning flow to the outer edge of the spiral trough. At the base of the spiral the concentrate is discharged close to the central tubular column and into a common concentrate collection launder under the spiral bank. There is another discharge hole further along the profile of the spiral where a middlings product can be collected and discharged into the common middlings launder mounted under the spiral bank. At the outer edge of the spiral

trough the tailings are discharged through a discharge hole again into a common tailings trough mounted below each spiral. Operation of the spirals is simple adjustments of each concentrate and middlings discharge hole by adjusting the cutters to each discharge hole. The operator needs to inspect the graduated settings on each cutter to ensure that they comply with settings determined by the metallurgist who in turn has decided them from assays on a spiral survey. The other inspection ensures that all spirals are receiving similar flow rates by checking the spiral troughs and clearing any blocked feed spigots. Maintenance consists of very occasional pressure cleaning of the polyurethane surface lining of the spiral and an annual inspection for wear of the lining coating thickness across the spiral trough.

FM1 Spiral separation of Hole 83/84 >30 micron LIMS non-magnetic product



The photograph shows the separation of dark grey hematite minerals to the central (concentrate) area of the spiral with reddish brown goethite segregated to the outer (tailings) and middlings area of the spiral trough.

The spiral assay results shown below were samples taken for a range of 3 cutter settings during the operation of the FM1 spiral. The results show poor selectivity of tin to the concentrates but high recovery of iron which is assumed to be coarser particles of hematite. There were no further tests

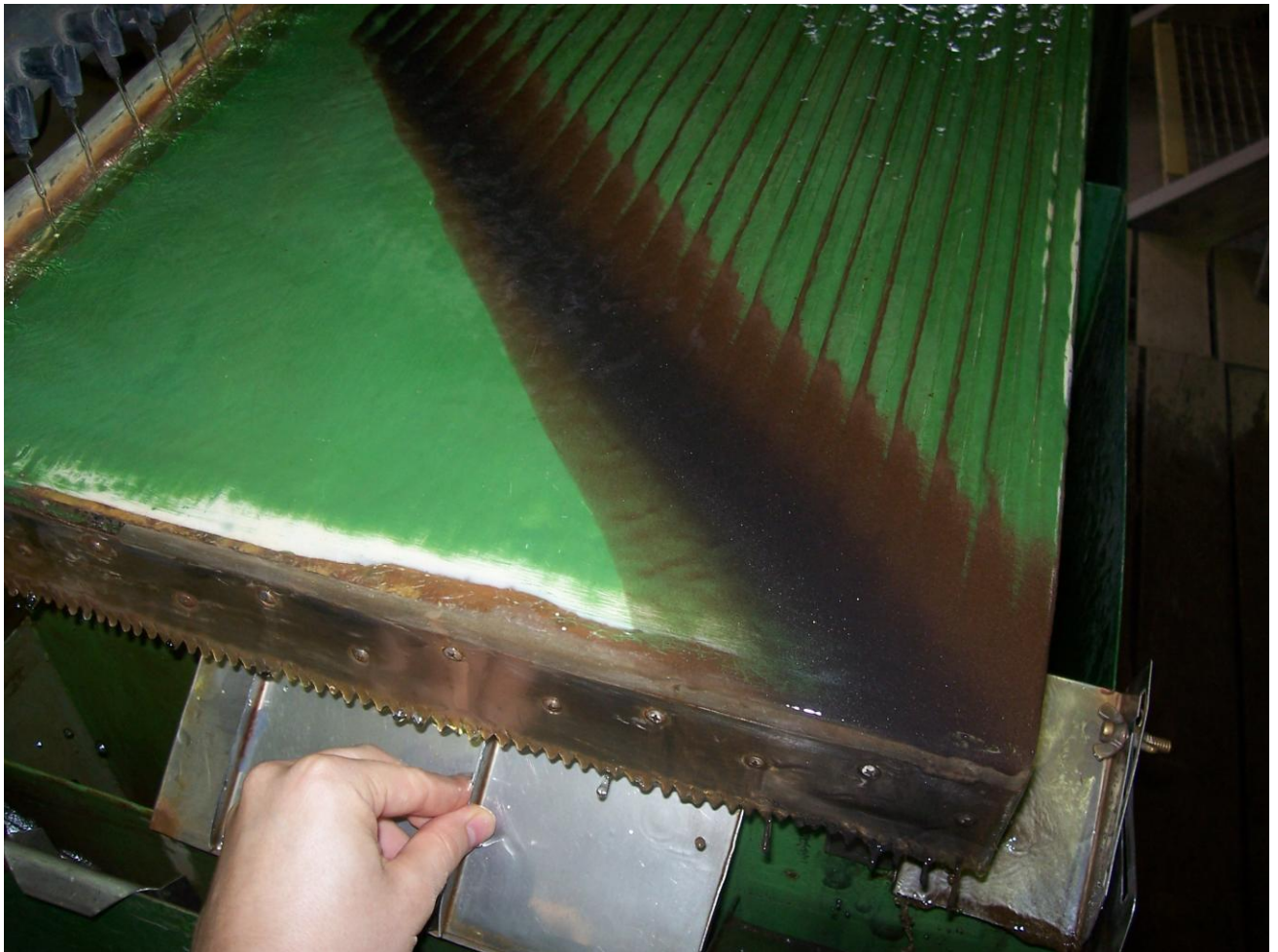
undertaken on treatment of minerals using the FM1 spiral. The use of spirals within the process is not considered suitable to recover tin concentrates.

Description	Fe %	SiO ₂ %	Al ₂ O ₃ %	TiO ₂ %	MgO %	Mn %	P %	CaO %	S %	K ₂ O %	Sn %
T301 Con	62.79	3.79	0.7	0.04	0.06	0.70	0.026	0.31	0.048	0.03	1.13
T302 Con	63.42	2.85	0.62	0.04	0.04	0.65	0.025	0.25	0.06	0.03	1.28
T303 Con	61.74	4.7	0.77	0.04	0.05	0.74	0.028	0.34	0.048	0.04	1.09
T301 Mid	46.97	22.9	1.36	0.04	0.12	0.77	0.045	0.59	0.016	0.08	0.83
T302 Mid	57.26	9.45	1.05	0.04	0.07	0.84	0.037	0.49	0.02	0.05	0.87
T303 Mid	41.65	29.3	1.49	0.04	0.1	0.72	0.047	0.6	0.012	0.1	0.84
T301 Tail	44.38	20	3.27	0.06	0.15	0.86	0.070	0.49	0.016	0.1	1.52
T302 Tail	41.51	25.7	3.04	0.06	0.15	0.78	0.064	0.51	0.012	0.11	1.39
T303 Tail	44.87	19.7	3.34	0.07	0.15	0.87	0.071	0.49	0.016	0.09	1.53

2. Holman Shaking Tables- These units have been used for tin ore processing for many years and their unique stroke action provides the optimum separation of fine cassiterite from finely ground products. They are fabricated in the UK by Holman-Wilfley Ltd and marketed in Australia by Downer EDI Mineral Technologies. These units are capable of recovering fine cassiterite below the limits of the FM1 spirals and test results on the cleaning tests on the FM1 concentrate product are shown below. It shows that a concentrate grade of 59.2%Sn content at a recovery value of 17.1% was achieved off the Hole 83/84 mineral sample previously upgraded from a Kelsey Jig treatment. This represented an overall recovery of 7.5% from the total feed from Hole 83/84. The results from the Hole 7 minerals is assumed to be higher in recovery as the testing on a screened >38 micron sample achieved 33% recovery of the tin content in the table feed to a 61.8%Sn concentrate. It has been assumed that Holman tables will be used to generate a >40%Sn concentrate off a mineral sizing feed down to >10 microns, which has previously been upgraded through a Kelsey Jig . The shaking tables will be used as a concentrate cleaning method for either a concentrate recovered from a J1300 Kelsey Jig section in the circuit.

The operational adjustment for each table consists of setting the water flow across each table surface, this maintains a flowing film of water across the horizontal riffles which are fitted to the table surface. A forward/reverse shaking motion is applied to the table at right angles to the flow of water across the table surface. The table surface is tilted to allow the water to flow across the table. The mineral slurry is fed to the top corner of the table and the shaking motion of the table forces the minerals to spread across the table the heavy minerals are trapped in the riffles and the lighter gangue minerals are washed over each riffle and eventually are discharged at the long edge of the table. The heavy concentrates are carried along the riffles, pushed forward by the shaking motion and they are finally discharged at the short side of the table at the opposite end to the feed point. The parameters of operation of the table is the waterflow, stroke length and frequency of the shaking motion on the table, feed rate of slurry to the table, tilt angle of the table top, and the

setting of the launder cutters to produce tailings, middlings and concentrates. In the case of Mt Garnet ores visual settings of concentrate cutters are easy as the colour difference of cassiterite, magnetite, hematite, goethite and gangue minerals is markedly different as can be seen on the attached photographs. The table tops required for fine mineral separation are specific to the fine mineral separation that is required in the process. The cassiterite band is clearly visible on the photograph below of the cleaner concentrate tabling of Hole 83/84 minerals. The brown strip at the head of the riffle discharge is the cassiterite, the hematite dark grey band is behind the cassiterite and the red-brown goethite band is being discharged along the tailings edge of the table.



This photograph shows the concentrate cutter is positioned to cut the light brown cassiterite minerals from the hematite (dark grey) and reddish-brown (goethite) going to tails on the right-hand edge of the table.

- Kelsey Jigs- The Kelsey Jig was invented and developed by Chris Kelsey during the 1980s and the first commercial units J650 and J1300 were commissioned at the Renison Bell Mine during the early 1990s. They have the ability to recover free cassiterite particles from a mixture of minerals with a wide SG range and at a sizing down to 10 microns and below (with respect to the high SG of cassiterite at 7.0). This allows our gravity recovery to perform at a lower mineral sizing than the fine spirals (FM1) previously tested. The laboratory testing unit is the J200, and this unit is used to provide indicative performance data for the commercial scale units, which the J1300 Kelsey Jig is now the smallest currently available unit. Downer EDI Mineral Technologies are now the manufacturers and marketers of the J1300 Kelsey Jig. The maximum capacity of the J1300 Kelsey jig is 30 tonnes/hr, but this is dependent on concentrate mass flow and tailings volumetric flow limits. The number of J1300 units required would need to be determined from testwork results, once concentrate mass yields and tails volumes are understood then 2 units may be included in the processing circuit. The capital, operating and maintenance costs for these units are significantly higher than a spiral circuit but they provide the benefit of a lower cassiterite mineral recovery size than the spiral circuit capability, and this in turn should enable a higher recovery of >40%Sn concentrates to be achieved. The secret of the jig is to subject the mineral feed to a centrifugal force of up to 50G and at the same time the minerals are introduced to a pulsed flow through a bed of coarse minerals termed 'ragging' with an SG between cassiterite and the gangue minerals. This allows the lighter SG particles to be mostly rejected at the entry to the ragging bed, while the heavier SG minerals, in particular cassiterite, pass through the ragging and then through the cylindrical ragging retention screen and into the concentrate hatches behind the screen. The concentrate minerals then exit through a single hole spigot in each concentrate hatch. The concentrate hatch is filled with pressurized water entering through another fixed size water nozzle at the base of the hatch. The water in each concentrate hatch is pulsed by a mechanical system deflecting a rubber diaphragm at the side of each hatch. The J1300 jig has 16 separate concentrate hatches. The hatches are formed into a single spinning rotor casing and the diaphragms to each hatch are pulsed from a main cam type casting and pulse arms which pulse each hatch in turn as the main rotor spins. The tailings (low SG minerals) are carried over the surface of the spinning bed of ragging and overflow into a circular rubber lined launder. The concentrates discharging from each hatch spigot are collected in an outer circular rubber lined launder. There is one electric motor driving the main hatch rotor and another motor driving the pulsing rotor. Each motor has variable speed adjustment in order that the pulse and spin speeds can be adjusted to maintain an acceptable grade and recovery of cassiterite concentrate. The jig ragging material typically used for cassiterite recovery is an accurately sized magnetite mineral. However a higher SG material may be required if significant high SG iron oxide minerals require to be rejected to tailings. The jig ragging retaining screen is constructed from stainless steel wedge wire material and, considering our p80 feed sizing of around 50 microns and pending testwork results, the aperture of this wedge wire mesh will probably need to be 200 microns. Consequently the minimum ragging mineral size will need to be $200/0.6 = 355$ microns, and the top size should be 500microns. Each jig needs to have a ragging recovery screen on the tailings discharge and the screen needs to recover ragging particles which are drawn into the jig tailings from the top of

the ragging bed. The screen oversize, usually ragging, is returned into the incoming feed in order to maintain an optimum ragging bed depth formed against the ragging screen. This description illustrates the intricate construction and operation of these units. Daily checking of the operations of these units requires experienced operators and sophisticated PLC control. Mineral Technologies run a service exchange system for each jig sold, which is based on an 8000hr maintenance interval, with a fee at each maintenance inspection interval. The units do require regular inspections on a shift operating basis to check for blocked concentrate spigots, which if blocked can cause damage to the hutch pulse levers or diaphragms. All new units are fitted with automatic screen cleaning units which require a short-term diversion of feed material while the ragging retention screen is cleaned to remove any pegged feed or ragging particles from the screen. Units are fitted with vibration detection to detect any out- of- balance movements which may be caused by blocked concentrate spigots/hutches. If the interlock between the vibration detectors and PLC does not stop the unit quickly enough, then the hutch pulse arms are fitted with shear pins which break when a hutch is blocked, to prevent diaphragm or pulse arm damage. The ragging recovery screen must be regularly checked for the presence of excess ragging on the screen and if not extra fresh ragging must be added.

In summary it is evident that this type of unit can provide recovery benefits for finer sized minerals as opposed to the spiral operation, but operator skills and maintenance requirements make the choice for this equipment dependent heavily on an improved recovery of cassiterite. Mineral Technologies currently offer KCJ Modular Plants, which incorporate all the essential ancillary equipment required for successful operation of a Kelsey Jig circuit.

J200 Kelsey Jig assay results on Hole 83/84 samples

Description	Fe %	SiO ₂ %	Al ₂ O ₃ %	MgO %	MnO %	P %	CaO %	S %	K ₂ O %	Sn %
J200 Feed (start)	56.42	10.4	1.14	0.11	0.94	0.034	0.45	0.036	0.04	1.06
T607 Con	64.26	1.55	0.53	0.04	0.71	0.021	0.16	0.012	0.01	2.16
T607 Tail	54.25	13.1	1.30	0.1	1.03	0.039	0.48	0.024	0.05	0.76
T608 Con	64.75	1.02	0.43	0.03	0.61	0.017	0.13	0.012	0.01	2.75
T608 Tail	54.95	12.4	1.28	0.1	1.02	0.038	0.46	0.028	0.05	0.79
T609 Con	63.14	0.9	0.37	0.02	0.54	0.015	0.14	0.012	0.01	4.36
T609 Tail	55.09	12.2	1.30	0.12	1.01	0.039	0.45	0.02	0.05	0.87
J200 Feed (End)	58.03	9.52	1.00	0.09	0.91	0.031	0.42	0.032	0.04	0.98

These tests were carried out on a bulk sample using zirconia ragging of 300-412 micron size, using a 200 micron ragging screen, run at maximum spin speed and high pulse rate. The 3 tests

were carried out at differing water addition rate to the concentrate hutch and at the highest waterflow rate the mineral pulp feed was doubled. The pulp density of the feed was maintained at 30-35% solids throughout the tests. These results demonstrate a high upgrading to concentrate of up to 4:1.

Final Metallurgical Balance on the Hole 83/84 Minerals using LIMS/Kelsey Jig/Tabling methods

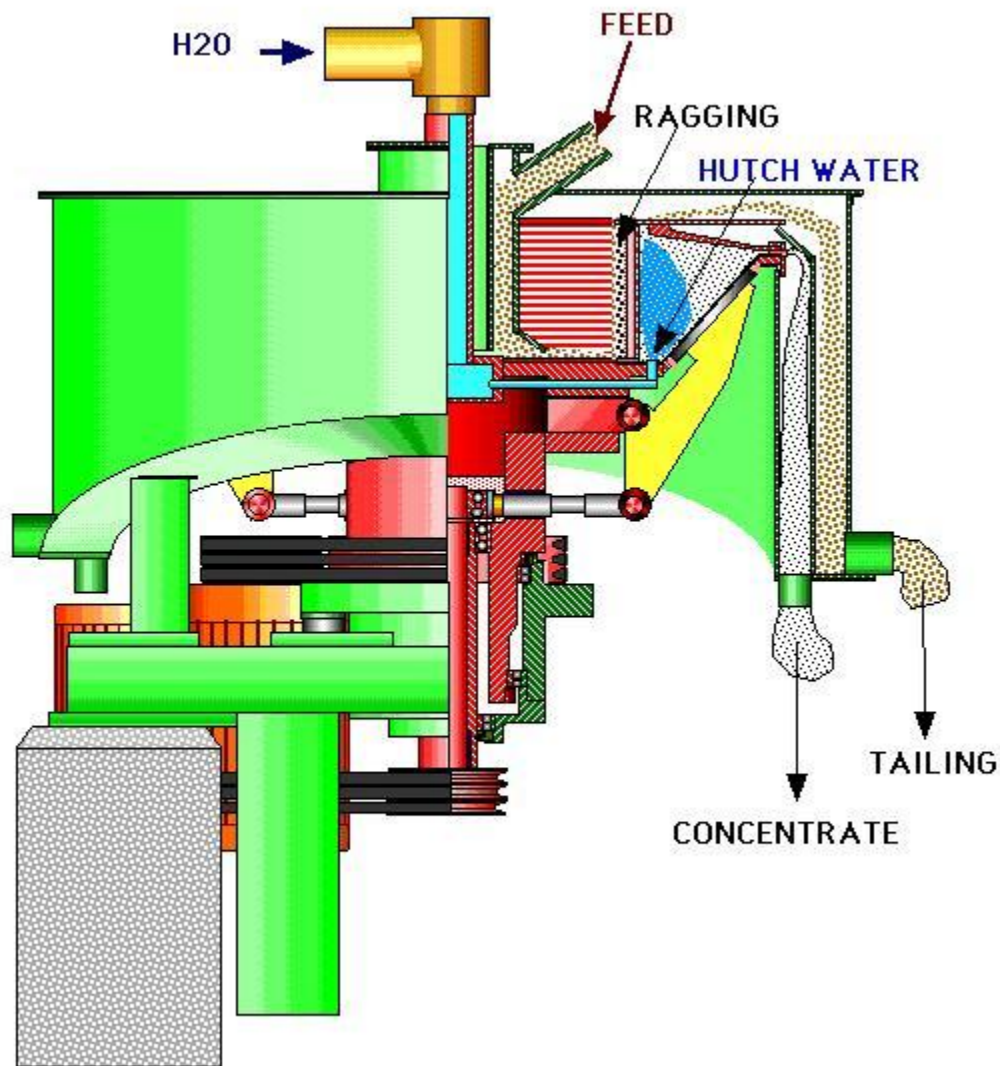
NAME	Weight %	Assay					Distribution				
		Fe %	SiO ₂ %	Al ₂ O ₃ %	LOI ₁₀₀₀ %	Sn %	Fe %	SiO ₂ %	Al ₂ O ₃ %	Sn %	
Cleaner LIMS Mag	45.7	67.7	0.88	0.30	-0.67	0.27	51.3	5.2	15.0	16.1	
Cleaner LIMS NM	54.3	61.0	3.99	0.87	1.84	1.88					
Kelsey Jig Tail	36.8	50.3	18.3	1.70	3.64	0.84	30.7	87.0	68.3	40.3	
Rougher table Tail	9.20	59.5	5.72	1.21	2.83	1.05	9.1	6.8	12.2	12.6	
Rougher Table Mid	7.36	64.7	0.95	0.52	0.80	1.50	7.9	0.9	4.2	14.4	
Cleaner Table Mid	0.92	60.5	0.42	0.33	0.43	7.61	0.9	0.1	0.3	9.1	
Cleaner Table Con	0.10	9.93	0.47	0.10	0.10	59.2	0.0	0.0	0.0	7.5	
Total	100.0	60.2	7.73	0.91	1.36	0.77	100.0	100.0	100.0	100.0	

These results reveal the following features of the LIMS and gravity treatment of Hole 83/84 minerals:

1. LIMS separation loses 16% of the tin to a high iron product. The high iron product contains low impurities of silica and alumina but with a tin content of 0.27%Sn it is not marketable for iron production. A further treatment process is required for this product which is discussed in section Y.
2. The J200 Kelsey Jig separation gave a tin loss to tail of over 40%. This value may be reduced if the sizing on the jig feed is reduced to >10 microns rather than the >15 micron cyclone cut used for the test. The bulk ore mixture test will use a >10micron cut for the Kelsey Jig feed, thus allowing more free cassiterite particles to be presented to the jig for recovery. The high zirconia ragging, with an SG of 6.2, provides greater selectivity during the jiggling action on the high SG hematite particles with an SG of 5.3, than the magnetite ragging with an SG of 5.15.
3. The rougher table tails would have to be recycled back to the jig for further recovery of the free cassiterite that may be present. The rougher table middlings provide another high iron mineral product which contains significant tin content that would be subjected to the tin fuming process that is discussed in section Y.
4. The cleaner table operation can be modified to lower the table concentrate grade to the target of 40%Sn, which would provide an increase in tin recovery to concentrate. Also the

cleaner table middling/tail would be recycled back to the rougher table in order to improve rougher table recoveries and grades.

Sectioned View of a J1300 Kelsey Jig



4. Falcon Centrifugal Concentrators- These units are produced and marketed by Sepro Systems based at Langley, British Columbia, Canada. Their Australian agents are Western Process Equipment, Western Australia. Basically they are a centrally fed centrifuge where the heavier mineral particles in the slurry feed are pushed against the outer wall of the bowl of the unit by centrifugal force of up to 600G. The minerals layer the surface of the bowl with the highest SG and coarsest particles closest to the wall. The feed pulp washing up the wall forces the lighter SG and finer particles to be washed from the mineral layer and they are discharged over the top of the spinning bowl as tailings. The concentrate or heavies are retained against the wall and can then be collected in an area under the top lip of the spinning bowl, and in the case of the

continuous feed C2000 Falcon concentrator, the dense concentrate slurry is discharged continuously through air pressure controlled spigots (termed muscles) into a circular concentrate launder. The capacity of the continuous Falcon concentrator C2000 unit is 60 tonnes per hour. This unit can exert up to 400G forces on the slurry particles and is capable of the selective recovery of concentrate particles down to 20 microns. This unit could be considered to provide a roughing stage in front of a J1300 Kelsey jig, the C2000 Falcon can cope with a cyclone underflow product after the LIMS treatment, and the Falcon concentrate of 20 tonnes/hr can feed the Kelsey jig. Both enhanced gravity units can complement each other within the circuit. The C2000 Falcon has lower capital and operating costs than a second Kelsey Jig, but lacks the ability to recover <20 micron particles which the Kelsey jig can achieve down to 10 microns.

The use of the UF Falcon on the tin flotation feed has been carried out on the SLon non magnetic product and results are shown in the table below. There is a unit which can cope with very fine mineral feed, the UF Falcon, which imposes gravitational force of up to 600G on the mineral particles of the feed slurry. The UF1500 production unit can process around 3 tonnes of minerals per hour and can produce cassiterite upgrades of 3:1 from feed to concentrate. The benefit of this unit would be felt when upgrading flotation feed minerals. Flotation feed can contain ultrafine particles, <3micron slimes, which get carried into the flotation froth, the use of this unit will be considered for desliming of flotation feed. The high G forces within the unit will ensure that the >3 micron cassiterite particles can be retained in the UF Falcon concentrate. The slime particles of <3 microns are rejected in the tail discharge from the bowl overflow during each batch processing.

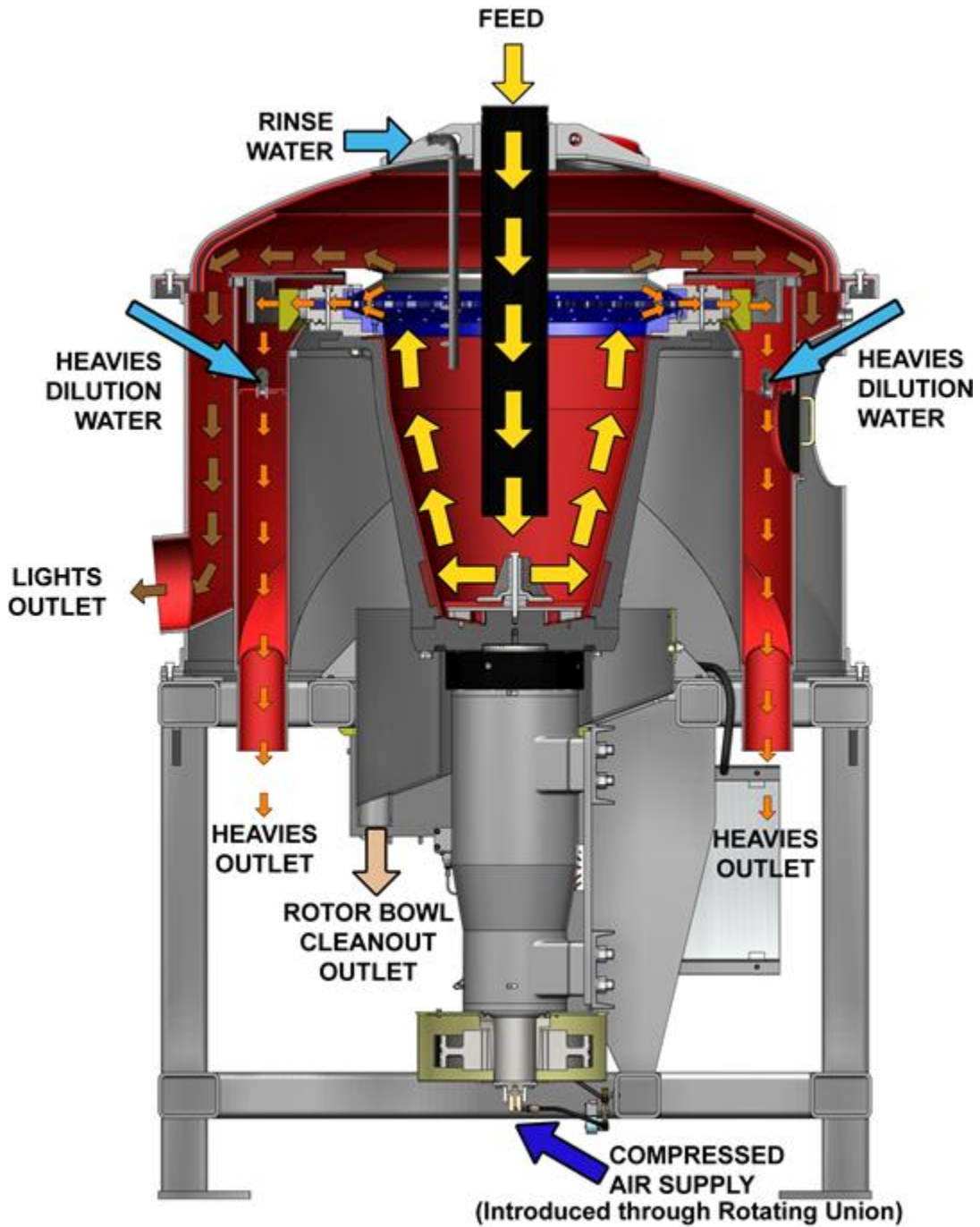
UF100 Falcon test results on a SLon Non Magnetic Product- Hole 83/84 Minerals with p80 of <10 microns

Product	Feed on Time (min)	Product Mass (gm)	Product Mass (%)	Calc Feed Mass (gm)	Calc Mass to Conc (gm)	Calc Mass to Conc (%)	Sn Tail (%)	Sn Dist (%)	Sn Cum Rec (%)	Sn Calc Conc (%)
Tail	3	472.6	64.0	738.6	266.0	36.0	0.97	44.6	55.4	2.14
F22 Conc		266.0	36.0				2.14	55.4		
Total		738.6	100.0				1.39	100		
Head							1.40			

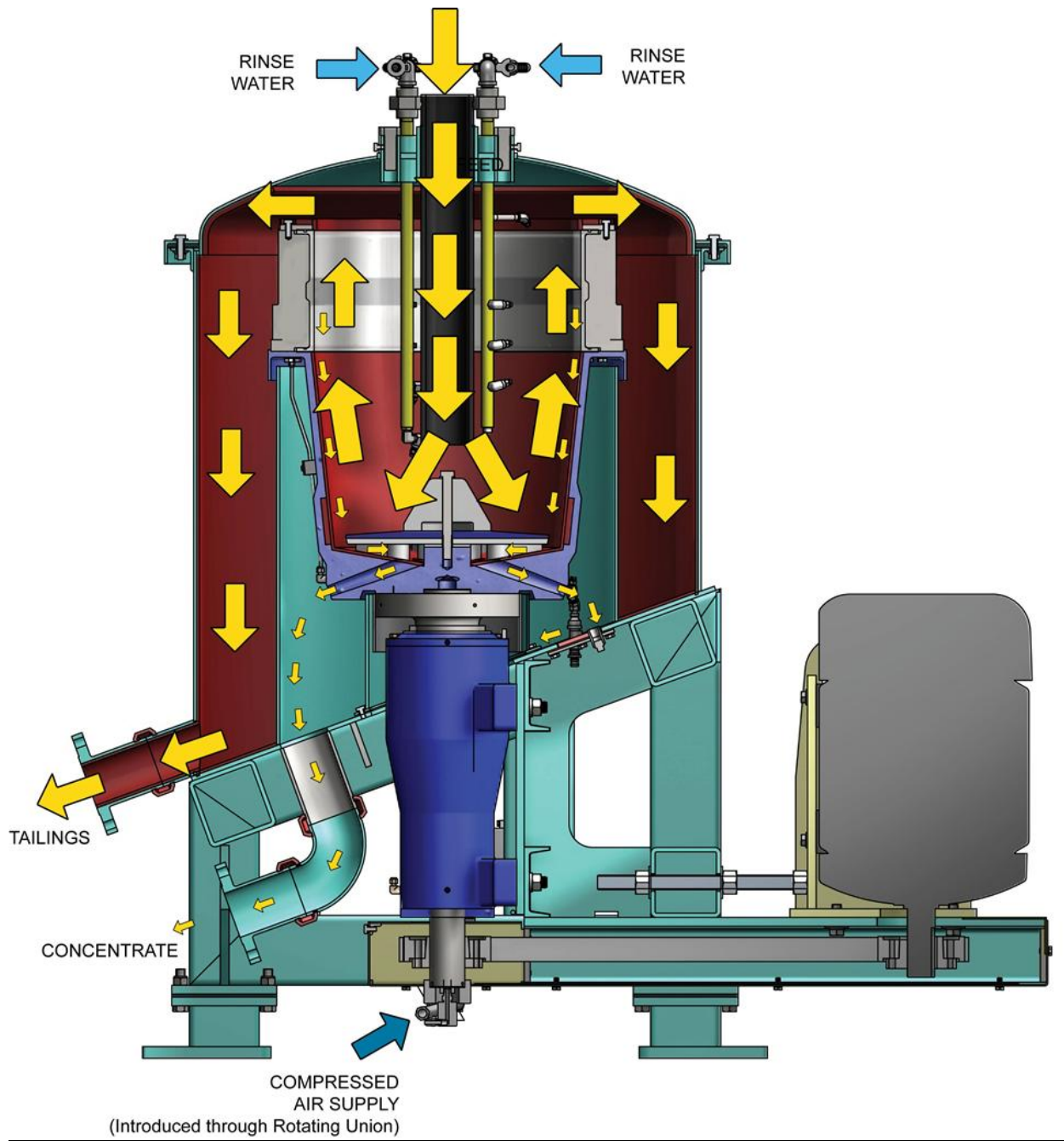
	Fe %	Dist %	SiO2 %	Dist %
Tail	34.81	59.0	25.2	73.1
F22 Conc	42.9	41.0	16.5	26.9
Calc Head	37.7	100.0	22.1	100.0
Assay Head	38.0		22.1	

These are results from a single stage test and demonstrate the ability of the UF Falcon to provide an upgrade to the SLon non magnetic product prior to the tin flotation. Further tests may allow an increase in recovery of tin to be achieved. A second stage scavenging unit would allow a higher tin recovery value to be achieved and consequently a lower tin grade tailing to be produced.

Schematic View of C2000 Falcon Continuous Concentrator



Schematic View of UF1500 Falcon Concentrator



V. Soluble Tin Leaching

The previous work carried out when RGC owned the leases, included a series of leach tests to assess the dissolution of iron and tin in 30% sulphuric acid solution at 95 degrees C over a series of leach times. It was considered that gravity and flotation methods would not allow this portion of the tin to be recovered from the ores. This soluble tin component was named as 'Gillianite' and was found to be strongly associated with the goethite content of the ores. Generally it is considered that the higher the goethite content of an ore from the Mt Garnet area, then the soluble tin portion will be higher. The recovery of a tin product from this component of the ores must be attempted in order to enhance the overall tin recovery. Hole 7 minerals contain significant goethite levels, with in excess of 30% of the iron associated with goethite and the remainder of iron represented as iron oxides. Hole 83/84 minerals have 90% of the iron as either magnetite or hematite with only 10% of the iron as goethite. Initial leach tests were carried out on a sample of Hole 7 minerals which had been subjected to WHIMS magnetic separation at 8000 gauss. This was the non-magnetic fraction with 65.5% of the iron associated as goethite and the remainder as iron oxides. The table below shows the assay results of the leach filtrate and leach residues.

The recovery of the soluble tin from the resultant leach filtrate is difficult as the level of iron sulphates within the filtrate is far too high to selectively recover the tin metal by electrolysis. Further testwork to selectively precipitate the iron hydroxide from the filtrate may prove to be method of allowing tin metal to be recovered from the leach solution. The Albion process developed by Xstrata for the recovery of copper from refractory oxide copper ores may offer a similar method for the recovery of tin. The Jarosite process may also offer a treatment route to selectively precipitate the iron sulphate from the leach filtrate and leave the tin in solution.

XRD Results on the Sulphuric Acid Leaching of Hole 7 Non-magnetic Minerals

Sample 512126 was the leach feed minerals, which was a WHIMS non-magnetic product from the treatment of Hole 7 minerals. They were the screen undersize at <38 microns. The assay was 0.75%Sn and 26.7%Fe.

Sample 512143 was the acid leach residue left after a 2 hour leach in 30% sulphuric acid maintained at a leach temperature of 90°C. The assay was 1.27%Sn and 12.6%Fe.

Sample 512139 was the tin flotation first cleaner concentrate product which assayed at 4.22%Sn and 53.8%Fe.

Sample 512142 was the tin flotation rougher tailing, which still showed high tin grade. The assay was 0.87%Sn and the iron was 6.9%Fe.

The XRD examination showed that a significant amount of goethite was dissolved, but hematite and goethite was floated with these minerals. Also a finer grind may release more cassiterite for flotation but the hematite will follow the cassiterite into the flotation concentrate.

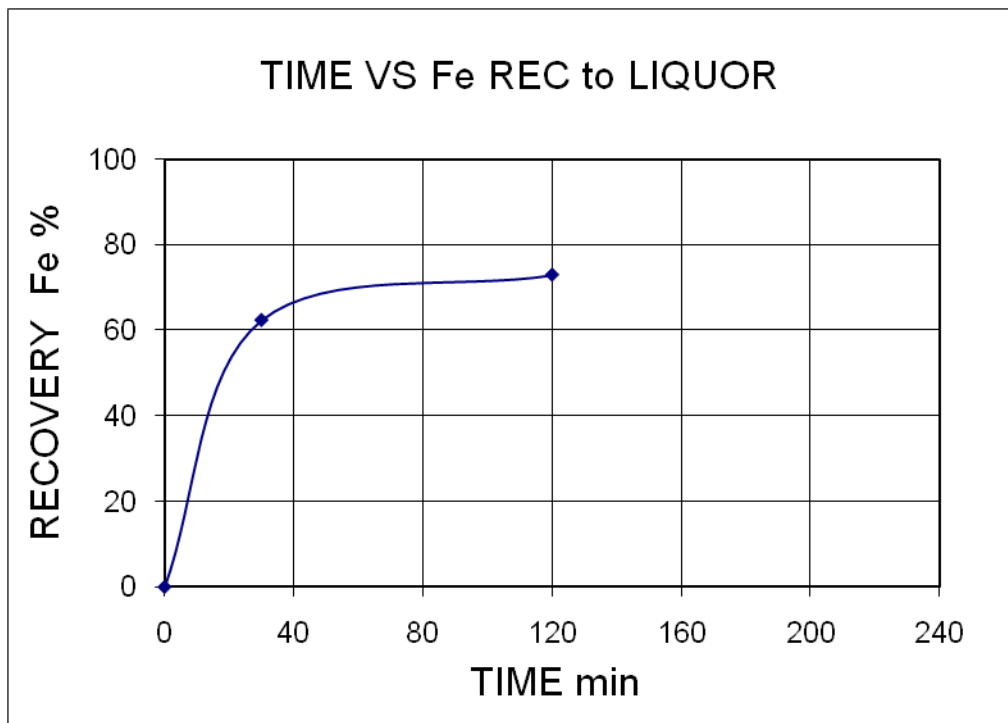
XRD Summary of Mineral Content of Hole 7 Leach and Tin Flotation Products

Phase	512126	512143	512139	512142
Cassiterite	1.5	2	6.1	1.3
Goethite	32.1	6.4	12.2	4.1
Hematite	11.9	13	72.8	7.1
Kaolinite T	34.1	38.3	1	48.9
Smectite clay	3.6	1.1	0	2.7
Muscovite	2.9	2.3	0.3	3.8
Orthoclase 1	2.2	6.3	0	5.7
Quartz	11.7	30.6	7.6	26.3

The details of the leach test are shown below and the graph shows that the iron dissolution slows considerably after 30 minutes. Note the high dissolution of copper from these minerals. Only a small proportion of the tin has dissolved from these minerals, indicating that most of the tin is as cassiterite in this Hole 7 sample.

TIME min	LIQUOR TENORS								REC TO LIQ		
	TEMP deg	Fe ²⁺ g/L	Fe ³⁺ g/L	FeT g/L	Sn mg/L	Cu g/L	FA g/L	SG	Fe %	Sn %	Cu %
0	90	0	0	0	0	0	300	1.148	0	0	0
30	90	0	0.0	33.5	35.4	1.47	0	1.256	62.3	5	91.9
120	90	2.5	37.3	39.8	42.0	1.59	136	1.256	72.9	5.3	97.8

Stream	Mass gm	Vol L	Fe		Sn		Cu	
			gm	REC	gm	REC	gm	REC
Start Liquor	1144.4	1	0.0	0.0	0.0	0.0	0.0	0.0
Final Liquor	1236.3	0.984	39.2	72.9	0.041	5.3	1.57	97.8
Leach Residue	115.6		14.6	27.1	0.73	94.7	0.03	2.2
Calc Feed	200.0		53.8	100.0	0.78	100.0	1.60	100.0
Assay Feed			53.5		0.75		1.54	



The tin flotation results on the leach residue minerals are detailed in the table below. The grade from the float has improved over a similar non leached sample but the recovery remains unacceptable at only 42.3% of the tin to the rougher concentrate. This needs to exceed 60% for leaching to be regarded as a success.

Tin Flotation Results on Leach Residue

PRODUCTS	WT g	WT %	Sn %	DIST	Fe %	DIST	SiO2 %	DIST
T21 RoC1	7.5	6.6	4.22	22.2	53.8	29.9	6.31	0.8
C2	4.9	4.3	3.25	11.2	34.7	12.6	24.7	2.2
C3	6.2	5.5	2.07	8.99	20.0	9.2	39.8	4.4
RoT	94.6	83.6	0.87	57.7	6.91	48.4	55.2	92.6
CALC	113.2	100.0	1.26	100.0	11.9	100.0	49.8	100.0
ASSAY HEAD			1.27		12.6		51.0	

CUM PRODUCTS	CUM Wt	WT %	Sn %	CUM	Fe %	CUM	SiO2 %	CUM
T21 RoC1	7.5	6.6	4.22	22.2	53.8	29.9	6.31	0.8
C2	12.4	11.0	3.84	33.3	46.2	42.4	13.6	3.0
C3	18.6	16.4	3.25	42.3	37.5	51.6	22.3	7.4
FEED	113.2	100.0	1.26	100.0	11.9	100.0	49.8	100.0

This report below described the mineralogical examination of two size fractions of the acid leach residue tin flotation tails, identifying significant free cassiterite remaining in both fractions.

Results

Composition

The following minerals occur:

- *Quartz*
- *Unidentified gangue*
- *Poorly crystallised, unidentified ferruginous material*
- *Hematite* (mainly in the +8µm fraction)
- *Cassiterite*
- Assumed contaminant pyrite, pyrrhotite, sphalerite, galena, chalcopyrite (mainly in the +19µm fraction)

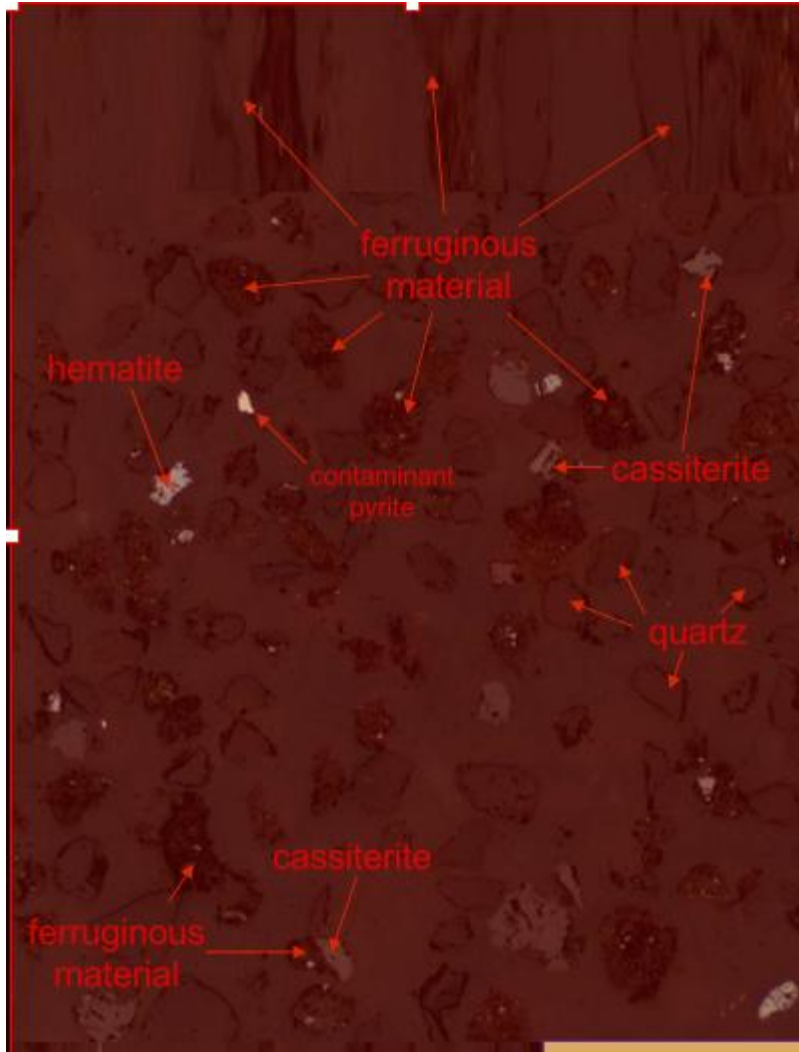
No other Sn minerals were observed.

Cassiterite liberation

The *cassiterite* present is mostly liberated, but some minor ultrafine *cassiterite* occurs within the *ferruginous material*.

Gillian Leach Residue Rghr Tail Cassiterite Distribution%

Fraction	Liberated	Binary with gangue	Ternary with ferruginous material
+19µm	84	13	3
+8µm	90	8	2



This photomicrograph shows the presence of free cassiterite particles within the +19 micron sample from the acid leach tin flotation tailings. At this stage it is assumed that the remaining free cassiterite particles have not been recovered from the leach residue because the collector has been exhausted due to the tendency of the hematite particles to be activated to float in preference to the cassiterite particles. The hematite must be minimized before the efficiency of cassiterite collection can be improved in the tin flotation process.

W. Tin Flotation

Due to the very fine size of the cassiterite minerals in both of the ores tested the majority of tin recovery must be achieved from flotation. The liberation of cassiterite must be achieved by grinding the ores to very fine size after the >10 micron liberated particles have been recovered by gravity methods. Initial flotation tests on Hole 7 minerals of non-magnetic WHIMS 8000 gauss which had been subjected to LIMS and WHIMS magnetic separation, after firstly being ground to a p80 of 53 microns, did not produce an acceptable concentrate tin grade. All concentrates showing heavy contamination with iron oxides. The majority of flotation tests reported below

used styrene phosphonic acid (SPA) as the collector at a float pH of 4.8-5.0 and after firstly conditioning the minerals with sodium silico fluoride (SSF). Efforts were made to depress the iron oxides with some success, but this tended to reduce the tin recovery down to unacceptable levels also. Further iron depression was attempted using sodium metabisulphite (SMBS) in order to use SO₂ to depress the iron oxides, with some indication of success this was used in later tests. In order to ensure that maximum cassiterite liberation was present in the float feed a test was run on <10 micron samples from Hole 83/84 minerals after first removing the ferromagnetic portion using a LIMS. This test again demonstrated the heavy contamination of iron oxides and goethite within the float concentrate.

The performance of the earlier flotation tests were disappointing with poor recoveries and grades, and it was concluded that the problem of iron oxide or goethite contamination in the float must be resolved prior to attempting to recover the cassiterite through flotation. The strategy was to fine grind to <10 microns, then either remove iron oxides by the use of the SLon equipment, and/or to use sulphuric acid leaching. The use of both methods after fine grinding would allow most cassiterite particles to be liberated, and minimize the goethite and iron oxides from the tin flotation feed.

Typical tin flotation result on Hole 7 non-magnetic minerals is tabled below:

PRODUCTS	WT g	WT %	Sn %	DIST	Fe %	DIST	SiO ₂ %	DIST
T07								
SnCl1Con1	24.3	3.56	3.91	15.8	50.8	6.69	6.67	0.68
SnCl1Con2	28.8	4.22	2.79	13.4	48.5	7.57	9.31	1.13
SnCl1Con3	35.2	5.15	1.98	11.6	44.4	8.47	13.6	2.01
Cl1Tail	120.7	17.7	0.79	15.9	29.3	19.2	28.5	14.4
RoTail	473.9	69.4	0.55	43.4	22.6	58.1	41.1	81.8
CALC	682.9	100.0	0.88	100.0	27.0	100.0	34.9	100.0
ASSAY HEAD								
CUM PRODS								
CUM PRODS	CUM Wt	WT %	Sn %	CUM	Fe %	CUM	SiO ₂ %	CUM
T07								
SnCl1Con1	24.3	3.56	3.91	15.8	50.8	6.69	6.67	0.68
SnCl1Con2	53.1	7.776	3.30	29.2	49.5	14.3	8.10	1.81
SnCl1Con3	88.3	12.9	2.78	40.8	47.5	22.7	10.3	3.82
Cl1Tail	209.0	30.6	1.63	56.6	37.0	41.9	20.8	18.2
FEED	682.9	100.0	0.88	100.0	27.0	100.0	34.9	100.0

An improvement in concentrate grade is achieved when the flotation feed minerals are sourced from a UF Falcon concentrate product after 400G force treatment, this is tabled below:

PRODUCTS	WT g	WT %	Sn %	DIST	Fe %	DIST	SiO2 %	DIST
T09 SnCl1C1	25.9	7.1	7.82	18.7	49.5	9.8	4.36	1.2
SnCl1C2	47.9	13.1	6.25	27.7	51.5	18.9	4.58	2.4
SnCl1C3	70.5	19.2	4.14	27.0	50.9	27.5	7.28	5.6
Cl1T	54.5	14.9	2.48	12.5	35.8	15.0	23.4	13.9
RoT	167.7	45.8	0.91	14.1	22.4	28.8	42.2	76.9
CALC	366.5	100.0	2.95	100.0	35.6	100.0	25.1	100.0
ASSAY HEAD			2.88		35.8		25.7	

CUM PRODUCTS	CUM Wt	WT %	Sn %	CUM	Fe %	CUM	SiO2 %	CUM
T09 SnCl1C1	25.9	7.1	7.82	18.7	49.5	9.8	4.36	1.2
SnCl1C2	73.8	20.1	6.80	46.4	50.8	28.7	4.50	3.6
SnCl1C3	144.3	39.4	5.50	73.4	50.9	56.2	5.86	9.2
Cl1T	198.8	54.2	4.67	85.9	46.7	71.2	10.7	23.1
FEED	366.5	100.0	2.95	100.0	35.6	100.0	25.1	100.0

This test was carried out on Hole 7 minerals with a p80 of 50microns, which had been subjected to magnetic separation. They were known to contain a high proportion of liberated cassiterite.

The suppression of iron oxides within the float was tested with both dextrin(starch) and sodium metabisulphite(SO2) both tests are shown below:

The dextrin suppression test below used the addition at 150 gm/tonne to the roughing stage of the float in order to minimize the iron oxides being carried into the cleaning stage.

PRODUCTS	WT g	WT %	Sn %	DIST	Fe %	DIST	SiO2 %	DIST
T6 SnCl1Con1	34.8	4.6	3.18	13.5	57.5	7.1	3.81	0.6
SnCl1Con2	51.4	6.8	2.80	17.6	56.8	10.4	4.62	1.1
SnCl1Con3	44.3	5.9	2.12	11.5	54.2	8.6	8.37	1.7
Cl1Tail	91.1	12.1	1.17	13.0	34.7	11.3	25.6	10.9
RoTail	534.4	70.7	0.68	44.4	32.8	62.6	34.4	85.7
CALC	756.0	100.0	1.08	100.0	37.0	100.0	28.4	100.0
ASSAY HEAD			1.08		38.7		29.4	

CUM PRODUCTS	CUM Wt	WT %	Sn %	CUM	Fe %	CUM	SiO2 %	CUM
T6 SnCl1Con1	34.8	4.6	3.18	13.5	57.5	7.1	3.81	0.6
SnCl1Con2	86.2	11.4	2.95	31.1	57.1	17.6	4.29	1.7
SnCl1Con3	130.5	17.3	2.67	42.6	56.1	26.1	5.68	3.5
Cl1Tail	221.6	29.3	2.05	55.6	47.3	37.4	13.9	14.3
FEED	756.0	100.0	1.08	100.0	37.0	100.0	28.4	100.0

Sodium metabisulphite (SMBS) was added to the standard tin flotation, this releases SO₂ into the pulp which prevents the iron oxide surfaces from being activated by the collector. The SMBS addition was made to the rougher float concentrate at a rate of 600 gm/tonne.

PRODUCTS	WT g	WT %	Sn %	DIST	Fe %	DIST	SiO ₂ %	DIST
T17 SnCl1 C1	22.0	2.7	2.15	7.26	40.0	3.8	17.4	1.5
SnCl1 C2	29.4	3.6	1.58	7.13	38.8	5.0	18.9	2.2
SnCl1 C3	83.1	10.1	1.21	15.4	35.6	12.9	21.0	7.0
Cl1T	264.1	32.1	0.69	28.0	28.1	32.2	30.3	32.3
RoT	423.1	51.5	0.65	42.2	25.0	46.1	33.3	56.9
CALC	821.7	100.0	0.79	100.0	28.0	100.0	30.1	100.0
ASSAY HEAD			0.73		27.2		29.6	

CUM PRODUCTS	CUM Wt	WT %	Sn %	CUM	Fe %	CUM	SiO ₂ %	CUM
T17 SnCl1 C1	22.0	2.7	2.15	7.3	40.0	3.8	17.4	1.5
SnCl1 C2	51.4	6.3	1.82	14.4	39.3	8.8	18.3	3.8
SnCl1 C3	134.5	16.4	1.44	29.8	37.0	21.7	20.0	10.8
Cl1T	398.6	48.5	0.94	57.8	31.1	53.9	26.8	43.1
FEED	821.7	100.0	0.79	100.0	28.0	100.0	30.1	100.0

As can be seen from both dextrin and SMBS additions neither suppressant can limit iron oxide flotation in the tin float significantly without a similar reduction in cassiterite recovery.

Removal of hematite from the tin flotation feed was considered to be able to improve the tin flotation stage. The table below shows a partial improvement in tin recovery with a standard SPA collector tin flotation test on a SLon non magnetic sample which had also been subjected to UF Falcon upgrading.

PRODUCTS	WT g	WT %	Sn %	DIST	Fe %	DIST	SiO ₂ %	DIST
T25 Sn RoCon1	15.5	6.0	3.66	11.0	45.2	6.3	9.31	3.1
Con2	36.6	14.3	2.71	19.1	47.3	15.5	9.26	7.2
Con3	50.3	19.6	2.18	21.2	48.3	21.8	9.81	10.5
RoTail	153.9	60.0	1.64	48.7	40.9	56.4	24.1	79.2
CALC	256.3	100.0	2.02	100.0	43.5	100.0	18.3	100.0
ASSAY HEAD			0.00		0.00		0.00	

CUM PRODUCTS	CUM Wt	WT %	Sn %	CUM	Fe %	CUM	SiO ₂ %	CUM
T25 Sn RoCon1	15.5	6.0	3.66	11.0	45.2	6.3	9.31	3.1
Con2	52.1	20.3	2.99	30.1	46.7	21.8	9.27	10.3
Con3	102.4	40.0	2.59	51.3	47.5	43.6	9.54	20.8
FEED	256.3	100.0	2.02	100.0	43.5	100.0	18.3	100.0

Summary of Tin Flotation Results

1. Iron oxide contamination is the most important factor in reducing both the grade and recovery of cassiterite in the tin flotation of Mt Garnet ores.
2. The high tin collector (SPA) consumption is probably linked also to the high proportion of iron oxides in the tin flotation feed. The flotation concentrate produced after acid leaching contained a high proportion of hematite, in excess of 70%, in the XRD examination of this product.
3. The tin flotation feed minerals must be ground to <10 microns in order to ensure that there is a maximum liberation of cassiterite within the mineral pulp.
4. The development of an alternative cassiterite collector is expected within the next 12 months. Previous results on small scale tests have indicated that it will be much more selective to cassiterite and less to iron oxides. The tests using styrene phosphonic acid (SPA) have always been influenced by the results showing high iron oxide contamination, particularly with hematite. It is hoped that as the project progresses that the new collector will become available and this should allow significant grade and recovery to be achieved in the tin flotation stage.
5. Tin flotation is a slow process and tests must aim to recover as much cassiterite throughout the roughing stage as possible. The cleaning stage can then be carried out in two or three stages to allow a maximum concentrate grade to be achieved. Each cleaning stage should return the tailing back to the previous cleaner or rougher stage to enable an adequate recovery to be maintained for the overall flotation process.

X. Reduction Roasting

The ferromagnetic product off the LIMS treatment of both Hole 7 and Hole 83/84 minerals, although in significantly differing proportions of 2% and up to 61% respectively, shows a very similar assay grade of 67.5-69%Fe, 1-2%SiO₂, <1%Al₂O₃, <0.1%P and 0.2-0.25%Sn content. The further treatment by conventional mineral dressing may not be economic but cannot provide a saleable iron product from the project.

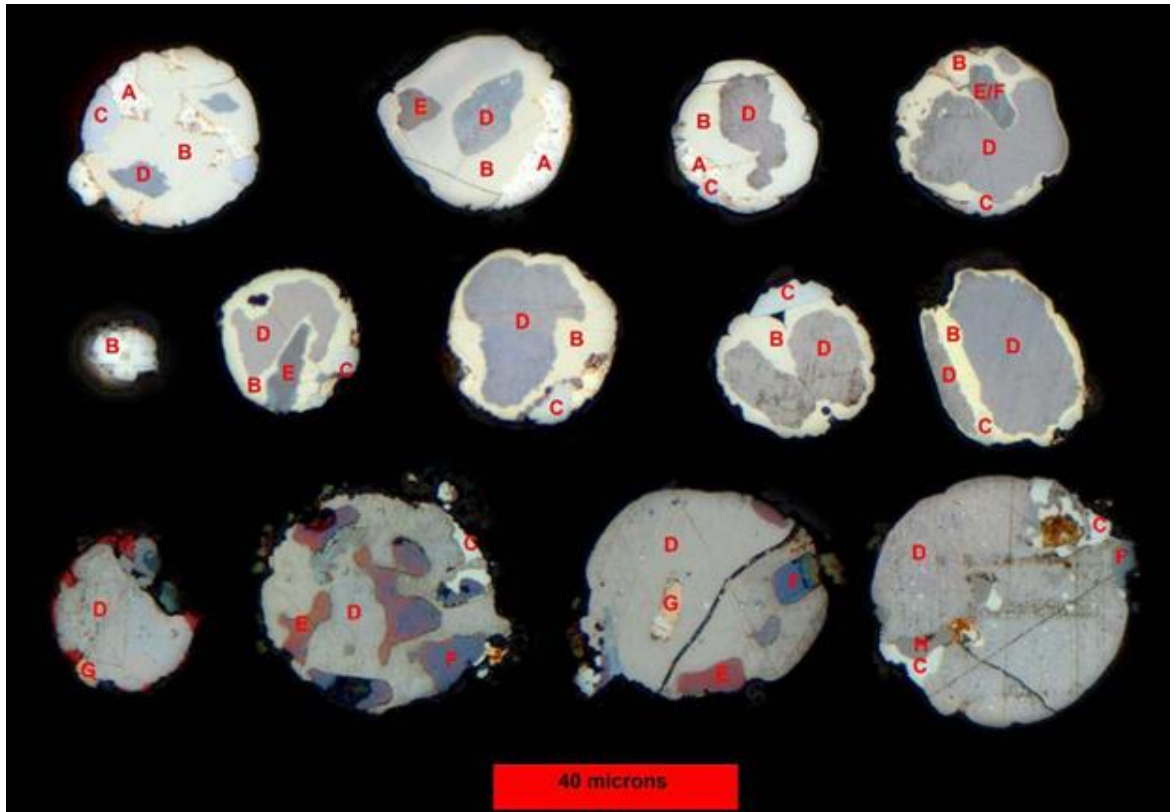
Further magnetic separation of the minerals after LIMS treatment, using high intensity magnetic separation with the SLon equipment has produced a magnetic product at 8000 gauss which contains 1.5%Sn . The tin in this mineral is known to be in the form of both locked cassiterite and soluble tin (Gillianite). The iron content of this magnetic product is shown to be 1.0%Sn and will therefore not allow efficient recovery by tin flotation of a clean cassiterite product.

A short series of sighter reduction roasts were run at the CSIRO in Perth on WHIMS magnetic products under the direction of METS consultants (Ref 11). The minerals were from Hole 7 ore sample and they had previously been subjected to LIMS separation prior to WHIMS treatment. The tests were run at a temperature of 800°C and the roast times were varied between 15

minutes and up to four hours. Each 10 gm sample of ore was mixed with 6gms of fine carbon. The ore sample was a mixture of WHIMS magnetic products with a p80 size of 53 microns. These tests were carried out in high alumina crucibles to prevent contamination, and at the end of each roast the samples were subjected to hand magnet separation to measure the degree of ferromagnetic transformation of the minerals. The longest roast of 4 hours had exhausted the carbon content of the mixture and the sample had reverted to mostly non-ferromagnetic hematite, whereas all of the other shorter tests had an increase in ferromagnetic portion from essentially none at the start of the test to exceed 76% on the 2 hour roast, all samples except the 4 hour roast had a ferromagnetic portion of >70%. The 4 hour roast, with total loss of carbon present in the mixture, caused the reversion of magnetite(Fe_3O_4) back to wustite (FeO) and then back to hematite (Fe_2O_3) as the reducing conditions maintained by CO from the carbon had been lost and oxidizing conditions returned. These shorter test results determined the conditions for a 500 gram sample to be subjected to a reduction roast and a 2 hour test at 800°C was run in order to generate a larger sample for more extensive testing.

The tests on the larger sample product were subjected to XRD examination, mineralogical examination, size distribution, and magnetic separation at 3 different size fractions. The conclusions reached on the changes to the product after roasting were:

1. There was an increase in magnetic response due to the conversion of hematite and goethite to magnetite or metallic iron.
2. The percentage of acid soluble tin increased from 6% before roasting to 52% after roasting, with the majority of the soluble tin in the LIMS 1000 gauss fraction indicating the formation of an iron-tin alloy.
3. Magnetic separation on the roast product allowed an upgrade of tin from an average grade of 1% to 2.2%Sn.
4. The reductive roast also produced metallic tin phases, predominantly with copper with compositions of 58-70%Cu and 38-39%Sn content. See photomicrograph below.



5.

Photomicrograph of the Spherical Particles in Reduction Roasted Product

- A. Shows presence of white soft tin metal.
 - B. Pale cream possibly pyrite FeS_2 .
 - C. Bluish white – unidentified.
 - D. Pale brown –possible cubanite $\text{Cu Fe}_2\text{S}_3$.
 - E. Orange brown-bornite Cu_5FeS_4 .
 - F. Blue- chalcocite Cu_2S .
 - G. Pale orange – bornite or mawsonite $\text{Cu}_6\text{SnFe}_2\text{S}_8$.
 - H. Mid brown- stannite $\text{Cu}_2\text{SnFeS}_4$.
6. Residual cassiterite was present in the roast product indicating that there was incomplete reduction in the roast.
 7. There were other copper rich alloy particles produced in the roast product which suggested there was reduction of stannite ($\text{Cu}_2\text{SnFeS}_4$) although none was detected in the pre-roast sample mineralogy report.

These conclusions show that there is evidence to indicate that there may be a possible process route for treatment of high intensity magnetic products. The SLon <10 micron magnetic product currently does not offer a conventional mineral separation route to recover tin as cassiterite concentrate. The reduction roast has shown that after reduction roasting the minerals become more magnetically susceptible and the tin in the roasted mineral shows significant evidence of improved acid solubility. There was also some evidence of increased particle size within the roasted sample, this may be due to particle sintering or growth due to the changes in individual particles. These latter changes may allow enhanced gravity treatment after roasting to be considered.

At this stage in the test program for Mt Garnet ores, the reduction roast will only be tested on the treatment of <10 micron SLon separated magnetic which currently only form 15% of the Hole 83/84 minerals. It is envisaged that tests can be carried out under fluidized bed roasting conditions at CSIRO test facilities at Clayton. These will enable a more concise testing regime to be developed at a variety of temperatures and reducing conditions, which should reduce all cassiterite or goethitic tin present to acid leachable tin metal, or gravity or magnetically separable tin iron alloy from this product. These tests will be carried out on the magnetic fraction from the SLon high intensity magnetic separation test on the <10 micron Hole 83/84 minerals.

Y. Treatment of LIMS Products and Tin Fuming Possibilities

The LIMS separation has indicated that a significant tonnage will be produced from the Mt Garnet ores with an estimate of 180000tonnes/annum forecast from an annual ore feed rate of 700000 tonnes. These are mainly clean magnetite/martite minerals and will contain in excess of 450 tonnes of tin at the 0.2-0.3%Sn assay produced in testwork on both Hole 7 and Hole 83/84 minerals. The other impurity content of these minerals, in particular the silica, alumina and phosphorus contents are acceptably low for iron producers. However the tin content will incur a penalty and limit sales. Consolidated Tin Mines need to investigate the removal and recovery of tin from this product.

A guide to possible treatment is demonstrated by the synthetic rutile process to separate the iron content from ilmenite minerals, and the reactions occurring in the tin fuming process. The ilmenite reduction process was developed during the 1970s and 1980s and involves the reduction roasting of ilmenite at 1250°C which converts the iron oxide in the ilmenite minerals to iron metal and the iron and titanium dioxide can then be separated by acid leaching. During the reduction roasting period elemental sulphur is added to the roast in order lock the manganese minerals present to manganese sulphide which prevents manganese from contaminating the acid leaching process that follows. The elevated temperatures to which the LIMS minerals can be subjected to are expected to allow reactions to occur in both the magnetite/martite and should allow diffusion of the sulphur and carbon monoxide gases through the magnetite particles. This should encourage the following reactions to occur through the particles and allow fuming of the stannous sulphide from the mineral particles.

The tin fuming reactions to enable cassiterite to be released from iron gangue minerals are as follows:

Carbon to carbon monoxide reaction is: $C + CO_2 \rightarrow 2CO$

Cassiterite reduction to stannous oxide is: $SnO_2 + CO \rightarrow SnO + CO_2$

Stannous oxide reaction with sulphide is: $SnO + FeS \rightarrow FeO + SnS$. Stannous sulphide is driven off as a vapour at 1250°C.

FeO is wustite and will be included in the magnetite/hematite particles which have released their cassiterite content at a reduction roasting temperature of 1250°C. The stannous sulphide is vaporised at this temperature and passes into the freeboard of the kiln. Extra air can be introduced into the freeboard of the kiln through air ducts similar to those used in a synthetic rutile kiln. The stannous sulphide (SnS) vapour is then oxidized back to stannic oxide (SnO₂) and this forms a fume which can ultimately be collected in a fabric filter Baghouse after gas cooling has been completed in the kiln flue. The sulphur dioxide has to be removed from the gas stream after the Baghouse using a lime scrubber. The reactions within the kiln gas stream are as follows:

Oxidation of stannous sulphide by excess air addition: $SnS + 2O_2 \rightarrow SnO_2 + SO_2$

Collection of SO₂ gas by lime scrubbing: $SO_2 + CaO \rightarrow CaSO_3$

Oxidation of calcium sulphite to insoluble calcium sulphate for stable disposal:
 $2CaSO_3 + O_2 \rightarrow 2CaSO_4$.

Tests are proposed to confirm roasting temperatures and reducing conditions required to maximize tin removal to fume and produce samples of fume and magnetite for probable quality of these products.

This method may also apply to treatment of the high intensity magnetic product from the SLon separation on the <10 micron minerals.

The reduction of Goethite to Wustite occurs at 360°C by the following reaction:
 $Fe(OH)_2 \rightarrow FeO + H_2O$

Then Wustite combines with Hematite to produce magnetite thus: $FeO + Fe_2O_3 \rightarrow Fe_3O_4$

The addition of iron pyrites and coal to these minerals to create reducing conditions enables the tin fuming reactions to attack cassiterite inclusions in both the goethite and hematite composite particles. This in turn allows a tin oxide fume to be collected and an iron oxide product to be produced from these minerals. This method will probably provide a more straightforward recovery of tin to an oxide fume. It is expected that the SLon separation process will generate 140000 tonnes of high hematite content minerals with an assay of approximately 0.55%Sn content. Therefore in this case roasting conditions may be significantly

different from those required for the LIMS treatment and would require a second kiln to be utilised but the tin fume product would be similar in grade and a common Baghouse can be used to recover the fume.

Currently tests are planned on both LIMS and SLon products from the bulk ore testing. A suitable laboratory for the testing is being sourced.

Z. Executive Summary and Flowsheet Design for Mt Garnet Ores – June 2010

The proposed flowsheet for the treatment of Mt Garnet ores is shown below. The process stages for 700000 tonnes/annum feed at 0.6%Sn ore are as follows:

1. **Ore Crushing-** The run of mine (ROM) ore will be transported to the Concentrator area to be tipped onto designated fingers for the variety of ores to be blended to feed the single toggle jaw crusher. A mobile rock breaker may be required to reduce large ore pieces to the maximum 400mm size to feed the jaw crusher. There is a choice of equipment for the crushing stage down to 10mm. The single toggle jaw crusher is the initial stage but this can be followed by the alternative of secondary and tertiary cone crushers, or a high pressure grinding roll (HPGR) unit taking the ore down to 10mm with the use of the three stage screen. The HPGR unit may reduce the power requirements of the following Ball Mill stage as it imparts micro cracks to the ore particles. The cost of a suitable capacity HPGR unit is significantly higher than two cone crushers, however there are second hand HPGR units available within Australia of similar cost. HPGR testing will be carried out when crushing parameter tests are undertaken on diamond core samples later this year.
2. **Trommel and Screening-** The comminution tests on the bulk ore sample, mixed from RC chip cores, have indicated that during the laboratory crushing process a significant amount of fines are produced prior to subjecting the sample to grinding. It is therefore anticipated that this will occur during operational processing in the crushing circuit. The production of a significant quantity of <75 micron minerals in the crushed product will enable this to be separated prior to milling and in turn enable the load on the ball milling circuit to be reduced. The method for fines separation is to pass the crushed ore through a 3mm trommel screen unit and then the trommel screen undersize would again be screened to <75 micron through a Stacksizer screen. The mixture of trommel oversize and Stacksizer oversize is ground through the ball mill, and the stacksizer undersize can be sent directly to the LIMS treatment.
3. **Ball Milling –** 10mm screened crushed ore is conveyed from the crushing plant to an emergency stockpile or crushed ore storage bin, which will allow extended downtime for the crushing plant for maintenance and improve the ore blending which would be initially achieved through the ROM pad organization of the feed to the crusher. The Ball Mill uses steel media and is rubber lined, and this runs in closed circuit with a 75mm Stacksizer screening system. The Stacksizer will reduce the recycling of <75 micron minerals back to the Ball Mill thereby reducing the power requirements for the Ball Mill and overgrinding of

the Ball Mill product. Currently the minimum screen sizing for a Derrick Stacksizer is 75 micron, and the screen undersize from this treatment is expected to achieve a p80 of around 50 microns. The alternative to the Stacksizer are hydrocyclones but their efficiency is much lower and a recirculating load of 300% is expected back to the Ball Mill.

4. LIMS Separation – There are several alternative suppliers of LIMS separation equipment and the main requirement will be to ensure a high field strength of 1300 gauss in order to maximize the removal of magnetite/martite from the <75 micron ground ore product. It is expected that the LIMS magnetic product may reach 180000 tonnes/annum and the tin grade will run at 0.25%Sn grade, which represents a 10% loss of tin from the feed of 0.6%Sn. The retreatment of this magnetic minerals by tin fuming, may only reduce this loss to 3%, but may enhance the sales of these minerals by reduction of the tin content to <0.1%Sn.
5. LIMS Non Magnetic Mineral Treatment – Cyclone separation of these minerals will allow >10 micron product to be subjected to enhanced gravity separation using two J1300 Kelsey jigs to produce a 1.8%Sn concentrate. The Kelsey Jig concentrate will then be upgraded to >40%Sn concentrate using 20 Holman fine roughing tables and 4 fine cleaning tables.
6. The tailings products from all of the gravity treatment are then returned to be mixed with the <10 micron cyclone product. The mixture is inert media ground to <10 micron sizing, using either an IsaMill or Deswik mill. A comparative test is required on both mill designs in order to assess the advantages of each unit. Suitable capacity units are commercially available for both mill designs, the choice should be evaluated on power input requirements to achieve the target sizing.
7. Inert Media ground LIMS Non Magnetics – These minerals are processed through either the Outotec SLon or Eriez GRZINM High Intensity Magnetic separators, which will produce magnetic product with high hematite content and non magnetic product. The HIMS circuit will require a roughing stage and cleaning stage for the magnetic stream, and a scavenging stage for the non magnetic stream. This configuration will ensure that both magnetic and non magnetic products are as clean as possible for further treatment. The magnetic product containing high levels of hematite, also contains recoverable levels of tin. The non magnetic product has a reduced level of hematite and high goethite content but contains free cassiterite which is recoverable by tin flotation directly or may require acid dissolution treatment or enhanced gravity treatment (UF Falcon) in order to complete the liberation of free cassiterite and removal of ultrafine slimes. The choice of equipment for this magnetic separation step will be made on laboratory testing of both products and the efficiency of separation of the iron oxides into the magnetic products. Essentially removal of hematite into the magnetic product will ensure efficient tin recovery in the following tin flotation treatment.
8. The SLon or GRZINM magnetic product will probably be treated by either reduction roasting at 800°C in order to produce a tin iron alloy within the mineral mixture to be recovered by

magnetic separation, or by tin fuming at 1250°C. The latter method is preferred and will probably give the highest recovery of tin to an oxide fume and a grade exceeding 70%Sn content. This roasting method would be adopted for both LIMS and SLon magnetic products.

9. The SLon non magnetic product will be treated by a choice of methods which will be evaluated to achieve maximum recovery of tin to a flotation product of 10-20%Sn grade. The use of tin flotation without pretreatment will be the preferable choice to give a direct method to produce a low grade tin concentrate, but currently this does not appear to provide a high grade and recovery. The efficiency of the tin flotation using styrene phosphonic acid (SPA) as the collector has shown limited selectivity with high recovery of hematite and some goethite to flotation concentrate. A more selective collector is being developed and should be available for testing later during the year. The strategy at this time is to improve tin flotation using SPA and this should allow higher recoveries and grades to be achieved when the alternative collector is available.
10. There is an acid soluble tin component of these minerals which can be present in the goethite. This tin is not recoverable by flotation and therefore in order to be recovered these minerals must be leached by a 30% sulphuric acid solution at 95°C. The leach residue can then be subjected to tin flotation to recover the liberated cassiterite. The filtrate will contain both iron and tin sulphate, this can either be neutralised with lime for disposal to the tailings dam or further treatment may be considered similar to the Albion process to produce a goethite precipitate. The evaporation of the filtrate containing tin may allow tin metal to be recovered by electrolysis. This method will be when the extent of soluble tin and resultant losses to flotation tails has been quantified.
11. The preferred alternative prior to tin flotation will be to treat all of the SLon non magnetic product through UF Falcon Concentrators to remove ultrafine (<3micron) slimes prior to processing the UF Falcon concentrates to tin flotation. This method would reduce the operating cost for the tin flotation and reduce the volume of minerals to be treated by flotation, and enable a higher grade of flotation concentrate to be produced. UF Falcon treatment to upgrade the tin flotation product is also required to maximize the low grade tin product to be sold for tin fuming metal extraction.
12. Concentrator Tailings grade must target a value less than 0.2%Sn in order to achieve a recovery value of tin of 75% from the ores. Currently a recovery value of 68% of tin metal in products is expected in three main saleable products with a tailings assay at 0.25%Sn.
13. All tailings will need to be neutralized with milk of lime to prevent acidic tailings being discharged to the tailings dam. The pH control of the thickener tailings should hold the discharge to a pH of >7.0 in order to prevent drainage from the tailings dam becoming acidic. Recycling of process water from the tailings thickener is important to maintain the water balance in the process and a source of fresh water is required for reducing contamination into the tin flotation circuit.

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