

TNT MINES LIMITED

ABN 67 107 244 039

RL10/1988

MOINA

ANNUAL REPORT TO 21 OCTOBER 2013

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ABSTRACT

A maiden JORC Inferred Resource of 24.6 million tonnes at 16% fluorspar (CaF_2), 0.1% tin (Sn), 0.1% tungstate (WO_3) and 17.2% iron (Fe) was estimated.

A final report on the metallurgical test work conducted at ALS AMMTEC in Burnie and Perth was received from Mintrex. The work demonstrated that a saleable tungsten concentrate could be produced, an acid grade quality fluorspar concentrate could be achieved but would require agglomeration to be saleable and a marginal magnetite concentrate was separated. No recoverable tin was separated.

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1.0 INTRODUCTION

1.1 Location and tenure

The Moina tenement is located approximately 40km south-west of Devonport, in north-west Tasmania (Figure 1). The 2 km² tenement is centred approximately 2 km south-west of the small town of Moina. The tenement area can be found on the Forth (1:100,000) LTIS map sheets.

Topographically the area is of variable relief with patches of rainforest, plantation and farmland. Vehicular access is good with Moina Road running through the tenement and numerous rough tracks giving 4WD access to most of the tenement. The land tenure is a mixture of State Forest and private freehold.

The owner of the tenement is Geotech International Pty Ltd (“Geotech”). That company has entered into an option agreement with TNT Mines (Moina) Pty Ltd, a wholly owned subsidiary of TNT Mines Limited (formerly part of the Minemakers Australia group).



Figure 1: Tenement location plan

1.2 Geology

Tenement geology is shown below in Figure 2 and is taken from Map 9 (1:25,000) Geology of the Winterbrook – Moina Area, of the Geological Survey of Tasmania’s Mt Read Volcanics Project 1989.

RL10/1988 is underlain by a thin sequence of Ordovician sediments. The Ordovician sedimentary package is a graded sequence of shallow water marine sediments with Roland Conglomerate at the base, overlain by medium to coarse grained Moina Sandstone, which in turn is overlain by Gordon Limestone. These three formations are conformable, gradational, and relatively thin, typically being in the range 50m to 150m thick. The sedimentary package dips gently north and has been lightly folded with fold axes trending NW sub parallel to the Bismuth Creek Fault. The sediments have been disrupted by a number of NW trending normal faults, principal of which is the Bismuth Creek Fault.

The Ordovician sediments are underlain in part by Cambrian volcanics and were intruded in Upper Devonian times by the Dolcoath Granite. A 2km wide stock of this leucogranite outcrops 3km to the east of Moina with an average composition of 40% orthoclase, 35% quartz, 20% plagioclase and 5% biotite. Gravity data indicates a west trending spine of this granite underlies RL10/1988 at depths of less than 1km. Drilling has revealed that beneath Moina the granite has been metasomatically altered to greisen. A Tertiary erosion surface, characterised by cemented gravels (graybilly) is patchily developed on the Ordovician sediments. Tertiary basalts, which are variably magnetic, cover substantial sections of the tenement area.

A large zone of hydrothermal alteration was associated with this granite spine. It caused dominantly iron and fluorine metasomatism of the Gordon Limestone and of calcareous beds in the Moina Sandstone and resulted in the formation of the Moina Skarn. These fluids were accompanied by variable amounts of tin, tungsten, bismuth, and molybdenum, which were fractionated from the granite; and by some precious metals and base metals either from the granite or leached from the Cambrian volcanics that lie between the sediments and the granite. This metasomatism resulted in a pocket of higher grade metamorphism turning the limestone to marble, the sandstone to quartzite, and indurating the conglomerate.

The Moina Skarn, with its associated tin-tungsten-fluorine veins and greisen, has been deposited in the roof above the Dolcoath Granite where it replaced Ordovician sediments. The skarn occurs as a thick horizontal plate roughly 1km in its longest dimension and up to 100m thick. It is separated from the granite's upper near horizontal contact by about 200m of the Moina Sandstone and replaces parts of the Gordon Limestone. The plumbing system for the mineralizing fluids was probably a series of east-west trending tension fractures, now tin-tungsten-quartz veins, associated with the major NW trending Bismuth Creek Fault and named the Shepherd and Murphy Vein Swarm. Emplacement of the granite was at shallow depths, probably less than 3km.

The main body of skarn is zoned and consists of:

- A top zone of a granular garnet-pyroxene-vesuvianite-fluorite skarn overlying the other units. This unit is relatively enriched in boron;
- The main skarn ("wrigglite") of fluorite-magnetite-vesuvianite (cassiterite-scheelite- adularia) and having a characteristic, fine grained (less than 0.2mm), rhythmic, finely layered, contorted structure;
- Within and near the base of the main skarn a granular, pale green pyroxene skarn occurs as thin units (less than 5cm) consisting of diopside-hedenbergite with very minor amounts of fluorite and garnet;
- A wollastonite-rich skarn may be present in places and can be a useful marker. It is probably derived from a silty/sandy facies of the limestone and consists of over 80% by volume of wollastonite with small amounts of garnet, pyroxene, vesuvianite and fluorite;
- A basal zone of granular garnet-pyroxene-vesuvianite-fluorite skarn;
-

However, the skarn is essentially variable depending on local factors that controlled the metasomatism. A number of distinctly different skarn types are found in limited quantities in other areas where metasomatic conditions varied. The two most notable are the pyrrhotite skarn and the sphalerite skarn. The former consists of medium to fine grained pyrrhotite, magnetite, fine grained

actinolite/chlorite, and minor fluorite; the latter of granular to massive andradite garnet with minor diopside containing conspicuous bands of closely spaced lenses of sphalerite with quartz.

The various skarn units can carry up to 25% (by weight) fluorite; 0.6% tin, 0.5% tungsten, 0.2% beryllium, 27.5% zinc, and 4.5 g/t gold. Tin, beryllium, and iron values increase toward the upper part of the skarn sequence but zinc, copper, and molybdenum values are erratic. Secondary zinc-copper-indium-cadmium-gold-sulphide-amphibole alteration of the primary fluorine-tin-beryllium oxide skarn is related to the Bismuth Creek Fault. When the primary wrigglite skarn is altered, tin is largely lost from that part of the skarn.

The hydrothermal fluids that extensively skarned the Gordon Limestone resulted in the formation of a number of known significant mineral deposits, including:

- The Shepherd & Murphy vein swarm, consisting of a set of east-west near vertical veins containing tin-tungsten-bismuth-molybdenum mineralisation.
- The fluorite-magnetite “wriggite deposit” in the basal section of the Gordon Limestone west of the Bismuth Creek Fault.
- The zinc-bismuth-gold mineralisation in the Hugo Skarn east of the Bismuth Creek Fault where the Hugo Thrust, which strikes E-W and dips north at 30°, has removed the top of the skarn and thrust older sediments over the top of the skarn.
- The auriferous pyrrhotite skarn west of the Shepherd & Murphy Mine.

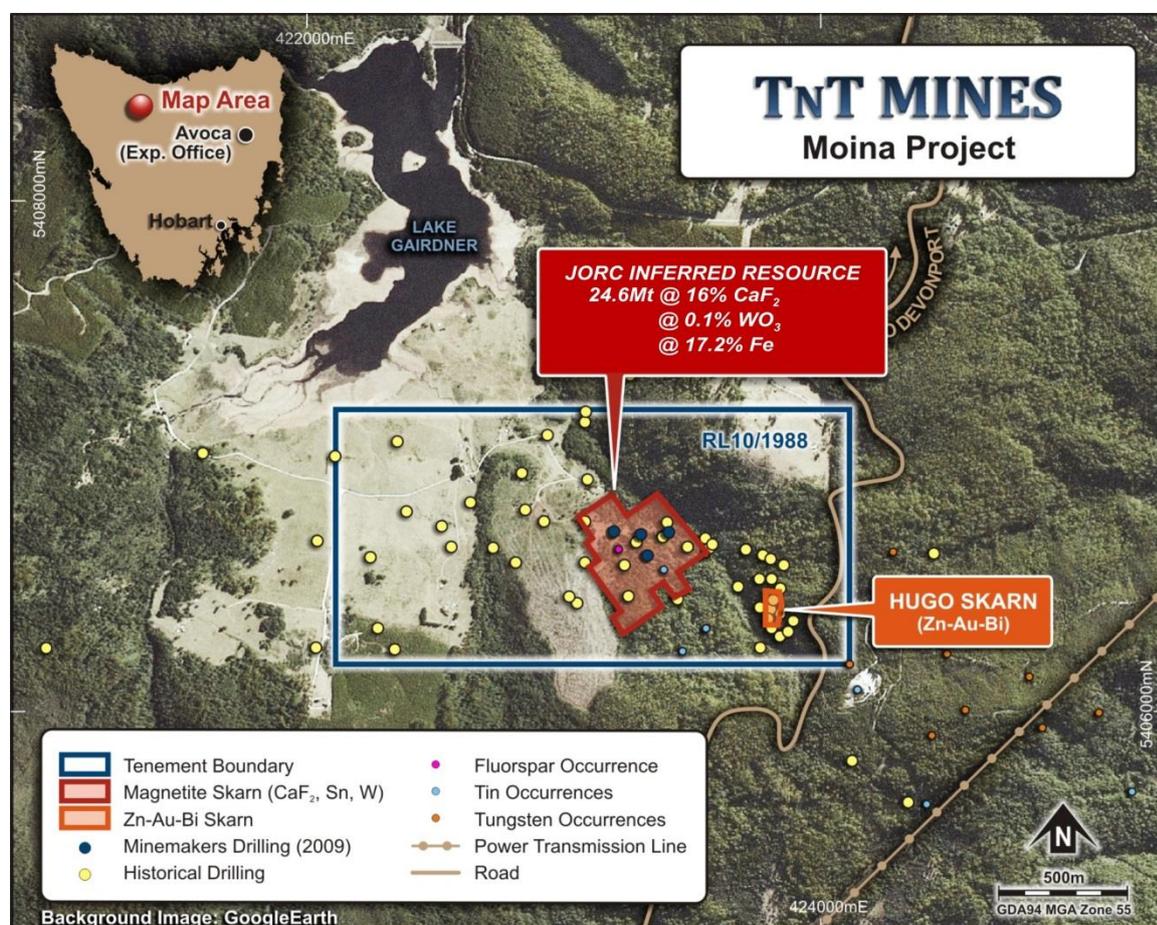


Figure 2: Tenement geology

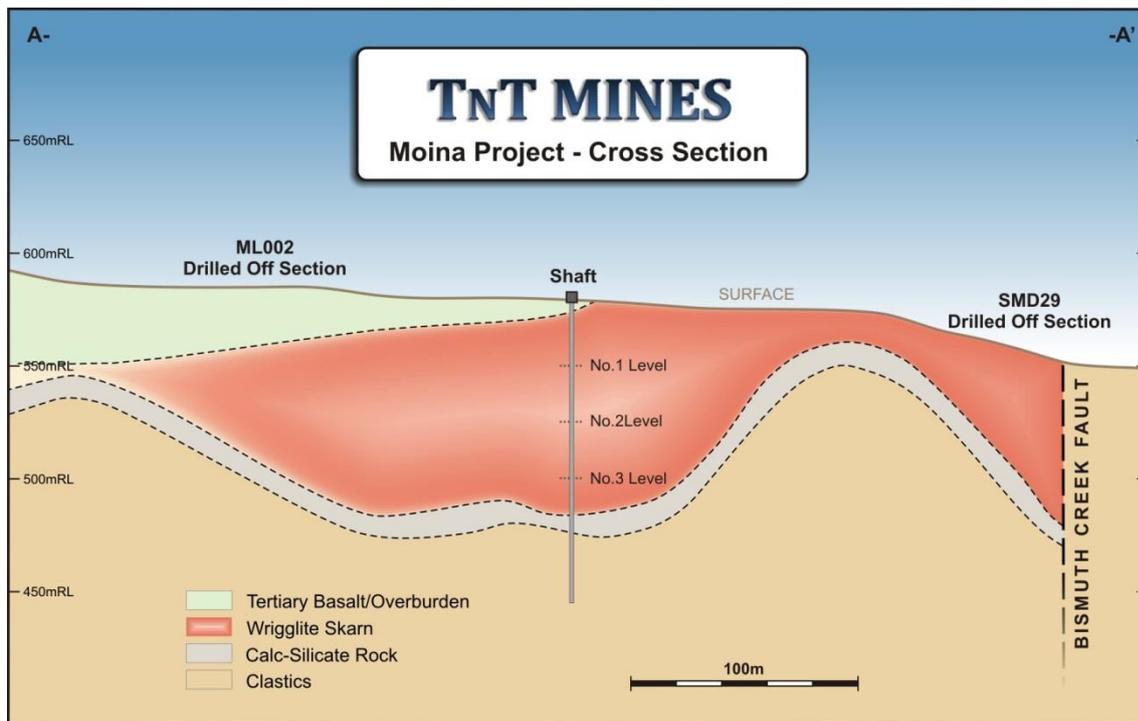


Figure 3: Section through Moina deposit

1.3 Exploration Rationale

The Moina fluor spar deposit has been known about for a long time but has remained undeveloped due to the ready availability of cheap, high quality fluor spar. This situation has changed over the past five years or so and the supply of high quality fluor spar has decreased and the price risen significantly.

The Moina fluor spar deposit has a pre-JORC resource estimate of 26.5Mt @ 18% fluor spar, 0.1% tungsten and 0.1% tin. The deposit also contains significant magnetite. Metallurgical test work carried out in the 1970s and 1980s was unable to define a clear pathway to generate a saleable product. TNT Mines believes that advances in processing technologies since then combined with dwindling fluor spar supply and consequent higher prices mean that the potential to develop an economically viable operation at the Moina deposit is now as high as it has ever been.

TNT Mines aims to:

- Undertake the necessary metallurgical test work for optimal circuit design.
- Drill out sufficient of the main deposit to at least JORC-compliant Indicated Resource status and to allow open-pit design optimization for, say, an initial 10 year operation.
- Assess the potential to market the bulk commodities fluor spar, magnetite, tungsten, tin and sulphides.
- Complete bankable feasibility study and, if economic, commission an open-cut mining and processing operation.

2.0 REVIEW OF PREVIOUS WORK

2.1 Previous exploration prior to TNT Mines Limited (formerly Minemakers TTT Pty Ltd)

Historical exploration before TNT Mines is well summarized in the 2011 annual report.

2.2 Exploration by TNT Mines Limited

TNT Mines has completed the following work:

2006-2011

- Review of literature
- Fatal flaw review to determine potential project viability
- Metallurgical test work carried out in Austria by tungsten producer Wolfram Bergbau Material was collected on-site (tailings dump) or from diamond drill core held at the MRT core store in Mornington
- Infill assaying using diamond drill core obtained from the MRT core store
- Davis Tube Recovery work on selected intervals of wrigglyite from diamond drill core held at the MRT core store in Mornington
- Drilling of four PQ/HQ-sized cored holes in 2009 to recover mineralisation for further metallurgy
- Analysis of 274 half PQ-sized core samples from 2009 drilling for F, Al, Bi, Ca, Cd, Cu, Fe, Mg, Mo, Pb, S, Sb, Si, Sn, W and Zn.
- QEMSCAN work. A compositing of selected core samples from the 2009 drilling was sent to SGS Lakefield in Ontario, Canada for QEMSCAN analysis. The purpose of this work was to investigate the mineral distributions, deportment of F and Sn, locking/association and grain size characteristics of the fluor spar, cassiterite, Bi-minerals, scheelite, sulphides and Fe-O oxides, and determine mineralogical parameters such as mineral release and grade recovery. The work has demonstrated that to obtain reasonable recoveries the ore would need to be ground to -20 μ . At this size, fluor spar would need to be agglomerated to produce a saleable product. The analytical work has also demonstrated that only 48% of the tin is present in cassiterite, the remainder contained in garnet or Sn-Fe oxides. Although the modelling suggested that recoveries, other than tin, would be relatively high at a fine grind it should be recognized that the QEMSCAN system is a 2-D modelling approach and with such fine-grained mineralisation the results should not be seen as conclusive. The modelled maximum recoveries achievable at a 20 μ regrind are F – 76.4%, WO₃ – 92%, Sn – 36% and Bi – 90%.
- A mining heritage survey was conducted over the area, at the request of MRT, by Gary Vines from Biosis Research. The conclusions drawn from the survey was that the site had significance at state level as an example of a distinctive and rare mining site reflecting a range of mineral extraction and processing technologies. In particular it represents an early example of tungsten production. Although the site is significant the report indicated that there was no impediment to further exploration drilling including track clearing and site preparation provided recommended precautions were observed.

2012

- 2008 DAVIS TUBE RECOVERY WORK REVIEW: A review of Davis Tube Recovery work undertaken in 2008 was carried out by Geos Minin. The review indicated that the DTR work had been carried out under sub-optimal test conditions and only a single stage grind had been used. There was no record of the parameters set for the DTR testing and no analysis of composite head grade. Recoveries of 25% magnetite with grades ranging from an

acceptable 62% in the minus 20 micron fraction to 50% in the 20-53 micron fraction were achieved. Geos Mining used the QEMSCAN modelling to show that, in theory, a much better recovery and grade would be achieved from fine grinding with recoveries of up to 55-65% at acceptable Fe grades. They recommended another round of DRT testing under stringently controlled test conditions using the 2009 core.

- **NEW METALLURGICAL TEST WORK:** A new metallurgical test work program was commissioned in May. This study was carried out at ALS-Ammtec Burnie, and ALS-Ammtec Perth under the management of John Glen. Metallurgical advice and interpretation came from Brian Povey of Mintrex and Ron Goodman. This is the first integrated study to look at producing magnetite, fluorspar, and scheelite concentrates.

The purpose of this first phase work is to determine the recoverability of fluorspar, magnetite and scheelite. Although QEMSCAN work indicated that very good recoveries could be achieved it is generally accepted that QEMSCAN modelling overestimates the likely actual recoveries of mineral constituents by up to 20%. Oxide flotation and gravity work was carried out at Burnie and detailed DTR work was carried out in Perth. Some magnetite separation work has also been carried out at Burnie.

A main composite was made up from core retained from the 2009 drill program and stored in Launceston. Table 1 shows the intervals selected for the composite. Analyses of the metre intervals from which the composites were selected are presented in Appendix 2. This composite was used for fluorspar, scheelite, and magnetite work at both Burnie and Perth laboratories. A second batch of six samples was sent to Ammtec in Perth for variability analysis but unfortunately the samples were composited and run as a single DTR test rather than as six individual tests.

Hole_ID	From	To	Hole_ID	From	To
MODD003	33.20	33.60	MODD001	66.50	66.75
MODD003	36.15	36.45	MODD001	65.85	66.15
MODD003	38.40	38.70	MODD001	62.85	63.10
MODD003	40.50	40.70	MODD001	48.05	48.25
MODD003	42.80	43.00	MODD001	46.20	46.50
MODD003	45.05	45.4	MODD001	41.95	42.25
MODD003	47.40	47.65	MODD001	40.50	40.80
MODD003	49.15	49.50	MODD001	37.15	37.45
MODD003	51.80	52.10	MODD001	34.40	34.75
MODD003	54.45	54.80	MODD001	31.25	31.50
MODD003	55.40	55.75	MODD001	29.70	29.95
MODD003	56.60	56.80	MODD001	27.55	27.80
MODD003	59.30	59.70	MODD001	25.80	26.20
MODD003	46.00	46.40	MODD004	71.60	71.90
MODD003	60.70	61.00	MODD004	69.30	69.55
MODD003	62.60	63.00	MODD004	67.55	67.75
MODD003	64.40	64.70	MODD004	65.60	65.90
MODD003	66.20	66.65	MODD004	64.10	64.40
MODD003	67.65	67.90	MODD004	62.40	62.70
MODD003	69.35	69.80	MODD004	60.50	60.70
MODD003	71.70	72.00	MODD004	59.35	59.65
MODD003	73.80	74.10	MODD004	56.15	56.40
MODD003	76.25	76.45	MODD004	54.60	54.90
MODD001	69.30	69.70	MODD004	53.20	53.40

Table 1: Main composite core intervals

Hole_ID	From	To
MODD001	30.50	30.75
MODD001	47.40	47.80
MODD003	32.60	32.80
MODD003	33.00	33.25
MODD003	49.60	49.85
MODD003	75.10	75.40
MODD004	64.70	65.00

Table 2: Second composite core intervals (DTR only)

A summary of the results obtained in the 2012 year are outline below:

Scheelite

A significant amount of scheelite is present in veins and gravity separation of the coarser vein scheelite (and vein fluorite) looks good. The finer groundmass scheelite is likely to end up in the fluorspar concentrate. It is likely that a saleable grade of 60-65% WO₃ suitable as APT feedstock will be achieved with an overall recovery of about 45%.

Magnetite

Basically, the main composite produced results that were not particularly encouraging. Six individual samples that were intended to be processed separately to get a feel for variability were composited, unfortunately, because of a lack of communication between Povey and the Ammtec Perth. This composite produced more encouraging results but probably because it has a higher grade to begin with. The main composite gave a 25% yield to a grade of 58% Fe and the smaller composite gave a 31% yield to a grade of 63% Fe.

John Glen believes that it may be possible to upgrade the magnetite concentrate by floating off some of the silicates that are reporting to the magnetite concentrate (because they contain fine inclusions of magnetite). This work would be carried out in the next phase of met work.

Fluorspar

Fluorspar flotation has been carried out on the non-magnetics stream and the Burnie lab has achieved a 95% CaF₂ concentrate with 80% recovery in a six cycle float. They are confident that this recovery will push up towards 90%. However, striking the balance between fluorine reporting to the mags and fluorine reporting to the non-mags is a critical factor in overall CaF₂ recovery and this will not be resolved with the current phase of test work. Floating silicates off form the mag concentrate, as discussed above, may significantly increase the overall CaF₂ recovery.

Cassiterite

It does not look like a saleable tin concentrate will be produced. There is no coarse vein tin and about half of the tin is locked up in garnet (and some in stokesite). Garnet contains up to 1% tin. Whether a garnet concentrate can be separated or not has not been addressed in the current work.

Bismuthinite

No work has been done on the recovery of the small amount of sulphide present.

- **JACOBS SCOPING STUDY:** Jacobs, a large international engineering company, were engaged to carry out a desktop scoping study on a mining operation at Moina. The study generate CAPEX and OPEX on an 800,000 tonnes per annum open pit mining operation producing magnetite, fluorspar and scheelite concentrates. The Jacobs study assumed a mining rate of 800,000 tonnes per annum with ore production based on the parameters outlined in Table 3.

Mineral	% in feed	Concentrate Grade	Recovery (%)	Dry tonnes per annum
CaF ₂	18.2	94% CaF ₂	68	86,369
Fe ₃ O ₄	21.9	67% Fe	70	150,100
WO ₃	0.12	65% WO ₃	58	703

Table 3: Ore Production Targets

Jacobs took input from Shaw Contracting, Mining One and Mancala Pty Ltd. for mining studies and used their own expertise for process plant estimation.

They considered three scenarios: Owner Mined and Concentrated, Contract Mining, and Dry Lease of Mining Equipment. The first scenario was the most expensive and the latter two were similar.

For contract mining the estimated cost were:

CAPEX	\$96.7M
Mining OPEX	\$12.34/t
Process plant OPEX	\$16.56/t
General and Administration	\$6.00/t

The Jacobs study was the first step in the proposed development of a mine at Moina. The assumptions made, particularly in relation to feed grade and recovery will be modified by the metallurgical test work being carried out at present and this will likely have some effect on the capital cost of plant construction. The mining costs will not be significantly affected by changes in these assumptions as long as the mining rate is unchanged.

3.0 WORK COMPLETED DURING THE REPORTING PERIOD

3.1 Mineral Resource Estimate

A maiden JORC Inferred Resource estimate was made by Mick McKeown of Mining One. Historical drill data, including Minemakers 2009 diamond drilling, was used for the estimate. The resource is 24.6Mt at 16%CaF₂, 0.1%WO₃, 0.1%Sn and 17.2% Fe. The full report is attached as Appendix 1.

3.2 Metallurgical test work

No metallurgical test work was carried out during the reporting period other than the production of reports by ALS Ammtec summarising the magnetic separation, gravity and flotation work carried out at the Burnie laboratory, a report on the magnetite separation work carried out at ALS Ammtec Perth and a report summarising the combined test work. These reports are attached as Appendices 2-4.

4.0 DISCUSSION OF RESULTS

4.1 Mineral Resource Estimate

The mineral resource estimate was the first JORC-compliant estimate of mineralisation conducted for the Moina skarn and the estimate compared favourably with the historic estimate, although a drop in grade from 18% to 16% was noted for CaF₂. There is no QAQC on the original CaF₂ analyses from drilling in the 1970s and 1980s and there remain some uncertainties in regard to the true grade. Fluorine is notoriously difficult to analyse. It is also apparent from QEMSCAN work that approximately 8% of the fluorine in the Moina system reports to minerals other than fluorite and would not be recoverable. The original resource quoted tungsten metal, W, whereas the new estimate quotes the oxide tungstate, WO₃.

Substantial further drilling with rigorous QAQC for fluorine would be required to lift the JORC category to Indicated or Measured.

The previous historic estimate for Moina was: 26Mt @ 18% CaF₂, 0.1%Sn and 0.1%W
The new Inferred Resource for Moina is: 24.6Mt @ 16% CaF₂, 0.1%Sn, 0.1% WO₃ and 17.2% Fe.

4.2 Metallurgical test work

A final report on the metallurgical test work conducted at ALS AMMTEC in Burnie and Perth was received from Mintrex (Appendix 4). The work demonstrated that a saleable tungsten concentrate could be produced, an acid grade quality fluorspar concentrate could be achieved but would require agglomeration to be saleable and a marginal magnetite concentrate was separated. No recoverable tin was separated. Based on the test work, the following comments represent the current status of Moina beneficiation work.

- Fluorspar – 48% recovery @ 75µ grind (potential for 72% CaF₂ recovery at final grind size of 25µ)
- Scheelite – 25% W recovery to 65% WO₃ grade (potential for 42% W recovery; separate scheelite at 500µ)
- Magnetite - Magnetism in composite sample were 25% by weight; needs to be ground to 80% passing -20µ; grade will only just reach 58.5% Fe.
- Tin – not recoverable through conventional processes. Reports to the magnetite and fluorspar concentrates.
- Bismuth – insufficient work undertaken to comment.

5.0 CONCLUSIONS AND FUTURE WORK

The maiden JORC Inferred Resource for Moina has confirmed the large size of the deposit and the tenor of the mineralisation, albeit with a small decrease in the CaF₂ grade. Any resource category upgrade will require a significant amount of drilling and more rigorous QAQC in regard to analyses of fluorine.

The summary reports and review of the metallurgical test work received this reporting year have essentially confirmed the preliminary results reported last year. The work has defined the style of plant design that will recover the three potentially economic minerals, fluorspar, magnetite and scheelite, and the potential recovery factors. No further test work was carried out this year due to a lack of funds.

Proposed work for next year includes:

- Test work to upgrade the magnetite concentrate through silicate flotation/regrinding and consequent increase in overall fluorine recovery through delivery of more fluorspar to non-magnetic stream. Budget – \$30,000
- Investigation of potential to produce a heavy media product suitable for the coal industry. Budget – \$20,000

Looking to the future, the following work will need to be undertaken:

- Define the magnetite resource by drilling and testing using typical magnetite techniques – magnetic susceptibility and Davis tube testing.
- Improve the fluorite recovery by recovering fluorite from the scheelite tails and the fine grinding magnetite tails
- Improve scheelite recovery at a coarser grind size of 500micron and reduce the losses to slimes.
- Rework the plant capital and operating costs based on the metallurgical recoveries determined by the recent and any future test work.
- Re-model mining costs.
- Conduct marketing studies on the magnetite and the fluorite to ensure that both will be sellable – the magnetite in terms of grade and impurities – and the fluorite in terms of size.

6.0 ENVIRONMENT

No other ground-disturbing exploration work was carried out at Moina during the reporting period. No rehabilitation of previous disturbance relating to mining or mineral exploration was undertaken.

7.0 REFERENCES

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APPENDIX 1 – Moina skarn deposit mineral resource estimate for TNT Mines Limited by Mining One Pty Ltd.



MOINA SKARN DEPOSIT

MINERAL RESOURCE REPORT

For

TNT MINES LIMITED

Job No. 1807_G
Doc No. 3366v2
Date: October 2012
Prepared by: M V McKeown

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Quality
ISO 9001
SAI GLOBAL

FINAL REPORT

Grid Convention

For this resource estimate, all coordinates relate to MGA94, zone 55.

Abbreviations used in this report

Comalco	Commonwealth Aluminium Corporation Limited
DTM	Digital terrain model
ID2	Inverse distance squared method
JORC	Joint Ore Reserve Committee (of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geosciences and Minerals Council of Australia)
Minemakers	Minemakers Limited
MRT	Mineral Resources Tasmania
Mt Lyell	The Mt Lyell Mining and Railway Company Limited
OK	Ordinary kriging
RL	Retention Licence
SIE	Specific ion electrode
TNT	TNT Mines Limited
XRF	X-Ray Fluorescence

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1 INTRODUCTION

1.1 Background

TNT Mines Limited (“TNT”) holds an option to acquire an 80% interest in Retention Licence (“RL”) 10/1988 at Moina in Tasmania (see Figure 1). The owner of the RL is Geotech International Pty Ltd. TNT is exploring the licence for tin, tungsten, fluorine and iron.

TNT completed four diamond drill holes on the Licence in 2008 and 2009 primarily to obtain samples for metallurgical test work. In addition, the results of that drilling confirmed the general reliability of historic data and has allowed for the estimation of Mineral Resources for the deposit and reporting of the Mineral Resources in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves prepared by the Joint Ore Reserve Committee of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geosciences and Minerals Council of Australia (“JORC Code”).

This report describes how the resource estimate was made.

The information in this report which relates to Mineral Resources is based on information compiled by Michael V. McKeown who is a Fellow of the Australasian Institute of Mining and Metallurgy. Michael McKeown is employed by Mining One Pty Ltd and he has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity he is undertaking to qualify as a Competent Person as defined in the 2004 Edition of the “Australasian Code for Reporting Exploration Results, Mineral Resources and Ore Reserves” (JORC Code). Michael McKeown consents to the inclusion in this report of the matters based on his information in the form and context in which it appears.

1.2 Property Location and Access

RL 10/1988 is located about 2 kilometres south-west of the settlement of Moina in North-West Tasmania. Access to the licence is by way of the sealed road to Cradle Mountain.

The licence is at an altitude of about 600m above sea level. The area is hilly to steep and is dissected by creeks which flow to the north-east.

1.3 Property Tenure

Mining One has not reviewed the type or status of RL 10/1988.

1.4 Brief Summary of Mining and Exploration

RL 10/1988 covers an outcropping magnetite-fluorite skarn, known as the Moina skarn, which carries minor amounts of tin and tungsten.

In 1893, quartz veins were discovered at the Shepherd and Murphy deposit (see Figure 2). Tin, tungsten and bismuth were produced irregularly and discontinuously from the quartz veins until 1956, the main periods of production being from 1907 to 1918 and from 1953 to 1956. Access into the Shepherd and Murphy mine was initially by adits and later by a shaft.

Interest in the Moina skarn followed a brief period of exploration by the Mt Lyell Mining and Railway Company Limited (“Mt Lyell”) in the early 1970s. Mt Lyell completed three diamond drill holes (see Table 1).

In the mid-1970s, the Commonwealth Aluminium Corporation Limited (“Comalco”) explored the property, principally as a potential source of fluorite for use in their aluminium smelter at Bell Bay in Northern Tasmania. Comalco undertook significant exploration, completed 15 diamond drill holes into the main skarn area (see Table 1), and estimated the tonnage and grade of the Moina skarn.

From 1980 to 1987, Billiton Ltd (“Billiton”) in joint venture with Comalco, completed 7 holes in an around the main skarn area. Billiton also investigated the nearby retrograde Hugo Skarn which, although within the current RL area, is not the subject of this resource estimate.

In 2006, Minemakers acquired an option over the licence and completed 4 diamond drill holes in 2008 to 2009.

Table 1: Diamond drilling on the Moina Deposit.
(after Drummond, 2006)

Years	Hole Numbers	Number of Holes	Total Length metres
1970 - 71	Mt Lyell Mining and Railway Co Ltd	3	861.1
1972	Department of Mines, Tasmania	1	325.0
1976 - 79	Comalco	15	1,645.5
1980-1987	Billiton	7	1268.4
2008 - 09	Minemakers Limited	4	370.5
Total		30	4,470.5

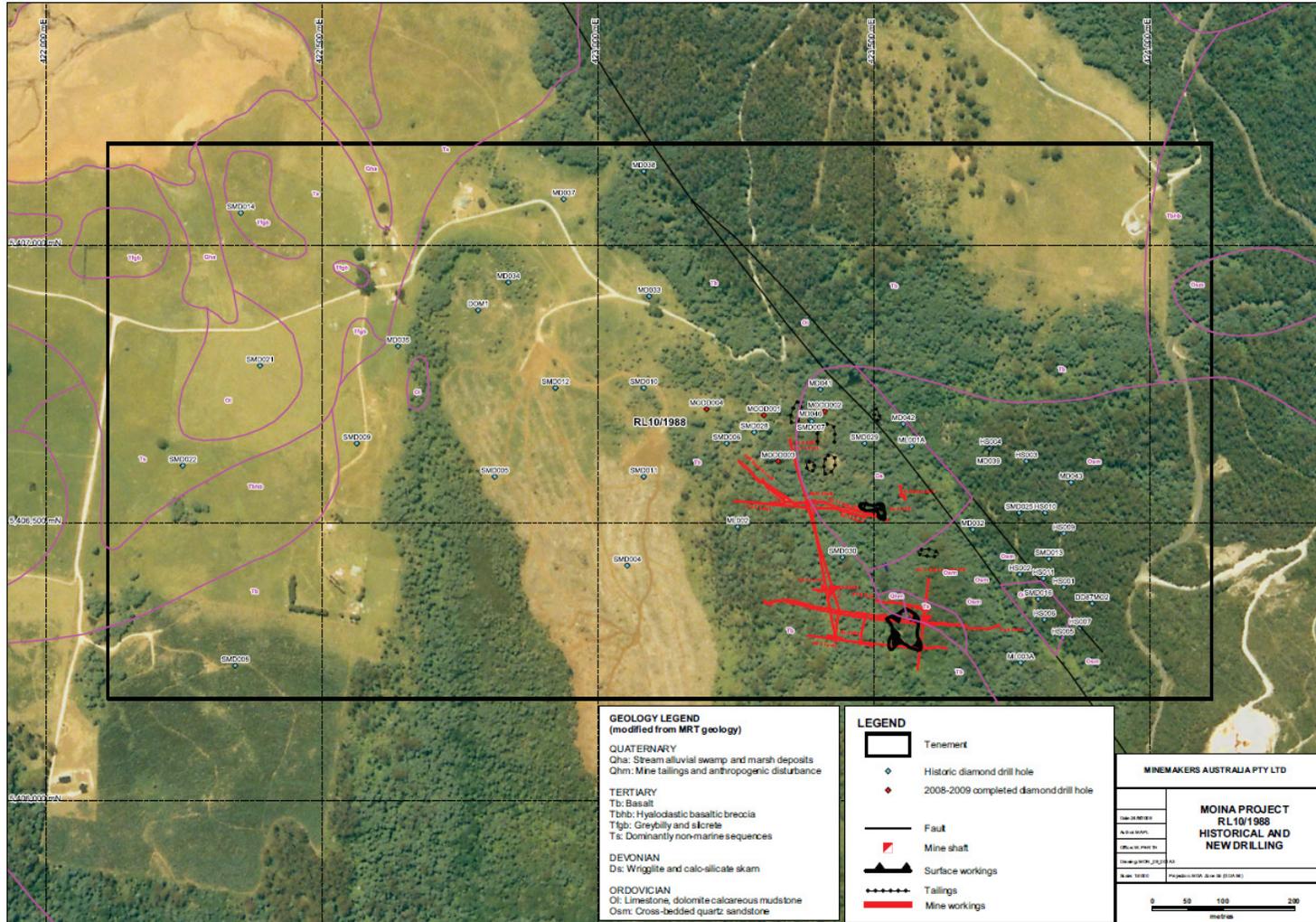


Figure 1: Retention Licence 10/1988.

2 GEOLOGY

2.1 Geology

The geology of the licence is shown in Figure 2 and was described by Drummond (2006):

“RL 8810 is underlain by a thin sequence of Ordovician sediments. The Ordovician sedimentary package is a graded sequence of shallow water marine sediments with Roland Conglomerate at the base, overlain by medium to coarse grained Moina Sandstone, which in turn is overlain by Gordon Limestone. These three formations are conformable, gradational, and relatively thin, typically being in the range 50m to 150m thick. The sedimentary package dips gently north and has been lightly folded with fold axes trending NW sub parallel to the Bismuth Creek Fault. The sediments have been disrupted by a number of NW trending normal faults, principal of which is that Bismuth Creek Fault.

The Ordovician sediments are underlain in part by Cambrian volcanics and were intruded in Upper Devonian times by the Dolcoath Granite. A 2km wide stock of this leucogranite outcrops 3km to the east of Moina with an average composition of 40% orthoclase, 35% quartz, 20% plagioclase, 5% biotite. Gravity data indicates a west trending spine of this granite underlies RL 8810 at depths of less than 1km. Drilling has revealed that beneath Moina the granite has been metasomatically altered to greisen. A Tertiary erosion surface, characterised by cemented gravels (graybilly) is patchily developed on the Ordovician sediments. Tertiary basalts, which are variably magnetic, cover substantial sections of the tenement area.

A large zone of hydrothermal alteration was associated with this granite spine. It caused dominantly iron and fluorine metasomatism of the Gordon Limestone and of calcareous beds in the Moina Sandstone and resulted in the formation of the Moina Skarn. These fluids were accompanied by variable amounts of tin, tungsten, bismuth, and molybdenum, which were fractionated from the granite; and by some precious metals and base metals either from the granite or leached from the Cambrian volcanics that lie between the sediments and the granite. This metasomatism resulted in a pocket of higher grade metamorphism turning the limestone to marble, the sandstone to quartzite, and indurating the conglomerate.”

2.2 Geological Interpretation

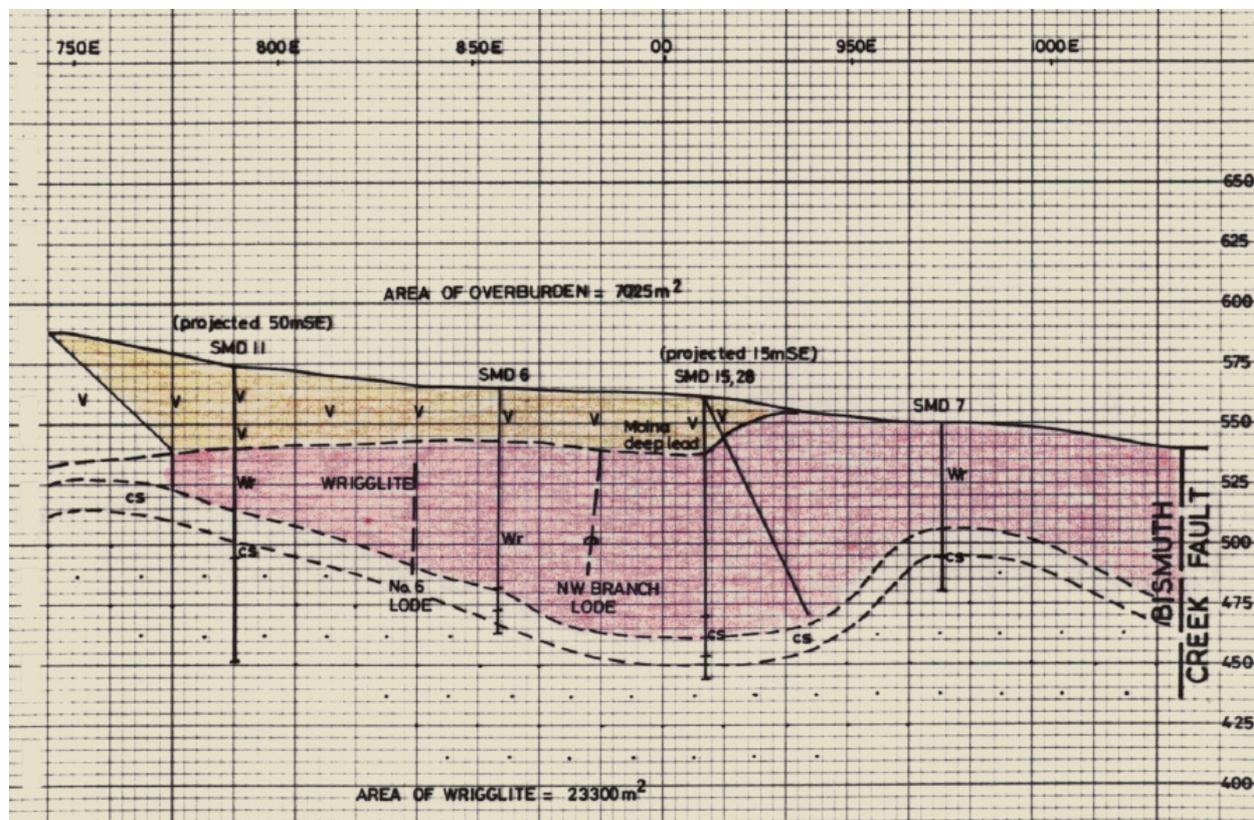
Askins (1979) made a comprehensive geological interpretation of the Moina skarn. The interpretation was based on five cross-sections oriented along about 055° and numbered B to F, for example, see Figure 3. The orientation of the sections was normal to the strike of the folding in the area.

The interpretation was made by hand and was based on outcrop mapping and structural contours of the bases of the principal lithological units. These cross-sections were digitised and formed the basis of the interpretation used for this report.

The digitised cross-sections were modified where necessary to match the lithologies in the current digital database that includes the results of the four holes drilled by Minemakers. Only small adjustments were necessary to achieve a good fit.

Askins reported that the extent of the skarn, which is essentially a magnetite skarn, was well delineated by the magnetic anomaly bounded by the 62500nT contour. The 62500nT contour on a recent plan of the magnetic field (WESTas_MAGA0.pdf) was digitised and used as a limit to the extent of the mineralisation for this report. This limit was beyond Askins' most north-westerly cross section, so, section G was created by duplicating section F just over 50m to the north-west (see Figure 4).

The skarn is known along strike for about 400m, across strike for about 500m and over a vertical range of about 100m.



**Figure 3: Cross-section through holes SMD11, SMD6, SMD19, SMD 20 and SM7
(Askins, 1979)**

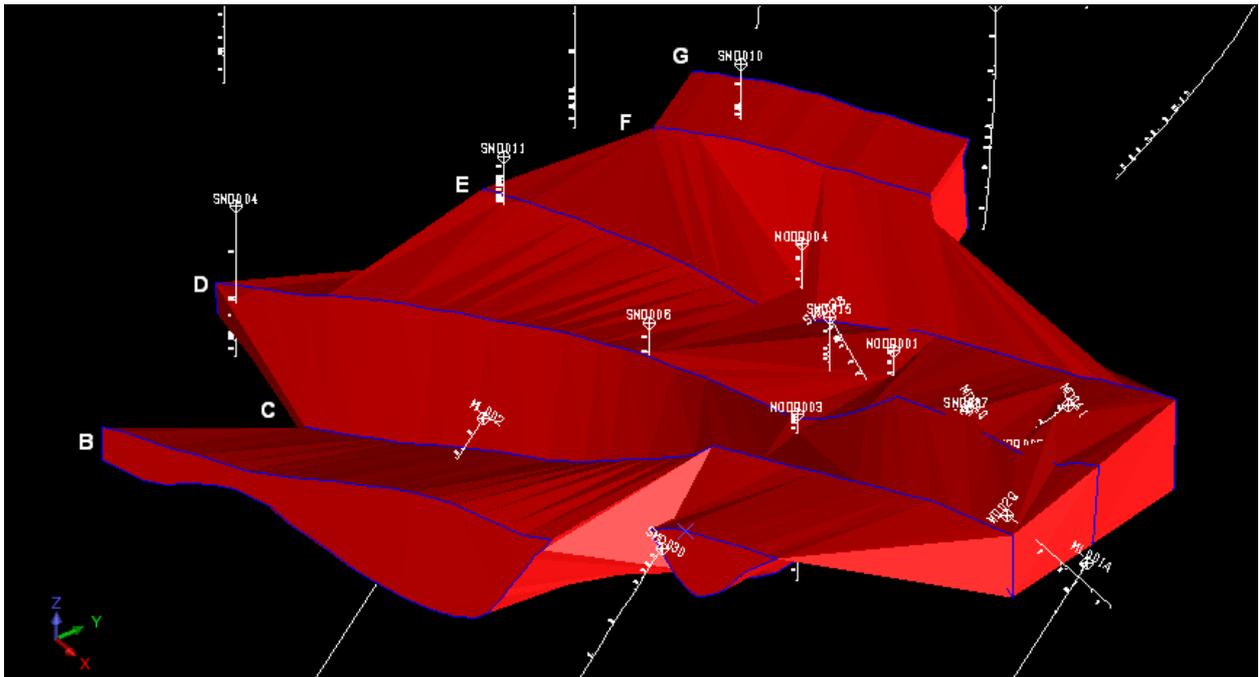


Figure 4: Oblique view of the skarn wireframe

The cross-section numbers are shown; the distance from section B to section G is just over 460m.
(*interp_skarn3.dtm*)

2.3 Mineralogy

The mineralogy of the Moina skarn was described by Drummond (2006):

“The main body of skarn is zoned and consists of :-

- A top zone of a granular garnet-pyroxene-vesuvianite-fluorite skarn overlying the other units. This unit is relatively enriched in boron;
- The main skarn ("wrigglite") of fluorite – magnetite - vesuvianite (cassiterite - scheelite - adularia) and having a characteristic, fine grained (less than 0.2mm), rhythmic, finely layered, contorted structure;
- Within and near the base of the main skarn a granular, pale green pyroxene skarn occurs as thin units (less than 5cm) consisting of diopside – hedenbergite with very minor amounts of fluorite and garnet;
- A wollastonite-rich skarn may be present in places and can be a useful marker. It is probably derived from a silty/sandy facies of the limestone and consists of over 80 % by volume of wollastonite with small amounts of garnet, pyroxene, vesuvianite, and fluorite;
- A basal zone of granular garnet-pyroxene-vesuvianite-fluorite skarn.”

3 DATABASE

3.1 Database Introduction

Both the historic and Minemakers' data are recorded in a very extensive library of reports held by Mineral Resources Tasmania ("MRT") both in hard copy and in the MRT Tiger Database.

In particular, TNT supplied a series of Excel spreadsheets containing the data relevant to this resource estimate:

- 1_Moina_Collars_Resource_2012.xls
- 2_Moina_Survey_Resource_2012.xls
- 3_Moina_Assays_Resource_2012.xls
- 4_Moina_Geology_Resource_2012.xls
- 5_Moina_Resource_Core_Recovery_2012.xls
- 8_Moina_Density_2009_Wrigglite.xls

The first four of these were used to create an Access database named Moina_database.mdb for use with Gemcom Surpac software.

It is commercially traditional to quote tungsten content as % WO₃ rather than % W and fluorine content as % CaF₂ rather than % F and this tradition has been maintained for this report.

3.2 Reliability of Minemakers Data

The JORC Code includes a list of Assessment Criteria for Sampling Techniques and Data. For the Minemakers' diamond drilling, the nature of defects, if any, in the Assessment Criteria and their likely impacts on the quality of this Mineral Resource report is listed in Table 2.

In Table 2, the Assessment Criteria for Sampling Techniques and Data have been categorised and described for the Minemakers diamond drilling using the terms from Table 7.

Table 2: Nature of Defects, Likely Adverse Impact on Resource Report Quality and Likely Adverse Impact on Subsequent Use of the Resource Report

Nature of Defects	Likely Adverse Impact on Resource Report Quality	Likely Adverse Impact on Subsequent Use of Resource Report
Acceptable: no defects	None	None
Defects tolerable	Low	No material impact
Defects not acceptable	Quality severely compromised	Severely compromised

Table 3: New data - Assessment Criteria for Sampling Techniques and Data

Sampling Techniques and Data	
Sample recovery.	<p>Acceptable.</p> <p>Core recoveries were recorded during core logging. No systematic core loss in mineralised zones was reported.</p>
Logging.	<p>Acceptable.</p> <p>Core logging was comprehensive and included detailed lithological and mineralogical logging.</p> <p>Core logging was appropriate to support the estimation of the resource, and mining and metallurgical studies.</p>
Sub-sampling techniques and sample preparation.	<p>Tolerable.</p> <p>Drill core was nominally PQ and HQ size.</p> <p>Drill core was sawn in half; the half core was crushed to -12mm before the sample was then split and pulverised. The effect on the reliability of Sn and W assays of splitting samples at this relatively coarse size should be investigated.</p>
Quality of assay data and laboratory tests.	<p>Acceptable.</p> <p>Samples were submitted to SGS Perth for analysis. Sn and W analysis methods were acceptable (see below). After a round of umpire F analyses, samples were submitted to AMDEL Adelaide for F analysis (see below).</p>
Verification of sampling and assaying.	<p>Tolerable (for reporting an Inferred Mineral Resource).</p> <p>Assays were undertaken by reputable laboratories: SGS, Perth and AMDEL, Adelaide. The results of the laboratory checks of duplicates and standards have been requested but are not currently available.</p>
Location of data points.	<p>Acceptable.</p> <p>Drill hole collars were picked up by GPS.</p> <p>Holes were vertical and were set up using a level. Given the large diameter of the holes (PQ and HQ) and the relatively short length, relatively low deviation would be anticipated over the lengths of the holes.</p>

Data spacing and Distribution.	Acceptable. Data spacing and distribution was adequate for the estimation and classification of Mineral Resources for this report (see 4.2 Geological Interpretation and 8.3 Mineral Resource Estimate).
Orientation of data in relation to geological structure.	Acceptable. The skarn is relatively massive and, although folded, has an overall flat dip and plunge. Generally, the holes were drilled at steep angles to the local dip and plunge of the skarn.
Audits or reviews.	The data was reviewed for internal inconsistencies (see 3.1 Database Introduction).

3.3 QA/QC for Analyses of Minemakers Samples

For Minemakers' samples, Sn and W assays were made by fused bead X-ray fluorescence ("XRF") which is the current industry standard for such assays.

Generally, accurate F assays have proved difficult for laboratories to achieve, especially for assays greater than about 5% F. Minemakers commissioned an inter-laboratory check by submitting F standards to three independent laboratories and the results are listed in Table 4 and graphed in Figures 3 to 5. Results from AMDEL, by the specific ion electrode ("SIE") method, were by far the most reliable and the F assays entered into the database were those made by AMDEL.

Table 4: Assays of F standards by AMDEL, AMMTEC and SGS
(P0268 Fluorine Standards Assays.xls)

Standard		AMDEL	Ammtec	SGS
ID	F %	F %	F %	F %
JKSP	17.3	16.2	2.51	5.29
HJ-5	2.34	2.20	2.20	2.37
HJ-20	9.51	9.50	9.33	7.44
HJ-10	5.23	14.10	10.20	8.72
JKD	47.24	44.00	17.20	8.21
JKC	37.43	36.00	23.10	18.40
2666-83	15.20	15.70	12.80	8.24
5133-89	1.98	1.90	1.91	2.04

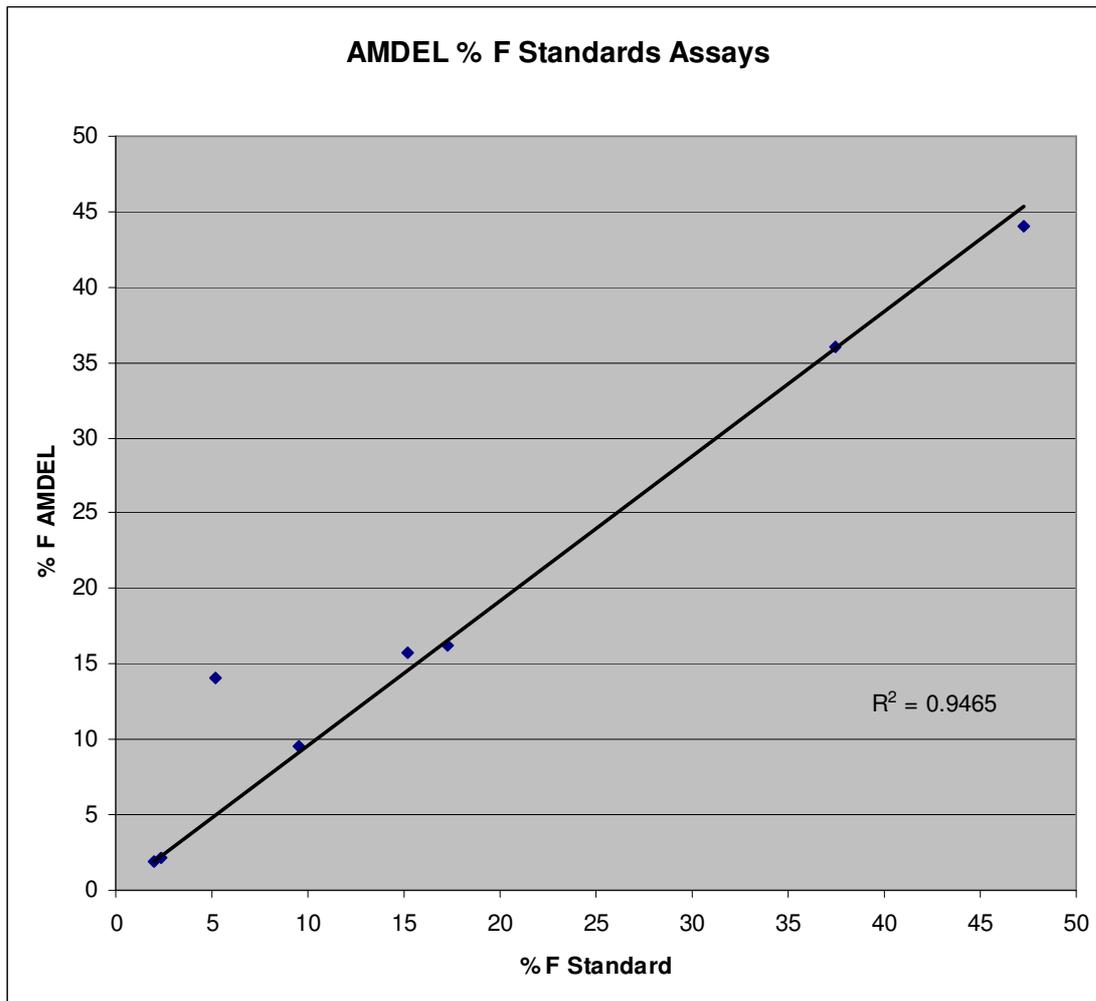


Figure 5: Assays of F standards by AMDEL
(P0268 Fluorine Standards Assays.xls)

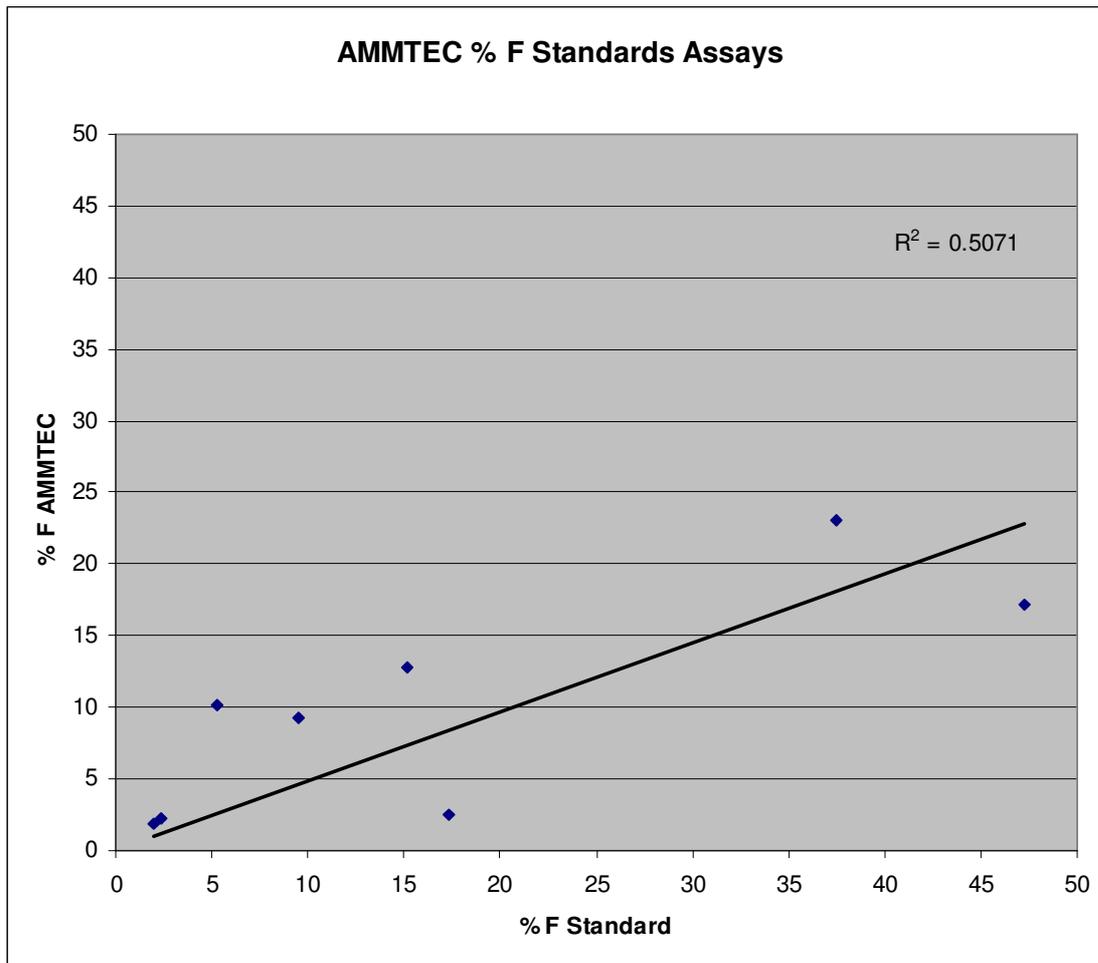


Figure 6: Assays of F standards by Ammtec
(P0268 Fluorine Standards Assays.xls)

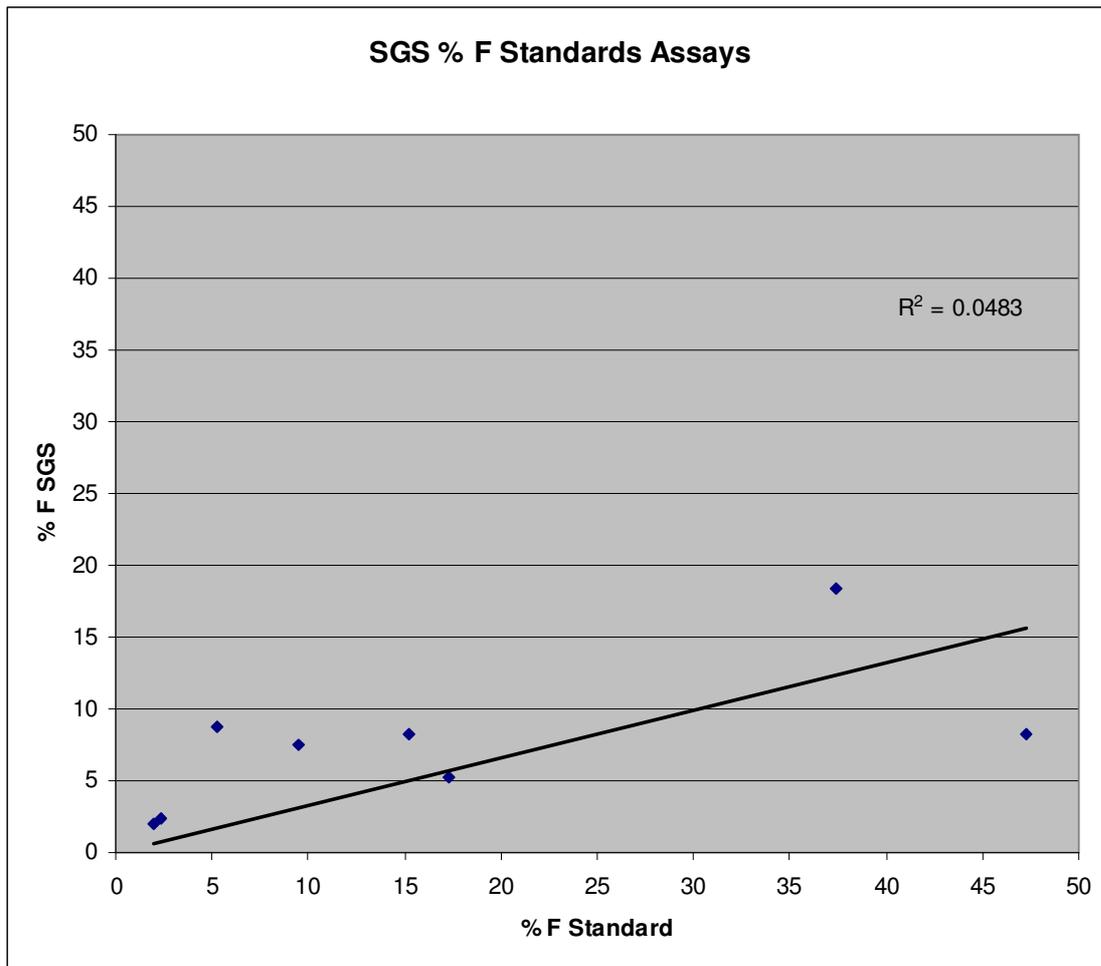


Figure 7: Assays of F standards by SGS
(P0268 Fluorine Standards Assays.xls)

3.4 Reliability of Historic Data

The JORC Code includes a list of Assessment Criteria for Sampling Techniques and Data. The nature of defects, if any, in the Assessment Criteria and their likely impacts on the quality of this Mineral Resource report are listed in Table 2.

In Table 5 the Assessment Criteria for Sampling Techniques and Data have been categorised and described for the WISMUT data using the terms from Table 2.

Table 5: Historic data – Assessment Criteria for Historic Sampling Techniques and Data

Historic Sampling Techniques and Data	
Sample recovery.	Acceptable. Core recoveries were recorded during core logging. Core losses of up to 30% occurred over short intervals but no systematic core loss in mineralised zones was reported.
Logging.	Acceptable. Core logging was comprehensive and included detailed lithological and mineralogical logging. Core logging was appropriate to support the estimation of the resource, and mining and metallurgical studies.
Sub-sampling techniques and sample preparation.	Acceptable. Details of the sampling techniques and sample preparation are not known. However, the results of the holes drilled by Minemakers confirmed the general tenor of the F, Sn, W and Fe grades in the historic holes indicating that the sampling techniques and sample preparation methods were appropriate.
Quality of assay data and laboratory tests.	Acceptable. Assays were made at reputable laboratories of Comalco and AMDEL. Comparison with the results of Minemakers' holes confirmed the general reliability of the results (see below).
Verification of sampling and assaying.	Acceptable. The results of the holes drilled by Minemakers confirmed the general reliability of the locations of the skarn in the historic holes and the general tenor of the F, Sn, W and Fe grades in the historic holes.

<p>Location of data points.</p>	<p>Acceptable.</p> <p>Drill holes were located on a cut grid and collar locations were originally recorded in local coordinates. The coordinates have since been transformed to MGA94, zone 55. Current plots indicate that the coordinates have been correctly transformed.</p>
<p>Data spacing and Distribution.</p>	<p>Acceptable.</p> <p>Data spacing and distribution was adequate for the estimation and classification of Mineral Resources for this report (see 2.2 Geological Interpretation and 4.5 Mineral Resource Estimate).</p>
<p>Orientation of data in relation to geological structure.</p>	<p>Acceptable.</p> <p>The skarn is relatively massive and, although folded, has an overall flat dip and plunge. Generally, the historic holes were drilled at steep angles to the local dip and plunge.</p>
<p>Audits or reviews.</p>	<p>Acceptable.</p> <p>The historic data was compiled by Minemakers independently of the companies which accumulated the data.</p>

Comparison of the mean Sn, W, F and Fe grades of samples from the Minemakers' holes and the historic holes is shown in Table 6. As mentioned above (see 2.2 Geological Interpretation), the locations of the skarn in the Minemakers' holes conformed well with the historic interpretation (Askins, 1979). The sampling and geological comparisons confirm the general reliability of the results of the historic drilling.

Further confirmation of the historic sampling results could be obtained by re-sampling historic core that has been preserved. Such samples should be taken over the same intervals as the original samples.

The issue of the general reliability of F assays has been discussed above (see QA/QC for Analyses of Minemakers' Samples). For the historic samples, not all F assays in the database have been attributed to a specific laboratory. Those that have, have been attributed to AMDEL or Comalco. The reliability of the Minemakers' F assays from AMDEL has been discussed. Comalco was a large industrial user of fluorite at Bell Bay and it would be reasonable to assume that F assays from the Comalco laboratory were reliable.

Table 6: Comparison of mean Sn, W, F and Fe grades of samples of skarn from historic and Minemakers' drill holes
(comps_skarn1_old_and_new.xls)

	Sn	W	F	Fe
old holes	1281	925	8.18	10.8
Minemakers holes	1240	718	6.91	15.7
	Sn	WO₃	CaF₂	Fe
old holes	1281	1166	16.79	10.8
Minemakers holes	1240	905	14.19	15.7

3.5 Database Audits and Reviews

For this report, the historic and the new data in the database were reviewed for:

- Holes which appeared to collar beyond the apparent limits of the drilling campaigns.
- Holes which appeared not to collar on the surface.
- Assays which exceeded those possible, that is, exceeded the metal content of pure minerals.
- Assays which exceeded those mentioned in existing reports.
- Overlapping sample depths.
- Holes which had records which exceeded the final depth recorded in the Collar table.

No material errors were found in the database. Minor errors were fixed.

4 RESOURCE ESTIMATE

4.1 Compositing Length

All samples for this estimate of Mineral Resources came from sampling of diamond drill core. A histogram of the lengths of samples taken in the skarn showed that most samples were taken over lengths of 1.0, so, samples were composited to a length of 1.0m.

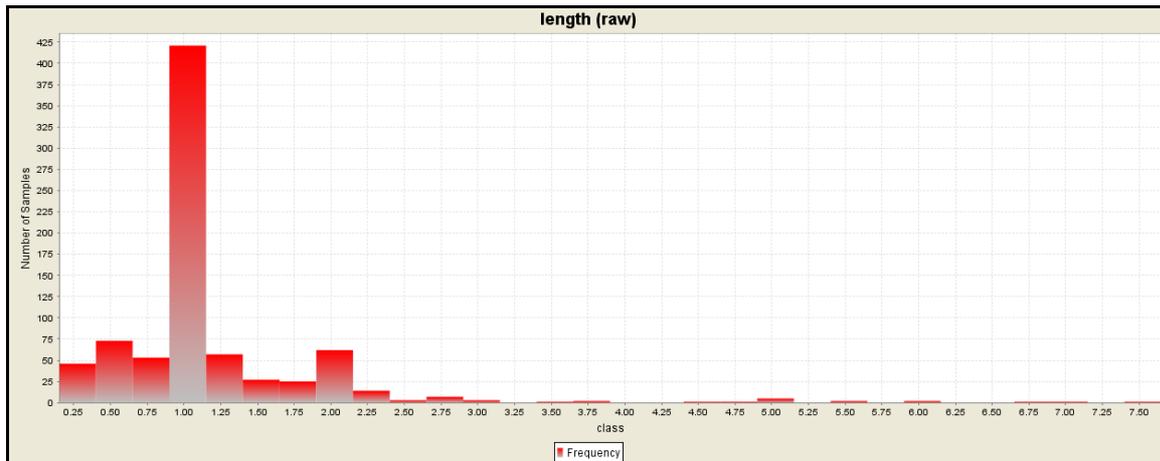


Figure 8: Histogram of sample lengths
(samples_skarn1.str)

4.2 Basic Statistics

4.2.1 Introduction

The mean grades for Sn, W, F and Fe of the composited samples from the skarn are listed in Table 7.

Table 7: Mean grades of composited samples in skarn
(comps_skarn1.str)

	Sn ppm	W ppm	F %	Fe %
number of samples	926	926	705	411
mean grade	1270	868	7.72	13.8

4.2.2 Basic Statistics for Sn

A histogram and log histogram for % Sn of the composited samples are shown in Figures 9 and 10. As expected, the % Sn distribution appears to be log-normal.

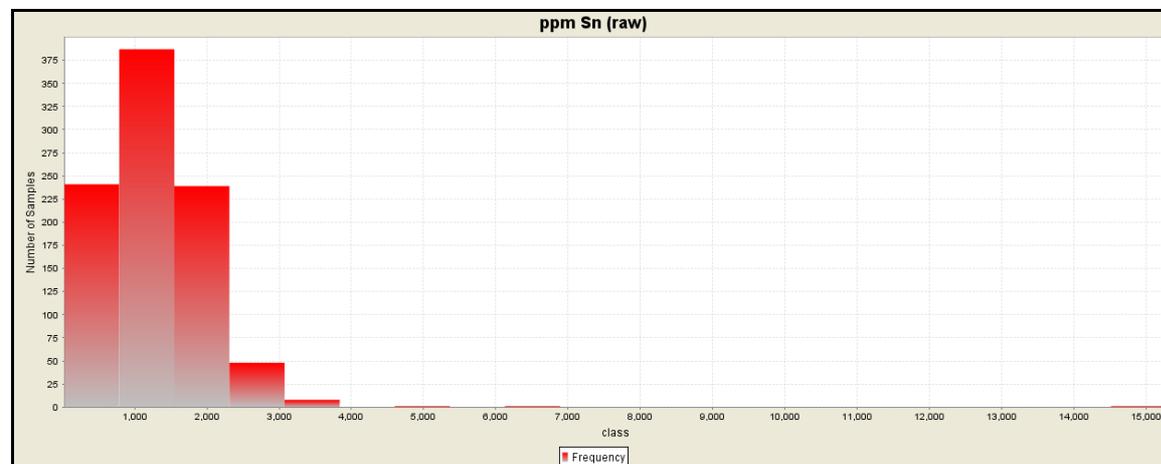
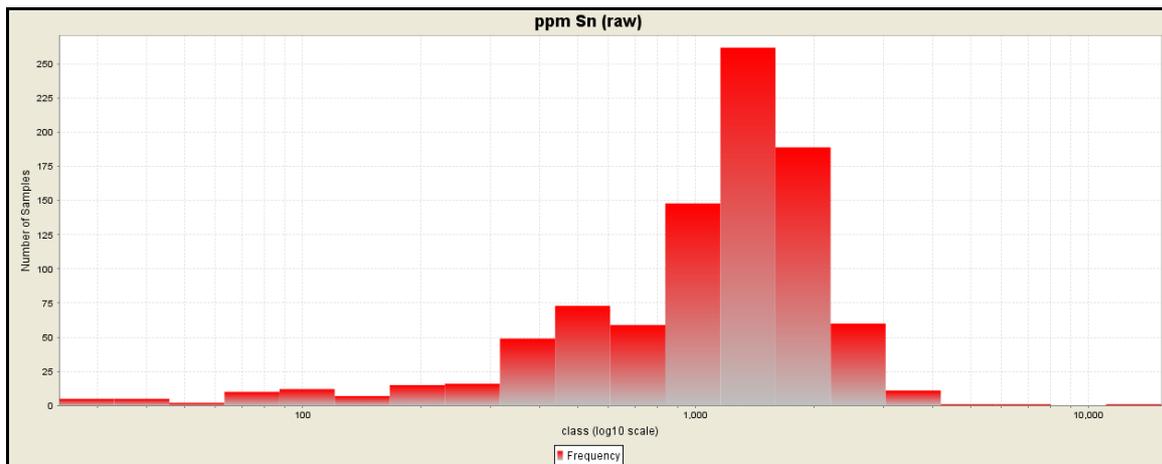


Figure 9: Histogram for % Sn of composited samples in skarn
(comps_skarn1.str)



**Figure 10: Log histogram for % Sn of composited samples in skarn
(comps_skarn1.str)**

4.2.3 Basic Statistics for W

A histogram and log histogram for % W of the composited samples are shown in Figures 11 and 12. As expected, the % W distribution appears to be log-normal.

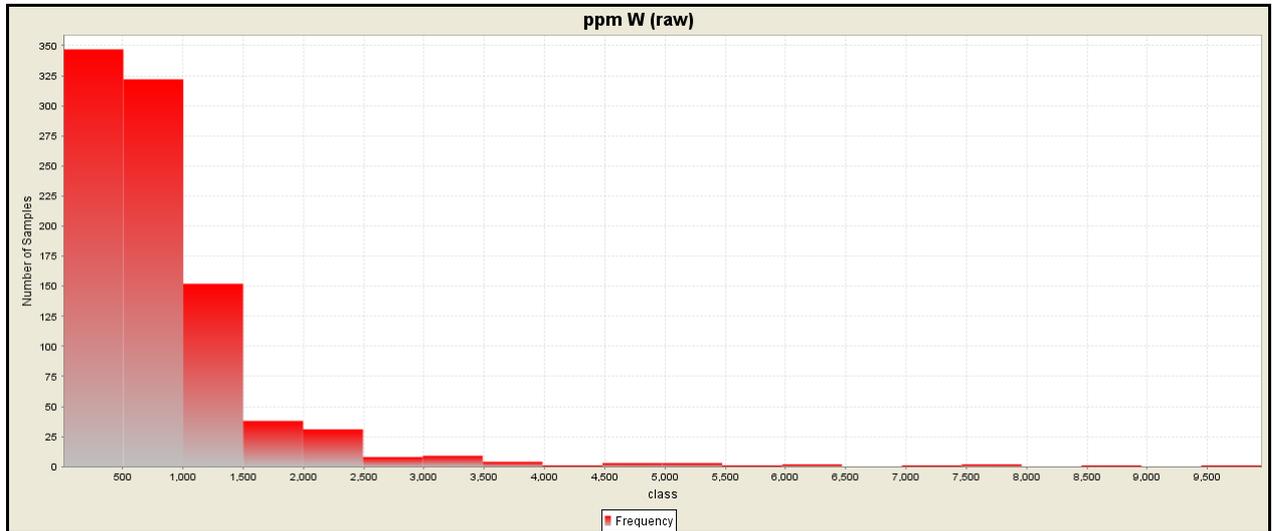


Figure 11: Histogram for % W of composited samples in skarn
(comps_skarn1.str)

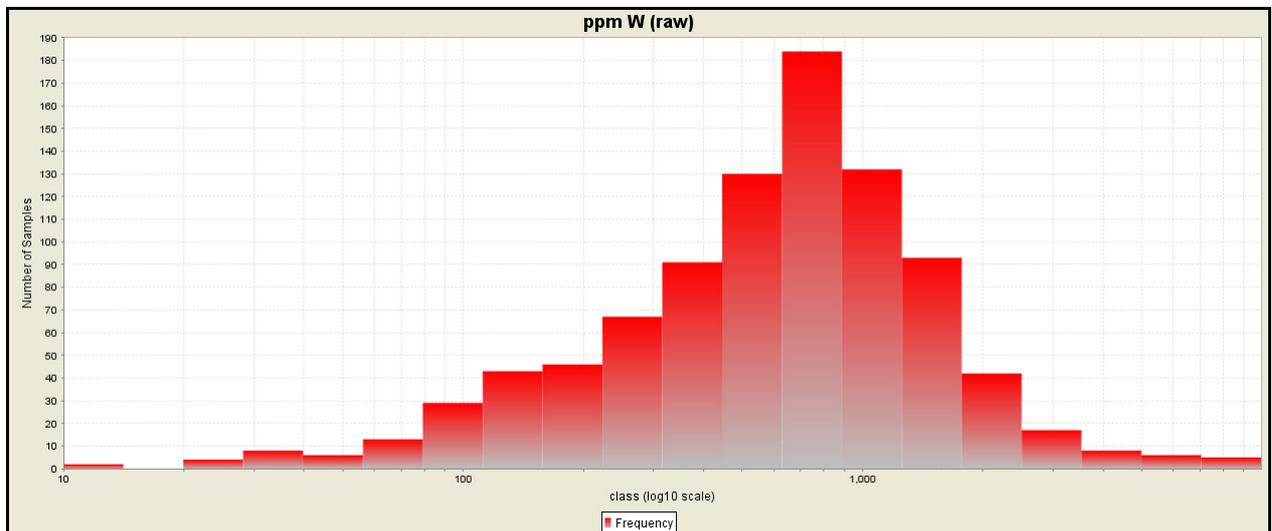
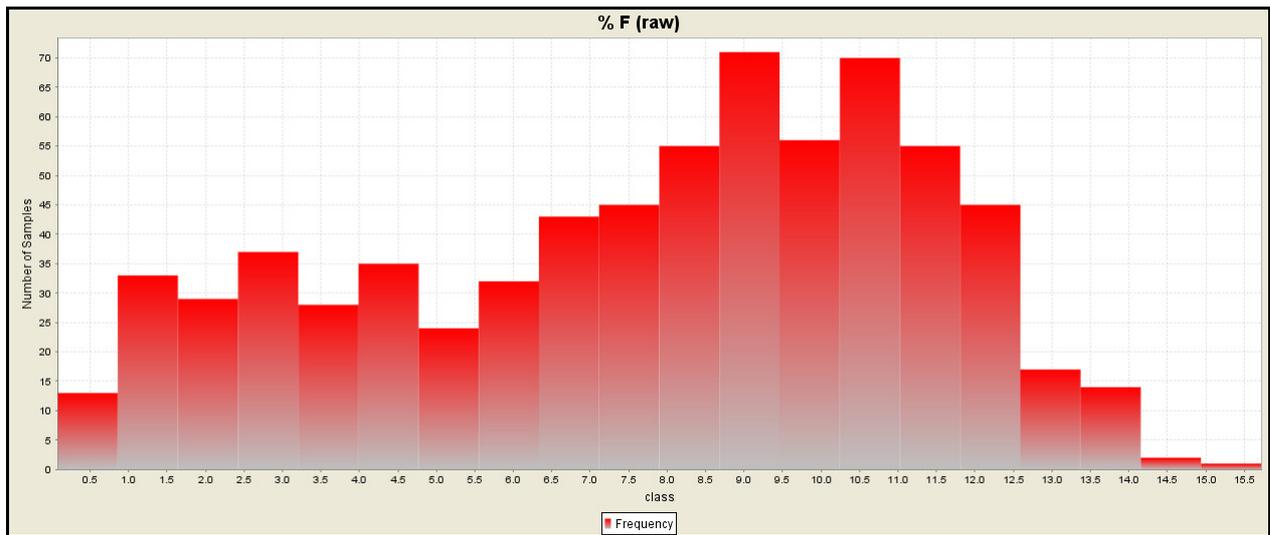


Figure 12: Log histogram for % W of composited samples in skarn
(comps_skarn1.str)

4.2.4 Basic Statistics for F

A histogram for % F is shown in Figure 13. The distribution does not appear to be particularly log-normal but does appear to be multi-modal, suggesting that there are two or more domains of F occurrence within the skarn. The domains could be related, for example, to particular mineral assemblages, perhaps in particular stratigraphic positions.

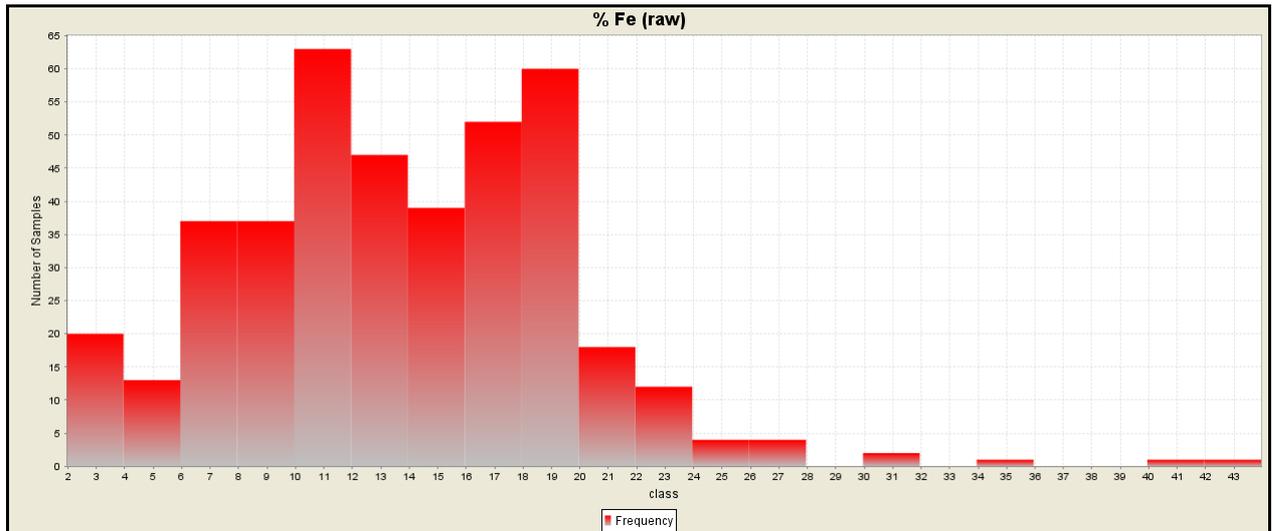


**Figure 13: Histogram for % F of composited samples in skarn
(comps_skarn1.str)**

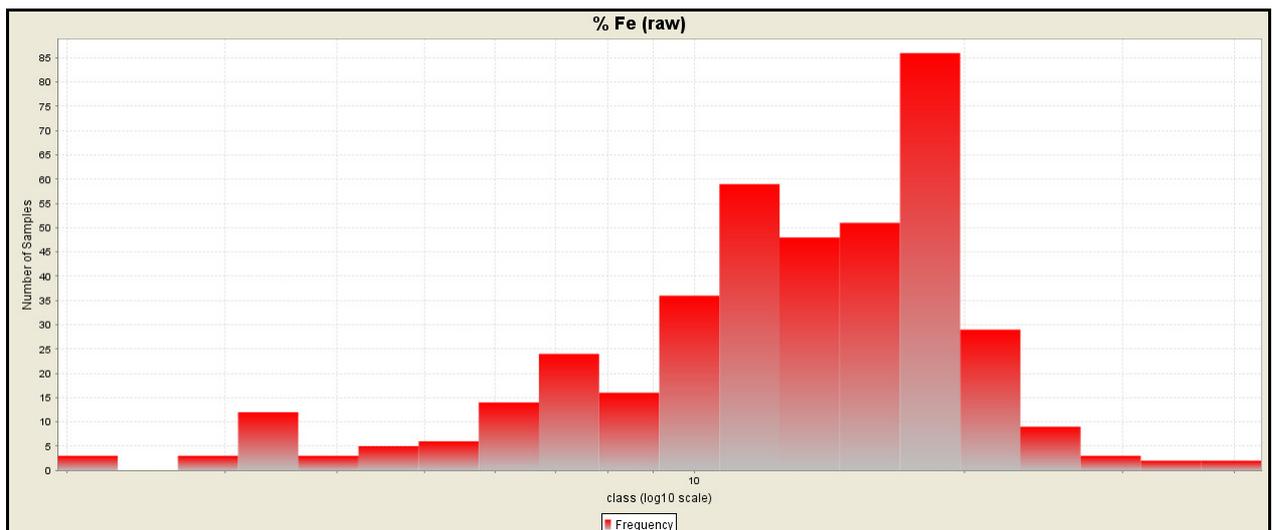
4.2.5 Basic Statistics for Fe

A histogram for % Fe is shown in Figure 14. As was the case for the distribution of % F, the distribution appears to be multi-modal, suggesting that there are two or more domains of Fe occurrence within the skarn. The domains could be related, for example, to particular mineral assemblages, perhaps in particular stratigraphic positions.

The distribution of % Fe appears to be somewhat log-normal (see Figures 14 and 15).



**Figure 14: Histogram for % Fe of composited samples in skarn
 (comps_skarn1.str)**



**Figure 15: Log histogram for % Fe of composited samples in skarn
 (comps_skarn1.str)**

4.3 Bulk Density

Previous workers have attributed a bulk density of 3.3 tonnes per cubic metre to the Moina skarn.

Using an Archimedes method, Minemakers estimated that the average density of 25 core samples from holes MODD001 to MODD004 was 3.1 tonnes per cubic metre. Somewhat unexpectedly, there was poor correlation between the Fe grade and density for these samples (see Figure 16).

In recognition of the wider spatial distribution of the historic drill holes compared to the Minemakers' drill holes, a density of 3.3 tonnes per cubic metre was retained for this estimate. Nevertheless, the matter of density should be further investigated in the future.

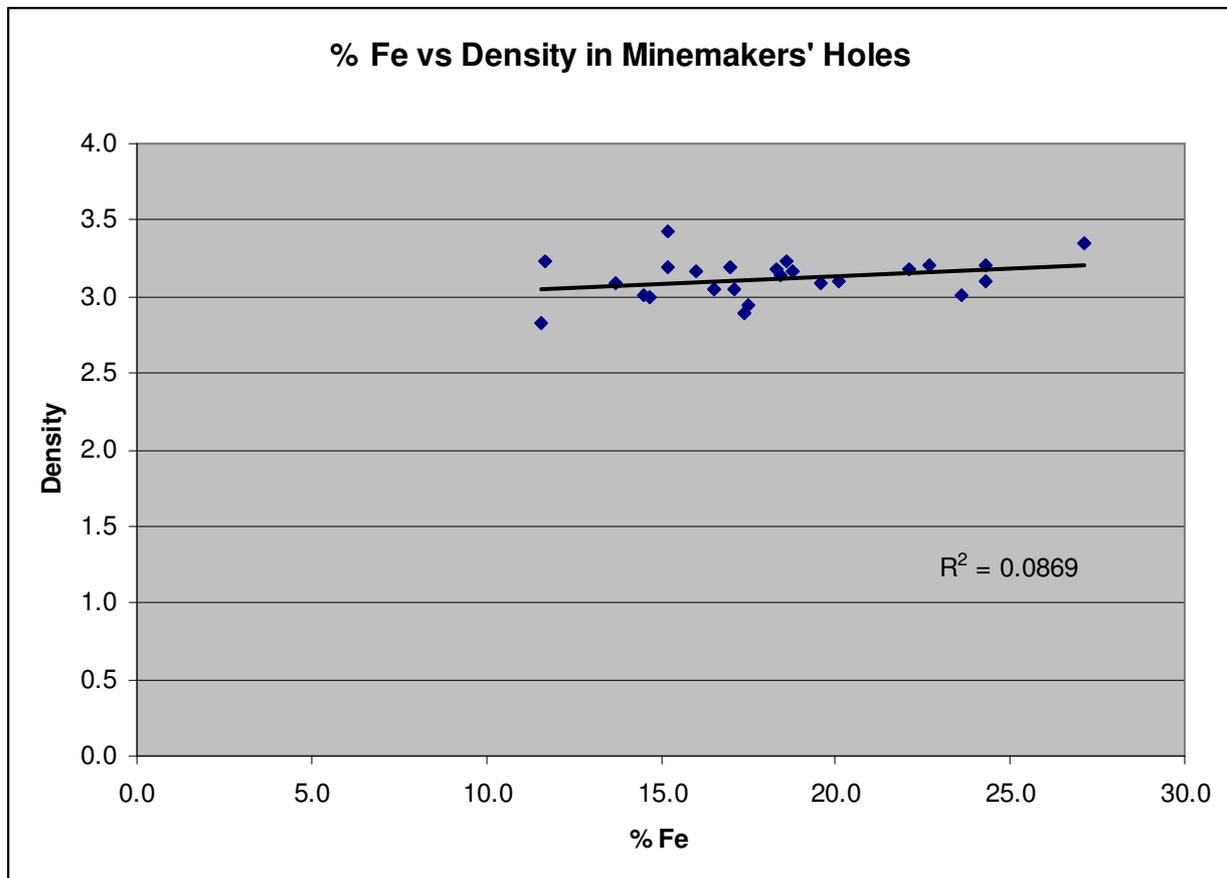


Figure 16: % Fe vs density for samples from Minemakers' holes MODD001 to MODD004

(8_Moina_Density_2009_Wrigglite.xls)

4.4 Global Resource Estimate

4.4.1 Wireframe and Block Model

A wireframe was created from the skarn boundaries on the six cross-sections described above (see 2.2 Geological Interpretation and Figure 4).

A Gemcom Surpac block model was created with the dimensions listed in Table 8.

The nominal skarn wireframe extended somewhat beyond the extent of the magnetite mineralisation (see 2.2 Geological Interpretation), so, blocks in the block model were flagged as skarn where the blocks occurred:

- within the nominal skarn wireframe named interp_skarn3.dtm, and
- within the 62500nT boundary string named mag_62500nt1.str.

Table 8: Block model extents and block dimensions
(momoina1201.mdl)

Direction	Minimum	Maximum	Block size
X	422,900E	423,650E	10m
Y	5,406,350N	5,406,950N	10m
Z	400RL	750RL	10m

4.4.2 Global Resource Estimate

There were insufficient data for successful variography, so, grades for Sn, W, F and Fe were interpolated into the block model using an inverse distance squared method (“ID2”).

ID2 is a widely used method when variography has been unsuccessful and geostatistical methods such as ordinary kriging (“OK”) cannot be implemented. ID2 provides good global estimates if the samples used to make the estimate are not clustered. Local estimates made using ID2 are not as reliable as OK estimates.

Grades were interpolated using only samples from within the skarn (see Table 9). A spherical search was used and the search radii were selected to ensure that all skarn blocks were informed (see Table 10).

Blocks flagged as skarn were attributed the density used by Askins (1979) of 3.3 tonnes per cubic metre.

Estimates were made on a dry tonnes basis.

The global resource was 24,654,300 tonnes at 1379ppm Sn, 1043ppm WO_3 , 15.99% CaF_2 and 17.2% Fe.

One check of the reliability of an ID2 global resource estimate is a comparison of the grades of the estimate with the average grades of the composited samples used to make the estimate. There was very good agreement between the grades of the ID2 estimate and the average grades of the composited samples (see Table 11).

A second check of the reliability of an ID2 global resource is a comparison of the grades of the estimate with the grades of a nearest neighbour (“NN”) estimate. There was very good agreement between the grades of the ID2 estimate and the average grades of the NN estimate (see Table 11).

Table 9: Skarn intercepts
(moina_samples_conversion.xls)

hole_id	depth_from	depth_to
ML002	insufficient assay data	insufficient assay data
MD040	14.20	145.00
MD041	10.00	124.00
MODD001	17.00	83.40
MODD002	0.00	53.00
MODD003	14.00	88.00
MODD004	31.40	89.00
SMD004	68.37	92.45
SMD006	21.35	83.05
SMD007	0.00	71.00
SMD010	38.75	86.00
SMD011	32.90	74.00
SMD015	33.00	100.70
SMD028	29.00	101.00
SMD029	0.00	23.25
SMD030	19.90	37.80

Table 10: Search ellipsoid parameters

Search Ellipsoid Parameters		
	Direction	Axis Radius
Major	00/000	200m
Semi-major	00/090	200m
Minor	-90/000	200m

**Table 11: Grades of global resource and composited samples
(comps_skarn1.str, momoina1201.mdl)**

	Sn ppm	W ppm	F %	Fe %
Block model ID2	1379	827	7.79	17.2
Mean grades of composited samples	1270	868	7.72	13.8
Block model NN	1378	814	7.56	16.5
	Sn ppm	WO₃ ppm	CaF₂ %	Fe %
Block model ID2	1379	1043	15.99	17.2
Mean grades of composited samples	1270	1095	15.85	13.8
Block model NN	1378	1026	15.52	16.5

4.4.3 Previous Estimate

A previous estimate of the resource was made by Askins (1979) before the JORC Code was introduced. The estimate was a global estimate based on length-weighted averages of all samples within the skarn which is an acceptable method for a global estimate. Askins estimated a global resource with 0.1% Sn, 0.1% W and 18% CaF₂. The Sn and W grades compare well with those estimated for this report (see Table 11). However, the CaF₂ grade of 18% estimated by Askins is somewhat higher than the grade of 15.99% estimated for this report.

One reason for the difference in the F grade of the two estimates is the lower CaF₂ grade of the samples from the holes drilled by Minemakers compared to those from the historic holes (see Table 12).

Table 12: Comparison of mean grades of samples of skarn from historic and Minemakers holes

(comps_skarn1_old_and_new.xls)

	Sn	W	F	Fe
old holes	1281	925	8.18	10.8
Minemakers holes	1240	718	6.91	15.7
	Sn	WO₃	CaF₂	Fe
old holes	1281	1166	16.79	10.8
Minemakers holes	1240	905	14.19	15.7

4.5 Mineral Resource Estimate

4.5.1 Mineral Resource Estimate

No assumptions regarding mining methods and mining factors or metallurgical processes and metallurgical assumptions were made for this report. Ore Reserves have not been estimated.

Within the spatial limits of the resource estimate, given the magnetic signature of the skarn and the 15 intercepts in drill holes, the extent and continuity of the skarn is reasonably assured. However, the continuity of F, Sn and W grades can only be assured with closer spaced drilling.

The resource estimate was made using historic data which, although it was reported by the reputable operators Mt Lyell, Comalco and Billiton, has not been verified. The data could be verified by re-sampling historic core.

In addition, at present, the results of laboratory checks of duplicates and standards are not currently available for the Minemakers' assay data. Copies of the checks and duplicate assays are being sought from the laboratories concerned.

Taking these considerations into account, the Mineral Resource has been classified as Inferred (see Table 13).

Table 13: Moina Mineral Resource 3 October 2012

Moina Mineral Resource 3 October 2012					
zero cut-off grades					
	Tonnage	% CaF₂	% Sn	% WO₃	% Fe
Inferred	24.6M	16.0	0.1	0.1	17.2
Total	24.6M	16.0	0.1	0.1	17.2

4.5.2 JORC Criteria for Estimation and Reporting of Mineral Resources

The criteria nominated in the JORC Code and the places in this report where they have been discussed are listed in Table 14.

Table 14: Criteria for Estimation and Reporting of Mineral Resources in JORC Code

Estimation and Reporting of Mineral Resources	
Database integrity.	3 DATABASE
Geological interpretation.	2.2 Geological Interpretation
Dimensions.	2.2 Geological Interpretation
Estimation and modelling techniques.	4.4 Global Resource Estimate
Moisture.	4.4 Global Resource Estimate
Cut-off parameters.	4.5.1 Mineral Resource Estimate
Mining factors or assumptions.	4.5.1 Mineral Resource Estimate
Metallurgical factors or assumptions.	4.5.1 Mineral Resource Estimate
Bulk density.	4.3 Bulk Density
Classification.	4.5.1 Mineral Resource Estimate
Audits or reviews.	This report was peer reviewed internally by Mining One.
Discussion of relative accuracy/confidence	4.5.1 Mineral Resource Estimate

5 REFERENCES

- 1) **Anonymous, 2011.** An investigation into mineralogical characteristics of the Moina fluorspar, tin, tungsten and bismuth deposit for the purpose of metallurgical beneficiation, Project 12547-001 Final Report, February 9, 2011. Unpublished Report by SGS Minerals Services for TNT Mines Limited.
- 2) **Askins, P.W., 1979.** E.I.7/74 – Moina, Areas Covered by Moina Sheets 1, 2, 3, 1979 Update and Moina Sheet A, Report on All Investigations to August, 1979. Unpublished Report for Commonwealth Aluminium Corporation Limited Exploration Department.
- 3) **Drummond, A., 2006.** Annual Report Retention Licence RL 10/1988, Moina, October 2006. Unpublished Report, Minemakers Limited.

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DOCUMENT REVIEW AND SIGN OFF

Version	Reviewer	Position	Signature	Date
2	S. Hutchin	Geology Manager		19/10/12

**APPENDIX 2 – Metallurgical assessment of Moina ore for TNT Mining. Report
No: TO728 September 2012. ALS Ammtec Burnie.**



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METALLURGICAL ASSESSMENT

OF

MOINA ORE

FOR

TNT MINING

REPORT NO: T0728

S ZHAO

SEPT 2012

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1. SUMMARY

Bagged dry split cores were received and fully blended to treat as a bulk sample for ball bond mill work index test, heavy liquid test, magnetic separation, gravity separation and flotation assessment for recovery of tungsten and fluorite values.

Analysis results indicated 23.6% CaO, 20.22% Fe, 25.18% SiO₂, 0.14% WO₃, 8.45% F and 0.17% Sn.

A Bond Ball Work Index yielded a value of 18.6 Kilowatt hours/dry tonne, indicating a hard to very hard value.

Heavy liquid assessment indicated a low mass rejection (8.4%) to float product, also indicating poor selectivity.

Magnetic separation

Gravity separation for scheelite recovery indicated 69% recovery of tungsten could be achieved to a low concentrate grade. Dressing of this concentrate improved the grade to 41% WO₃.

Tungsten flotation yielded a low grade tungsten concentrate with poor selectivity performance in the float.

Fluorite flotation results indicated acid grade fluorite (>95% CaF₂) could be achieved with a simple reagent scheme and flotation conditions. The best flotation results (T13) showed 80.6% recovery could be achieved at 95% grade. The main loss of fluorite was as locked grains reporting to the magnetite product in magnetic separation.

2. SAMPLE RECEIVED

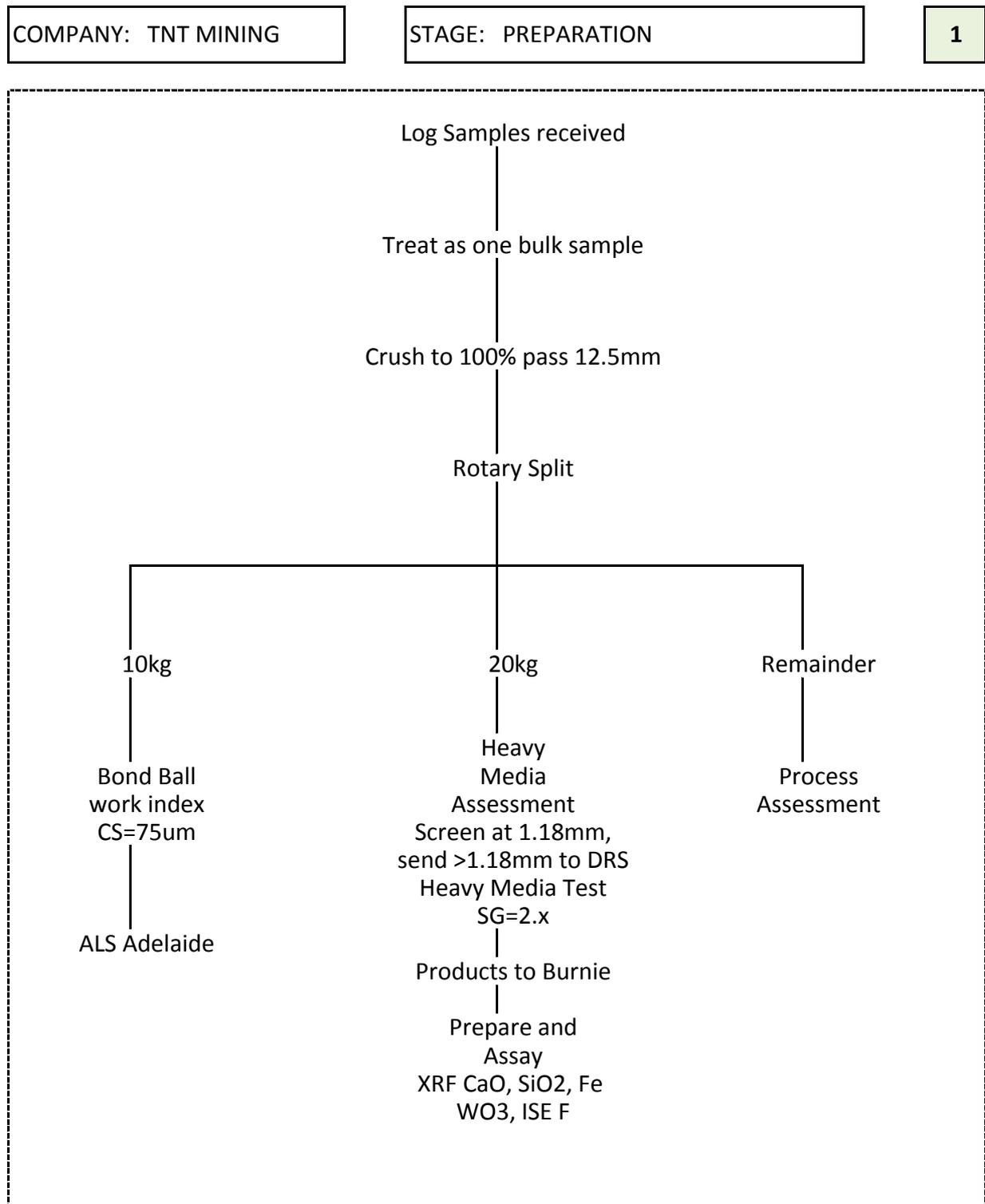
Samples were received as bagged dry split core. Samples were fully blended and treated as one bulk sample. The total weight of the received samples was 128.21kg. Individual intersection weights are appended as below.

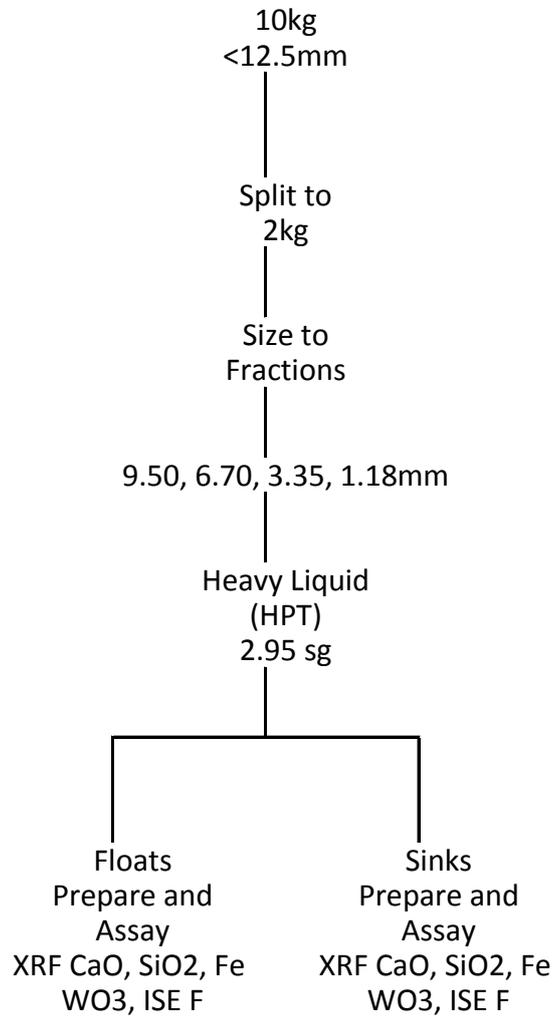
Sample Name		MASS (kg)
Bag1	USC 1	3.37
	11	3.53
	20	4.38
	26	2.75
	41	2.9
Bag 2	9	2.54
	10	3.88
	12	2.32
	13	3.77
	14	3.29
	21	2.44
Bag 3	2	2.57
	3	2.45
	4	1.97
	5	1.68
	6	2.98
	7	2.35
	8	3.09
	37	1.89
Bag 4	15	3.01
	16	3.87
	17	2.76
	18	3.75
	19	2.22
	35	2.15
	36	2.10
Bag 5	22	2.17
	23	2.18
	24	3.66
	25	2.14
	27	2.03
	28	2.33
	29	2.38
	30	3.04
Bag6	USC 32	2.62
	33	2.94
	38	2.76
	39	2.33
	46	2.49
	47	2.53
Bag 7	31	2.63
	34	2.22
	40	2.06
	42	2.69
	43	2.62
	44	2.10
	45	2.50
	48	1.78

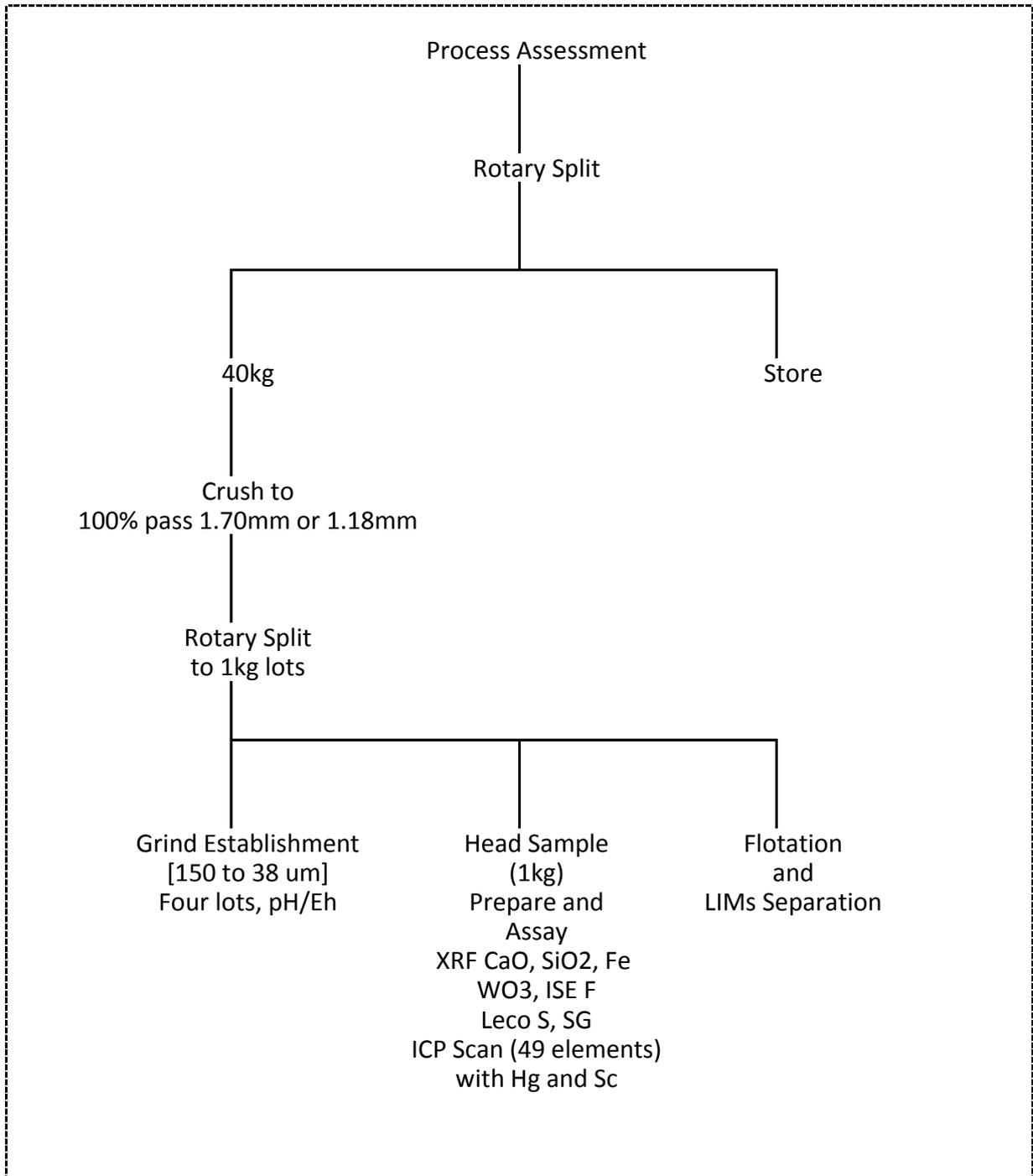
3. TESTWORK FLOWSHEET

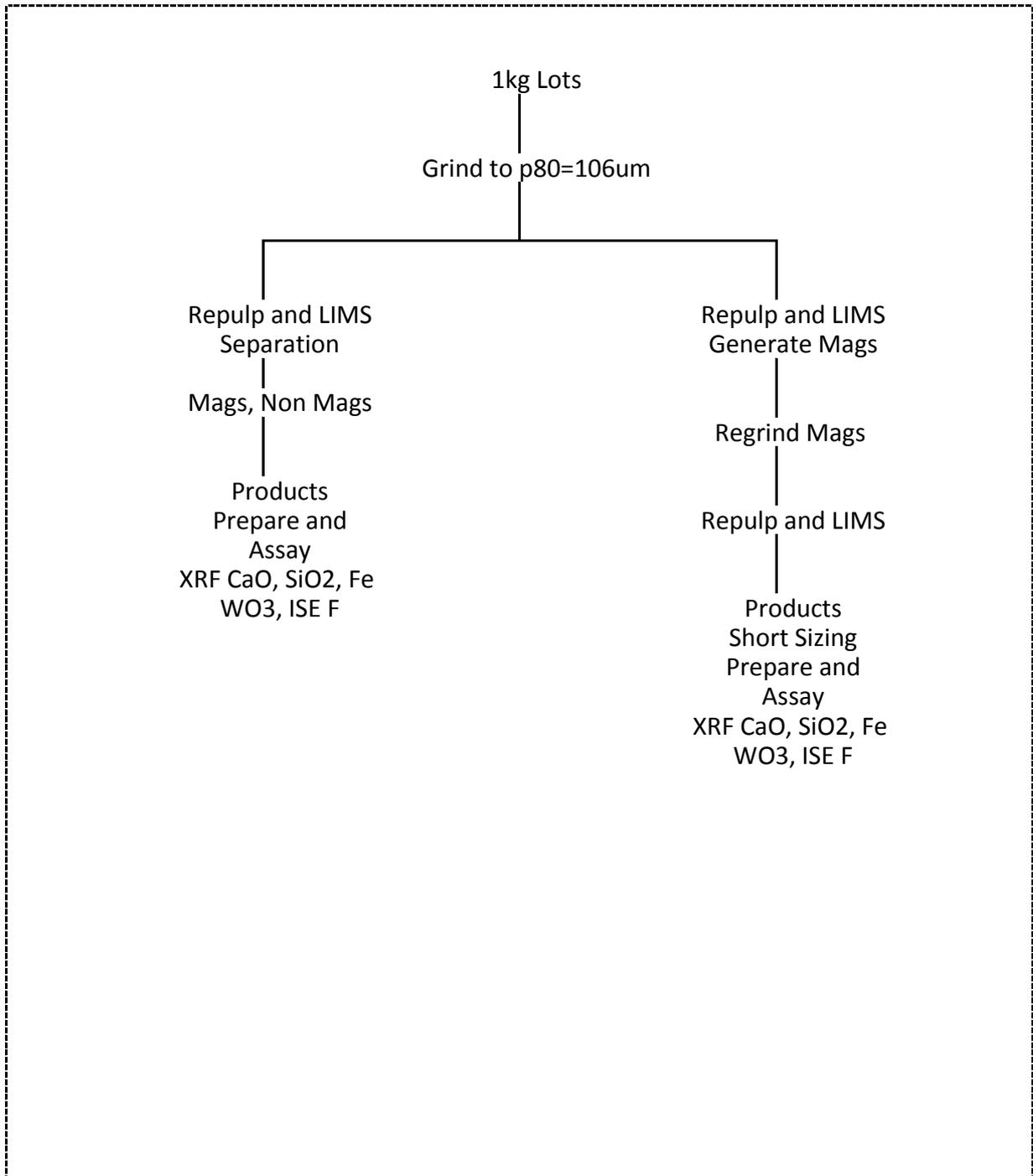
The following diagrams summarise the test program performed.

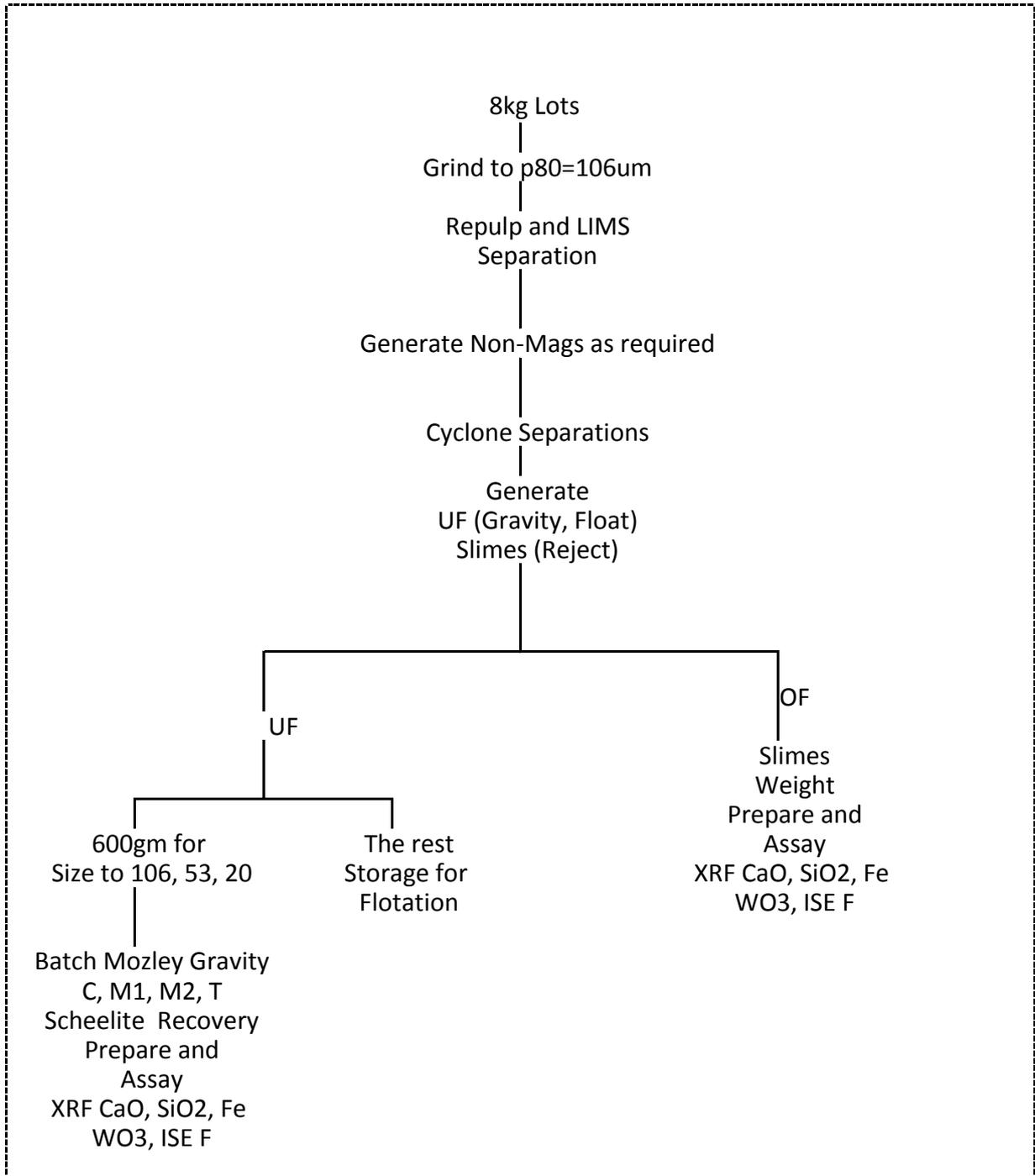
DIAGRAM 1: TESTWORK











Rougher Flotation: Scheelite
2 Floats

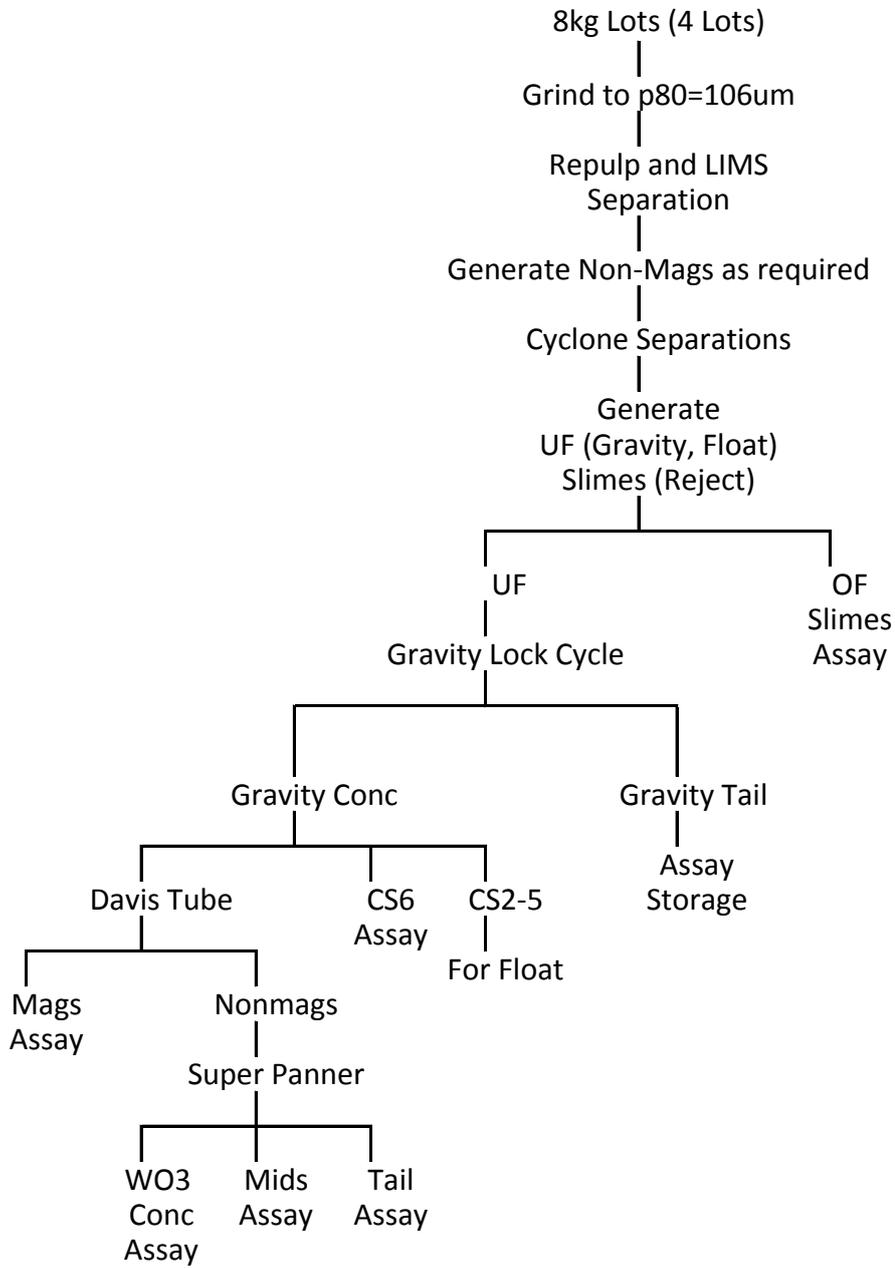
[Na₂CO₃/Na₂SiO₃/Dextrin]
[FS2]
[MIBC/76A]
Combinations

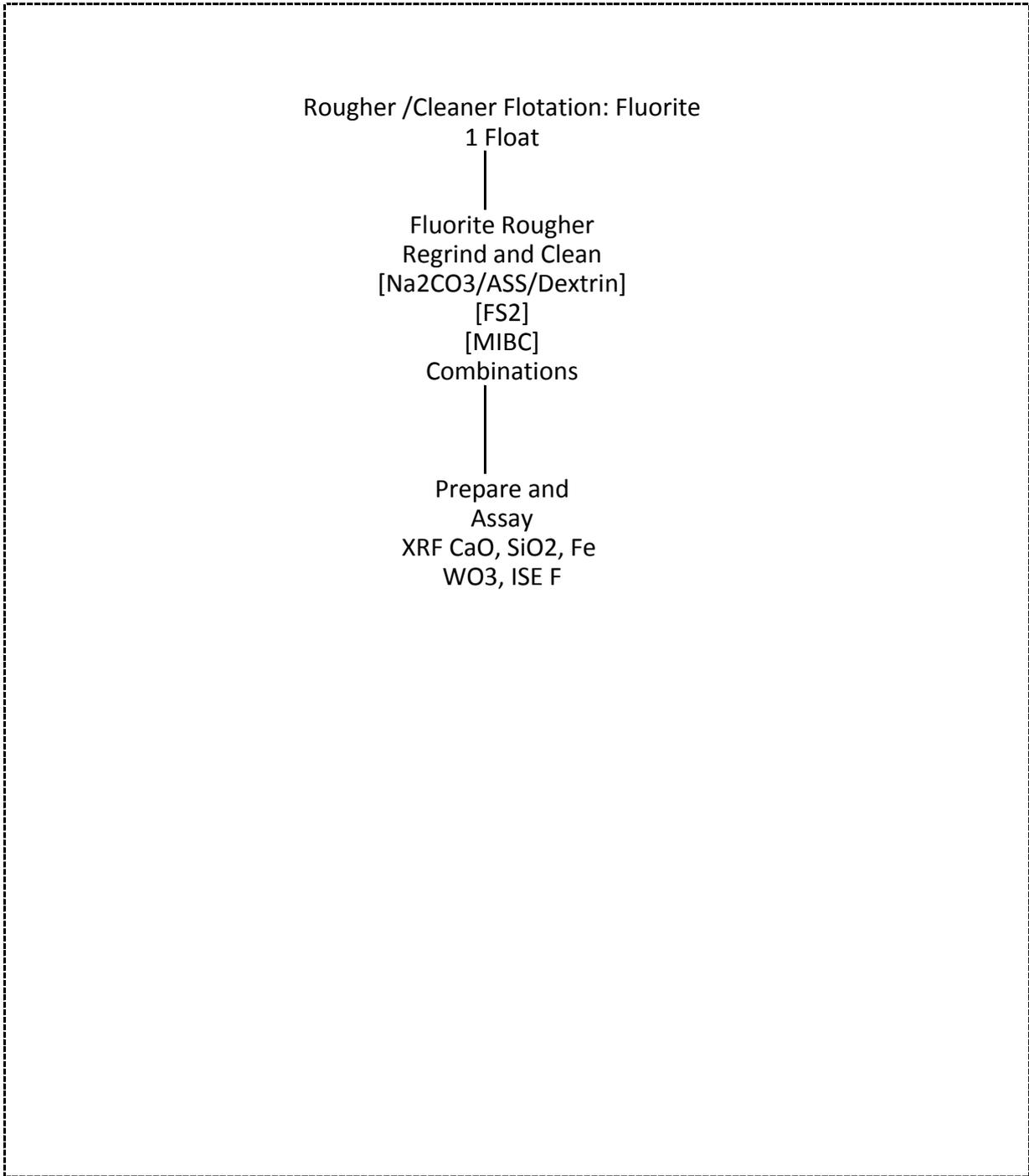
Prepare and
Assay
XRF CaO, SiO₂, Fe
WO₃, Sn

Rougher /Regrind/Cleaner Flotation: Fluorite
3 Floats

Fluorite Rougher
Regrind and Clean
[Na₂CO₃/ASS/Dextrin]
[FS2]
[MIBC]
Combinations

Prepare and
Assay
XRF CaO, SiO₂, Fe
WO₃, ISE F





4. TEST METHODS

SAMPLE PREPARATION:

Samples were crushed in a single toggle lab jaw crusher to 100% passing 12.5mm. 10kg of sample was split for Bond Ball Work Index test. 20kg of sample was split for Heavy Media assessment. 10kg of sample was further split to 2kg lots for Heavy Liquid testing. Another 40kg of sample was crushed to 100% passing 1.18mm, rotary split to 1kg lots and cold stored for further LIMS separation and flotation tests.

MAGNETIC SEPARATION

- Separations were performed on a SALA laboratory wet rotating drum magnet.
- The drum operates with a susceptibility of 950 gauss at the drum surface.
- One lot of each sample was grind to p80=106um and repulped and pumped at 2lpm rate to the rotating drum hopper.
- Mags are attracted and lifted out of the pulp for removal by scrapper and spray water. Mags were filtered to cake and submitted for assay.
- Non mags are removed by diversion through a 3mm spigot below the rotating drum and feed to further tests.

GRAVITY SEPARATION

- 600gm representative sample from sulphide tail was sized to fractions: 106, 53, 20, -20um.
- Sample of -20um was submitted for assay.
- Fractions were separated by gravity on a Mozley Gravity Table to generate concentrate, middling and tailing product.
- Representative samples of C1, M1, M2, T were submitted for assay.

TUNGSTEN FLOTATION

Tungsten rougher flotation tests were performed under the following conditions:

- Flotation feed was non mags underflow product.
- P80 of feed was 102um.
- Flotation rougher performed in a 1.2L Agitair style laboratory cell.
- Dilute reagents were added and conditioned prior to flotation.
- Air rate and time were recorded for each concentrate.

FLOTATION REAGENTS USED

The following reagents were used in this test program:

- H₂SO₄, pH modifier.
- PAX, collector, Potassium Amyl Xanthate, Orica.
- FS2, collector, Clariant.
- STP, dispersant, Sodium Tripolyphosphate
- Guar, depressant.
- Dextrin, depressant.
- ASS, depressant, Acidized Sodium Silicate.
- Na₂SiO₃, depressant, Sodium Silicate, N40, Orica.
- MIBC, frother, Methyl Iso Butyl Carbinol, Shell.

FLUORITE FLOTATION

Fluorite rougher/cleaner flotation tests were performed under the following conditions:

- Flotation feed of T09,10,12 was non mags underflow product with p80 102um.
- Flotation feed of T13 was CS2-CS5 from gravity lock cycle tests.
- Flotation rougher performed in a 1.2L Agitair style laboratory cell.
- Dilute reagents were added and conditioned prior to flotation.
- Air rate and time were recorded for each concentrate.

FLOTATION REAGENTS USED

The following reagents were used in this test program:

- H₂SO₄, pH modifier, 98%.
- Na₂CO₃, pH modifier.
- Dextrin, depressant.
- ASS, depressant, Acidized Sodium Silicate, pH 1.75.
- FS2, collector, Clariant.
- MIBC, frother, Methyl Iso Butyl Carbinol, Shell.

5. TESTS PERFORMED

Table 1 summarises feed sources, test type and basic conditions for the tests performed.

TABLE 1: TEST SUMMARY

TEST NO	ORE	FEED	TEST	Note
T01	Moina	-12.5mm	Heavy Liquid Test	SG=2.95
T02	Moina	P80=106um	Drum Mag Separation with Reground	1000Gauss
T03	Moina	P80=106um	Drum Mag Separation	1000Gauss
T04	Moina	P80=200um	Drum Mag Separation	1000Gauss
T05	Moina	P80=106um	Mag Separation and Hydrocyclone	1000Gauss
T06	Moina	UF from T05	Mozley Separation	
T07	Moina	UF from T05	WO3 Flotation	Dextrin, FS2, MIBC, 76A
T08	Moina	UF from T05	WO3 Flotation	Na ₂ SiO ₃ , STP, Dextrin, FS2, MIBC
T09	Moina	UF from T05	Fluorite Flotation	Dextrin, ASS, Na ₂ CO ₃ , FS2, MIBC
T10	Moina	UF from T05	Fluorite Flotation	Dextrin, ASS, Na ₂ CO ₃ , FS2, MIBC
T11	Moina	Ro Conc from T10	Mag Separation	
T12	Moina	UF from T05	Fluorite Flotation	Dextrin, ASS, Na ₂ CO ₃ , FS2, MIBC
T13	Moina	CS2-5	Fluorite Flotation	Dextrin, ASS, Na ₂ CO ₃ , FS2, MIBC, H ₂ SO ₄
T14	Moina	Gravity LC Combined Conc	Davis Tube Test	3000Gauss
T15	Moina	Non mags from T14	Super Panner Test	
T16	Moina	-12.5mm	Ball Bond Work Index	
T17	Moina	-1.18mm (1kg, 8kg) -1.7mm (1kg)	Grind Establishments	
T18	Moina	P80=106um	Gravity Lock Cycle Feed Preparation (Mag Separation and Hydrocyclone)	
LC1	Moina	UF from T18	Gravity Separation Lock Cycle (6 cycles)	

6. RESULTS

BOND BALL Mill WORK INDEX

The bond ball mill work index is 18.6 Kilowatt hours/dry tonne, a high value indicating the ore is hard.

HEAVY LIQUID SEPARATION

Heavy liquid separation was performed on the 2kg of representative sample (100% passing 12.5mm screen). The test results show that 8.41% of sample was floated, which indicated the separation is ineffective. The test results were summarised as below:

TABLE 2: HEAVY LIQUID SEPARATION SUMMARY

FRACTIONS mm	WT %	Overall		Overall	
		%WO3	dist	%Sn	dist
Total Floats	8.41	0.13	7.32	0.05	2.92
Total Sinks	80.75	0.15	77.06	0.16	85.97
Total Fines	10.84	0.22	15.61	0.15	11.11
CALC FEED	100.00	0.15	100.00	0.15	100.00

MAGNETIC SEPARATION

Table 3 below details results of the magnetic separation. Some 41.8% of mass could be removed with the loss of 24.1% of WO₃ and 31.8% of F.

Table 4 shows some 36.86 of magnetic could be removed with the loss of 17.9% of WO₃ and 18.8% of F by applying regrind on magnetics after the first magnetic separation.

TABLE 3: MAGNETIC SEPARATION

PRODUCT				WO ₃		ISE F	
NAME	Wt (gm)	Calc (gm)	Wt (%)	WO ₃ (%)	DIST (%)	ISE F (%)	DIST (%)
Non Magnetics	1165.3	1165.3	58.34	0.18	75.9	12.40	68.2
Magnetics	832.3	832.3	41.66	0.08	24.1	8.10	31.8
TOTAL	1997.6		100.00	0.14	100.0	10.61	100.0

TABLE 4: MAGNETIC SEPARATION WITH REGROUND

PRODUCT				WO ₃		ISE F	
NAME	Wt (gm)	Calc (gm)	Wt (%)	WO ₃ (%)	DIST (%)	ISE F (%)	DIST (%)
Non Magnetics	1127.7	1127.7	57.33	0.19	75.6	10.40	70.6
Re Grind Non Magnetics	114.3	114.3	5.81	0.16	6.5	15.50	10.7
Re Grind Magnetics	725.2	725.2	36.86	0.07	17.9	4.30	18.8
TOTAL	1967.20		100.0	0.14	100.0	8.45	100.0

MAGNETIC SEPARATION AND CYCLONE CLASSIFICATION

Table 5 details the results of magnetic separation and cyclone classification. Some 39.8% of mass and 57.9% of WO₃ were recovered in product of non mags underflow.

TABLE 5: MAGNETIC SEPARATION AND CYCLONE CLASSIFICATION

PRODUCT				CaO		Iron	
NAME	Wt (gm)	Calc (gm)	Wt (%)	CaO (%)	DIST (%)	Fe (%)	DIST (%)
Magnetics	3689.9	3689.9	47.25	16.70	34.97	33.50	79.03
OF	1005.6	1005.6	12.88	27.90	15.92	7.85	5.05
UF	3113.5	3113.5	39.87	27.80	49.11	8.00	15.92
TOTAL	7809.0		100.0	22.57	100.0	20.03	100.0

Silica		WO ₃		Sn	
SiO ₂ (%)	DIST (%)	WO ₃ (%)	DIST (%)	Sn (%)	DIST (%)
18.80	35.11	0.09	30.88	0.16	43.61
28.10	14.30	0.12	11.22	0.14	10.40
32.10	50.59	0.20	57.90	0.20	45.99
25.30	100.0	0.14	100.0	0.17	100.0

GRAVITY SEPARATION OF TUNGSTEN

Mozley gravity separation was performed on UF (underflow from cyclone classification) to determine gravity amenability to tungsten recovery and to produce a small high-grade concentrate. Gravity separation results are summarised in Tables 6 below.

TABLE 6: MOZLEY GRAVITY SEPARATION SUMMARY

PRODUCT	Wt (%)	WO ₃ (%)	DIST (%)
CONC	2.1	5.42	69.0
MIDS 1	15.2	0.09	8.5
MIDS 2	27.3	0.06	9.5
TAILS	55.4	0.04	13.0
TOTAL	100.0	0.16	100.0

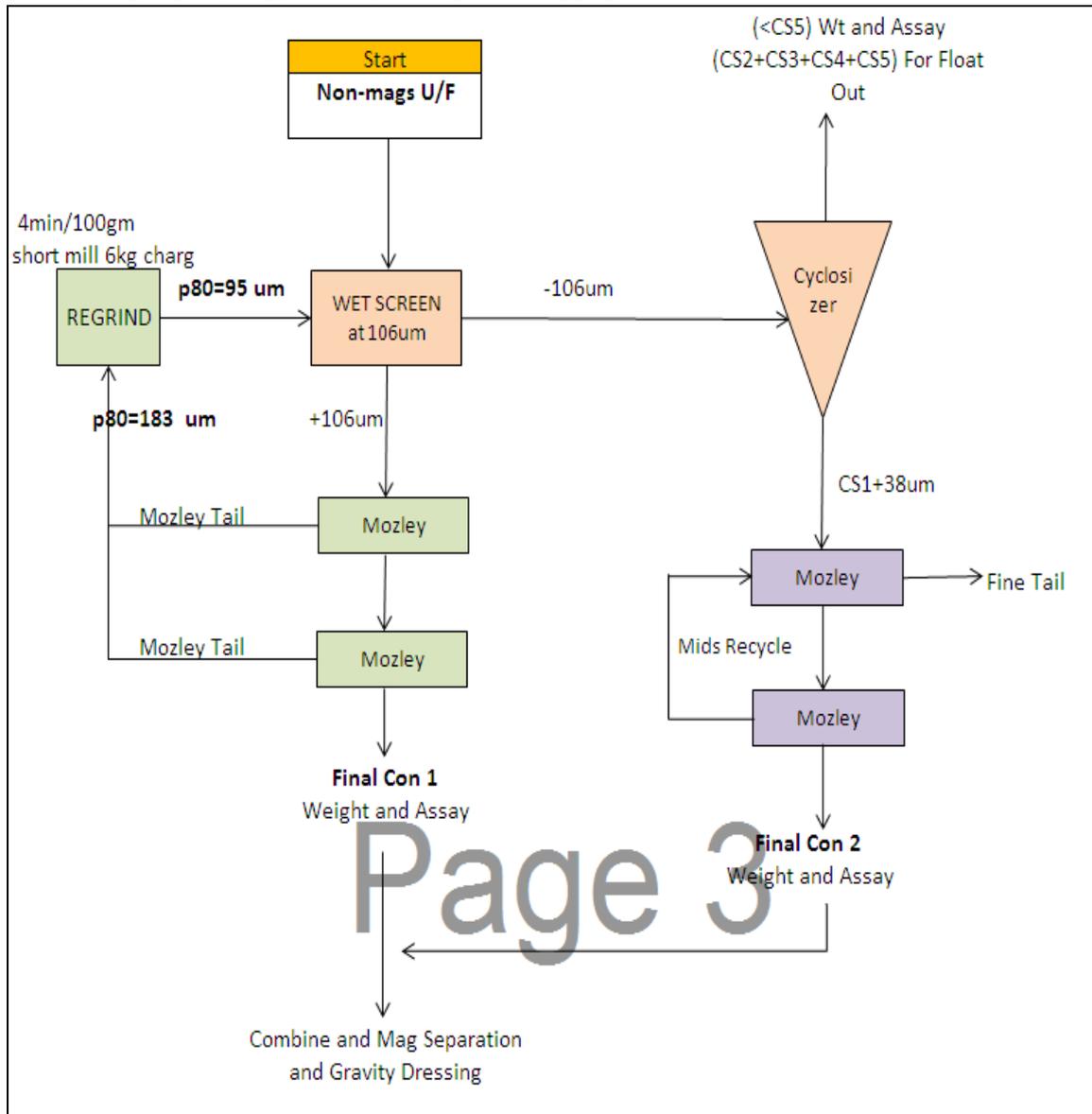
Results indicate that 69.0% of WO₃ was recovered at 5.42% grade. Some 13% of tungsten remained tailings.

GRAVITY LOCK CYCLE TEST

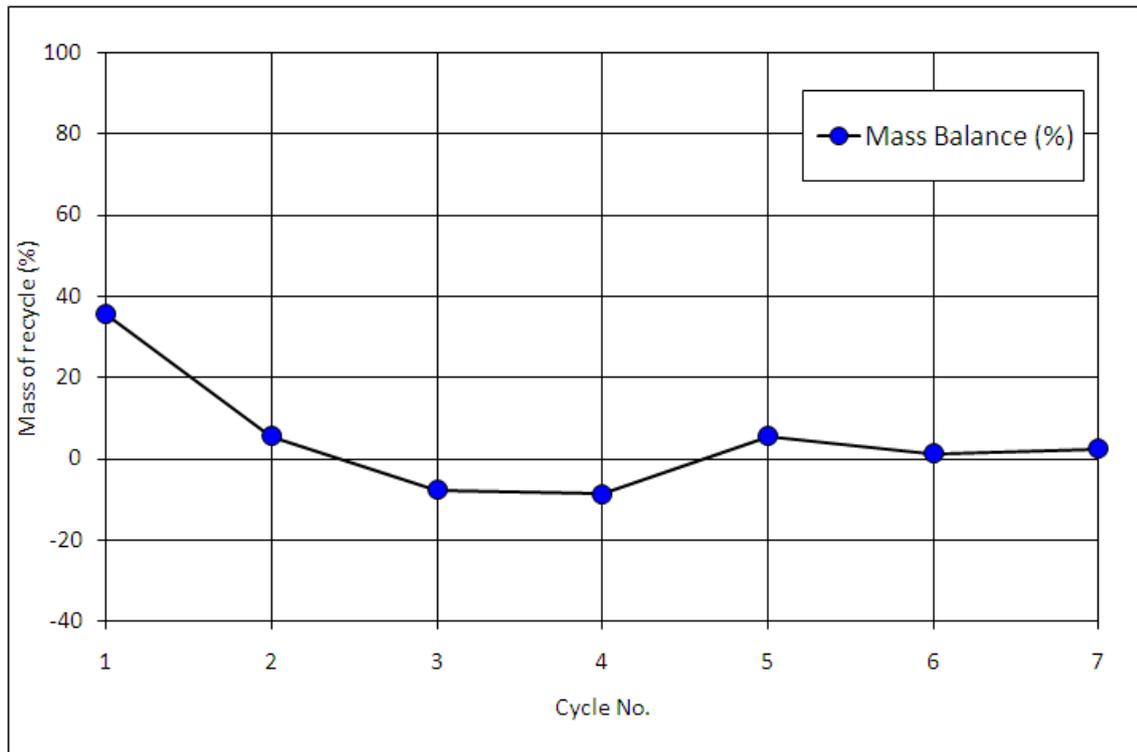
Seven cycles were performed in a gravity locked cycle test. The feed was from non mags underflow stream. The flow sheet was showed as diagram 1. Test results indicate 42.5% of WO₃ was recovered to concentrate at some 10% grade. The overall recovery of WO₃ was 24.6%.

3.9% of Sn was recovered to the concentrate at grade of some 0.8%. Some 89.8% of CaO was remained in the out stream of fine tail and CS2-CS5.

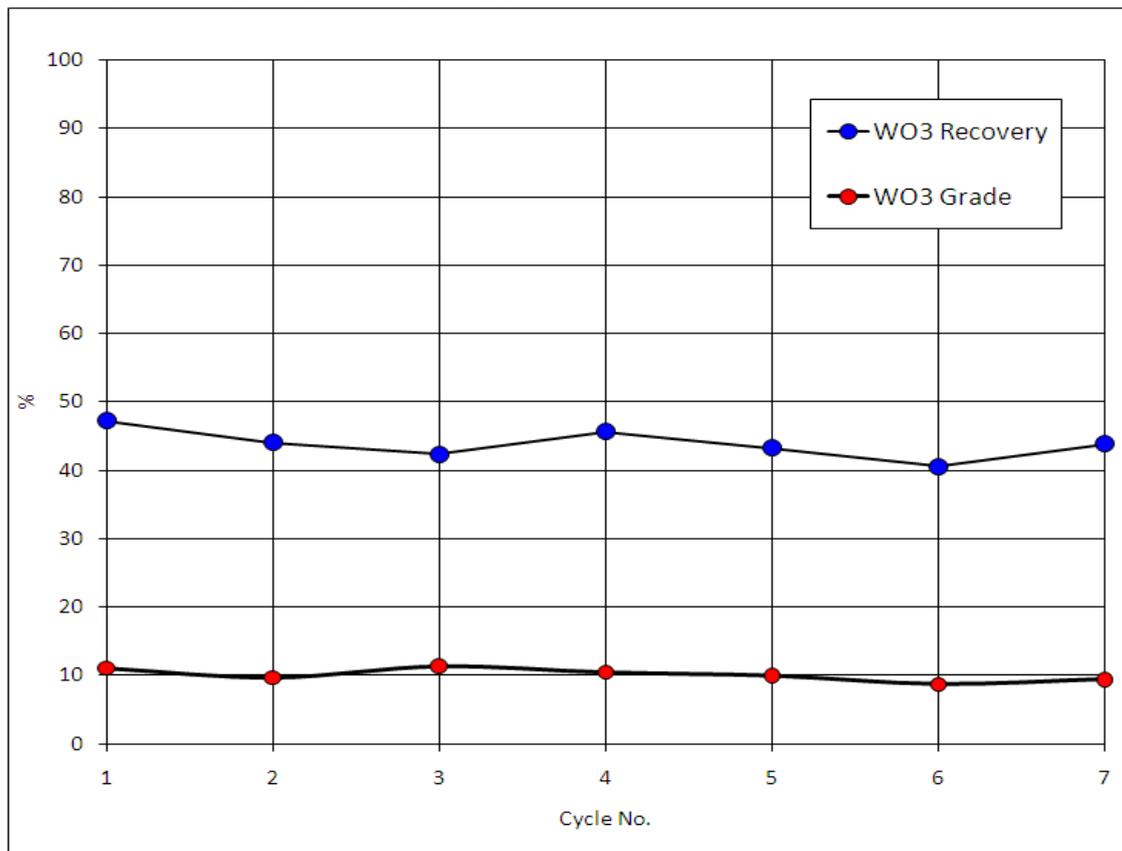
DIAGRAM 1: FLOWSHEET OF GRAVITY LOCK CYCLE TEST



GRAPH 1: MASS BALANCE OF GRAVITY LOCK CYCLE



GRAPH 2: WO3 RECOVERY AND GRADE



DAVIS TUBE TEST

The concentrates generated from the gravity lock cycle test were combined for magnetic dressing by Davis Tube as detailed in Table 7. The results indicate some 20.19% of magnetics were removed with some 11.6% loss of WO₃. The overall recovery of WO₃ is 21.7%.

TABLE 7: DAVIS TUBE TEST SUMMARY

PRODUCT				WO ₃		Iron	
NAME	Wt (gm)	Calc (gm)	Wt (%)	WO ₃ (%)	DIST (%)	Fe (%)	DIST (%)
Non Magnetics	8.3	8.3	79.81	11.06	88.4	18.51	64.7
Magnetics	2.1	2.1	20.19	5.73	11.6	40.00	35.3
TOTAL	10.40		100.00	9.98	100.0	22.85	100.0

SUPER PANNER TEST

A Super panner test was performed on non mags samples from Davis tube test to produce a small high-grade tungsten concentrate. Super panner test results are summarised in Tables 8 below. The results indicate that some 71.8% recovery was achieved from non mags feed at some 41.20% grade. The overall recovery of WO₃ was 15.6%. However, it should be noted the final WO₃ concentrate was only 1.6gm and a the limit for sensible testing.

TABLE 8: SUPER PANNER TEST SUMMARY

PRODUCT	WT (gm)	Wt (%)	WO ₃ (%)	DIST (%)
CONC	1.60	19.3	41.20	71.8
MIDS 1	2.46	29.6	9.06	24.3
MIDS 2	1.87	22.5	1.03	2.1
TAILS	2.37	28.6	0.69	1.8
TOTAL	8.30	100.0	11.06	100.0

TUNGSTEN FLOTATION

Tungsten flotation tests results are presented below, Table 9 summarises the results for the two rougher floats.

TABLE 9: TUNGSTEN FLOTATION SUMMARY

Test No.	CUM Wt	WT %	CaO %	Rec %	SiO2 %	Rec %	Fe %	Rec %	WO3 %	Rec %	Sn %	Rec %
T07	131.5	83.0	33.4	93.9	28.3	75.7	7.65	80.8	0.21	96.2	0.16	84.7
T08	258.8	53.8	38.8	69.9	24.6	41.9	6.77	46.2	0.25	74.2	0.15	48.5

Tungsten flotation results indicated the reagent scheme applied did not work well on the separation of tungsten and gangue minerals. The tungsten grade was poor (0.25%) and the selectivity was not good as well.

FLUORITE FLOTATION

Fluorite flotation tests results are presented below. Table 10 summarises the results for the four floats.

TABLE 10: FLUORITE FLOTATION SUMMARY

Test No.	WT	WT	CaO		SiO ₂		Fe	
	g	%	%	DIST	%	DIST	%	DIST
T09	95.6	21.2	61.3	44.3	7.40	4.9	2.27	5.7
T10	82.1	17.2	64.7	37.4	5.07	2.7	1.37	3.0
T12	36.8	7.7	70.2	18.3	2.84	0.7	1.08	1.0
T13	75.5	22.0	72.6	46.4	1.98	1.5	0.64	1.8

WO ₃		F		CaF ₂	
%	DIST	%	DIST	%	DIST
0.42	50.4	41.2	58.7	84.6	58.7
0.21	21.3	-	-	-	-
0.05	2.2	47.1	29.2	96.7	29.2
0.07	9.1	47.1	61.5	95.0	80.6

The feed of T09, 10, 12 were from the underflow product of T05 (non mags underflow).

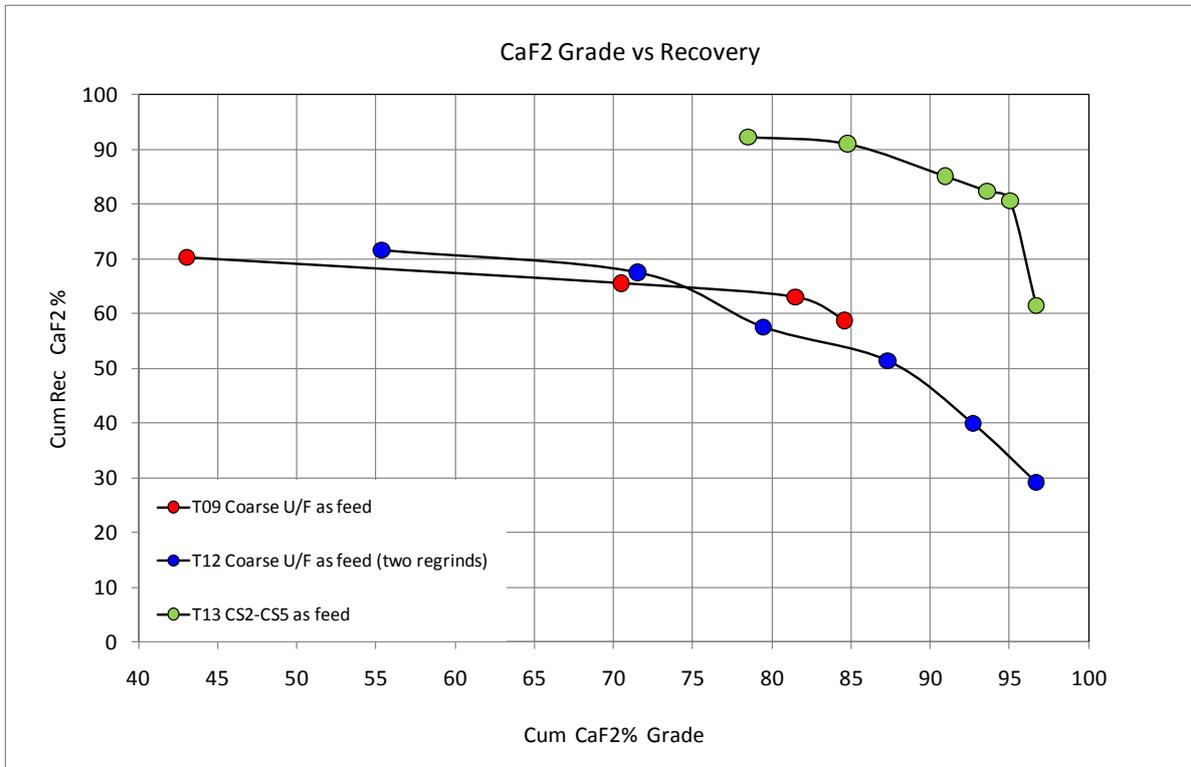
The slurry temperature of T09 in both rougher and cleaner floats was about 30 ° C and the pH was about 8. Reground on the rougher concentration for 2min/100gm was performed. The test results showed that 58.7% recovery at 84.6% grade was achieved by T09.

T10 was conducted by three rougher floats and three cleaner floats, which was as same as T09. Reground on rougher concentration was 2min/100gm as well. After regrinding, magnetic separation was performed on the reground sample. The last cleaner float was conducted at pH 10.8 and ambient temperature. The assay result of CaO indicated the recovery was slightly improved by 2.6%.

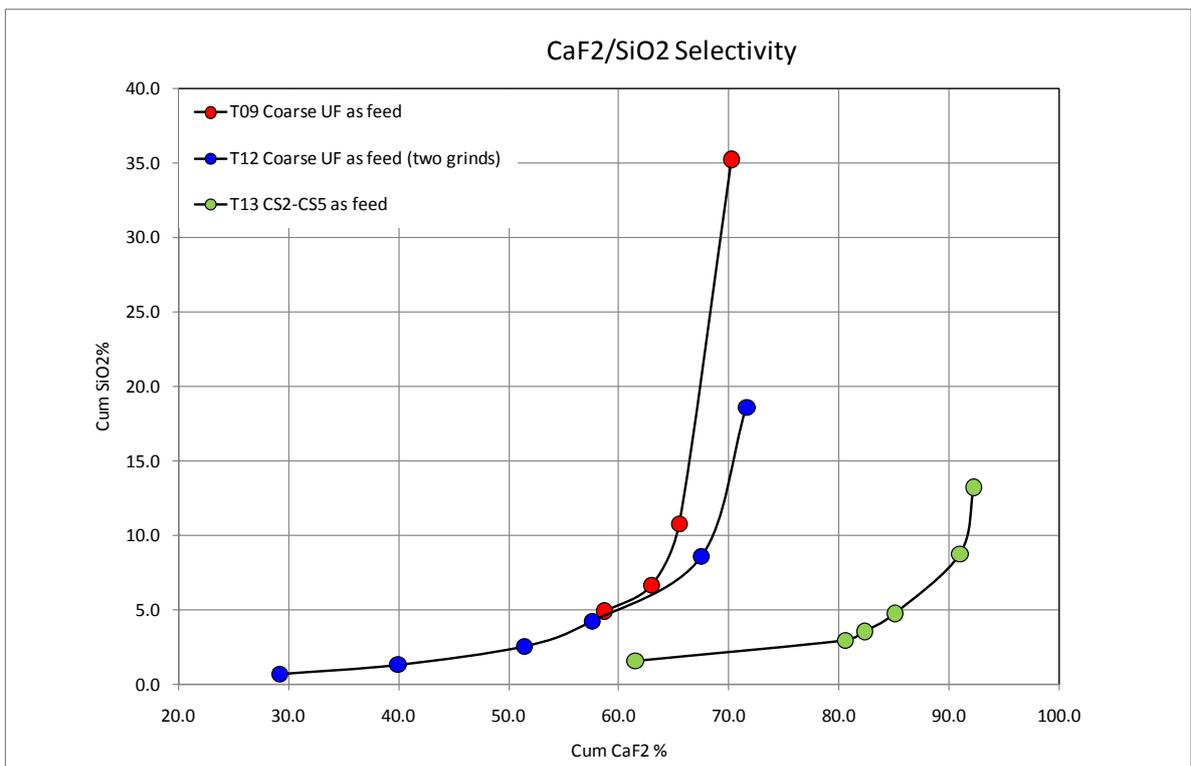
T12 was conducted by three rougher floats and five cleaner floats. Before rougher float, the feed was reground at 2min/100gm. Further regrinding was conducted on the rougher concentrate at 1min/100gm without further mags separation. Rougher 1-3 and Cleaner 1-2 were performed at about pH 8 and 30 ° C. Cleaner 3-5 was at pH 10.8 and ambient temperature. The test results showed that the grade of final concentrate was 96.7 with 29.2% of recovery.

The feed of T13 was from the out stream of CS2-CS5 of gravity lock cycle test. The float process was without further regrind because the feed sample was very fine (less than 20um). Three rougher floats and five cleaner floats were conducted . Rougher 1-3 and cleaner 1-2 were conducted at pH 8 and 30 ° C. Cleaner 3 and 4 were at pH 10.8 and ambient temperature. Cleaner 5 was at pH2.5 and ambient temperature. The test results showed that the grade of CaF₂ was 95.0% with 80.6% recovery in cleaner 4 concentrate. By cleaner 5, the final concentrate achieved 96.7% grade with 61.5% recovery.

GRAPH 3: FLUORITE FLOTATION CaF2 GRAD VS RECOVERY



GRAPH 4: FLUORITE FLOTATION CaF2/SiO2 SELECTIVITY



7. RECOMMENDATIONS

According to current test results and analysis, the following further tests for Moina ore are recommended:

- Magnetic separation – poor mags grade and loss of fluorine. Assess a flotation based cleaning regime to remove non magnetics.
- Perform “whole” of circuit test to demonstrate overall departments of WO₃ and F and define a circuit mass balance.
- Perform agglomeration testing of fluorite concentrate.
- Submit concentrates for detailed elemental analysis.
- Reagent, temperature and pH value optimization for fluorite flotation
- Perform variability assessment of defined ore “domains” or production periods. Use a simplified lab routine based on above outcomes.

8. APPENDICES

A1 BOND BALL MILL WORK

BOND BALL MILL CLOSED CIRCUIT GRINDABILITY :**75 MICROMETERS**

SAMPLE IDENTITY	COMPOSITE USC 01 TO 48
CLIENT	TNT MOINA ORE
PROJECT No	P0399
DATE	May-12

PERIOD	REVS OF MILL	WT OF 700 mls (g)	WT OF NEW FEED (g)	WT OF O/SIZE (g)*	WT OF U/SIZE (g)*	NET WT OF U/SIZE (g)*	NET WT OF U/SIZE PER REV (g)	CIRCUITING LOAD (%)	WT OF FRESH FEED ADDED TO NEXT CYCLE (g)	WT OF U/SIZE IN FEED TO NEXT CYCLE (g)
1	250	1309.5		1072.6	236.9	179.9	0.7197	453	236.9	10.3
2	507	1309.5	236.5	932.3	377.2	366.9	0.7237	247	377.2	16.4
3	495	1309.5	376.8	920.1	389.4	373.0	0.7534	236	389.4	16.9
4	475	1309.5	388.7	912.2	397.3	380.4	0.8008	230	397.3	17.3
5	446	1309.5	396.9	934.5	375.0	357.7	0.8019	249	375.0	16.3
6	447	1309.5	374.6	935.4	374.1	357.7	0.8003	250	374.1	16.3

Note : * = Ex grinding mill

PRODUCT IN THE FEED	4.35 (%)
BULK DENSITY	1.8707 (t/m ³)
IDEAL POTENTIAL PRODUCT	374.1 (g)
AVERAGE EQUILIBRIUM CIRC LOAD	250 (%)
AVERAGE PRODUCT	0.801 (g/rev)
80 % PASSING FEED SIZE	2743 (µm)
80 % PASSING PRODUCT SIZE	54 (µm)

BOND BALL MILL WORK INDEX (Kilowatt hours / dry tonne) :	18.6
--	------

BOND BALL MILL GRINDABILITY TEST FEED AND PRODUCT SIZINGS

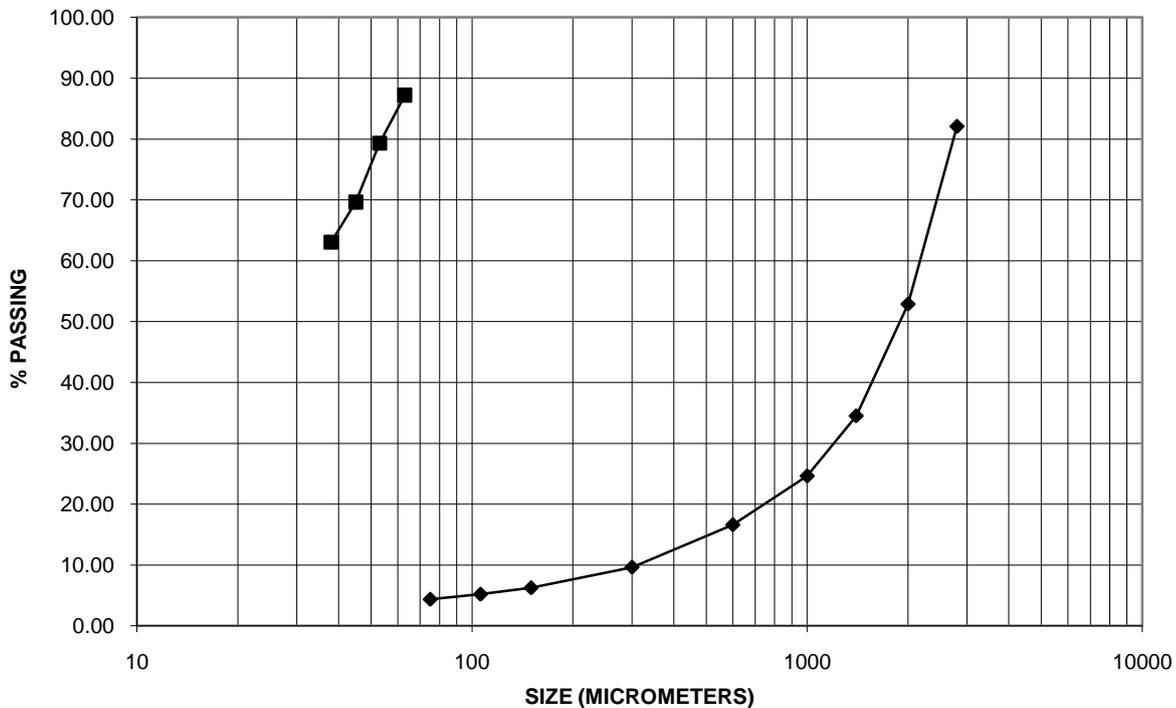
COMPOSITE USC 01 TO 48

P0399

FEED TO PERIOD No. 1			
Size (mm)	Weight (g)	Retained (%)	Passing (%)
3.350		0.00	100.00
2.800	179.7	17.93	82.07
2.000	292.8	29.21	52.87
1.400	184.1	18.37	34.50
1.000	98.9	9.87	24.63
0.600	80.5	8.03	16.60
0.300	69.8	6.96	9.64
0.150	34.1	3.40	6.24
0.106	10.5	1.05	5.19
0.075	8.4	0.84	4.35
-0.075	43.6	4.35	
TOTAL	1002.4	100.00	
F 80 (µm)		2743	

EQUILIBRIUM PRODUCTS			
Size (mm)	Weight (g)	Retained (%)	Passing (%)
0.075		0.00	100.00
0.063	47.9	12.78	87.22
0.053	29.6	7.90	79.32
0.045	36.2	9.66	69.66
0.038	24.7	6.59	63.07
-0.038	236.5	63.07	
TOTAL	375.0	100.00	
P 80 (µm)		54	

BOND BALL MILL GRINDABILITY : FEED & PRODUCT SIZE DISTRIBUTION



A2 HEAVY LIQUID TESTS



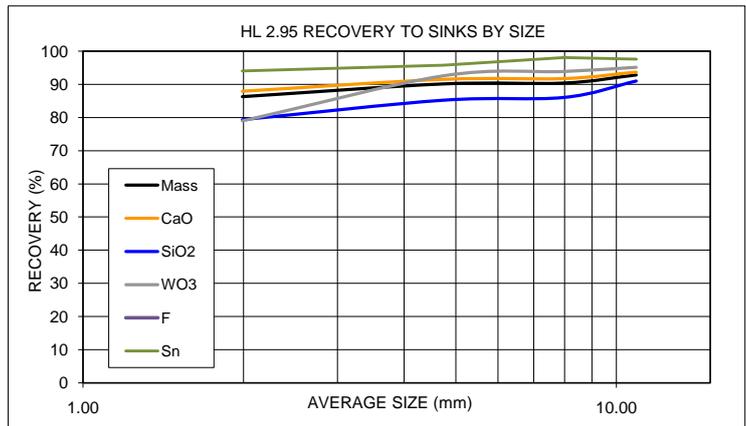
PROJECT	T0728
SAMPLE	TNT Moina ore
FROM TEST NO	T01
DATE	050512
TECHNICIAN	ID

CONDITIONS

LIQUID	hpt soln
MEASURED SG	2.95
TEMPERATURE	35
METHOD	beaker

SIZING

SIZE mm	WEIGHTS			
	gm	(%)	%PASS	p80
12.50	0.0	0.00	100.0	10.65
9.50	713.1	32.39	67.6	
6.70	413.0	18.76	48.9	
3.35	528.0	23.98	24.9	
1.18	309.1	14.04	10.8	
<1.18	238.7	10.84	0.0	
TOTAL	2201.9	100.00		



SEPARATIONS

SIZE mm	WT gm	WT %	Fraction									
			%CaO	dist	%SiO2	dist	%Fe	dist	%WO3	dist	%Sn	dist
Float+9.50	15.70	2.31	22.40	6.3	33.80	9.0	6.57	2.5	0.10	4.9	0.05	2.3
Sink+9.50	204.10	30.07	25.50	93.7	26.30	91.0	19.55	97.5	0.15	95.1	0.16	97.7
Calc 9.50 Total	219.80	32.39	25.28	100.0	26.84	100.0	18.62	100.0	0.15	100.0	0.15	100.0
Float+6.70	19.50	1.80	20.60	8.3	36.70	13.9	6.79	3.2	0.08	6.1	0.03	2.0
Sink+6.70	183.40	16.95	24.30	91.7	24.10	86.1	21.60	96.8	0.13	93.9	0.16	98.0
Calc 6.70 Total	202.90	18.76	23.94	100.0	25.31	100.0	20.18	100.0	0.13	100.0	0.15	100.0
Float+3.35	20.70	2.37	20.00	8.5	37.80	14.8	7.15	3.7	0.11	7.4	0.06	4.2
Sink+3.35	189.00	21.61	23.60	91.5	23.90	85.2	20.60	96.3	0.15	92.6	0.15	95.8
Calc 3.35 Total	209.70	23.98	23.24	100.0	25.27	100.0	19.27	100.0	0.15	100.0	0.14	100.0
Float+1.18	28.60	1.93	20.70	12.0	38.90	20.6	6.81	5.0	0.25	21.0	0.06	6.0
Sink+1.18	179.30	12.11	24.10	88.0	23.90	79.4	20.50	95.0	0.15	79.0	0.15	94.0
Calc 1.18 Total	207.90	14.04	23.63	100.0	25.96	100.0	18.62	100.0	0.16	100.0	0.14	100.0
<1.18 Fines	238.70	10.84	24.20		25.60		18.40		0.22		0.15	
CALC FEED	1079.0	100.00	24.19		25.92		19.05		0.15		0.15	

SUMMARY

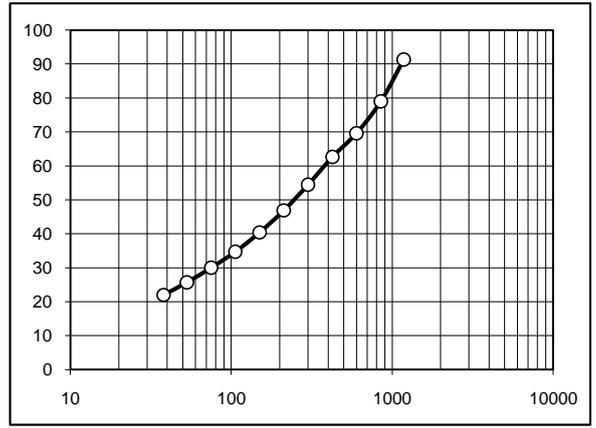
FRACTIONS mm	Picno SG	WT %	Overall									
			%CaO	dist	%SiO2	dist	%Fe	dist	%WO3	dist	%Sn	dist
Float+9.50	2.00	2.31	22.40	2.14	33.80	3.02	6.57	0.80	0.10	1.51	0.05	0.79
Float+6.70	2.00	1.80	20.60	1.53	36.70	2.55	6.79	0.64	0.08	0.94	0.03	0.37
Float +3.35	2.00	2.37	20.00	1.96	37.80	3.45	7.15	0.89	0.11	1.70	0.06	0.97
Float +1.18	2.00	1.93	20.70	1.65	38.90	2.90	6.81	0.69	0.25	3.16	0.06	0.79
Sink+9.50	4.00	30.07	25.50	31.70	26.30	30.52	19.55	30.87	0.15	29.53	0.16	32.88
Sink+6.70	4.00	16.95	24.30	17.03	24.10	15.76	21.60	19.23	0.13	14.43	0.16	18.53
Sink +3.35	4.00	21.61	23.60	21.08	23.90	19.93	20.60	23.38	0.15	21.22	0.15	22.15
Sink +1.18	4.00	12.11	24.10	12.06	23.90	11.16	20.50	13.03	0.15	11.89	0.15	12.41
Total Floats	2.00	8.41	20.95	7.29	36.72	11.92	6.84	3.02	0.13	7.32	0.05	2.92
Total Sinks	4.00	80.75	24.53	81.87	24.84	77.37	20.40	86.51	0.15	77.06	0.16	85.97
Total Fines	3.50	10.84	24.20	10.84	25.60	10.71	18.40	10.47	0.22	15.61	0.15	11.11
CALC FEED	3.64	100.00	24.19	100.00	25.92	100.00	19.05	100.00	0.15	100.00	0.15	100.00

SIZE (av) mm	SIZE mm	Wt Rec	CaO Rec	SiO2 Rec	Fe Rec	WO3 Rec	Sn Rec
10.90	9.50	92.86	93.7	91.0	97.48	95.1	97.7
7.98	6.70	90.39	91.7	86.1	96.77	93.9	98.0
4.74	3.35	90.13	91.5	85.24	96.34	92.6	95.8
1.99	1.18	86.24	88.0	79.4	94.97	79.0	94.0

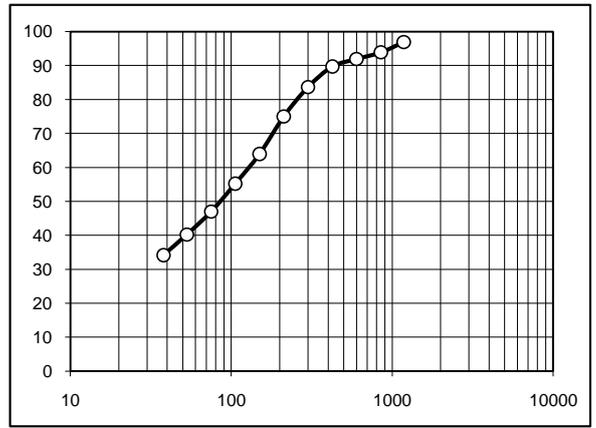
A3 GRIND ESTABLISHMENT

PROJECT	T0728
ORE	Moina
DATE	150512
TECHNICIAN	SP

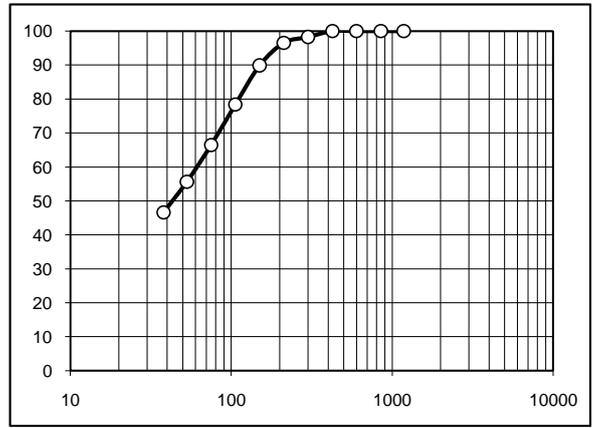
Moina		SIZE um	WEIGHTS			SIZE p80
G1			gm	(%)	%PASS	
Feed Size mm	1.700	1180	10.31	8.67	91.3	877
		850	14.67	12.34	79.0	
Mill Length	Short	600	11.14	9.37	69.6	
Media Type	Balls	425	8.31	6.99	62.6	
Media kg	12.00	300	9.71	8.17	54.5	
Solids kg	1.00	212	9.00	7.57	46.9	
Water kg	0.50	150	7.72	6.49	40.4	
Speed rpm	50	106	6.77	5.69	34.7	
TIME min	10	75	5.58	4.69	30.0	
		53	5.13	4.32	25.7	
		38	4.40	3.70	22.0	
SUB			26.14	21.99		
TOTAL			118.88	100.00		



Moina		SIZE um	WEIGHTS			SIZE p80
G2			gm	(%)	%PASS	
Feed Size mm	1.700	1180	3.43	3.09	96.9	263
		850	3.37	3.04	93.9	
Mill Length	Short	600	2.19	1.97	91.9	
Media Type	Balls	425	2.43	2.19	89.7	
Media kg	12.00	300	6.71	6.05	83.7	
Solids kg	1.00	212	9.61	8.66	75.0	
Water kg	0.50	150	12.24	11.03	64.0	
Speed rpm	50	106	9.71	8.75	55.2	
TIME min	20	75	9.15	8.25	47.0	
		53	7.50	6.76	40.2	
		38	6.70	6.04	34.2	
SUB			37.91	34.17		
TOTAL			110.95	100.00		



Moina		SIZE um	WEIGHTS			SIZE p80
G3			gm	(%)	%PASS	
Feed Size mm	1.700	1180	0.00	0.00	100.0	112
		850	0.00	0.00	100.0	
Mill Length	Short	600	0.00	0.00	100.0	
Media Type	Balls	425	0.00	0.00	100.0	
Media kg	12.00	300	2.01	1.74	98.3	
Solids kg	1.00	212	2.00	1.73	96.5	
Water kg	0.50	150	7.69	6.65	89.9	
Speed rpm	50	106	13.32	11.51	78.4	
TIME min	30	75	13.79	11.92	66.5	
		53	12.55	10.85	55.6	
		38	10.41	9.00	46.6	
SUB			53.94	46.62		
TOTAL			115.71	100.00		



PROJECT	T0728
ORE	Moina
DATE	150512
TECHNICIAN	SP

GRIND CONDITIONS

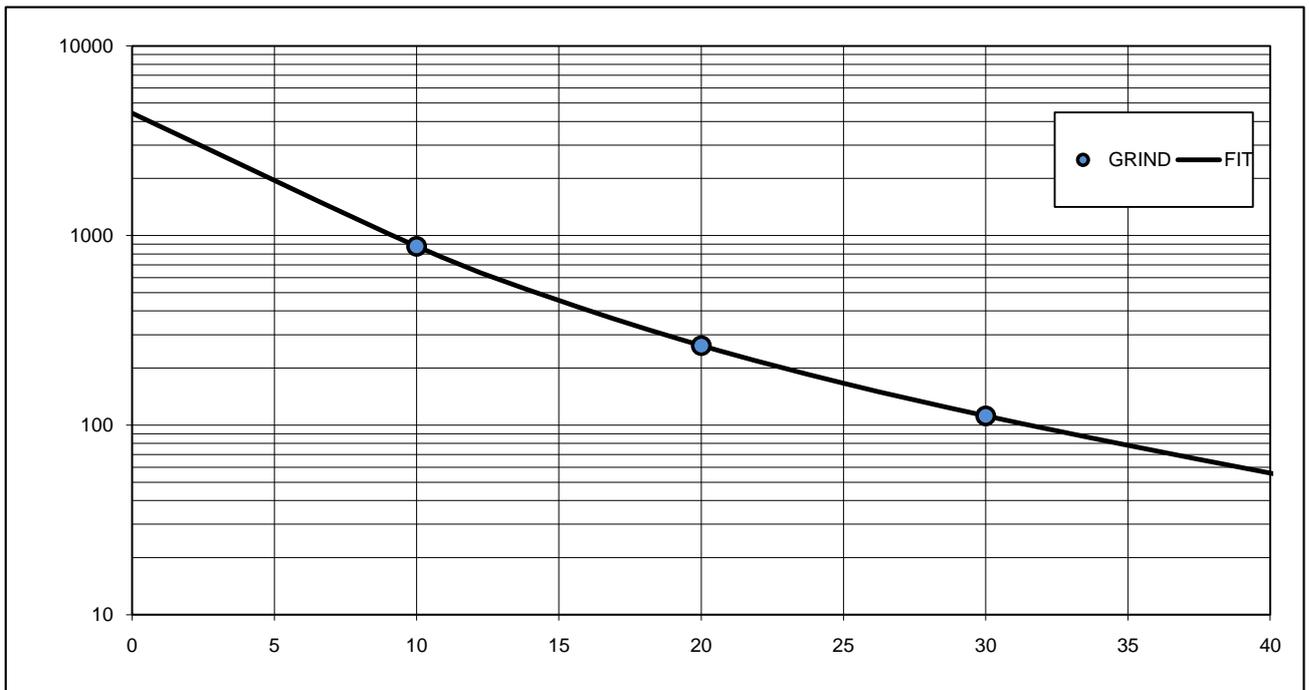
ORE	Moina
FEED mm	1.700
TYPE	Short
MEDIA	Balls
MEDIA kg	12.00
SOLIDS kg	1.00
WATER kg	0.50

GRIND DATA

GRIND	GRIND (min)	GRIND p80	FIT p80	DIFF
NIL	10	877	877	6E-07
G1	20	263	263	1E-05
G2	30	112	112	1E-05
G3				
G4				
G5				
G6				
G7				
			SUM	2E-05

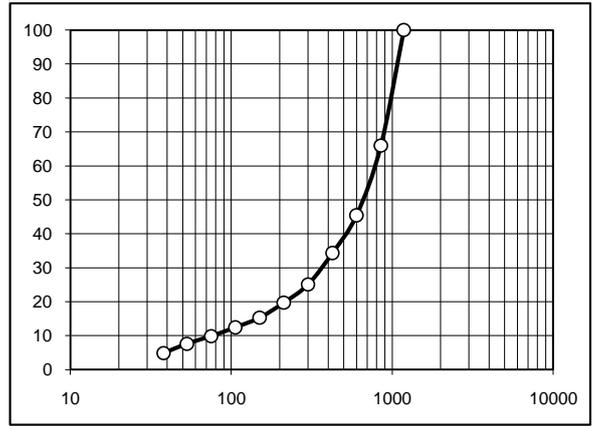
START VALUES	FIT
1000	3744.2
0.12	0.20
200	666.7
0.02	0.06

EST (min)	EST p80
30.7	106

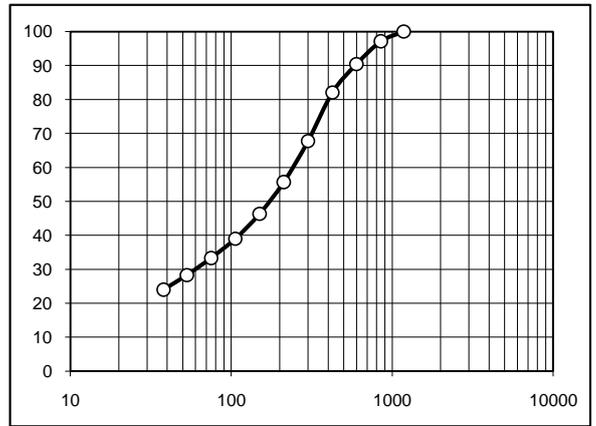


PROJECT	T0728
ORE	Moina
DATE	160512
TECHNICIAN	SP

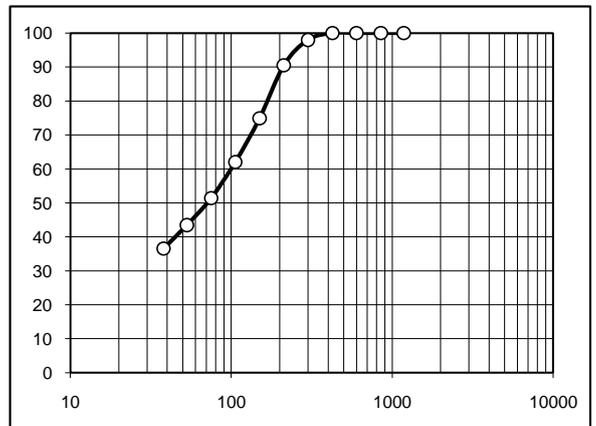
Moina		SIZE um	WEIGHTS			SIZE p80
G0			gm	(%)	%PASS	
Feed Size mm	1.180	1180	0.00	0.00	100.0	986
		850	35.14	34.03	66.0	
Mill Length	Short	600	21.20	20.53	45.4	
Media Type	Balls	425	11.45	11.09	34.4	
Media kg	6.00	300	9.58	9.28	25.1	
Solids kg	1.00	212	5.54	5.37	19.7	
Water kg	0.50	150	4.56	4.42	15.3	
Speed rpm	50	106	2.94	2.85	12.4	
TIME min	10	75	2.65	2.57	9.9	
		53	2.33	2.26	7.6	
		38	2.85	2.76	4.9	
		SUB	5.02	4.86		
		TOTAL	103.26	100.00		



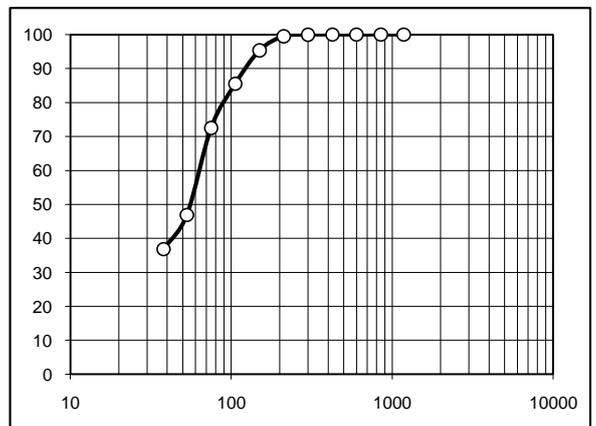
Moina		SIZE um	WEIGHTS			SIZE p80
G1			gm	(%)	%PASS	
Feed Size mm	1.180	1180	0.00	0.00	100.0	407
		850	3.17	2.88	97.1	
Mill Length	Short	600	7.38	6.71	90.4	
Media Type	Balls	425	9.18	8.35	82.1	
Media kg	6.00	300	15.74	14.31	67.8	
Solids kg	1.00	212	13.28	12.07	55.7	
Water kg	0.50	150	10.28	9.35	46.3	
Speed rpm	50	106	8.07	7.34	39.0	
TIME min	20	75	6.24	5.67	33.3	
		53	5.56	5.06	28.3	
		38	4.67	4.25	24.0	
		SUB	26.42	24.02		
		TOTAL	109.99	100.00		



Moina		SIZE um	WEIGHTS			SIZE p80
G2			gm	(%)	%PASS	
Feed Size mm	1.180	1180	0.00	0.00	100.0	170
		850	0.00	0.00	100.0	
Mill Length	Short	600	0.00	0.00	100.0	
Media Type	Balls	425	0.00	0.00	100.0	
Media kg	6.00	300	2.38	2.01	98.0	
Solids kg	1.00	212	8.87	7.49	90.5	
Water kg	0.50	150	18.45	15.59	74.9	
Speed rpm	50	106	15.24	12.88	62.0	
TIME min	20	75	12.58	10.63	51.4	
		53	9.41	7.95	43.4	
		38	8.16	6.89	36.6	
		SUB	43.26	36.55		
		TOTAL	118.35	100.00		



Moina		SIZE um	WEIGHTS			SIZE p80
G3			gm	(%)	%PASS	
Feed Size mm	1.180	1180	0.00	0.00	100.0	93
		850	0.00	0.00	100.0	
Mill Length	Short	600	0.00	0.00	100.0	
Media Type	Balls	425	0.00	0.00	100.0	
Media kg	6.00	300	0.05	0.04	100.0	
Solids kg	1.00	212	0.55	0.48	99.5	
Water kg	0.50	150	4.76	4.11	95.4	
Speed rpm	50	106	11.39	9.84	85.5	
TIME min	30	75	15.04	13.00	72.5	
		53	29.67	25.64	46.9	
		38	11.64	10.06	36.8	
		SUB	42.63	36.84		
		TOTAL	115.73	100.00		



PROJECT	T0728
ORE	Moina
DATE	150512
TECHNICIAN	SP

GRIND CONDITIONS

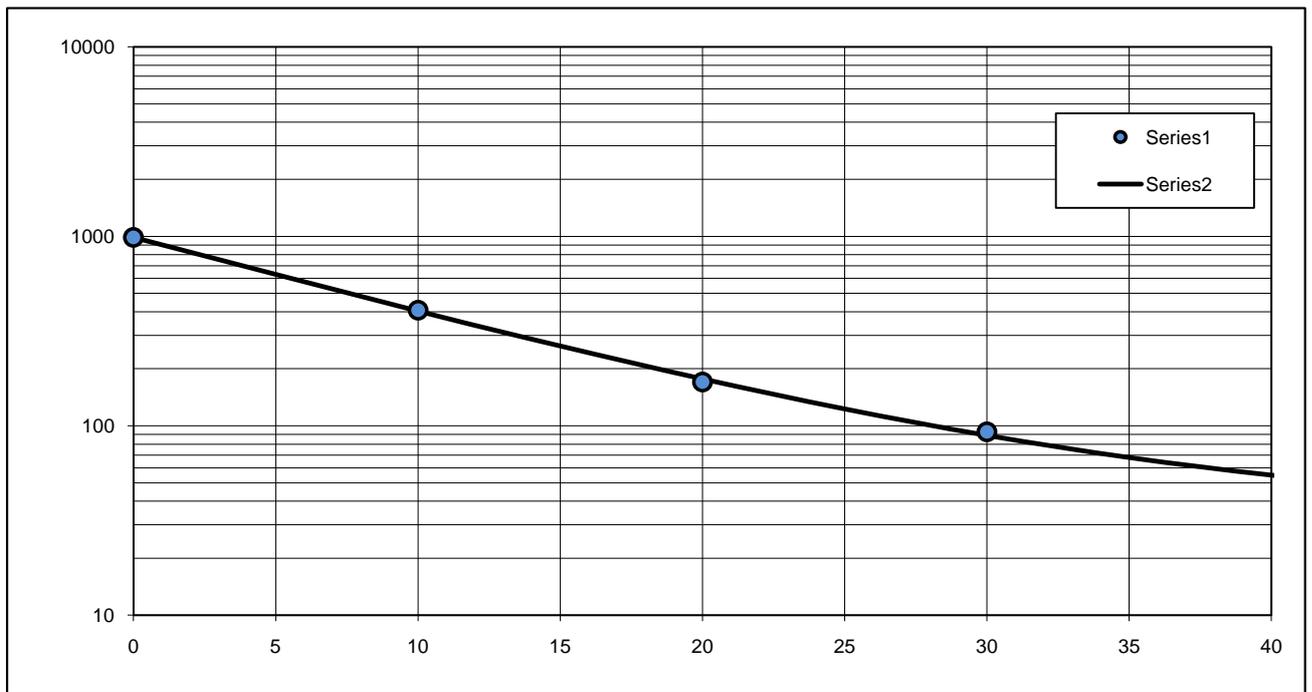
ORE	Moina
FEED mm	1.180
TYPE	Short
MEDIA	Balls
MEDIA kg	6.00
SOLIDS kg	1.00
WATER kg	0.50

GRIND DATA

GRIND	GRIND (min)	GRIND p80	FIT p80	DIFF
NIL	0	986	987	3E-01
G1	10	407	403	1E+01
G2	20	170	177	5E+01
G3	30	93	89	2E+01
G4				
G5				
G6				
G7				
			SUM	8E+01

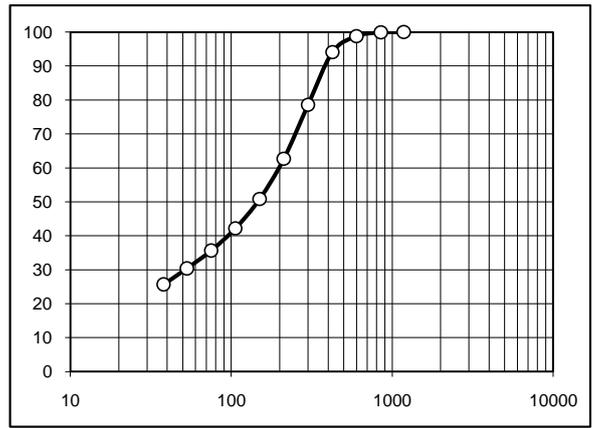
START VALUES	FIT
1000	953.3
0.12	0.09
200	33.3
0.02	0.00

EST (min)	EST p80
27.2	106

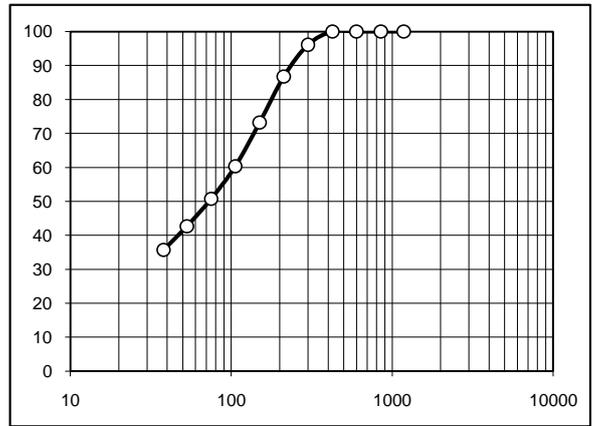


PROJECT	T0728
ORE	Moina Comp
DATE	10612
TECHNICIAN	SP

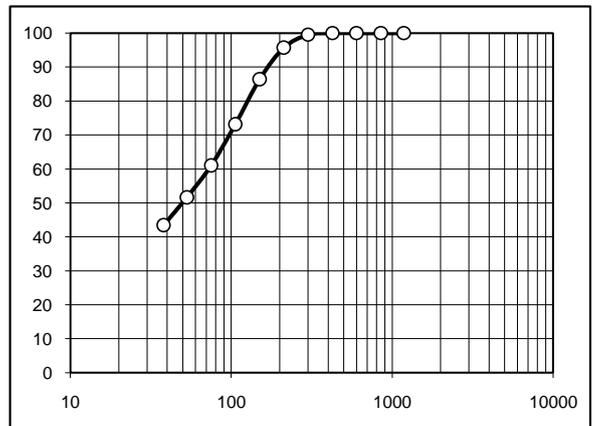
Moina Comp		SIZE um	WEIGHTS			SIZE p80
G0			gm	(%)	%PASS	
Feed Size mm	1.180	1180	0.00	0.00	100.0	311
		850	0.15	0.09	99.9	
Mill Length	PIG	600	1.98	1.13	98.8	
Media Type	Balls	425	8.32	4.73	94.1	
Media kg	40.0	300	27.22	15.47	78.6	
Solids kg	8.00	212	28.02	15.92	62.7	
Water kg	4.00	150	20.84	11.84	50.8	
Speed rpm	50	106	15.19	8.63	42.2	
TIME min	10	75	11.51	6.54	35.7	
		53	9.23	5.25	30.4	
		38	8.33	4.73	25.7	
		SUB	45.18	25.67		
		TOTAL	175.97	100.00		



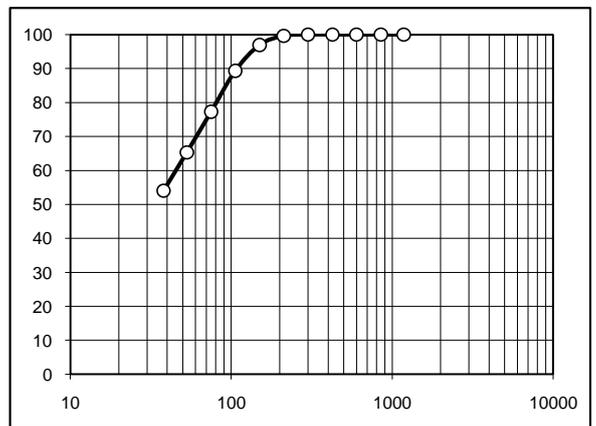
Moina Comp		SIZE um	WEIGHTS			SIZE p80
G1			gm	(%)	%PASS	
Feed Size mm	1.180	1180	0.00	0.00	100.0	181
		850	0.00	0.00	100.0	
Mill Length	PIG	600	0.00	0.00	100.0	
Media Type	Balls	425	0.00	0.00	100.0	
Media kg	40.00	300	4.81	3.88	96.1	
Solids kg	8.00	212	11.62	9.37	86.7	
Water kg	4.00	150	16.79	13.55	73.2	
Speed rpm	50	106	15.93	12.85	60.3	
TIME min	15	75	11.88	9.58	50.8	
		53	10.03	8.09	42.7	
		38	8.61	6.95	35.7	
		SUB	44.28	35.72		
		TOTAL	123.95	100.00		



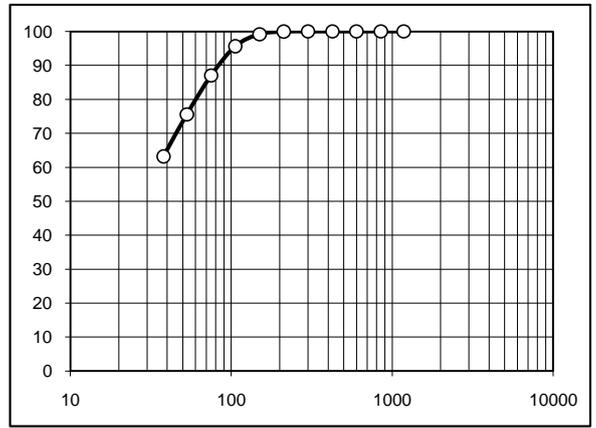
Moina Comp		SIZE um	WEIGHTS			SIZE p80
G2			gm	(%)	%PASS	
Feed Size mm	1.180	1180	0.00	0.00	100.0	129
		850	0.00	0.00	100.0	
Mill Length	PIG	600	0.00	0.00	100.0	
Media Type	Balls	425	0.00	0.00	100.0	
Media kg	40.00	300	0.84	0.49	99.5	
Solids kg	8.00	212	6.54	3.80	95.7	
Water kg	4.00	150	15.97	9.29	86.4	
Speed rpm	50	106	22.81	13.27	73.2	
TIME min	20	75	20.85	12.13	61.0	
		53	16.12	9.37	51.7	
		38	14.10	8.20	43.5	
		SUB	74.72	43.45		
		TOTAL	171.95	100.00		



Moina Comp		SIZE um	WEIGHTS			SIZE p80
G3			gm	(%)	%PASS	
Feed Size mm	1.180	1180	0.00	0.00	100.0	82
		850	0.00	0.00	100.0	
Mill Length	PIG	600	0.00	0.00	100.0	
Media Type	Balls	425	0.00	0.00	100.0	
Media kg	40.00	300	0.03	0.02	100.0	
Solids kg	8.00	212	0.56	0.33	99.6	
Water kg	4.00	150	4.54	2.71	96.9	
Speed rpm	50	106	12.69	7.58	89.4	
TIME min	30	75	20.17	12.05	77.3	
		53	20.05	11.98	65.3	
		38	18.88	11.28	54.0	
		SUB	90.44	54.04		
		TOTAL	167.36	100.00		



Moina Comp		SIZE um	WEIGHTS			SIZE p80
G5			gm	(%)	%PASS	
Feed Size mm	1.180	1180	0.00	0.00	100.0	62
		850	0.00	0.00	100.0	
Mill Length	PIG	600	0.00	0.00	100.0	
Media Type	Balls	425	0.00	0.00	100.0	
Media kg	40.00	300	0.03	0.01	100.0	
Solids kg	8.00	212	0.09	0.04	99.9	
Water kg	4.00	150	1.52	0.76	99.2	
Speed rpm	50	106	7.11	3.55	95.6	
TIME min	40	75	17.28	8.62	87.0	
		53	22.91	11.43	75.6	
		38	24.76	12.36	63.2	
		SUB	126.66	63.22		
		TOTAL	200.36	100.00		



PROJECT	T0728
ORE	Moina
DATE	10612
TECHNICIAN	SP

GRIND CONDITIONS

ORE	Moina Comp
FEED mm	1.180
TYPE	PIG
MEDIA	Balls
MEDIA kg	40.00
SOLIDS kg	8.00
WATER kg	4.00

INSTRUCTIONS

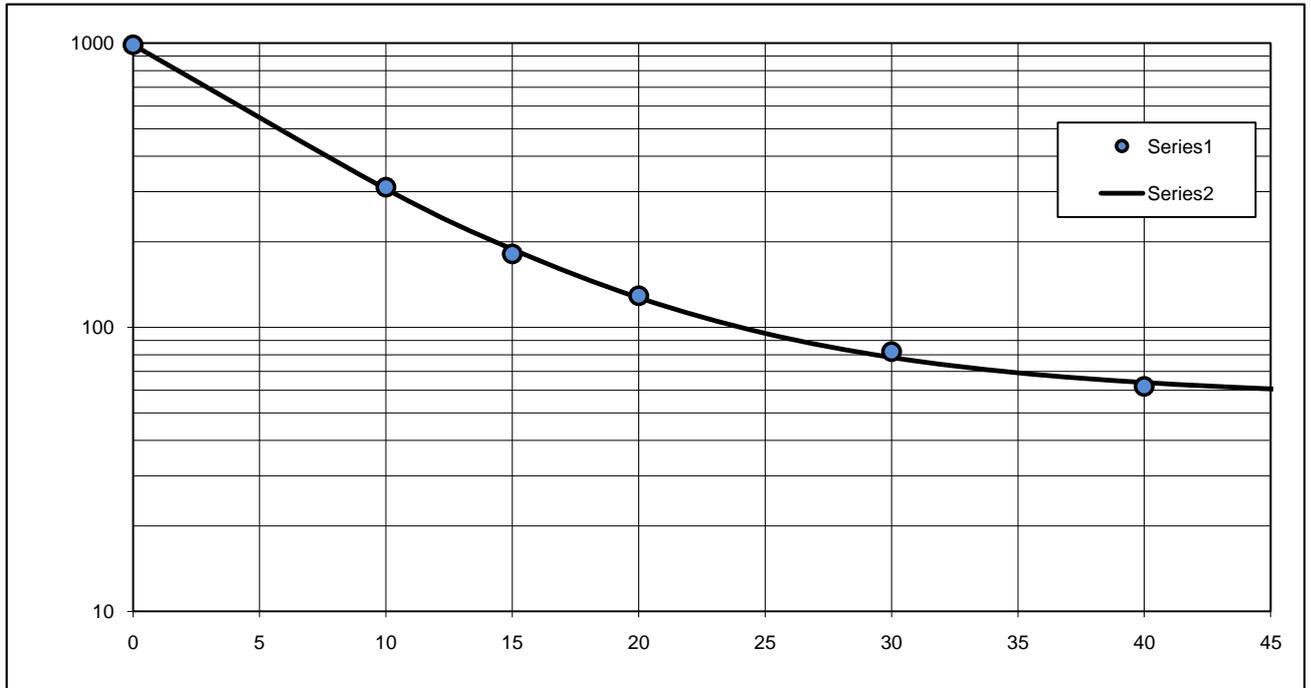
FIRST GRIND IS ALWAYS MINIMUM OF 8 MINUTES
 SECOND GRIND RUN FOR FIVE MIN THEN ADD BALLS AND START TIMING
 ENTER THE GRIND TIMES AND P80S IN TIME ORDER
 USE TOOLS-SOLVER-SOLVE TO DETERMINE GRIND CURVE
 SET TARGET CELL: W24 SET TO MIN USING: X19:X22

GRIND DATA

GRIND	GRIND (min)	GRIND p80	FIT p80	DIFF
NIL	0	986	986	7E-02
G1	10	311	307	2E+01
G2	15	181	188	5E+01
G3	20	129	127	3E+00
G4	30	82	78	1E+01
G5	40	62	64	4E+00
G6				
G7				
			SUM	9E+01

START VALUES	FIT
1000	917.2
0.12	0.13
200	69.1
0.02	0.00

EST (min)	EST p80
23	106



A4 MAGNETIC SEPARATIONS

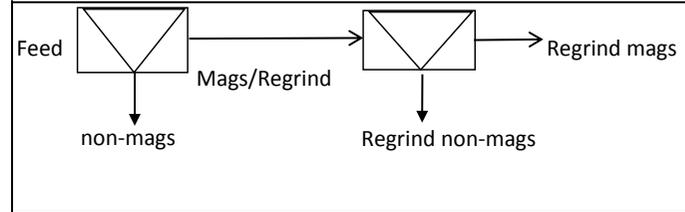


Ammtec Burnie

SALA MAG SEPARATION REPORT SHEET

PROJECT	T0728
TEST NO	T02
DATE	060612
TECH	ID

TEST TYPE	SALA Drum Magnet
Solids (g)	2000.0
Water (L)	25.0
Flow Rate (Lpm)	2.0
Density (%)	8.0
Time (min)	15.0
Pump 01 Output (%)	0.0
Pump 02 Output (%)	0.0
Comments	Gravity Feed



Regrind Details:	
Time (min)	7.5
Media (kg)	6
Solids (g)	839.5

START MATERIAL	
Moina ore	
P80 = 106um	
START WEIGHT (gm)	2000
FROM TEST NO	

MAGNETIC SEPARATION RESULTS

PRODUCT				CaO		Iron		Silica		WO3		ISE F		Sn	
NAME	Wt (gm)	Calc (gm)	Wt (%)	CaO (%)	DIST (%)	Fe (%)	DIST (%)	SiO2 (%)	DIST (%)	WO3 (%)	DIST (%)	ISE F (%)	DIST (%)	Sn (%)	DIST (%)
Non Magnetics	1127.7	1127.7	57.33	29.40	71.4	8.02	22.7	30.60	69.7	0.19	75.6	10.40	70.6	0.18	61.7
Re Grind Non Magnetics	114.3	114.3	5.81	29.30	7.2	9.46	2.7	28.40	6.6	0.16	6.5	15.50	10.7	0.15	5.2
Re Grind Magnetics	725.2	725.2	36.86	13.65	21.3	40.90	74.6	16.25	23.8	0.07	17.9	4.30	18.8	0.15	33.1
TOTAL	1967.20		100.00	23.59	100.0	20.22	100.0	25.18	100.0	0.14	100.0	8.45	100.0	0.17	100.0

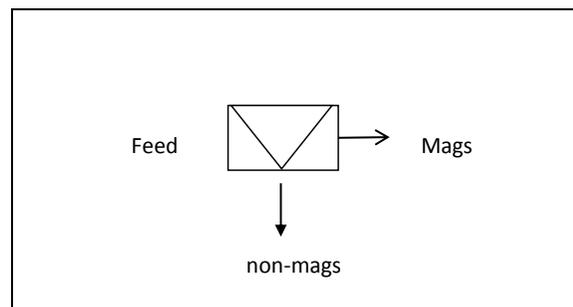


Ammtec Burnie

SALA MAG SEPARATION REPORT SHEET

PROJECT	T0728
TEST NO	T03
DATE	060612
TECH	ID

TEST TYPE	SALA Drum Magnet
Solids (g)	2000.0
Water (L)	25.0
Flow Rate (Lpm)	2.0
Density (%)	8.0
Time (min)	15.0
Pump 01 Output (%)	0.0
Pump 02 Output (%)	0.0
Comments	Gravity Feed



START MATERIAL	
Moina ore	
P80 = 106um	
START WEIGHT (gm)	2000
FROM TEST NO	

MAGNETIC SEPARATION RESULTS

PRODUCT				CaO		Iron		Silica		WO3		ISE F		Sn	
NAME	Wt (gm)	Calc (gm)	Wt (%)	CaO (%)	DIST (%)	Fe (%)	DIST (%)	SiO2 (%)	DIST (%)	W03 (%)	DIST (%)	ISE F (%)	DIST (%)	Sn (%)	DIST (%)
Non Magnetics	1165.3	1165.3	58.34	28.60	71.2	8.00	24.6	32.00	71.0	0.18	75.9	12.40	68.2	0.15	60.0
Magnetics	832.3	832.3	41.66	16.20	28.8	34.30	75.4	18.30	29.0	0.08	24.1	8.10	31.8	0.14	40.0
TOTAL	1997.60		100.00	23.43	100.0	18.96	100.0	26.29	100.0	0.14	100.0	10.61	100.0	0.15	100.0

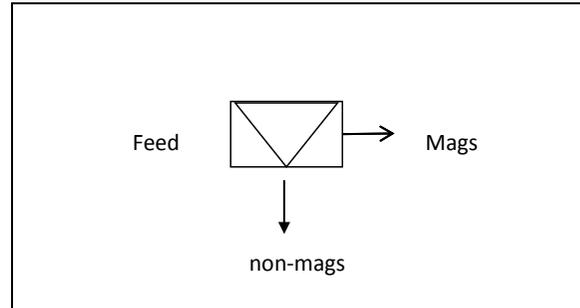


Ammtec Burnie

SALA MAG SEPARATION REPORT SHEET

PROJECT	T0728
TEST NO	T04
DATE	060612
TECH	ID

TEST TYPE	SALA Drum Magnet
Solids (g)	1000.0
Water (L)	20.0
Flow Rate (Lpm)	2.0
Density (%)	8.0
Time (min)	10.0
Pump 01 Output (%)	0.0
Pump 02 Output (%)	0.0
Comments	Gravity Feed



START MATERIAL	
Moina ore	
P80 = 200um	
START WEIGHT (gm)	2000
FROM TEST NO	

MAGNETIC SEPARATION RESULTS

PRODUCT			CaO		Iron		Silica		WO3		ISE F		Sn		
NAME	Wt (gm)	Calc (gm)	Wt (%)	CaO (%)	DIST (%)	Fe (%)	DIST (%)	SiO2 (%)	DIST (%)	WO3 (%)	DIST (%)	ISE F (%)	DIST (%)	Sn (%)	DIST (%)
Non Magnetics	560.3	560.3	55.54	29.20	67.9	8.14	23.0	30.30	66.7	0.19	74.8	13.00	76.1	0.15	55.5
Magnetics	448.5	448.5	44.46	17.25	32.1	34.10	77.0	18.90	33.3	0.08	25.2	5.10	23.9	0.15	44.5
TOTAL	1008.80		100.00	23.89	100.0	19.68	100.0	25.23	100.0	0.14	100.0	9.49	100.0	0.15	100.0

A5 MAGNETIC SEPARATION AND CYCLONE CLASSIFICATION



Ammtec Burnie

LAB GENERAL TEST DATA

SAMPLE	Moina ore, p80=122
TEST TYPE	Mag sep ,Cyclone

PROJECT	T0728
TEST NO	T05
DATE	120612
PERFORMED	ID

Grind big mill (pig) 8Kg P80 =122						Total
			Mags			3689.9
			Non Mags feed for cyclone			
Run 1						
Vortex	Spigot	KPA				
8	9	200				
U/F	Damp cake mass		3471.0			
Tare	Net wet	Gross Dry	MC %		Calc dry	3113.5
12.8	151.4	148.6	10.3			
O/F	Damp cake mass		1349.8			
Tare	Net wet	Gross Dry	MC %		Calc dry	1005.6
13.0	186.3	151.8	25.5			

MAGNETIC SEPARATION - CYCLONE RESULTS

PRODUCT				CaO		Iron		Silica		WO3		Sn	
NAME	Wt (gm)	Calc (gm)	Wt (%)	CaO (%)	DIST (%)	Fe (%)	DIST (%)	SiO2 (%)	DIST (%)	WO3 (%)	DIST (%)	Sn (%)	DIST (%)
Magnetics	3689.9	3689.9	47.25	16.70	35.0	33.50	79.0	18.80	35.1	0.09	30.9	0.16	43.6
OF	1005.6	1005.6	12.88	27.90	15.9	7.85	5.0	28.10	14.3	0.12	11.2	0.14	10.4
UF	3113.5	3113.5	39.87	27.80	49.1	8.00	15.9	32.10	50.6	0.20	57.9	0.20	46.0
TOTAL	7809.00		100.00	22.57	100.0	20.03	100.0	25.30	100.0	0.14	100.0	0.17	100.0

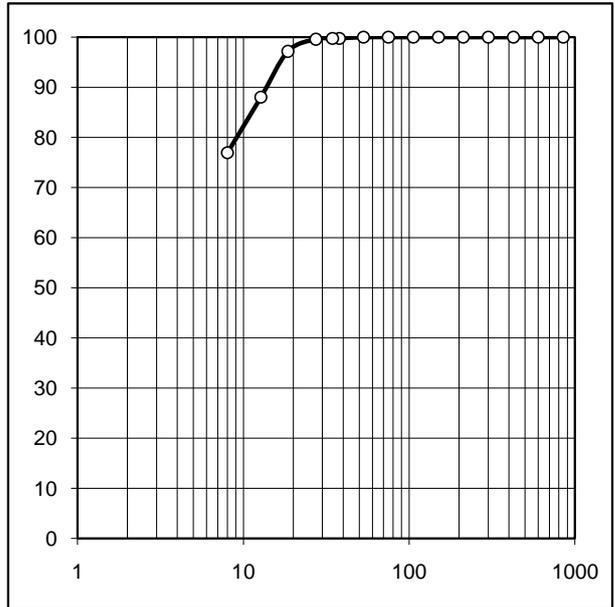


Metallurgy

PROJECT	T0728
SAMPLE NAME	Cyclone O/F
TEST NO	T05
DATE	190612
TECHNICIAN	SP

SIZING

Product Sized T05		SIZE um	WEIGHTS		
			gm	(%)	%PASS
P80		850	0.00	0.00	100.0
		600	0.00	0.00	100.0
		425	0.00	0.00	100.0
		300	0.00	0.00	100.0
		212	0.00	0.00	100.0
		150	0.00	0.00	100.0
		106	0.00	0.00	100.0
		75	0.00	0.00	100.0
		53	0.00	0.00	100.0
		38	0.23	0.23	99.8
CYCLOSIZER	CS1	34	0.03	0.03	99.7
FLOW	185	CS2	27	0.15	99.6
TEMP	21	CS3	19	2.42	97.2
SG	2.80	CS4	13	9.16	88.0
MINS	20	CS5	8	11.06	77.0
		SUB	76.95	76.95	0.0
		TOTAL	100.00	100.00	



ANALYSES

SIZE um	WT %	CaO		SiO2		Fe		WO3		Sn		F	
		%	dist	%	dist	%	dist	%	dist	%	dist	%	dist
38	0.23		0.0		0.0		0.0		0.0		0.0		0.0
34	0.03		0.0		0.0		0.0		0.0		0.0		0.0
27	0.15	33.8	0.2	27.7	0.1	7.1	0.1	0.2	0.3	0.2	0.2	5.8	0.1
19	2.42	33.7	2.9	27.2	2.3	7.3	2.3	0.2	3.3	0.2	2.5	9.1	2.7
13	9.16	32.5	10.6	27.6	8.9	7.7	8.9	0.2	11.0	0.2	10.8	11.5	12.7
8	11.06	24.4	9.6	30.9	12.1	8.6	12.0	0.1	11.5	0.2	12.2	11.6	15.4
<8	76.95	27.9	76.6	28.1	76.5	7.9	76.7	0.1	73.9	0.1	74.4	12.1	112.5
CALC Head	100.00	28.0	100.0	28.3	100.0	7.9	100.0	0.1	100.0	0.1	100.0	8.3	143.4
ASSAY		27.9		28.1		7.9		0.1		0.1		10.8	

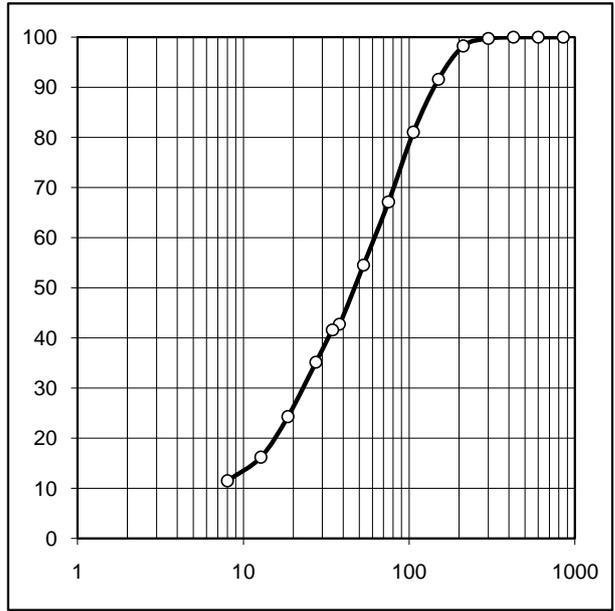


Metallurgy

PROJECT	T0728
SAMPLE NAME	Cyclone U/F
TEST NO	T05
DATE	190612
TECHNICIAN	SP

SIZING

Product Sized T05		SIZE um	WEIGHTS		
			gm	(%)	%PASS
P80	104	850	0.00	0.00	100.0
		600	0.00	0.00	100.0
		425	0.00	0.00	100.0
		300	0.39	0.27	99.7
		212	2.09	1.47	98.3
		150	9.44	6.64	91.6
		106	14.99	10.55	81.1
		75	19.79	13.93	67.1
		53	17.88	12.58	54.5
		38	16.78	11.81	42.7
CYCLOSIZER	CS1	34	1.61	1.13	41.6
FLOW 185	CS2	27	9.17	6.45	35.2
TEMP 21	CS3	19	15.40	10.84	24.3
SG 2.80	CS4	13	11.51	8.10	16.2
MINS 20	CS5	8	6.69	4.71	11.5
	SUB		16.35	11.51	0.0
	TOTAL		142.09	100.00	



ANALYSES

SIZE um	WT %	CaO		SiO2		Fe		WO3		Sn		F	
		%	dist	%	dist								
300	0.27	23.6	0.2	38.3	0.3	7.7	0.3	0.40	0.5	0.11	0.2	5.48	0.2
212	1.47	24.0	1.3	38.2	1.7	7.3	1.3	0.15	1.1	0.14	1.3	5.67	1.0
150	6.64	24.5	5.9	36.7	7.6	7.6	6.3	0.11	3.7	0.14	5.7	5.26	4.3
106	10.55	25.3	9.6	35.6	11.7	7.7	10.2	0.13	6.9	0.15	9.8	6.08	7.8
75	13.93	26.4	13.2	34.8	15.1	8.0	13.9	0.16	11.1	0.16	13.8	6.78	11.5
53	12.58	28.5	12.9	34.0	13.3	8.0	12.6	0.22	13.8	0.17	13.2	7.34	11.3
38	11.81	30.5	13.0	31.3	11.5	7.8	11.4	0.22	13.0	0.15	10.9	8.97	12.9
34	1.13	35.6	1.5	23.4	0.8	10.0	1.4	2.31	13.1	0.35	2.5	11.10	1.5
27	6.45	35.8	8.3	25.9	5.2	7.6	6.1	0.27	8.7	0.18	7.2	11.70	9.2
19	10.84	33.8	13.2	27.6	9.3	7.2	9.7	0.18	9.8	0.17	11.4	12.30	16.3
13	8.10	34.0	9.9	27.3	6.9	7.5	7.6	0.18	7.3	0.16	8.0	11.45	11.3
8	4.71	32.9	5.6	27.2	4.0	7.8	4.6	0.15	3.5	0.16	4.7	13.20	7.6
<8	11.51	13.1	5.4	35.1	12.6	10.2	14.6	0.13	7.5	0.16	11.4	3.52	4.9
CALC Head	100.00	27.8	100.0	32.1	100.0	8.0	100.0	0.2	100.0	0.2	100.0	8.2	100.0

A6 MOZLEY GRAVITY SEPARATION

PROJECT	T0728
TEST NO	T06
DATE	140611
TECHNICAL	ID

METHOD	
Stage 1	mozley panner on size fractions
Stage 2	feed into conc, tail and mids
Stage 3	mids into tail, mids 2 and mids 1

SAMPLE	
Cyclone U/F from T05	

GRAVITY SEPARATION RESULTS

FRACTION (um)	PRODUCT		WO3		Fe		SiO2		CaO		Sn		
	PROD	WT (gm)	WT (%)	WO3 (%)	DIST (%)	Fe (%)	DIST (%)	SiO2 (%)	DIST (%)	CaO (%)	DIST (%)	Sn (%)	DIST (%)
>106	CONC	1.40	0.38	1.92	4.5	13.60	0.7	27.70	0.3	30.70	0.4	0.31	0.8
	MIDS 1	10.75	2.94	0.10	1.8	9.88	3.7	31.30	2.8	29.80	3.1	0.20	3.8
	MIDS 2	41.96	11.49	0.06	4.2	8.79	12.9	33.80	11.8	27.00	10.9	0.16	11.7
	TAIL	35.41	9.70	0.06	3.6	5.72	7.1	40.40	11.9	20.30	6.9	0.10	6.2
>53	CONC	2.21	0.61	6.60	24.4	13.90	1.1	20.00	0.4	31.10	0.7	0.45	1.7
	MIDS 1	33.72	9.23	0.08	4.5	9.39	11.1	28.40	7.9	36.00	11.6	0.22	12.9
	MIDS 2	29.67	8.13	0.05	2.5	8.92	9.2	31.70	7.8	29.60	8.4	0.18	9.3
	TAIL	64.66	17.71	0.04	4.3	6.55	14.8	39.40	21.1	21.40	13.3	0.11	12.4
>20	CONC	4.00	1.10	6.00	40.1	12.85	1.8	19.85	0.7	33.00	1.3	0.61	4.3
	MIDS 1	11.12	3.05	0.12	2.2	8.27	3.2	24.00	2.2	39.30	4.2	0.19	3.7
	MIDS 2	28.05	7.68	0.06	2.8	8.07	7.9	27.20	6.3	35.40	9.5	0.17	8.3
	TAIL	102.19	27.99	0.03	5.1	7.47	26.6	31.70	26.9	30.50	29.9	0.14	25.0
TOTAL	CALC	365.14	100.00	0.16	100.0	7.85	100.0	33.04	100.0	28.59	100.0	0.16	100.0

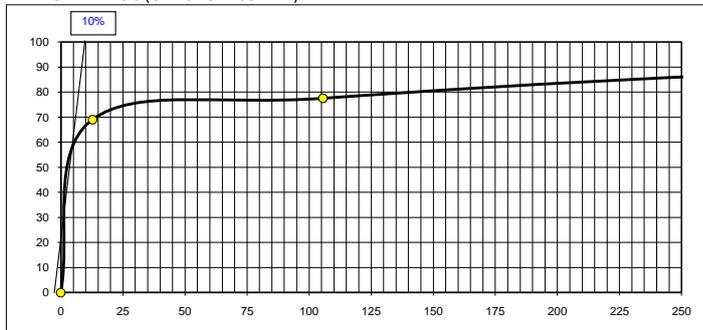
GRAVITY SEPARATION BALANCE

PRODUCT	Wt (%)	WO3 (%)	DIST (%)	CUM DIST	UNITS	CUM UNIT
CONC	2.1	5.42	69.0	69.0	12.72	12.7
MIDS 1	15.2	0.09	8.5	77.5	92.89	105.6
MIDS 2	27.3	0.06	9.5	87.0	166.57	272.2
TAILS	55.4	0.04	13.0	100.0	337.99	610.2
TOTAL	100.0	0.16	100.0			

SUMMARY

FRACTION	WT (gm)	WT (%)	PASSING	WO3(%)	DIST (%)
>106	89.52	19.8	80.2	0.09	6.9
>53	130.26	28.8	51.5	0.16	17.6
>20	145.36	32.1	19.4	0.21	24.8
<20	87.70	19.4	0.0	0.70	50.6
TOTALS	452.84	100.0		0.27	100.0
HEAD				0.69	

RELEASE ANALYSIS (UNITS VS RECOVERY)

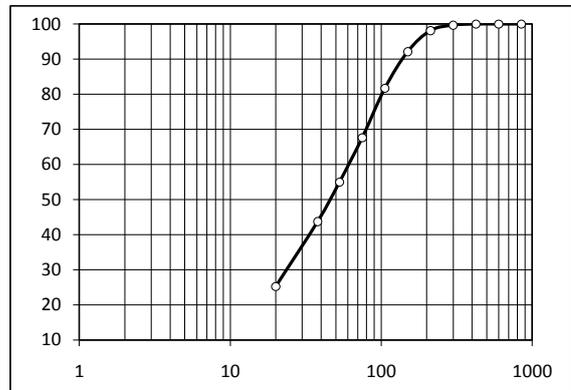




Ammtec Burnie
GENERAL SIZING REPORT SHEET

PROJECT	T0728
SAMPLE	Mozley feed
TEST NO	T06
DATE	130612
TECHNICIAN	ID

Sample Mozley feed	SIZE um	WEIGHTS			P80 um
		gm	(%)	% Pass	
	850	0.00	0.00	100.0	102
	600	0.00	0.00	100.0	
	425	0.00	0.00	100.0	
	300	1.60	0.32	99.7	
	212	7.67	1.55	98.1	
	150	29.69	6.00	92.1	
	106	51.60	10.42	81.7	
	75	69.66	14.07	67.6	
	53	62.66	12.66	55.0	
	38	55.58	11.23	43.7	
	20	91.75	18.54	25.2	
	SUB	124.77	25.2	0.0	
	TOTAL	494.98	100.0		



A7 GRAVITY LOCK CYCLE



AMMTEC BURNIE

LOCK CYCLE DATA SHEET

PROJECT	T0728
LC01	Cycle 1
DATE	080812
BY	ID

Stream	Dry Wt (g)	Damp Cake (g)	Time min/sec	Notes
New Feed	----	551	----	
+106 Mozley Feed	----	113.9	----	
+38 Mozley Feed	----	192.1	----	
+CS1 Mozley Feed	----	14.3	----	
Final Con 1	1.1	----	----	
+106 Gravity tail	----	106.7	----	
Regrind time	----	----	3	Short mill 6 Kg Charge, 180sec/100gm
+106 Gravity tail regrind	----	25	----	
-106 Gravity tail regrind	----	90.3	----	
< CS5	13.36	----	----	
CS1 + 38 Mozley tail	----	136.8	----	
CS1 + 38 Mozley Tail WET / DRY	Tare	Net Wet	Gross Dry	
	4.77	10.01	14.08	
CS1 + 38 Mozley Mids	----	66.7	----	
Final Con 2	2.39	----	----	
CS2 + CS5	----	160.5	----	for float (freezer)
CS2 + CS5 WET / DRY	Tare	Net Wet	Gross Dry	
	4.74	10.45	13.75	



PROJECT	T0728
LC01	Cycle 2
DATE	090812
BY	ID

Stream	Dry Wt (g)	Damp Cake (g)	Time min/sec	Notes
New Feed	----	551.3	----	
+106 Mozley Feed	----	122.5	----	
+38 Mozley Feed	----	255	----	
+CS1 Mozley Feed	----	11.6	----	
Final Con 1	1.58	----	----	
+106 Gravity tail	----	118.1	----	
Regrind time	----	----	3.15	Short mill 6 Kg Charge, 180sec/100gm
+106 Gravity tail regrind	----	33.9	----	
-106 Gravity tail regrind	----	95.7	----	
< CS5	43.9	----	----	
CS1 + 38 Mozley tail	----	225	----	
CS1 + 38 Mozley Tail WET / DRY	Tare	Net Wet	Gross Dry	
----	4.82	9.35	13.25	
CS1 + 38 Mozley Mids	----	105.8	----	
Final Con 2	2.16	----	----	
CS2 + CS5	----	190.7	----	for float (freezer)
CS2 + CS5 WET / DRY	Tare	Net Wet	Gross Dry	
----	4.74	9.29	12.75	



PROJECT	T0728
LC01	Cycle 3
DATE	100812
BY	ID

Stream	Dry Wt (g)	Damp Cake (g)	Time min/sec	Notes
New Feed	----	551.7	----	
+106 Mozley Feed	----	116.1	----	
+38 Mozley Feed	----	252.8	----	
+CS1 Mozley Feed	----	13.1	----	
Final Con 1	1.05	----	----	
+106 Gravity tail	----	114.7	----	
Regrind time	----	----	3.15	Short mill 6 Kg Charge, 180sec/100gm
+106 Gravity tail regrind	----	43.4	----	
-106 Gravity tail regrind	----	84.1	----	
< CS5	39.2	----	----	
CS1 + 38 Mozley tail	----	281.6	----	
CS1 + 38 Mozley Tail WET / DRY	Tare	Net Wet	Gross Dry	
----	4.76	10.04	13.99	
CS1 + 38 Mozley Mids	----	76.3	----	
Final Con 2	2	----	----	
CS2 + CS5	----	198.5	----	for float (freezer)
CS2 + CS5 WET / DRY	Tare	Net Wet	Gross Dry	
----	4.81	11.12	14.33	



PROJECT	T0728
LC01	Cycle 4
DATE	110812
BY	ID

Stream	Dry Wt (g)	Damp Cake (g)	Time min/sec	Notes
New Feed	----	550.3	----	
+106 Mozley Feed	----	126.1	----	
+38 Mozley Feed	----	248.4	----	
+CS1 Mozley Feed	----	12.2	----	
Final Con 1		----	----	
+106 Gravity tail	----		----	
Regrind time	----	----		Short mill 6 Kg Charge, 180sec/100gm
+106 Gravity tail regrind	----		----	
-106 Gravity tail regrind	----		----	
< CS5		----	----	
CS1 + 38 Mozley tail	----	285.7	----	
CS1 + 38 Mozley Tail WET / DRY	Tare	Net Wet	Gross Dry	
----	4.77	9.98		
CS1 + 38 Mozley Mids	----		----	
Final Con 2	2.23	----	----	
CS2 + CS5	----	197.2	----	for float (freezer)
CS2 + CS5 WET / DRY	Tare	Net Wet	Gross Dry	
----	4.71	11.03		



LOCK CYCLE DATA SHEET

PROJECT	T0728
LC01	Cycle 5
DATE	130812
BY	ID

Stream	Dry Wt (g)	Damp Cake (g)	Time min/sec	Notes
New Feed	----	549.9	----	
+106 Mozley Feed	----	107.6	----	
+38 Mozley Feed	----	249.6	----	
+CS1 Mozley Feed	----	8.9	----	
Final Con 1	1.3	----	----	
+106 Gravity tail	----	103.8	----	
Regrind time	----	----	3	Short mill 6 Kg Charge, 180sec/100gm
+106 Gravity tail regrind	----	21.7	----	
-106 Gravity tail regrind	----	84.6	----	
< CS5	21.1	----	----	
CS1 + 38 Mozley tail	----	242.2	----	
CS1 + 38 Mozley Tail WET / DRY	Tare	Net Wet	Gross Dry	
----	4.76	9.74	13.58	
CS1 + 38 Mozley Mids	----	46.2	----	
Final Con 2	2.26	----	----	
CS2 + CS5	----	201.6	----	for float (freezer)
CS2 + CS5 WET / DRY	Tare	Net Wet	Gross Dry	
----	4.74	11.02	13.97	



PROJECT	T0728
LC01	Cycle 6
DATE	150812
BY	ID

Stream	Dry Wt (g)	Damp Cake (g)	Time min/sec	Notes
New Feed	----	550.2	----	
+106 Mozley Feed	----	103.1	----	
+38 Mozley Feed	----	236	----	
+CS1 Mozley Feed	----	8.9	----	
Final Con 1	1.54	----	----	
+106 Gravity tail	----	99.4	----	
Regrind time	----	----	4	Short mill 6 Kg Charge, 240sec/100gm
+106 Gravity tail regrind	----	13.3	----	
-106 Gravity tail regrind	----	88.7	----	
< CS5	41.5	----	----	
CS1 + 38 Mozley tail	----	241.7	----	
CS1 + 38 Mozley Tail WET / DRY	Tare	Net Wet	Gross Dry	
----	4.77	10.02	13.83	
CS1 + 38 Mozley Mids	----	35.7	----	
Final Con 2	2.27	----	----	
CS2 + CS5	----	199.6	----	for float (freezer)
CS2 + CS5 WET / DRY	Tare	Net Wet	Gross Dry	
----	4.74	10.29	13.52	



PROJECT	T0728
LC01	Cycle 7
DATE	200812
BY	ID

Stream	Dry Wt (g)	Damp Cake (g)	Time min/sec	Notes
New Feed	----	550.6	----	
+106 Mozley Feed	----	105.6	----	
+38 Mozley Feed	----	236.3	----	
+CS1 Mozley Feed	----	11.4	----	
Final Con 1	1.24	----	----	
+106 Gravity tail	----	102.2	----	
Regrind time	----	----	4	Short mill 6 Kg Charge, 240sec/100gm
+106 Gravity tail regrind	----	----	----	
-106 Gravity tail regrind	----	----	----	
< CS5	51.4	----	----	
CS1 + 38 Mozley tail	----	223.2	----	
CS1 + 38 Mozley Tail WET / DRY	Tare	Net Wet	Gross Dry	
----	5.13	8.9	13.36	
CS1 + 38 Mozley Mids	----		----	
Final Con 2	2.57	----	----	
CS2 + CS5	----	206.1	----	for float (freezer)
CS2 + CS5 WET / DRY	Tare	Net Wet	Gross Dry	
----	4.74	11.27	13.91	



Ammtec Burnie
LOCKED CYCLE GRAVITY TEST

PROJECT	T0728
ORE TYPE	COMP 1 (USC 1-48)
FEED FROM	Non Mags T02, T03, T04
TECH	ID

PROJECT	T0728
TEST NO	LC01
DATE	
TECH	ID

CYCLE NO		Cycle 1	Cycle 2	Cycle 3	Cycle 4	Cycle 5	Cycle 6	Cycle 7
Gravity final conc 1 (+106um Conc) dry wt and assays	gm	1.10	1.58	1.05	1.35	1.30	1.54	1.24
	%CaO	28.30	30.20	28.90	30.10	29.50	30.00	27.80
	%Fe	16.60	15.50	17.00	16.55	16.00	14.55	16.80
	%Sn	0.41	0.33	0.44	0.37	0.34	0.32	0.33
	%WO3	3.71	2.04	2.75	2.57	2.17	1.61	2.48
Gravity final conc 2 (CS1+38umConc) dry wt and assays	gm	2.39	2.16	2.00	2.23	2.26	2.29	2.57
	%CaO	17.15	15.25	14.20	15.05	14.90	16.35	16.50
	%Fe	26.50	25.40	26.60	25.80	26.30	26.30	26.60
	%Sn	1.12	1.12	1.12	1.08	1.06	1.03	1.01
	%WO3	14.50	15.25	15.95	15.25	14.45	13.45	12.80
Mozley Fine Tail (Mozley CS1+38um Tail) dry wt and assays	gm	127.22	202.50	259.07	265.70	220.40	217.53	205.34
	%CaO	25.30	27.20	29.30	29.40	27.40	27.40	28.10
	%Fe	7.46	7.89	8.36	8.27	7.93	7.88	7.90
	%Sn	0.13	0.15	0.16	0.17	0.16	0.17	0.15
	%WO3	0.02	0.04	0.05	0.05	0.06	0.05	0.03
CS2-CS5 (FLOAT FEED) Note: combined sample) dry wt and assays	gm	138.03	164.00	170.71	163.68	169.34	169.66	166.94
	%CaO	35.10	35.90	34.50	33.60	33.20	34.00	34.10
	%Fe	7.82	8.37	7.91	7.79	7.72	7.76	7.90
	%Sn	0.17	0.19	0.18	0.16	0.14	0.16	0.16
	%WO3	0.20	0.19	0.19	0.18	0.18	0.18	0.17
<CS5 (Note: combined sample) dry wt and assays	gm	13.36	43.90	39.20	43.50	21.10	41.50	51.40
	%CaO	15.65	27.40	15.65	15.65	21.10	14.45	19.55
	%Fe	11.20	9.15	10.55	10.30	10.25	10.60	10.25
	%Sn	0.18	0.14	0.17	0.16	0.15	0.15	0.15
	%WO3	0.20	0.11	0.14	0.14	0.13	0.15	0.13
COMBINED CONCENTRATE Calculated	gm	3.49	3.74	3.05	3.58	3.56	3.83	3.81
	%CaO	20.66	21.57	19.26	20.73	20.23	21.84	20.18
	%Fe	23.38	21.22	23.30	22.31	22.54	21.58	23.41
	%Sn	0.90	0.79	0.89	0.81	0.80	0.74	0.79
	%WO3	11.10	9.67	11.41	10.47	9.97	8.69	9.44

New Feed (gm) (damp)	551.0	551.3	551.7	550.3	549.9	550.2	550.6
Regrind D/C Cake (gm) (damp)	106.7	118.1	114.7	126.5	103.8	99.4	102.2
Regrind D/C Dry	0.0	90.7	100.4	97.5	107.5	88.2	84.5
CS1+38um Mozley Mids cake (gm) (damp)	66.7	105.8	76.3	31.6	46.2	35.7	55.0
CS1+38um Mozley Mids cake Dry	0.0	56.7	89.9	64.9	26.9	39.3	30.3
Cycle Feed	551.0	698.7	742.0	712.7	684.3	677.7	665.4
Concentrates Made (gm)	3.5	3.7	3.1	3.6	3.6	3.8	3.8
CS1+38um Mozley Tails	127.2	202.5	259.1	265.7	220.4	217.5	205.3
Cyclosizer -CS2	151.4	207.9	209.9	207.2	190.4	211.2	218.3
Cycle Discharge	282.1	414.1	472.0	476.5	414.4	432.5	427.5

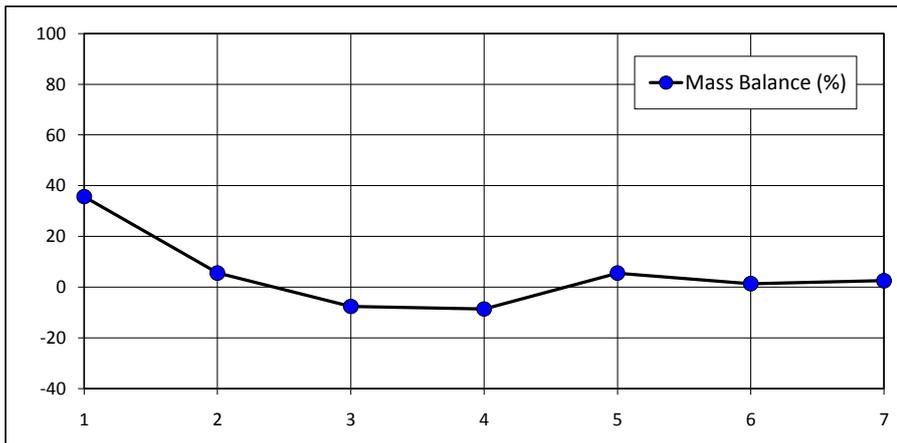
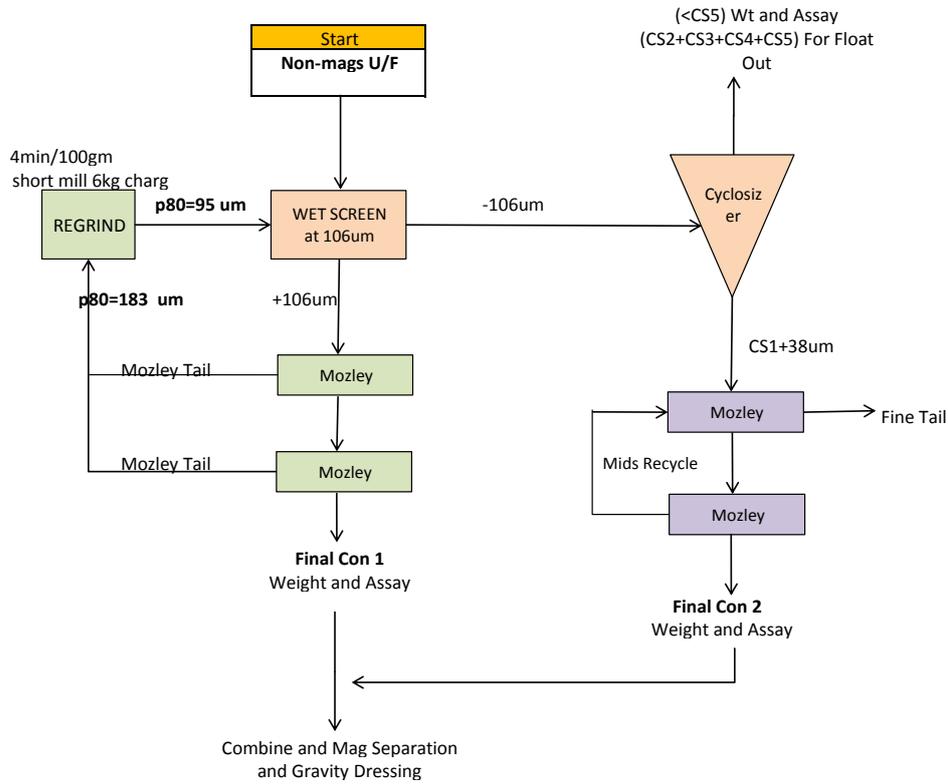
Fine Mids Recycle	
gm	48.5
%CaO	35.30
%Fe	10.90
%Sn	0.24
%WO3	0.11
%F	

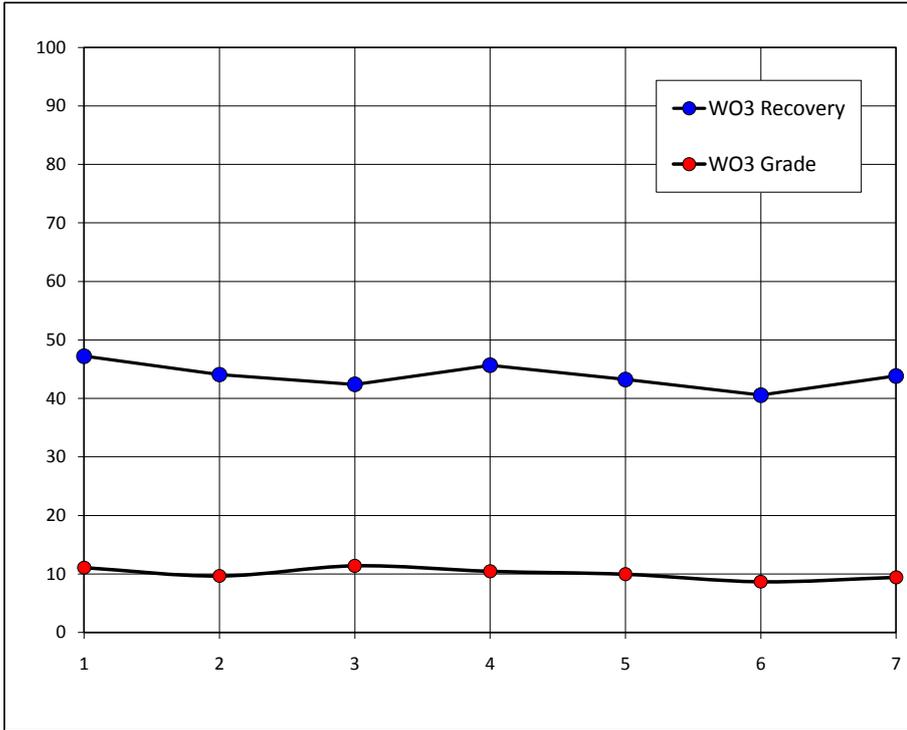
End Regrind Feed		ASSAY
gm		102.2
%CaO		23.90
%Fe		8.95
%Sn		0.15
%WO3		0.05

NEW FEED	CALC	ASSAY
gm	3069.9	
%CaO	29.49	29.70
%Fe	8.34	8.06
%Sn	0.17	
%WO3	0.19	0.18

DISTRIBUTIONS		AV (C5-7)	FROM NEW FEED (%)						
			Cycle 1	Cycle 2	Cycle 3	Cycle 4	Cycle 5	Cycle 6	Cycle 7
MASS	Final Conc 1	0.3	0.3	0.4	0.2	0.3	0.3	0.4	0.3
	Final Conc 2	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.6
	Fine Tail	48.9	29.0	46.2	59.1	60.6	50.3	49.6	46.8
	CS2-CS5	38.5	31.5	37.4	38.9	37.3	38.6	38.7	38.1
	<CS5 Slimes	8.7	3.0	10.0	8.9	9.9	4.8	9.5	11.7
	Calc Recycle	3.1	35.7	5.6	-7.6	-8.6	5.5	1.4	2.5
WO3	Final Conc 1	3.4	5.0	3.9	3.5	4.2	3.4	3.0	3.7
	Final Conc 2	39.1	42.2	40.2	38.9	41.5	39.8	37.5	40.1
	Fine Tail	12.3	3.1	9.9	15.8	16.2	16.1	13.3	7.5
	CS2-CS5	50.3	33.6	52.0	52.0	48.6	49.7	51.0	50.3
	<CS5 Slimes	6.4	3.3	5.9	6.7	7.4	3.3	7.6	8.1
	Calc Recycle	-11.5	12.8	-11.9	-16.9	-17.9	-12.4	-12.4	-9.8
OA WO3 RECOVERY (%)		42.2	47.2	44.1	42.4	45.7	43.2	40.6	43.8
Fe	Final Conc 1	0.6	0.5	0.7	0.5	0.6	0.6	0.6	0.6
	Final Conc 2	1.7	1.7	1.5	1.5	1.6	1.6	1.6	1.9
	Fine Tail	46.3	25.9	43.7	59.2	60.0	47.8	46.8	44.3
	CS2-CS5	35.9	29.5	37.5	36.9	34.8	35.7	36.0	36.0
	<CS5 Slimes	10.8	4.1	11.0	11.3	12.2	5.9	12.0	14.4
	Calc Recycle	4.7	38.3	5.7	-9.3	-9.3	8.4	2.9	2.8
Sn	Final Conc 1	0.6	0.6	0.7	0.6	0.7	0.6	0.7	0.6
	Final Conc 2	3.3	3.7	3.3	3.1	3.3	3.3	3.2	3.5
	Fine Tail	46.9	22.6	41.5	56.6	61.7	48.2	50.5	42.1
	CS2-CS5	35.3	32.1	42.6	42.0	35.8	32.4	37.1	36.5
	<CS5 Slimes	7.8	3.3	8.4	9.1	9.5	4.3	8.5	10.5
	Calc Recycle	6.0	37.8	3.5	-11.4	-11.0	11.2	0.0	6.8
CaO	Final Conc 1	0.3	0.2	0.4	0.2	0.3	0.3	0.4	0.3
	Final Conc 2	0.3	0.3	0.3	0.2	0.3	0.3	0.3	0.3
	Fine Tail	45.8	24.9	42.6	58.7	60.4	46.7	46.1	44.6
	CS2-CS5	44.0	37.5	45.5	45.5	42.5	43.5	44.6	44.0
	<CS5 Slimes	5.3	1.6	9.3	4.7	5.3	3.4	4.6	7.8
	Calc Recycle	4.3	35.5	2.0	-9.4	-8.8	5.8	4.0	3.0

LOCKED CYCLE TEST FLOWSHEET

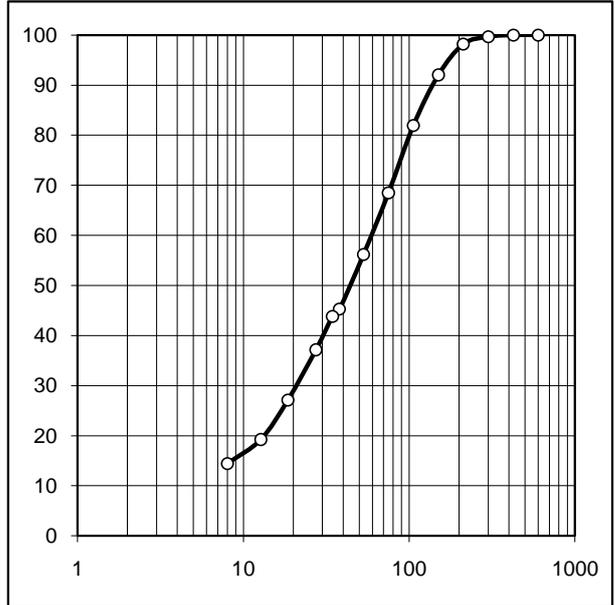




PROJECT	T0728
SAMPLE NAME	Feed Head
TEST NO	LC01
DATE	90812
TECHNICIAN	SP

SIZING

Product Sized LC01		SIZE um	WEIGHTS			
			gm	(%)	%PASS	
P80	101	850	0.00	0.00	100.0	
		600	0.00	0.00	100.0	
		425	0.00	0.00	100.0	
		300	0.47	0.34	99.7	
		212	2.00	1.47	98.2	
		150	8.38	6.15	92.0	
		106	13.74	10.08	82.0	
		75	18.35	13.46	68.5	
CYCLOSIZER	CS1	34	2.01	1.47	43.8	
		27	9.07	6.65	37.1	
		19	13.66	10.02	27.1	
		13	10.75	7.89	19.2	
		8	6.55	4.81	14.4	
FLOW	185	CS2	27	9.07	6.65	37.1
TEMP	21	CS3	19	13.66	10.02	27.1
SG	2.80	CS4	13	10.75	7.89	19.2
MINS	20	CS5	8	6.55	4.81	14.4
		SUB	19.67	14.43	0.0	
		TOTAL	136.31	100.00		


ANALYSES

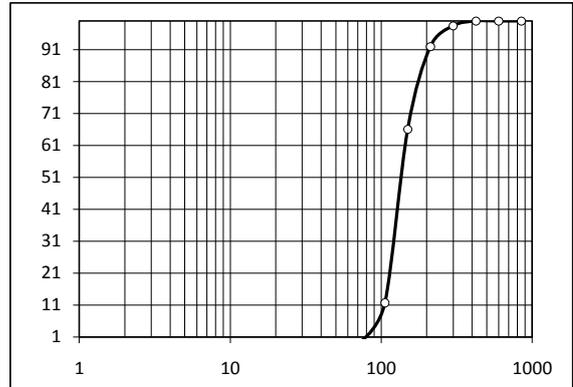
SIZE um	WT %	CaO		SiO2		Fe		WO3	
		%	dist	%	dist	%	dist	%	dist
212	1.81	24.4	1.5	38.0	2.1	7.8	1.7	0.1	1.3
150	6.15	24.8	5.2	36.7	7.0	7.9	5.9	0.1	2.4
106	10.08	25.3	8.7	36.5	11.5	8.2	10.0	0.1	6.2
75	13.46	26.2	12.0	36.0	15.1	8.6	14.1	0.2	11.4
53	12.33	28.4	11.9	35.0	13.4	8.6	12.9	0.2	11.8
38	10.89	30.6	11.4	32.1	10.9	8.1	10.7	0.2	12.2
34	1.47	31.7	1.6	20.9	1.0	9.2	1.6	1.6	13.1
27	6.65	35.3	8.0	26.6	5.5	7.9	6.4	0.2	8.6
19	10.02	33.6	11.5	28.8	9.0	7.6	9.3	0.2	11.3
13	7.89	34.6	9.3	27.8	6.8	7.7	7.4	0.2	8.4
8	4.81	33.4	5.5	28.7	4.3	8.1	4.7	0.2	4.3
<8	14.43	27.3	13.4	29.8	13.4	8.6	15.1	0.1	8.9
CALC Head	100.00	29.3	100.0	32.1	100.0	8.2	100.0	0.2	100.0
ASSAY		29.7		31.7		8.1		0.2	



Ammtec Burnie
GENERAL SIZING REPORT SHEET

PROJECT	T0728
SAMPLE	Mozley Tail
TEST NO	LC01 Cyc 7
DATE	220812
TECHNICIAN	SP

Sample Mozley Tail	SIZE um	WEIGHTS			P80 um
		gm	(%)	% Pass	
	850	0.00	0.00	100.0	183
	600	0.00	0.00	100.0	
	425	0.00	0.00	100.0	
	300	1.44	1.50	98.5	
	212	6.28	6.52	92.0	
	150	24.96	25.93	66.1	
	106	52.34	54.37	11.7	
	75	11.29	11.73	0	
	53	0.23	0.24	0	
	38	0.03	0.03	0	
	20	0.00	0.00	0	
	SUB	0.00	0.0	0	
	TOTAL	96.26	100		





Ammtec Burnie
GENERAL SIZING REPORT SHEET

PROJECT	T0728
SAMPLE	Mozley Tail +106
TEST NO	LC01 Cyc 7 .1/6/4
DATE	220812
TECHNICIAN	SP

Sample Mozley Tail +106	SIZE um	WEIGHTS			P80 um
		gm	(%)	% Pass	
	850	0.00	0.00	100.0	95
	600	0.00	0.00	100.0	
	425	0.00	0.00	100.0	
	300	0.26	0.26	99.7	
	212	0.12	0.12	99.6	
	150	1.42	1.44	98.2	
	106	10.93	11.12	87.0	
	75	19.28	19.61	67	
	53	17.30	17.60	50	
	38	12.85	13.07	37	
	20	0.00	0.00	37	
	SUB	36.14	36.8	0	
	TOTAL	98.30	100		

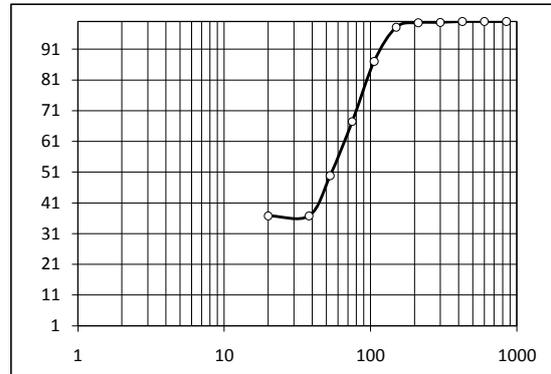
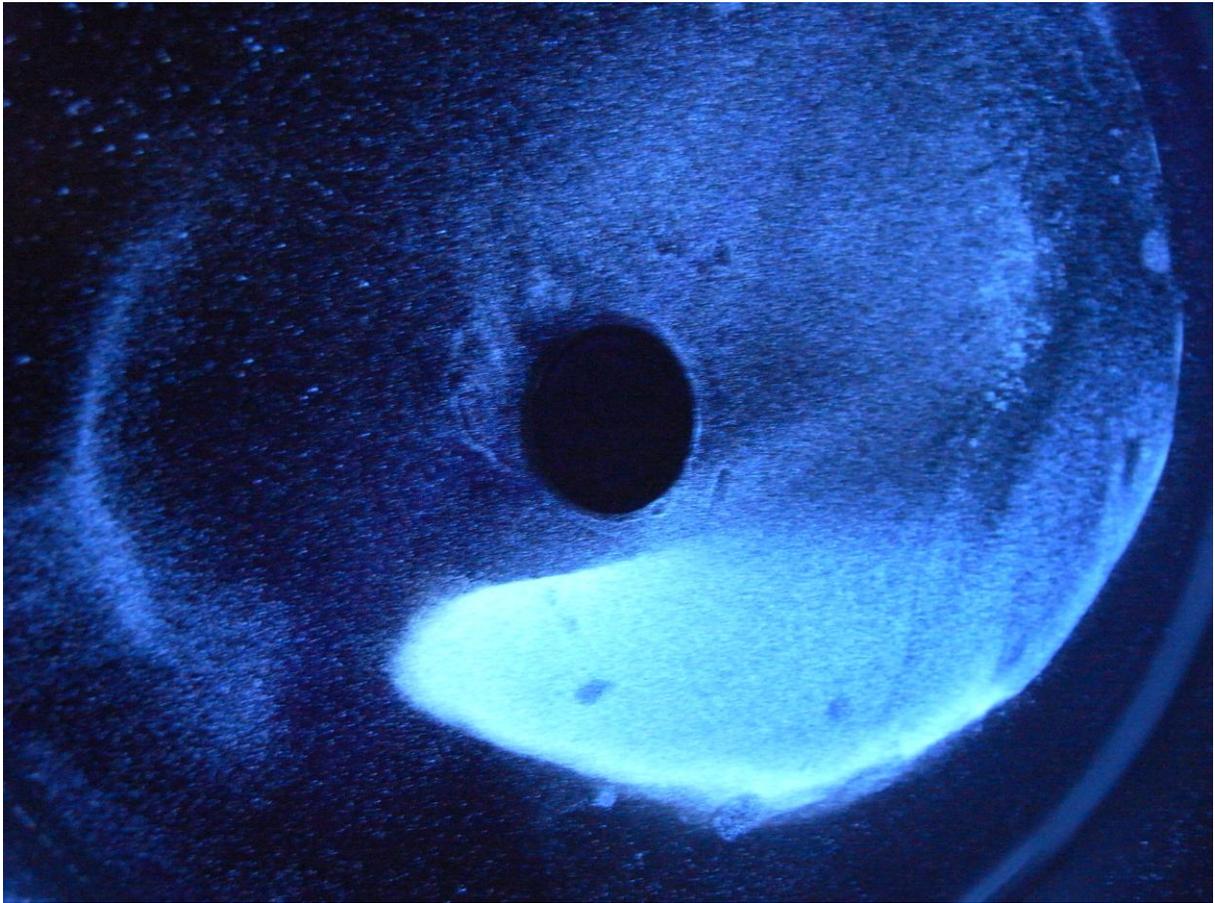


IMAGE OF TUNGSTEN CONCENTRATE OF GRAVITY LOCK CYCLE UNDER UV LIGHT



A8 DAVIS TUBE TEST

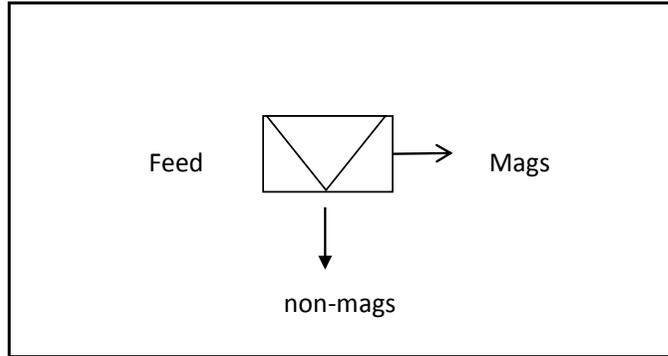


Ammtec Burnie

DAVIS TUBE TEST REPORT SHEET

PROJECT	T0728
TEST NO	T14
DATE	020912
TECH	SZ

TEST TYPE	Davis Tube
Electric Current (A)	1.4
Water Flowrate (mls/min)	570.0
Run Time (min)	15.0
Sample Start Weight (gms)	10.0
Comments	



START MATERIAL	
Moina ore	
Gravity conc	
START WEIGHT (gm)	10.4
FROM TEST NO	LC 1

MAGNETIC SEPARATION RESULTS

PRODUCT				CaO		Iron		Sn		WO3	
NAME	Wt (gm)	Calc (gm)	Wt (%)	CaO (%)	DIST (%)	Fe (%)	DIST (%)	Sn (%)	DIST (%)	WO3 (%)	DIST (%)
Non Magnetics	8.3	8.3	79.81	22.43	87.5	18.51	64.7	0.90	87.2	11.06	88.4
Magnetics	2.1	2.1	20.19	12.65	12.5	40.00	35.3	0.52	12.8	5.73	11.6
TOTAL	10.40		100.00	20.46	100.0	22.85	100.0	0.82	100.0	9.98	100.0
Head				20.30		22.50		0.84		10.45	

A9 SUPER PANNER TEST

PROJECT	T0728
TEST NO	T15
DATE	30912
TECHNICAL	ID

METHOD	
Stage 1	Super panner on size fractions
Stage 2	feed into conc, tail and mids
Stage 3	mids into tail, mids 2 and mids 1

SAMPLE	
Gravity Conc-Sulphide Tail-	
Davis Tube Non mags	
Moina	

GRAVITY SEPARATION RESULTS

FRACTION (um)	PRODUCT			WO3		Fe		Sn		CaO	
	PROD	WT (gm)	WT (%)	WO3 (%)	DIST (%)	Fe (%)	DIST (%)	Sn (%)	DIST (%)	CaO (%)	DIST (%)
>38	CONC	1.60	19.28	41.20	71.8	15.35	16.0	2.58	55.3	11.50	9.9
	MIDS 1	2.46	29.64	9.06	24.3	24.20	38.7	0.74	24.4	18.60	24.6
	MIDS 2	1.87	22.53	1.03	2.1	17.75	21.6	0.38	9.5	27.60	27.7
	TAIL	2.37	28.55	0.69	1.8	15.35	23.7	0.34	10.8	29.70	37.8
TOTAL	CALC	8.30	100.00	11.06	100.0	18.51	100.0	0.90	100.0	22.43	100.0

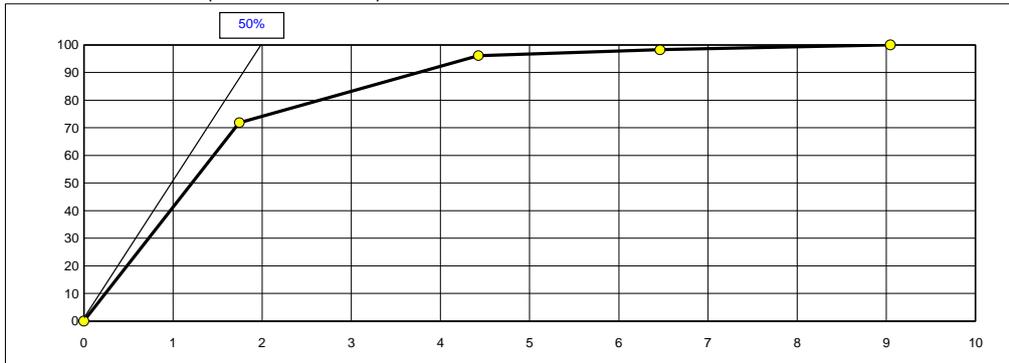
WO3 GRAVITY SEPARATION BALANCE

PRODUCT	Wt (%)	WO3 (%)	DIST (%)	CUM DIST	UNITS	CUM UNIT
CONC	19.3	41.20	71.8	71.8	1.74	1.7
MIDS 1	29.6	9.06	24.3	96.1	2.68	4.4
MIDS 2	22.5	1.03	2.1	98.2	2.04	6.5
TAILS	28.6	0.69	1.8	100.0	2.58	9.0
TOTAL	100.0	11.06	100.0			

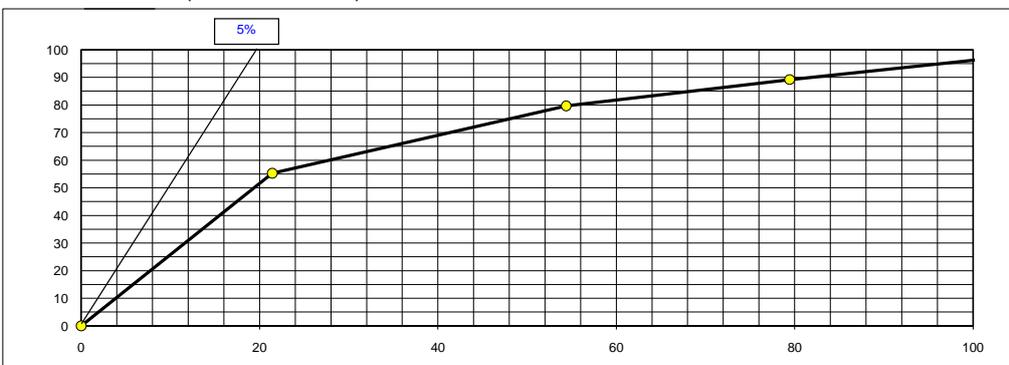
Sn GRAVITY SEPARATION BALANCE

PRODUCT	Wt (%)	Sn (%)	DIST (%)	CUM DIST	UNITS	CUM UNIT
CONC	19.3	2.58	55.3	55.3	21.43	21.4
MIDS 1	29.6	0.74	24.4	79.7	32.95	54.4
MIDS 2	22.5	0.38	9.5	89.2	25.05	79.4
TAILS	28.6	0.34	10.8	100.0	31.75	111.2
TOTAL	100.0	0.90	100.0			

WO3 RELEASE ANALYSIS (UNITS VS RECOVERY)



Sn RELEASE ANALYSIS (UNITS VS RECOVERY)

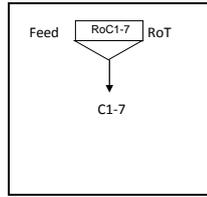


A10 TUNGSTEN FLOTATION



Ammtec Burnie
ROUGHER/CLEANER FLOTATION REPORT SHEET

Milling		Primary	Regrind
Mill	type	0	
Media	type	0	
Media	kg	0	
Solids	g	0	
Water	g	0	
Time	min	0	
Speed	rpm	0	
Lime	g	0.0	
End pH	pH	0.0	
End p80	µm	0	



PROJECT	T0728
TEST NO	T07
DATE	090712
TECHNICIAN	SZ

PRODUCT FLOATED
Molina ore
None mags U/F

NOTES
Standard sulphide rougher
WO3 Rougher
FS2 with 10% OT75

Float Cell	Volume	Regrind	Power
Rougher	1.5	Start	
0		Finish	
0		W/h	
Speed	800	kWh/t	

	pH	1 PAX g/t	1 Na2SiO3 g/t	5 SodaAsh g/t	100.0 STP g/t	1.0 FS2 g/t	Cum FS2 g/t	0.25 Guar g/t	1 Dextrin g/t	100 MIBC g/t	100 76A g/t	Cond Time min	Air L/min	Float Time min	Cum Float Time	Wet Wt g	Con % Solids
Condition	8.4											2					
Condition	8.5	63						158				1					
Sulphide Con	8.5									126	44					0	
Condition	8.5		631									2					
Condition	9.8			32								1					
Condition	10.2				631							1					
Condition	10.2					32	32					3					
WO3 Ro C1	10.2					189	221		631	126	44	7	4-10	1.0	1.0	100	2
Condition	9.8					316	537					3	7-10	2.5	3.5	100	12
WO3 Ro C2	8.0					316	852			63	44	2	7-10	2.5	6.0	100	12
Condition	8.0					631	1483					5	7-10	1.0	7.0	45	18
WO3 Ro C3	8.0					631	2115			63		5	7-10	1.5	8.5	120	15
Condition	8.0					631	2746			126		1	7-10	1.5	10.0	115	13
WO3 Ro C4	8.0					1262	4008					5	7-10	1.5	11.5	130	13
Condition	8.0					1894	5902					1	7-10	2.5	14.0	210	9
WO3 Ro C5	8.0											5	7-10	4.5	18.5	500	6
Condition	8.0											1					
WO3 Ro C6	8.0											1					
Condition	8.0											5					
WO3 Ro C7	8.0											1					
Condition	8.0											5					
WO3 Ro C8	8.0											1					
Condition	8.0											5					
WO3 Ro C9	8.0											1					
REAGENT TOTALS (g/t)		63	631	32	631	852		158	631	316	133						

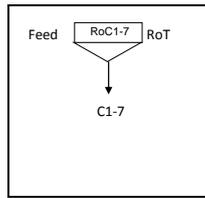
PRODUCTS	WT g	WT %	CaO %	DIST %	SiO2 %	DIST %	Fe %	DIST %	WO3 %	DIST %	Sn %	DIST %
WO3 Ro C1	1.9	1.2	29.1	1.18	17.5	0.7	13.3	2.0	0.15	1.0	0.10	0.8
C2	12.4	7.8	61.7	16.4	6.83	1.7	2.57	2.6	0.54	23.4	0.07	3.5
C3	11.7	7.4	53.2	13.3	10.6	2.5	3.43	3.2	0.49	20.0	0.09	4.3
C4	8.1	5.1	47.5	8.23	16.7	2.8	4.98	3.2	0.44	12.4	0.13	4.3
C5	18.0	11.4	35.6	13.7	25.4	9.3	7.56	10.9	0.30	18.9	0.15	11.0
C6	15.3	9.7	30.0	9.83	31.6	9.9	8.67	10.7	0.16	8.6	0.18	11.2
C7	17.2	10.9	27.9	10.3	34.9	12.2	9.45	13.1	0.10	6.0	0.22	15.4
C8	19.1	12.1	24.8	10.1	35.5	13.8	9.76	15.0	0.06	4.0	0.18	14.0
C9	27.8	17.5	18.2	10.8	40.4	22.9	9.01	20.1	0.02	1.9	0.18	20.3
RoT	26.9	17.0	10.7	6.13	44.3	24.3	8.88	19.2	0.04	3.8	0.14	15.3
CALC	158.4	100.0	29.5	100.0	31.0	100.0	7.86	100.0	0.18	100.0	0.16	100.0

CUM PRDGS	CUM Wt	WT %	CaO %	CUM %	SiO2 %	CUM %	Fe %	CUM %	WO3 %	CUM %	Sn %	CUM %
WO3 Ro C1	1.9	1.2	29.1	1.2	17.5	0.7	13.3	2.0	0.15	1.0	0.10	0.8
C2	14.3	9.0	57.4	17.6	8.24	2.4	4.00	4.6	0.49	24.4	0.07	4.3
C3	26.0	16.4	55.5	30.9	9.30	4.9	3.74	7.8	0.49	44.4	0.08	8.6
C4	34.1	21.5	53.6	39.1	11.1	7.7	4.04	11.1	0.48	56.9	0.09	12.9
C5	52.1	32.9	47.4	52.8	16.0	17.0	5.25	22.0	0.42	75.7	0.11	23.8
C6	67.4	42.6	43.4	62.6	19.6	26.8	6.03	32.7	0.36	84.3	0.13	35.0
C7	84.6	53.4	40.3	72.9	22.7	39.1	6.72	45.7	0.31	90.3	0.15	50.4
C8	103.7	65.5	37.4	83.0	25.0	52.9	7.28	60.7	0.26	94.3	0.15	64.4
C9	131.5	83.0	33.4	93.9	28.3	75.7	7.65	80.8	0.21	96.2	0.16	84.7
CALC FEED	158.4	100.0	29.5	100.0	31.0	100.0	7.86	100.0	0.18	100.0	0.16	100.0



Amtec Burnie
ROUGHER/CLEANER FLOTATION REPORT SHEET

Milling		Primary	Regrind
Mill	type	0	
Media	type	0	
Media	kg	0	
Solids	g	0	
Water	g	0	
Time	min	0	
Speed	rpm	0	
Lime	g	0.0	
End pH	pH	0.0	
End p80	µm	0	



PROJECT	T0728
TEST NO	T08
DATE	160712
TECHNICIAN	SZ

PRODUCT FLOATED
Molna ore
None mags U/F

NOTES
WO3 Rougher
FS2 with 10% OT75

Float Cell	Volume	Regrind	Power
Rougher	2.7	Start	
0		Finish	
0		W/h	
Speed	800	kWh/t	

	pH	1 Na2SiO3 g/t	5 SodaAsh g/t	100.0 STP g/t	1.0 FS2 g/t	Cum FS2 g/t	0.25 Guar g/t	1 Dextrin g/t	100 MIBC g/t		Cond Time min	Air L/min	Float Time min	Cum Float Time	Wet Wt g	Con % Solids
Condition	8.5										2					
Condition	8.5	1143						416			2					
Condition	10.0			416							1					
Condition	9.9				416	416					5					
Ro C1	9.7				416	831			42		7	3-8	3.0	3.0	150	37
Condition	9.6	831			416							3-8	2.0	5.0	100	30
Ro C2	9.8										2					
Condition	9.7	831			624	1455						3-10	2.0	7.0	100	14
Ro C3	9.7							416			7					
Condition	9.3				416	1871			21		5	5-10	2.0	9.0	200	5
Ro C4	9.3										5					
Condition	8.5				416	2286					5	5-10	3.0	12.0	200	10
Ro C5	8.5										5					
Condition	7.9				416	2702					5	5-10	3.0	15.0	300	21
Ro C6	7.9				312	3014					5	5-10	4.0	19.0	500	14
Ro C7	7.8								21							
REAGENT TOTALS (g/t)		2806		416	3014	12575		831	83							

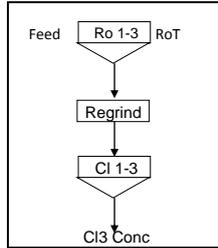
PRODUCTS	WT g	WT %	CaO %	DIST	SiO2 %	DIST	Fe %	DIST	WO3 %	DIST	Sn %	DIST
WO3 Ro C1	55.9	11.6	63.6	24.7	5.84	2.1	1.79	2.6	0.46	29.9	0.06	4.3
C2	29.5	6.1	52.9	10.9	13.5	2.6	3.78	2.9	0.45	15.4	0.09	3.4
C3	13.7	2.8	40.4	3.85	21.4	1.9	5.76	2.1	0.41	6.5	0.11	1.9
C4	9.4	2.0	32.0	2.09	26.6	1.6	7.30	1.8	0.33	3.6	0.14	1.7
C5	19.1	4.0	31.3	4.16	28.4	3.6	7.68	3.9	0.37	8.2	0.13	3.2
C6	61.8	12.8	27.8	12.0	32.3	13.1	9.05	14.8	0.09	6.5	0.19	15.1
C7	69.4	14.4	25.3	12.2	36.8	16.8	9.91	18.1	0.05	4.0	0.21	18.8
RoT	222.3	46.2	19.5	30.1	39.7	58.1	9.17	53.8	0.10	25.8	0.18	51.5
CALC	481.1	100.00	29.9	100.0	31.6	100.0	7.88	100.0	0.18	100.0	0.16	100.0

CUM PRODS	CUM Wt	WT %	CaO %	CUM	SiO2 %	CUM	Fe %	CUM	WO3 %	CUM	Sn %	CUM
WO3 Ro C1	55.9	11.6	63.6	24.7	5.84	2.1	1.79	2.6	0.46	29.9	0.06	4.3
C2	85.4	17.8	59.9	35.6	8.49	4.8	2.48	5.6	0.46	45.3	0.07	7.7
C3	99.1	20.6	57.2	39.5	10.3	6.7	2.93	7.7	0.45	51.8	0.08	9.7
C4	108.5	22.6	55.0	41.6	11.7	8.3	3.31	9.5	0.44	55.5	0.08	11.4
C5	127.6	26.5	51.5	45.7	14.2	11.9	3.96	13.3	0.43	63.7	0.09	14.6
C6	189.4	39.4	43.7	57.7	20.1	25.1	5.62	28.1	0.32	70.1	0.12	29.7
C7	258.8	53.8	38.8	69.9	24.6	41.9	6.77	46.2	0.25	74.2	0.15	48.5
CALC FEED	481.1	100.00	29.9	100.0	31.6	100.0	7.88	100.0	0.18	100.0	0.16	100.0

A11 FLUORITE FLOTATION



Milling		Primary	Regrind
Mill	type	0	
Media	type	0	
Media	kg	0	
Solids	g	0	
Water	g	0	
Time	min	0	
Speed	rpm	0	
Lime	g	0.0	
End pH	pH	0.0	
End p80	µm	0	



PROJECT	T0728
TEST NO	T09
DATE	230712
TECHNICIAN	SZ

PRODUCT FLOATED
Moina ore
None mags U/F

NOTES
FS2 with 10% OT75 Regrind: 2min/100g

Float Cell	Volume	Regrind Power
Rougher	2.7	Start
Cleaner	1.2	Finish
0		W/h
Speed	800	kWh/t

	pH	1 Dextrin g/t	1 ASS g/t	1 Na2CO3 g/t	1.0 FS2 g/t	100 MIBC g/t				Cond Time min	Air L/min	Float Time min	Cum Float Time	Wet Wt g	Con % Solids
Condition	8.0	443	443							7					
Ro C1	8.4			89	111	89				5	3-9	3.0	3.0	350	
Condition	8.0	443	443							7					
Ro C2	8.0			66	111					5	3-9	1.5	4.5	130	
Condition	8.0									5	3-9	1.5	6.0	100	
Ro C3	8.0				111					5	3-9	1.5	6.0	100	
Regrind	8.0									5					
Condition	8.0	443	443			89				5	2-6	3.0	9.0	400	
Cl1	8.0									5					
Condition	8.0	443	443							1	3-9	3.0	12.0	250	
Cl2	8.0			133						5					
Condition	8.0	443	443							1	3-9	3.0	12.0	250	
Cl3	8.1			222						1	2-7	3.0	15.0	230	
REAGENT TOTALS (g/t)		2216	2216	510	332	177									

PRODUCTS	WT g	WT %	CaO %	DIST	SiO2 %	DIST	Fe %	DIST	WO3 %	DIST	F %	DIST	Sn %	DIST	CaF2 %	DIST
CaF2 Cl3 Con	95.6	21.2	61.3	44.3	7.40	4.9	2.27	5.7	0.42	50.4	41.2	58.7	0.06	8.3	84.6	58.7
Cl3 Tail	10.8	2.4	39.3	3.21	23.3	1.7	7.32	2.1	0.50	6.8	26.4	4.3	0.12	1.9	54.2	4.3
Cl2 Tail	21.6	4.8	33.3	5.44	27.6	4.1	7.95	4.5	0.31	8.4	7.90	2.5	0.15	4.7	16.2	2.5
Cl1 Tail	96.7	21.4	22.5	16.4	36.5	24.5	11.2	28.3	0.12	14.6	3.30	4.8	0.16	22.5	6.77	4.8
Ro Tail	226.5	50.2	17.9	30.6	41.2	64.7	10.1	59.5	0.07	19.9	8.80	29.7	0.19	62.6	18.1	29.7
CALC	451.2	100.0	29.3	100.0	32.0	100.0	8.48	100.0	0.18	100.0	14.9	100.0	0.15	100.0	30.5	100.0
ASSAY			0.00		0.00		0.00		0.00		0.00		0.00		0.00	

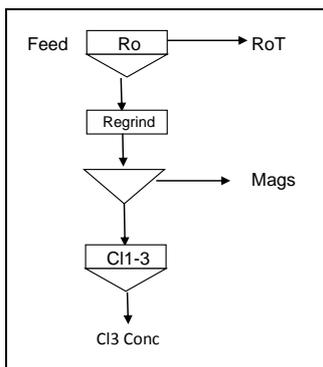
CUM PRODS	CUM Wt	WT %	CaO %	CUM	SiO2 %	CUM	Fe %	CUM	WO3 %	CUM	F %	CUM	Sn %	CUM	CaF2 %	CUM
CaF2 Cl3 Con	95.6	21.2	61.3	44.3	7.40	4.9	2.27	5.7	0.42	50.4	41.2	58.7	0.06	8.3	84.6	58.7
Cl3 Tail	106.4	23.6	59.1	47.5	9.01	6.7	2.78	7.7	0.43	57.1	39.7	63.0	0.07	10.2	81.5	63.0
Cl2 Tail	128.0	28.4	54.7	52.9	12.2	10.8	3.65	12.2	0.41	65.6	34.3	65.5	0.08	14.9	70.5	65.5
Cl1 Tail	224.7	49.8	40.9	69.4	22.6	35.3	6.90	40.5	0.28	80.1	21.0	70.3	0.11	37.4	43.1	70.3
Ro Tail	451.2	100.0	29.3	100.0	32.0	100.0	8.48	100.0	0.18	100.0	14.9	100.0	0.15	100.0	30.5	100.0



ALS Ammtec Burnie
ROUGHER/CLEANER FLOTATION REPORT SHEET

Milling		Primary	Regrind
Mill	type	0	
Media	type	0	
Media	kg	0	
Solids	g	0	
Water	g	0	
Time	min	0	
Speed	rpm	0	
Lime	g	0.0	
End pH	pH	0.0	
End p80	µm	0	

Float Cell	Volume	Regrind Power
Rougher	2.7	Start
Cleaner	1.2	Finish
0		W/h
Speed	800	kWh/t



PROJECT	T0728
TEST NO	T10
DATE	02/08/12
TECHNICIAN	SZ

PRODUCT FLOATED
Moina ore
None mags U/F

NOTES
Feed 540g damp cake FS2 with 10% OT75
Regrind: 2min/100g
No F assay results

	pH	1 Dextrin g/t	2 ASS g/t	100 Na2CO3 g/t	1.0 FS2 g/t	100 MIBC g/t			Cond Time min	Air L/min	Float Time min	Cum Float Time	Wet Wt g	Con % Solids
Condition Ro C1	8.4	419	838		84	84			7					
Condition Ro C2	8.0	419	838	42	63				5	1-7	3.0	3.0	310	
Condition Ro C3	8.0				63				5	3-9	1.5	4.5	100	
Condition Regrind Lims	8.0								5	3-9	1.5	6.0	45	
Condition Cl1	8.8	419	838						7					
Condition Cl2	8.1					84				2-7	4.0	10.0	270	
Condition Cl3	8.7	419	838						1	3-9	3.0	13.0	210	
Condition Cl3	8.7	419	838						7					
Condition Cl3	10.8			3288					1	2-7	3.5	16.5	200	
REAGENT TOTALS (g/t)		838	1675	42	209	84								

PRODUCTS	WT g	WT %	CaO %	DIST	SiO2 %	DIST	Fe %	DIST	WO3 %	DIST	F %	DIST	Sn %	DIST
CaF2 Cl3 Con	82.1	17.2	64.7	37.4	5.07	2.7	1.37	3.0	0.21	21.3		#DIV/0!	0.05	5.5
Cl3 Tail	21.3	4.5	45.9	6.88	18.1	2.5	4.61	2.6	1.60	42.2		#DIV/0!	0.10	2.9
Cl2 Tail	24.6	5.2	36.0	6.23	25.3	4.1	6.52	4.2	0.24	7.3		#DIV/0!	0.15	4.9
Cl1 Tail	67.0	14.0	21.7	10.2	35.5	15.6	9.57	16.9	0.08	6.6		#DIV/0!	0.17	15.2
Ro con mags	5.4	1.1	37.9	1.44	21.5	0.8	10.5	1.5	0.29	1.9		#DIV/0!	0.14	1.0
Ro Tail	277.1	58.0	19.4	37.8	40.8	74.3	9.86	71.9	0.06	20.6		#DIV/0!	0.19	70.5
CALC	477.5	100.0	29.8	100.0	31.9	100.0	7.96	100.0	0.17	100.0	0.00	#DIV/0!	0.16	100.0
ASSAY			0.00		0.00		0.00		0.00		0.00		0.00	

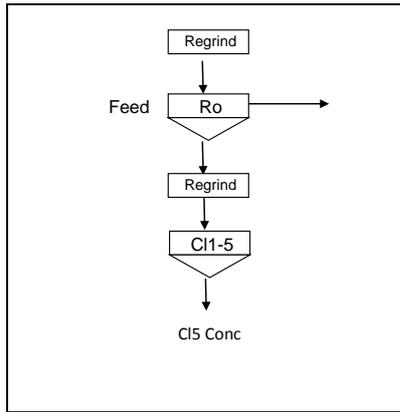
CUM PRODS	CUM Wt	WT %	CaO %	CUM	SiO2 %	CUM	Fe %	CUM	WO3 %	CUM	F %	CUM	Sn %	CUM
CaF2 Cl3 Con	82.1	17.2	64.7	37.4	5.07	2.7	1.37	3.0	0.21	21.3		#DIV/0!	0.05	5.5
Cl3 Tail	103.4	21.7	60.8	44.3	7.74	5.3	2.04	5.5	0.50	63.5		#DIV/0!	0.06	8.3
Cl2 Tail	128.0	26.8	56.1	50.5	11.1	9.3	2.90	9.8	0.45	70.8		#DIV/0!	0.08	13.3
Cl1 Tail	195.0	40.8	44.3	60.7	19.5	25.0	5.19	26.6	0.32	77.5		#DIV/0!	0.11	28.5
Ro con mags	200.4	42.0	44.1	62.2	19.5	25.7	5.33	28.1	0.32	79.4		#DIV/0!	0.11	29.5
Ro Tail	477.5	100.0	29.8	100.0	31.9	100.0	7.96	100.0	0.17	100.0		#DIV/0!	0.16	100.0



ALS Ammtec Burnie
ROUGHER/CLEANER FLOTATION REPORT SHEET

Milling		Reg 1	Reg 2
Mill	type		
Media	type		
Media	kg		
Solids	g	460	156
Water	g	307	77
Time	min	4'36"	3'07"
Speed	rpm	50	50
Lime	g	0.0	0.0
End pH	pH	0.0	0.0
End p80	µm	0	0

Float Cell	Volume	Regrind Power
Rougher	2.7	Start
Cleaner	1.2	Finish
0		W/h
Speed	800	kWh/t



PROJECT	T0728
TEST NO	T12
DATE	080812
TECHNICIAN	SZ

PRODUCT FLOATED
Moina ore
None mags U/F

NOTES
FS2 with 10% OT75
Regrind1 : 1min/100g
Regrind2 : 2min/100g

	pH	1 Dextrin g/t	2 ASS g/t	100 Na2CO3 g/t	1.0 FS2 g/t	100 MIBC g/t			Cond Time min	Air L/min	Float Time min	Cum Float Time	Wet Wt g	Con % Solids
Regrind 1														
Condition	8.1	420	841	105				7						
Ro C1	8.0				84	84		5	1-7	2.5	2.5		230	
Condition	8.1	420	841					7						
Ro C2	8.0				84			5	3-9	1.5	4.0		140	
Condition	8.0													
Ro C3	8.0				63			5	3-9	1.5	5.5		50	
Regrind 2														
Condition	8.0	420	841	21				7						
Cl1	8.1					84				2-7	4.0	9.5	270	
Condition	8.7	420	841	105				7						
Cl2	8.5							1	3-9	3.0	12.5		220	
Condition	8.7	420	841	2985				7						
Cl3	10.8							1	2-7	6.0	18.5		180	
Condition		420	1261	3994						1-5	4.5	23.0	130	
Cl4	10.8													
Condition		420	1682	4835				7						
Cl5	10.8							1	2-7	3.5	26.5		90	
REAGENT TOTALS (g/t)		841	1682	105	231	84								

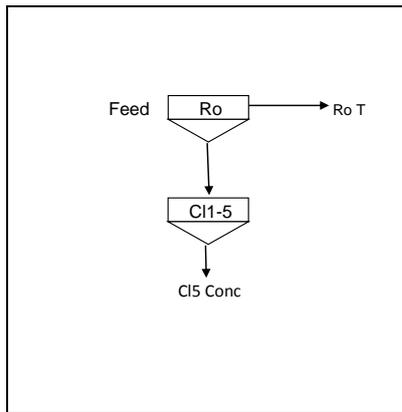
PRODUCTS	WT g	WT %	CaO %	DIST	SiO2 %	DIST	Fe %	DIST	WO3 %	DIST	F %	DIST	CaF2 %	DIST
CaF2 Cl5 Con	36.8	7.7	70.2	18.3	2.84	0.7	1.08	1.0	0.05	2.2	47.1	29.2	96.7	29.2
Cl5 Tail	15.7	3.3	66.7	7.43	5.94	0.6	1.64	0.6	0.16	3.0	40.6	10.7	83.3	10.7
Cl4 Tail	19.3	4.1	65.0	8.90	9.76	1.2	2.54	1.2	0.42	9.6	35.4	11.5	72.7	11.5
Cl3 Tail	16.5	3.5	49.2	5.76	15.4	1.7	3.92	1.6	0.98	19.2	22.1	6.1	45.4	6.1
Cl2 Tail	26.8	5.6	39.0	7.41	24.5	4.3	6.45	4.3	0.36	11.4	22.1	10.0	45.4	10.0
Cl1 Tail	42.6	9.0	20.5	6.19	35.6	10.0	11.0	11.8	0.11	5.6	5.70	4.1	11.7	4.1
Ro Tail	318.0	66.8	20.4	46.0	38.7	81.4	9.93	79.4	0.13	49.0	5.30	28.4	10.9	28.4
CALC	475.7	100.0	29.6	100.0	31.8	100.0	8.36	100.0	0.18	100.0	12.5	100.0	25.6	100.0
ASSAY			0.00		0.00		0.00		0.00		0.00		0.00	

CUM PRODS	CUM Wt	WT %	CaO %	CUM	SiO2 %	CUM	Fe %	CUM	WO3 %	CUM	F %	CUM	CaF2 %	CUM
CaF2 Cl5 Con	36.8	7.7	70.2	18.3	2.84	0.7	1.08	1.0	0.05	2.2	47.1	29.2	96.7	29.2
Cl5 Tail	52.5	11.0	69.2	25.7	3.77	1.3	1.25	1.6	0.08	5.2	45.2	39.9	92.7	39.9
Cl4 Tail	71.8	15.1	68.0	34.6	5.38	2.6	1.59	2.9	0.17	14.8	42.5	51.4	87.3	51.4
Cl3 Tail	88.3	18.6	64.5	40.4	7.25	4.2	2.03	4.5	0.32	34.0	38.7	57.6	79.5	57.6
Cl2 Tail	115.1	24.2	58.6	47.8	11.3	8.6	3.06	8.8	0.33	45.4	34.8	67.5	71.5	67.5
Cl1 Tail	157.7	33.2	48.3	54.0	17.8	18.6	5.20	20.6	0.27	51.0	27.0	71.6	55.4	71.6
Ro Tail	475.7	100.0	29.6	100.0	31.8	100.0	8.36	100.0	0.18	100.0	12.5	100.0	25.6	100.0



Milling		Reg 1	Reg 2
Mill	type		
Media	type		
Media	kg		
Solids	g		
Water	g		
Time	min		
Speed	rpm		
Lime	g		
End pH	pH		
End p80	µm		

Float Cell	Volume	Regrind	Power
Rougher	2.7	Start	
Cleaner	1.2	Finish	
0		W/h	
Speed	800	kWh/t	



PROJECT	T0728
TEST NO	T13
DATE	230812
TECHNICIAN	SZ

PRODUCT FLOATED
Moina ore
CS2-CS5

NOTES
FS2 with 10% OT75

	pH	1 Dextrin g/t	2 ASS g/t	100 Na2CO3 g/t	1.0 FS2 g/t	100 MIBC g/t	98 H2SO4 g/t		Cond Time min	Air L/min	Float Time min	Cum Float Time	Wet Wt g	Con % Solids
Condition	8.3	584	1168	146					7					
Ro C1	8.0				117	117			5	1-7	2.0	2.0	230	
Condition	8.1	584	1168	204					7					
Ro C2	8.0				117				4	4-9	1.5	3.5	90	
Condition	8.0													
Ro C3	8.0				88				3	3-9	1.5	5.0	30	
Condition	8.0	584	1168	204					7					
Cl1	8.0					58				2-9	4.0	9.0	340	
Condition	8.3	584	1168	292					7					
Cl2	8.0					29								
Condition	8.6	584	1168	12234					1	3-9	3.5	12.5	250	
Cl3	10.8								7					
Condition	10.8	584	1752	11942					1	2-7	4.5	17.0	300	
Cl4	10.8					58				1-5	5.0	22.0	310	
Condition	2.5	584	2336						7					
Cl5	2.5						2146		1	2-5	3.0	25.0	220	34
REAGENT TOTALS (g/t)		4088	9927	25022	321	263	2146							

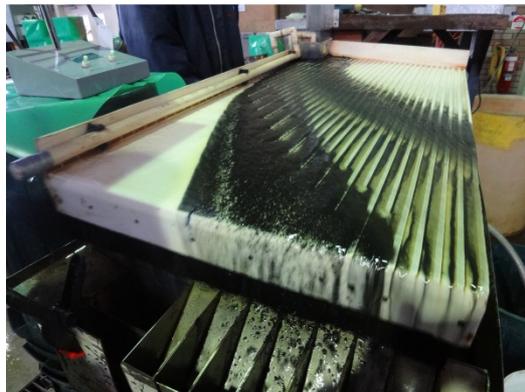
PRODUCTS	WT g	WT %	CaO %	DIST	SiO2 %	DIST	Fe %	DIST	WO3 %	DIST	F %	DIST	CaF2 %	DIST
CaF2 Cl5 Con	75.5	22.0	72.6	46.4	1.98	1.5	0.64	1.8	0.07	9.1	47.1	61.5	96.7	61.5
Cl5 Tail	25.1	7.3	66.5	14.1	5.35	1.4	1.55	1.4	0.19	8.2	43.9	19.1	90.1	19.1
Cl4 Tail	3.8	1.1	50.3	1.62	15.4	0.6	4.03	0.6	1.73	11.3	27.0	1.8	55.4	1.8
Cl3 Tail	6.7	2.0	43.5	2.47	17.7	1.2	5.06	1.2	3.46	39.8	24.0	2.8	49.3	2.8
Cl2 Tail	16.3	4.8	39.3	5.43	23.7	4.0	6.40	3.8	0.60	16.8	20.9	5.9	42.9	5.9
Cl1 Tail	12.1	3.5	21.1	2.16	36.0	4.5	11.0	4.9	0.21	4.4	6.00	1.3	12.3	1.3
Ro Tail	203.0	59.3	16.2	27.8	41.4	86.8	11.6	86.3	0.03	10.5	2.20	7.7	4.52	7.7
CALC	342.5	100.0	34.5	100.0	28.3	100.0	7.97	100.0	0.17	100.0	16.9	100.0	34.6	100.0

CUM PRODS	CUM Wt	WT %	CaO %	CUM	SiO2 %	CUM	Fe %	CUM	WO3 %	CUM	F %	CUM	CaF2 %	CUM
CaF2 Cl5 Con	75.5	22.0	72.6	46.4	1.98	1.5	0.64	1.8	0.07	9.1	47.1	61.5	96.7	61.5
Cl5 Tail	100.6	29.4	71.1	60.6	2.82	2.9	0.87	3.2	0.10	17.3	46.3	80.6	95.0	80.6
Cl4 Tail	104.4	30.5	70.3	62.2	3.28	3.5	0.98	3.8	0.16	28.6	45.6	82.3	93.6	82.3
Cl3 Tail	111.1	32.4	68.7	64.6	4.14	4.8	1.23	5.0	0.36	68.4	44.3	85.1	90.9	85.1
Cl2 Tail	127.4	37.2	64.9	70.1	6.65	8.7	1.89	8.8	0.39	85.2	41.3	91.0	84.8	91.0
Cl1 Tail	139.5	40.7	61.1	72.2	9.19	13.2	2.68	13.7	0.37	89.5	38.2	92.3	78.5	92.3
Ro Tail	342.5	100.0	34.5	100.0	28.3	100.0	7.97	100.0	0.17	100.0	16.9	100.0	34.6	100.0

**APPENDIX 3 – Combined Process Testwork on Moina Deposit. Report No:
P1307 Mintrex.**

TNT MINES LIMITED

COMBINED PROCESS TESTWORK ON MOINA DEPOSIT



Rev	Date	Description	Prep	Check	PM Approval
0	23/7/12	Draft Report	BCP		

EXECUTIVE SUMMARY

A composite sample of ore from Moina, which is a complex magnetite, scheelite, fluorite deposit containing Bi and Sn, has been tested at AMMTEC in Burnie and in Perth. The Burnie lab concentrated on the non magnetic fraction while the Perth laboratory concentrated on the magnetic fraction.

The aim of the testwork was to assess the separation of the various elements from the magnetite stream, assess the final grade and quality of magnetite and to assess the ability to upgrade the scheelite, fluorite and hopefully Sn to a marketable grade.

The testwork indicates that down to 1mm that there is little liberation of the valuable minerals. Both the magnetic separation testwork and gravity testwork indicates a split of elements following weight splits. At 500micron the scheelite and fluorite start to preferentially report to non magnetics with 65% of the CaF₂ and 70% of the W reports to non magnetics but only 53% of the weight. By contrast the Sn follows the weight split down to a grind of 75micron.

Gravity testwork at 106micron shows a sectional yield of 69% of the W reporting to a gravity concentrate of 5.5%WO₃ which is expected to upgrade relatively easily to a marketable grade of 65%WO₃. The laboratory net yield of WO₃ from ore is 32.2% with a value of \$18.70/t ore but with considerable scope for improvement.

The fluorite can be upgraded by flotation at the same size – producing some material at 80%CaF₂ but averaging 60%CaF₂ at reasonable yields. The expectation is that this 60.2% grade can be increased to the normal market range of 75 to 80% with less than 10% losses.

The estimated laboratory fluorite overall yield is 31.6% to a product at 80% CaF₂ and has a value of \$25.50/t in ore. Again there is plenty of scope for improvement.

The magnetite needs to be ground to 80% -20micron to reach even a minimum grade of 58%Fe and this with impurities of Sn and alkali minerals (K₂O and Na₂O). Assuming that a market can be found this material is worth \$23.75/t of ore in the ground. A second test sample directly from drill core gave superior results with magnetite weight yield increasing to 31% from 25% and magnetite grade reaching 63%Fe in concentrate.

The overall recoverable valuable minerals based on this preliminary testwork have a value of \$67.85/t ore with no allowance for Sn and Bi. This value is regarded as a conservative value with considerable scope for enhancing scheelite and fluorite yield.

It is recommended that gravity work be conducted on the non magnetics at 500micron to assist in recovering more scheelite and float work to prove that the fluorite reaches a marketable grade. An iron ore marketing expert should be consulted on the marketability of the magnetite given the level of impurities in the final product.

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1.0 INTRODUCTION

TNT Mines have access to a tungsten and fluorspar resource at Moina in northern Tasmania which has significant magnetite content and smaller quantities of Sn and Bi. Some preliminary testwork has been conducted on this material which paralleled a concept study to determine the possible viability of the resource.

Two testwork programmes have been developed on a composite sample of samples selected from available drill core.

- One series of tests were conducted in Burnie looking at the tungsten and fluorspar recovery and grades
- A second series of tests, conducted in Perth, treated the ore resource as a magnetite resource and then assessed the ability to separate magnetite from the other minerals.

While these two programmes addressed the issues from a different perspectives the results overlap to a certain extent and provide confirmatory analysis

2.0 TEST PROGRAMME

2.1 Sample Source

The laboratory in Burnie was working on a composite sample made up of subsamples of each of four drill holes. These were combined into a single sample and crushed to -3mm for testing.

A subsample of this, called COMP1, was sent to the AMMTEC Perth laboratory for testwork on the magnetite side.

In addition, since the magnetite work needed some results at coarser sizes and there was a desire to measure the variability of grade and concentrate of magnetite once ground to a given size, 6 subsamples were taken from the original drill core for DTR (Davis Tube Recovery) work. Unfortunately due to some miscommunication with the Perth Laboratory these coarse samples were crushed and combined as feed stock for the dry magnetic magnetite work so the extra value of testing variability was lost. However, the combined sample did provide a different reference point of data from the COMP1.

2.2 Gravity and float W and CaF₂ Testwork

The testwork concentrated on three aspects:

- Coarse size separation of material – aiming at a rod mill product size. A gravity separation using heavy liquid was tested to see if there was a split of valuables and a clean tails
- Testwork to determine the release of tungsten from the magnetite – and upgrading work to produce an upgraded tungsten product
- Testwork to determine the release of fluorite and its upgrading by flotation

2.3 Magnetite Testwork

The major part of a testwork programme for magnetite is the relationship between grade and grind – and the size at which tailings is released. A grade/grind curve is developed between 80% passing 500micron and 80% passing 25micron:

This data can be used to:

- Determine the best grind to achieve a sellable grade
- Indicate the size at which clean tailings can be produced – and the proportion thereof
- Determine the size at which the non magnetic element fraction reports to non magnetics

The latter is an important issue for a project like this with values in the magnetic fraction and values in the non magnetic fraction since this then determines the type of equipment which can be used for product concentration.

The most useful tool on the magnetite side is a Davis Tube which can rapidly and accurately separate out a magnetic fraction. The result can be directly scaled to a real plant and is not so susceptible to operator error as a laboratory scale magnetic separator. The machine consists of a glass tube containing the sample which is rotated and oscillated in a magnetic field to allow the collection of the magnetic fraction and its cleaning.



Laboratory Davis Tube

The Davis Tube is used in two ways:

- **DTR – Davis Tube Recovery.**

This is a two stage process where a sample is stage ground to a particular size and the ground product is washed in the Davis Tube. Normally the most important part is the pulverizing which is designed to give a constant size regardless of ore hardness.

For this project the DTR was carried out at 100% passing 38micron – which normally gives 80% passing 25 micron. In reality the sample were slightly overpulverized and was more closely measured at 80% passing 20micron.

- **DTW – Davis Tube Wash**

In this test the sample is ground to a predetermined size and then the sample is washed in the Tube. This can be carried out with samples as coarse as 80% passing 500micron all the way to 80% passing 25micron. This work was carried on this range with dry magnetic separation used to cover coarser fractions.

2.4 Magnetite Testwork Procedure

2.4.1 DTRs

DTR testwork was carried in duplicate on the ore samples received – such samples providing a reference for the basic ore quality.

2.4.2 Grind Curve

Samples of the COMP1 sample were ground to 80% passing 500micron, 250micron, 150micron, 75micron, 45micron, 32micron and 25micron. The samples were DTW'd to produce a concentrate for each sizes which was then assayed.

The data was also analysed to determine yield of each element to magnetics/non magnetics

2.4.3 Coarse Size DMS

Samples crushed to -4mm and -2mm – nominally 80% passing 3mm and 1.5mm were dry magnetically separated. The concentrate was then pulverized to allow a determination of the magnetic Fe and hence magnetic Fe yield.

Unfortunately this work was carried out on a different sample to COMP1 – and the results are significantly different in terms of weight yield and product grade.

2.4.4 Tabling of Products

This test was designed to simulate a preconcentration step at 2mm – which might be the product of a rod mill. A sample was crushed to below 2mm – part was dry magnetically separated to produce a magnetic concentrate and a non magnetic fraction. Tabling tests were then carried out on the full crushed material and the non magnetic fraction. The concept was to test whether the rod mill product would be sent directly to a gravity separation stage or whether a magnetic concentration would need to be carried out first to avoid overcrowding of the heavies.

3.0 RESULTS AND ANALYSIS

The detailed results are in the attached ALS-AMMTEC report. This section of the report covers the critical results and their interpretation.

3.1 DTR results

The core sample was significantly different from COMP1 as can be seen below – the magnetic Fe is 20.1% compared to 14.3% and for a given grind gives a higher grade and higher weight recovery (31.7%) with a grade of 63.2%Fe containing 77% of the Fe. The non magnetics contained 95% of the CaF₂, 82% of the Sn, 93% of the W and 90% of the Bi.

The critical results for COMP1 at a grind of 100% passing 38micron – which was subsequently measured at 80% passing 20micron – were that the sample had an average weight recovery of 24.1% to a concentrate of grade 56.9Fe, 8.4%SiO₂ and containing 70% of the Fe while the non magnetics contained 90% of the CaF₂, 80% of the Sn, 92% of the W and 93% of the Bi.

In the table below the core results are the top table and then the results of duplicate tests on COMP1.

DTR testwork on Drill Core and COMP1

A14451-DTR(3000 gauss)-P100 38µm CORE -TNT MINES											
LIMS	Wt.	Fe	SiO ₂	Al ₂ O ₃	CaO	LOI1000	Sn	W	Bi	F	Mag Fe
PRODUCT	DISTn.	Grade	Grade	Grade	Grade	Grade	Grade	Grade	Grade	Grade	
	(%)	(%)	(%)	(%)	(%)	(%)	(%)	ppm	ppm	%	
DT MAGS	31.7	63.20	5.32	1.97	2.600	-2.17	0.100	140.0	150.0	0.94	20.1
DT N-MAGS	68.2	8.69	30.35	11.44	24.43	3.06	0.20	828.64	589.56	9.51	
TOTAL	100.0	26.0	22.4	8.43	17.500	1.40	0.170	610.0	450.0	6.79	
A14451-DTR(3000 gauss)-P100 38µm COMP1 SAMPLE A-TNT MINES											
LIMS	Wt.	Fe	SiO ₂	Al ₂ O ₃	CaO	LOI1000	Sn	W	Bi	F	
PRODUCT	DISTn.	Grade	Grade	Grade	Grade	Grade	Grade	Grade	Grade	Grade	
	(%)	(%)	(%)	(%)	(%)	(%)	(%)	ppm	ppm	%	
DT MAGS	25.1	57.00	8.44	3.16	5.520	-1.50	0.098	250.0	180.0	1.49	14.3
DT N-MAGS	74.9	7.06	30.69	11.07	28.59	1.50	0.15	1505.21	834.31	8.97	
TOTAL	100.0	19.6	25.1	9.08	22.800	0.75	0.140	1190.0	670.0	7.09	
A14451-DTR(3000 gauss)-P100 38µm COMP1 SAMPLE B -TNT MINES											
LIMS	Wt.	Fe	SiO ₂	Al ₂ O ₃	CaO	LOI1000	Sn	W	Bi	F	
PRODUCT	DISTn.	Grade	Grade	Grade	Grade	Grade	Grade	Grade	Grade	Grade	
	(%)	(%)	(%)	(%)	(%)	(%)	(%)	ppm	ppm	%	
DT MAGS	23.1	56.80	8.38	3.19	5.540	-1.74	0.095	200.0	INS	INS	13.1
DT N-MAGS	76.8	8.41	30.13	10.85	27.99	1.50	0.15	1487.84	#####	####	
TOTAL	100.0	19.6	25.1	9.08	22.800	0.75	0.140	1190.0	670.0	7.09	

The two samples are significantly different – though nominally from the same source. The full chemistry is shown on the next page. The W from the drill core sample was only 610ppm rather than 1190ppm of COMP1 and the fluorite content is 25% lower.

A14451-DAVIS TUBE RECOVERY(3000 gauss)-P100 38µm CORE - TNT MINES																				
LIMS PRODUCT	PRODUCT WEIGHT (g)	Wt. DISTn. (%)	Fe Grade (%)	SiO ₂ Grade (%)	Al ₂ O ₃ Grade (%)	TiO ₂ Grade (%)	MnO Grade (%)	CaO Grade (%)	P Grade (%)	S Grade (%)	MgO Grade (%)	K ₂ O Grade (%)	Na ₂ O Grade (%)	Zn Grade (%)	LOI1000 Grade (%)	Sn Grade (%)	W Grade ppm	Bi Grade ppm	F Grade %	Mag Fe
DT MAGS	6.4	31.7	63.20	5.32	1.97	0.071	0.290	2.600	0.005	0.031	0.640	0.400	0.180	0.011	-2.17	0.100	140.0	150.0	0.94	20.1
DT N-MAGS	13.7	68.2	8.69	30.35	11.44	0.14	0.60	24.43	0.03	0.11	4.07	2.83	0.77	0.02	3.06	0.20	828.64	589.56	9.51	
TOTAL	20.0	100.0	26.0	22.4	8.43	0.120	0.500	17.500	0.020	0.087	2.980	2.060	0.580	0.015	1.40	0.170	610.0	450.0	6.79	
A14451-DAVIS TUBE RECOVERY(3000 gauss)-P100 38µm COMP1 SAMPLE A-TNT MINES																				
LIMS PRODUCT	PRODUCT WEIGHT (g)	Wt. DISTn. (%)	Fe Grade (%)	SiO ₂ Grade (%)	Al ₂ O ₃ Grade (%)	TiO ₂ Grade (%)	MnO Grade (%)	CaO Grade (%)	P Grade (%)	S Grade (%)	MgO Grade (%)	K ₂ O Grade (%)	Na ₂ O Grade (%)	Zn Grade (%)	LOI1000 Grade (%)	Sn Grade (%)	W Grade ppm	Bi Grade ppm	F Grade %	Mag Fe
DT MAGS	5.0	25.1	57.00	8.44	3.16	0.088	0.470	5.520	0.007	0.140	0.880	0.450	0.250	0.019	-1.50	0.098	250.0	180.0	1.49	14.3
DT N-MAGS	15.0	74.9	7.06	30.69	11.07	0.14	0.68	28.59	0.03	0.25	3.54	2.51	0.89	0.04	1.50	0.15	1505.21	834.31	8.97	
TOTAL	20.0	100.0	19.6	25.1	9.08	0.130	0.630	22.800	0.022	0.220	2.870	1.990	0.730	0.034	0.75	0.140	190.0	670.0	7.09	
A14451-DAVIS TUBE RECOVERY(3000 gauss)-P100 38µm COMP1 SAMPLE B - TNT MINES																				
LIMS PRODUCT	PRODUCT WEIGHT (g)	Wt. DISTn. (%)	Fe Grade (%)	SiO ₂ Grade (%)	Al ₂ O ₃ Grade (%)	TiO ₂ Grade (%)	MnO Grade (%)	CaO Grade (%)	P Grade (%)	S Grade (%)	MgO Grade (%)	K ₂ O Grade (%)	Na ₂ O Grade (%)	Zn Grade (%)	LOI1000 Grade (%)	Sn Grade (%)	W Grade ppm	Bi Grade ppm	F Grade %	Mag Fe
DT MAGS	4.6	23.1	56.80	8.38	3.19	0.087	0.470	5.540	0.007	0.120	0.890	0.450	0.230	0.018	-1.74	0.095	200.0	1487.84	INS	13.1
DT N-MAGS	15.4	76.8	8.41	30.13	10.85	0.14	0.68	27.99	0.03	0.25	3.47	2.45	0.88	0.04	1.50	0.15	1487.84	###	###	
TOTAL	20.0	100.0	19.6	25.1	9.08	0.130	0.630	22.800	0.022	0.220	2.870	1.990	0.730	0.034	0.75	0.140	190.0	670.0	7.09	

The critical issue is that the core sample magnetic concentrates may be an acceptable saleable product while the COMP1 concentrate is very marginal and will struggle to find a market.

The other issues with the magnetite are the high level of Sn in concentrate – 0.1% - the normal penalty level is 0.02%. The combined alkali metals (K₂O and Na₂O) are also high at 0.7% - 0.07% being a common target limit. It is not clear that the alkali metal contract limits are hard and fast – most ores have low values of these and these are reflected in contracts. Phosphorus levels are low <0.02% with normal contract limits at 0.07%.

Finally the silica and alumina levels are high for magnetite – normally the Al₂O₃ is below 0.5% and the silica is controlled to below 5% by grinding or reverse flotation.

3.2 Relationship with grind

The change of grade and weight recovery with grind is usually an important issue for the magnetite since it has a large impact on the plant design and operating cost. This is less critical in this project where the non magnetics may need grinding to the same size as the magnetite so the early production of a disposable tailings is difficult.

Magnetic Grades

p80	Wt.	Fe	SiO ₂	Al ₂ O ₃	CaO	MgO	K ₂ O	Na ₂ O	LOI-1000	Mag Fe
Feed	100.0	19.6	25.1	9.08	22.8	2.870	1.99	0.730	0.75	
500	46.8	32.30	19.40	7.14	16.7	2.25	1.35	0.600	0.36	15.1
250	43.9	34.60	18.50	6.84	15.5	2.16	1.28	0.570	0.38	15.2
150	41.8	35.10	18.00	6.76	15.3	2.15	1.28	0.560	0.52	14.7
75	34.9	40.70	15.60	5.92	12.7	1.90	1.12	0.490	0.16	14.2
45	28.6	48.70	12.10	4.58	8.9	1.45	0.85	0.370	-1.49	13.9
32	27.2	51.10	10.90	4.14	7.8	1.30	0.76	0.330	-1.40	13.9
25	27.2	51.70	10.80	4.07	7.6	1.28	0.74	0.320	-1.76	14.1

The improvement in grade as the grind increases can be clearly seen. The following table looks at the non magnetic fraction.

Non Magnetic Grades

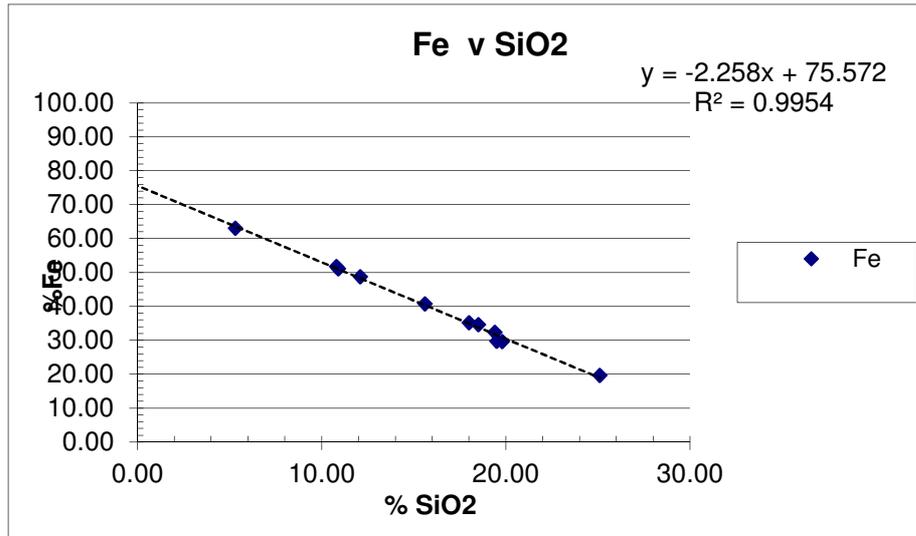
p80	Wt.	Fe	SiO ₂	Al ₂ O ₃	CaO	Snppm	Wppm	Bippm	F %
Feed	100.0	19.6	25.1	9.08	22.8	0.14	1190	670	7.1
500	53.2	8.42	30.12	10.79	28.2	0.14	1577	916	8.2
250	56.1	7.84	30.27	10.84	28.5	0.14	1613	929	8.3
150	58.2	8.49	30.19	10.74	28.2	0.14	1584	899	8.1
75	65.2	8.31	30.18	10.77	28.2	0.14	1543		8.3
45	71.4	7.93	30.31	10.88	28.4	0.15	1519	846	8.7
32	72.8	7.81	30.42	10.93	28.4	0.15	1504	842	8.7
25	72.8	7.61	30.44	10.95	28.6	0.15	1507	845	8.8

In this case there is little change in the non magnetic grade as the grind decreases but most valuable elements show a 20% increase in grade compared to the feed.

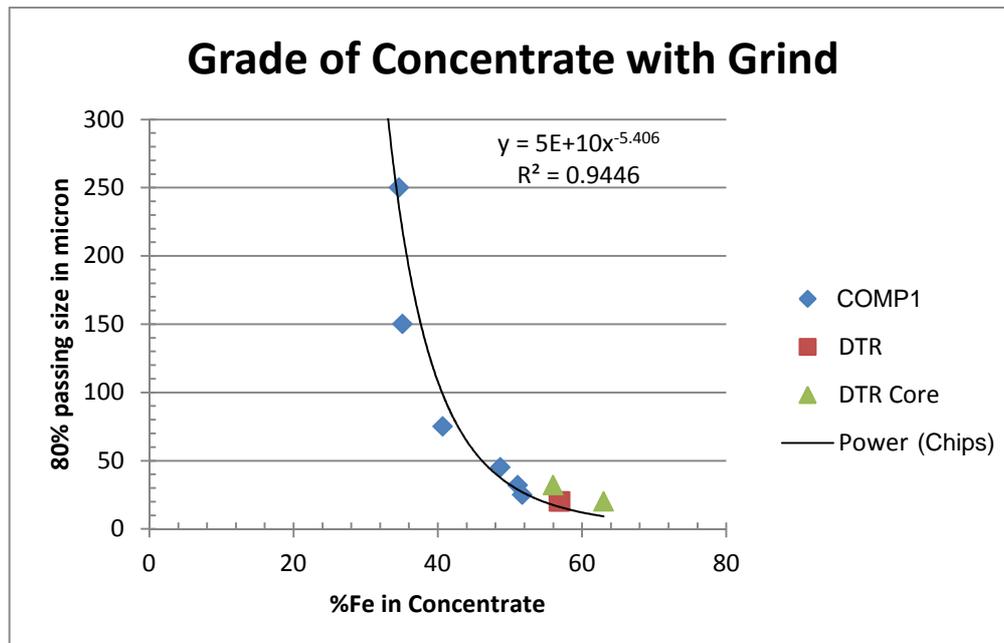
The data can be analysed in a number of ways.

3.3 Fe v SiO2

This is a commonly drawn graph. There is a clear relationship between Fe and SiO2 in concentrate. The trend line equation normally has an intercept on the Y axis of close to 72.4 – which is the Fe content in pure magnetite. It is unusual to see a value greater than 72.4% - presumably caused by the fact that not many samples have SiO2 values below 8% which would correct the graph. It is interesting to see that the feed sample at 25%SiO2 sits on the same graph.



3.4 Fe v Grind



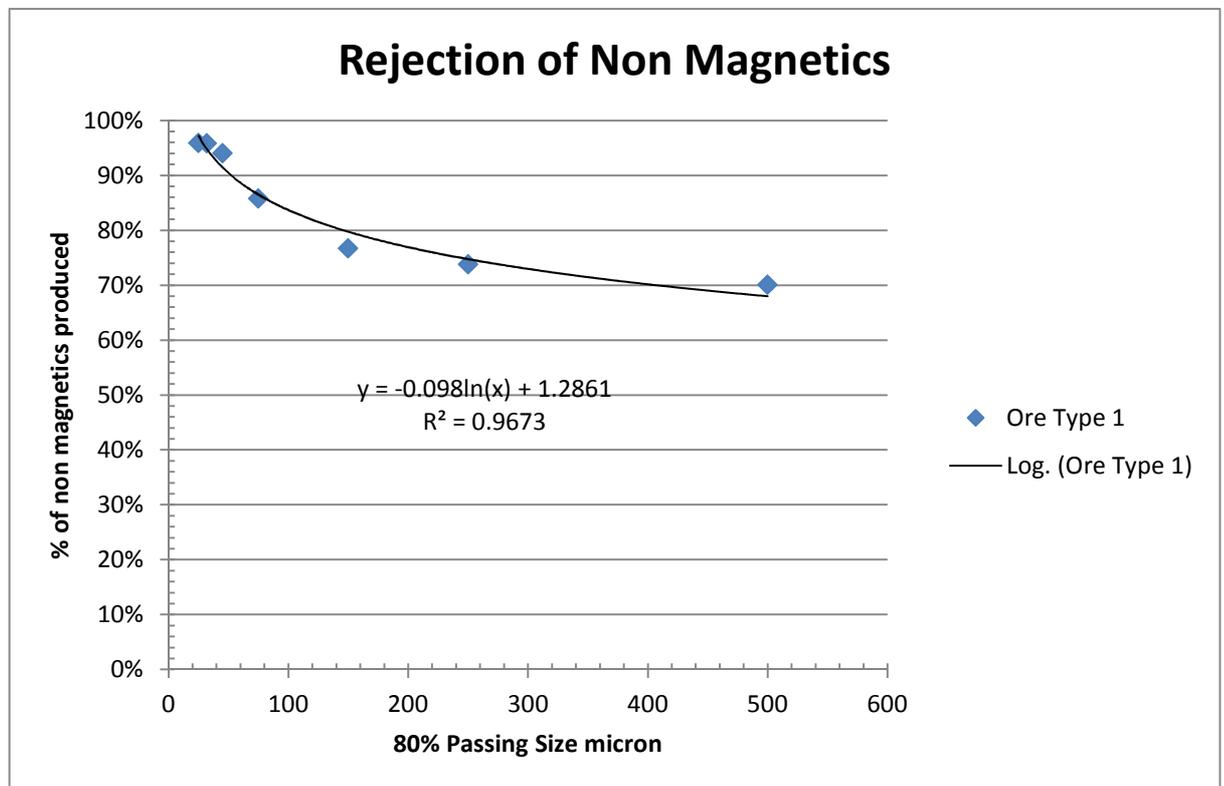
The different DTR points have also been plotted - the DTR from COMP1 sits on the line but the DTR from drill core (one is the DTR and one a DTW from the dry magnetic testwork) sit above the line – reflecting the better quality.

It can be seen that to achieve a minimum grade of 60%Fe that a grind of 80% passing 20micron is required – which is rather fine.

3.5 Production of Non Magnetics

This next graphs looks at the production of non magnetics relative to the more perfect DTR result at 20micron.

It can be seen that 70% of the non magnetics can be produced at a grind of 80% passing 500micron – so if only the non magnetics are of interest then the magnetics could be removed at this point – and the non magnetics treated for fluorspar and W recovery. The yield of those elements to non magnetics is covered in the following graphs.



3.6 Distribution of Elements to Magnetics

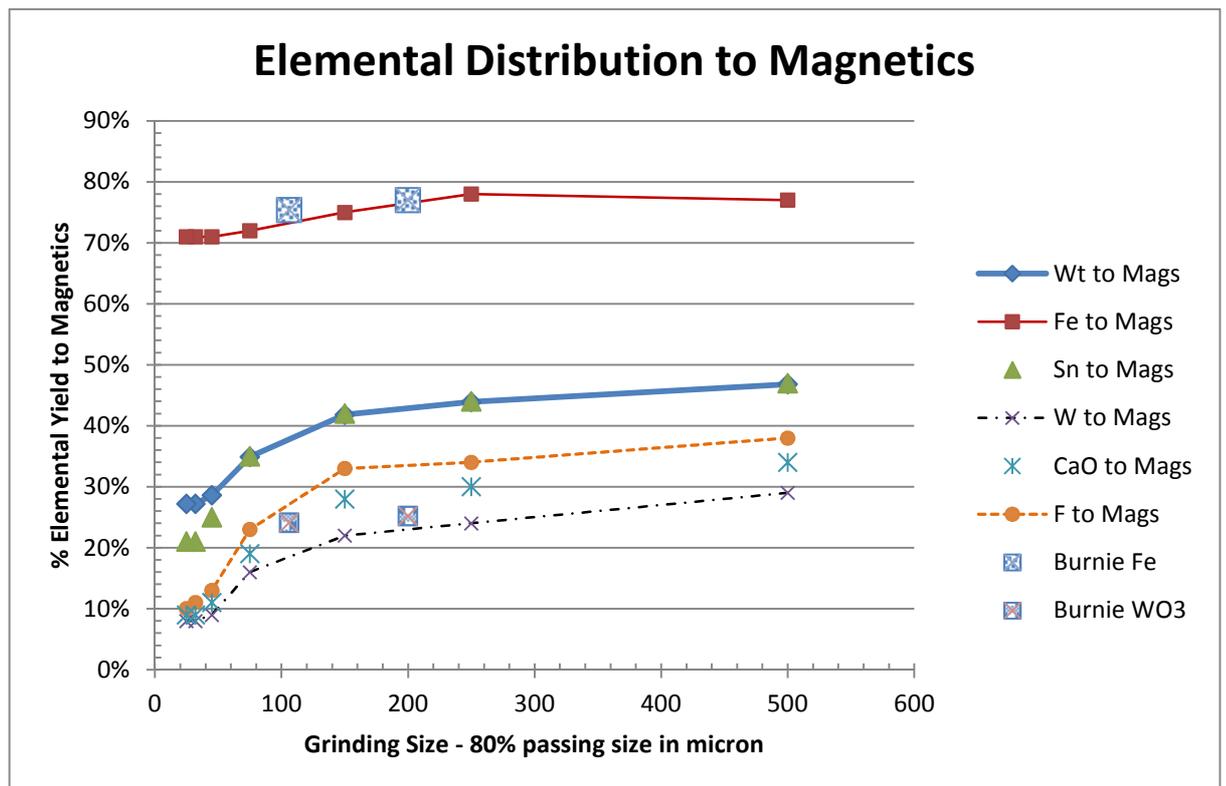
In this section the distribution of elements to magnetics and non magnetics is assessed – to determine the degree of liberation. Essentially the data is looking at the way the different elements move differently to the weight split.

In this first graph the relationship of distribution to magnetics against size has been calculated. The weight split to magnetics is also shown – as the solids line in the centre. The Fe is showing a yield to magnetics well above the weight split – showing that it is being upgraded to magnetics with the other elements showing an

offset below the weight split line – showing preferential distribution to non magnetics.

Sn is an unusual one – it follows the weight split until the sample is ground to 80% below 75micron at which point it starts to preferentially report to the non magnetics.

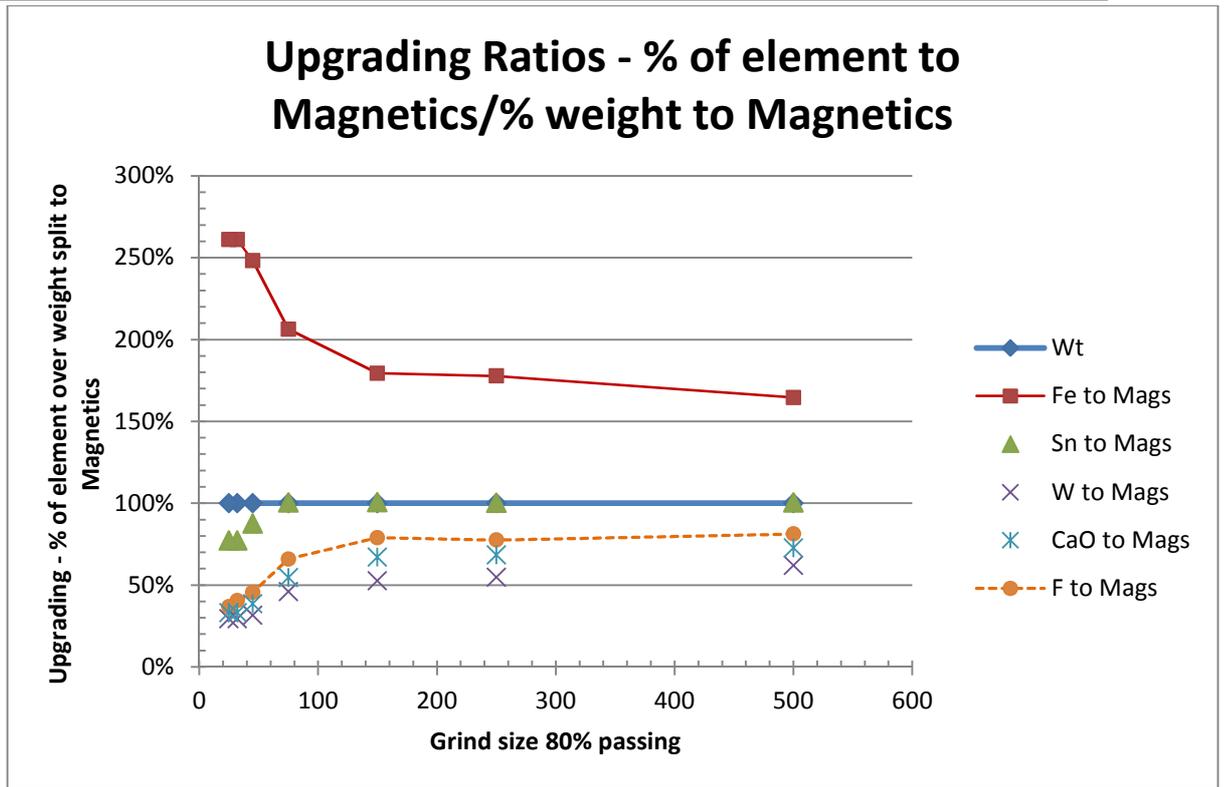
On this graph are two points from the data derived in the Burnie laboratory from the same COMP1 sample – the data is essentially the same – minor differences reflecting the use of the cleaner Davis Tube technique at AMMTEC in Perth.



In the next graph the upgrading is calculated relative to the weight split – ie the proportion of the element reporting to magnetics divided by the weight to magnetics.

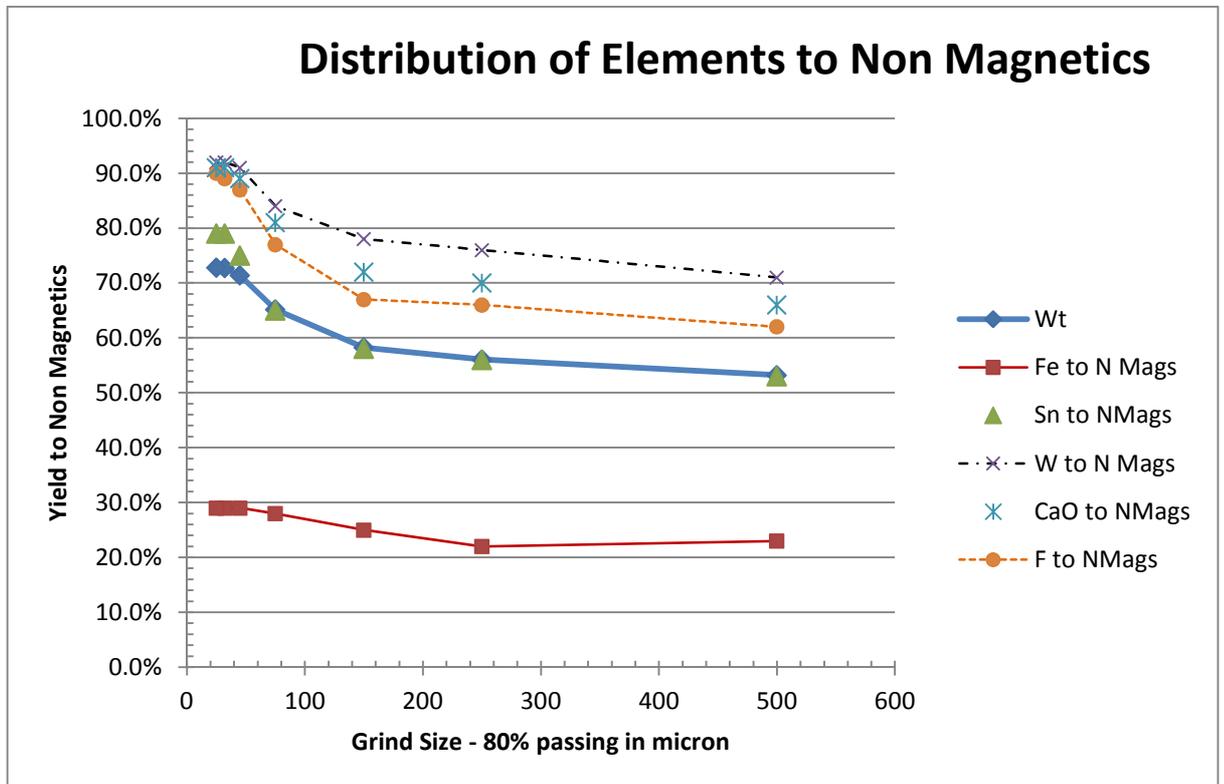
This graph clearly shows the magnetics upgrading rapidly as the grind decreases and also shows that once the size drops below 150micron that the W, and F are being liberated out of the magnetics. The Sn does not start to be released until 80% passing 75micron.

Similar graphs drawn in terms of non magnetics draws more or less straight lines - the non magnetics are not being upgraded by the techniques used



3.7 Distribution of Elements to Non Magnetics

While this data is related to the data above it is a more relevant graph if the target is the W and CaF_2



At 80% passing 500micron then 65% of the fluorspar and 70% of the W report to non magnetics compared to 53% of the feed weight. If the fluorspar or W can be recovered to an acceptable grade after concentration at this size this may reduce the plant capital and operating costs – though sacrificing the magnetite.

3.8 Operation at Coarser Sizes

Four tests have been carried out at coarser sizes – coarser than 80% passing 500micron:

- A heavy liquid separation at 2.9SG at a size of 80% passing 9.5mm
- Table/Gravity testing at 100% passing 2mm
- Dry magnetic separation at 4mm
- Dry magnetic separation at 2mm.

The data from these tests have been compared with the coarse wet magnetic separation data in the following table.

Distribution of elements to Mag or Heavies at Coarse Sizes

	80% Size	Split to Mags and Heavies					
	mm	Wt	Fe	WO ₃	Sn	CaO	F
Gravity	9.5	90.6%	96.6%	83.0%		91.8%	
Gravity	1.5	85.5%	87.5%	84.7%	84.7%	83.4%	83.3%
DryMS	3	88.2%	96.1%	85.7%	90.1%	84.4%	88.5%
DryMS	1.5	87.9%	95.5%	85.4%	89.2%	84.5%	87.8%
WMS	0.5	46.8%	77.0%	29.0%	47.0%	34.0%	38.0%
WMS	0.25	43.9%	78.0%	24.0%	44.0%	30.0%	34.0%

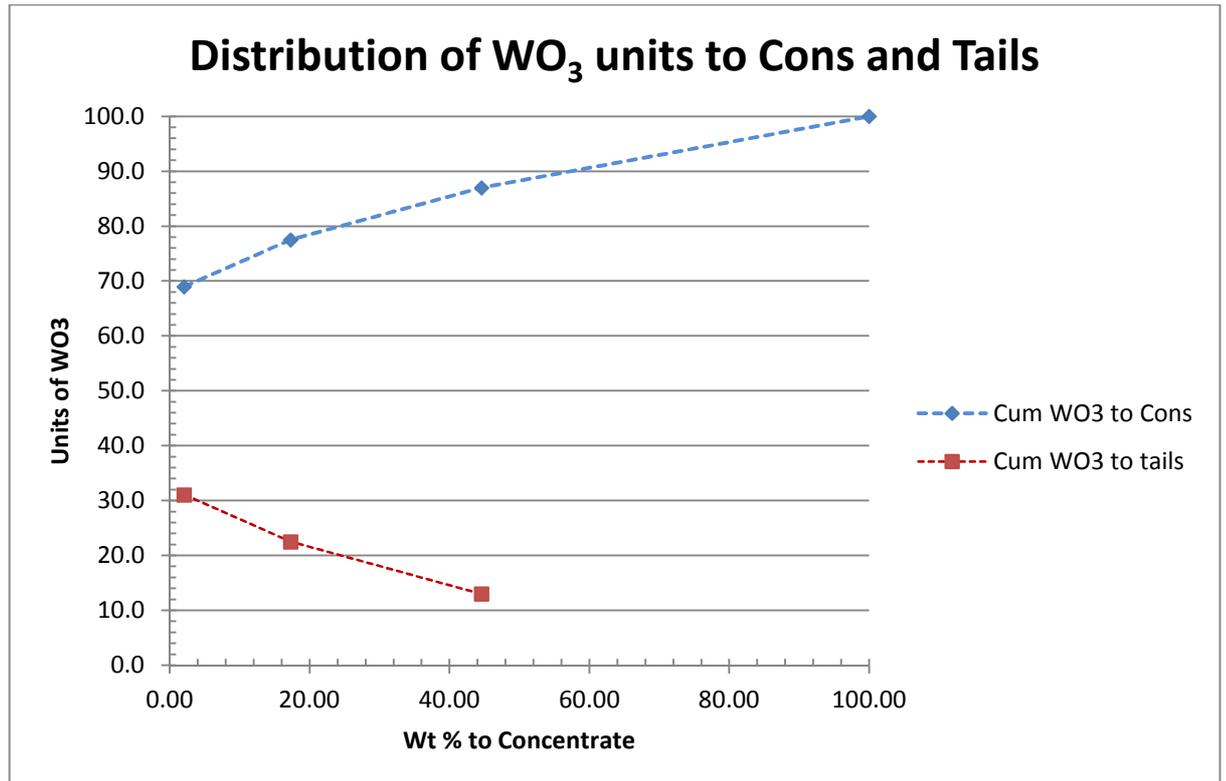
From this data the results at sizes coarser than 2mm indicate that material follows weight and there is no real tailings stream which can be rejected without losing valuable components. The element weight split to heavies is the same as the weight split. There is a slight trend in the gravity data at 9.5mm where W reports to a fines fraction – the samples went 10% fines (<1.18mm) (and not counted in the table above) and the W distribution was 14% to the fines.

Once below 500micron then the WO₃ reports preferentially to the non magnetics as does the fluorite while the Sn continues to follow weight. It is unfortunate that the transition from 1.5mm to 500micron is so abrupt – a mid point is needed really to assess the real trend

3.9 Upgrading of Non Magnetics

3.9.1 Gravity Upgrading

A sample of ore was ground to 80% passing 200micron and magnetically concentrated to remove the magnetite. The sample was then reground to approximately 75micron, re-concentrated and deslimed to give a final feed to the gravity section of 80% passing 104micron. The non magnetic fraction was then split into three sizes and tested on a Bartley Mozely super panner to produce a concentrate and tailings.



The result was that three sized concentrates were produced – together containing 2% of the feed weight but containing 70% of the WO₃. This is a good result showing good liberation

GRAVITY SEPARATION BALANCE

PRODUCT	Wt (%)	WO ₃ (%)	DIST (%)
CONC	2.1	5.42	69.0
MIDS 1	15.2	0.09	8.5
MIDS 2	27.3	0.06	9.5
TAILS	55.4	0.04	13.0
TOTAL	100.0	0.16	100.0

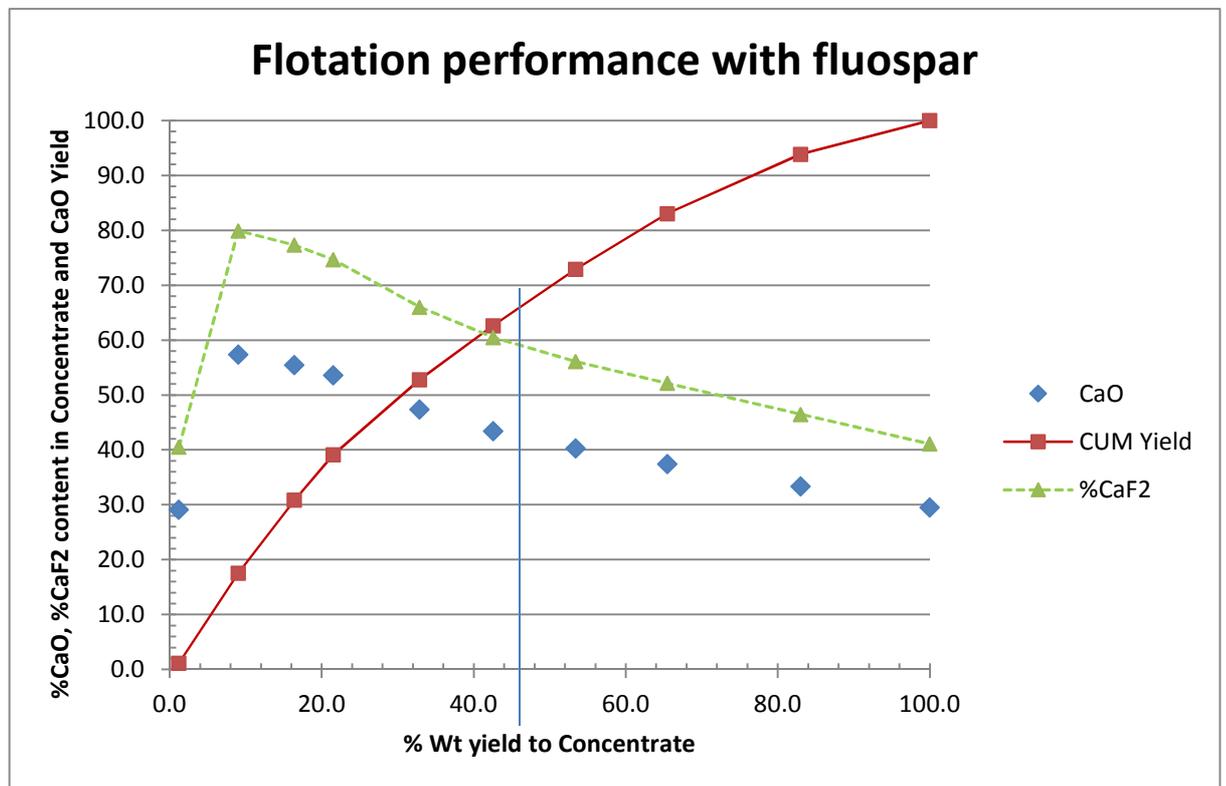
At the regrind section approximately 82.1% of the WO₃ reported to non magnetics and since the non magnetics were deslimed with a 28% weight loss it is assumed

that the WO_3 yield to gravity feed were $(82.1\% \times 72\%)$ 59.1%, subject to assay confirmation of the deslime cyclone WO_3 split. This gives a combined WO_3 yield to gravity concentrate of 40.8% into a 5.42% WO_3 grade.

The distribution of CaO – as a guide to CaF_2 - to the gravity concentrate was only 2.4% - following the gravity weight split. 78.6% of the CaO reported to the non magnetic fraction and making the same assumptions about losses at desliming then 55.2% of the CaF_2 would report to gravity tails.

3.9.2 Flotation Testwork

The same sample as the gravity feed sample was also subjected to flotation – first a sulphide float and then an oleac acid float. While this was designated as a scheelite float it is probably more useful to look at the fluorite float characteristics. As a scheelite float the best concentrate was only 0.5% - which did not match the efficacy of a gravity circuit.



This graph follows the fluorspar – a concentrate of nominally 60% CaF_2 can be produced at a yield of 65% with a weight yield of 46% of the feed weight. With the yield to gravity tails of 55.2% then the overall CaF_2 yield to this grade of concentrate would be 35.9%.

Commercial metallurgical grade fluorite is 60 to 85% fluorite so this product just meets this grade.

4.0 DISCUSSION

4.1 Production of Magnetite

Magnetites are normally a premium product with grades of 66 to 69%Fe with the only impurity being silica – and occasionally sulphur. Silica levels are normally at two levels – upto 5% as blast furnace feed and less than 2% as Direct reduction furnace feed. Concentrates are normally ground to 80% below 40micron but occasionally as low as 25micron both of which are followed by pelletisation. There is also a recent trend to mixing magnetite with iron ore fines on a sinter strand. The concentrate is too fine really but often the stream is micropelletised to allow concentrate to be used and allows the burning of magnetite to hematite to save carbon energy on the sinter strand.

There is also a minor use for magnetite as a coal washery medium though this normally supplied at 80% -40micron. There is a grade of maxi fine concentrate at 95% -45micron but even this is defined as less than 2% silica and greater than 69%Fe.

On a cursory review of this magnetite concentrate the material does not meet normal grades:

- It will need to very finely ground – 80% -20micron
- The grade will only just reach 58%Fe
- Silica and alumina levels are exceeding current targets – nominally for BF feed of <5%SiO₂ and <2.5%Al₂O₃.
- Impurities such as Sn, K₂O and Na₂O may provide avenues for resistance for buyers

The bright spots are:

- There is considerable value in the non magnetics and the multi product nature of the mine can make it very profitable. The main purpose of the magnetite is to essentially pay for the overall development and provide suitable cashflow and materials can be heavily discounted if necessary to sell.
- There is no phosphorus in the concentrate
- If necessary the bulk of the magnetite can be removed at 500micron – leaving a non magnetic fraction for regrinding to reach a suitable product. The magnetite product would be stockpiled for future use. At this size 70% of the non magnetics are produced.

Current prices of Fe ore are \$122/dmt for 61.5%Fe and \$93.5/55%Fe. This indicates a price of \$108/t CFR in China and perhaps \$95/t FOB in Australia. The value of recoverable magnetite at 25% weight recovery in the ore is thus \$23.75/t.

4.2 Production of Scheelite

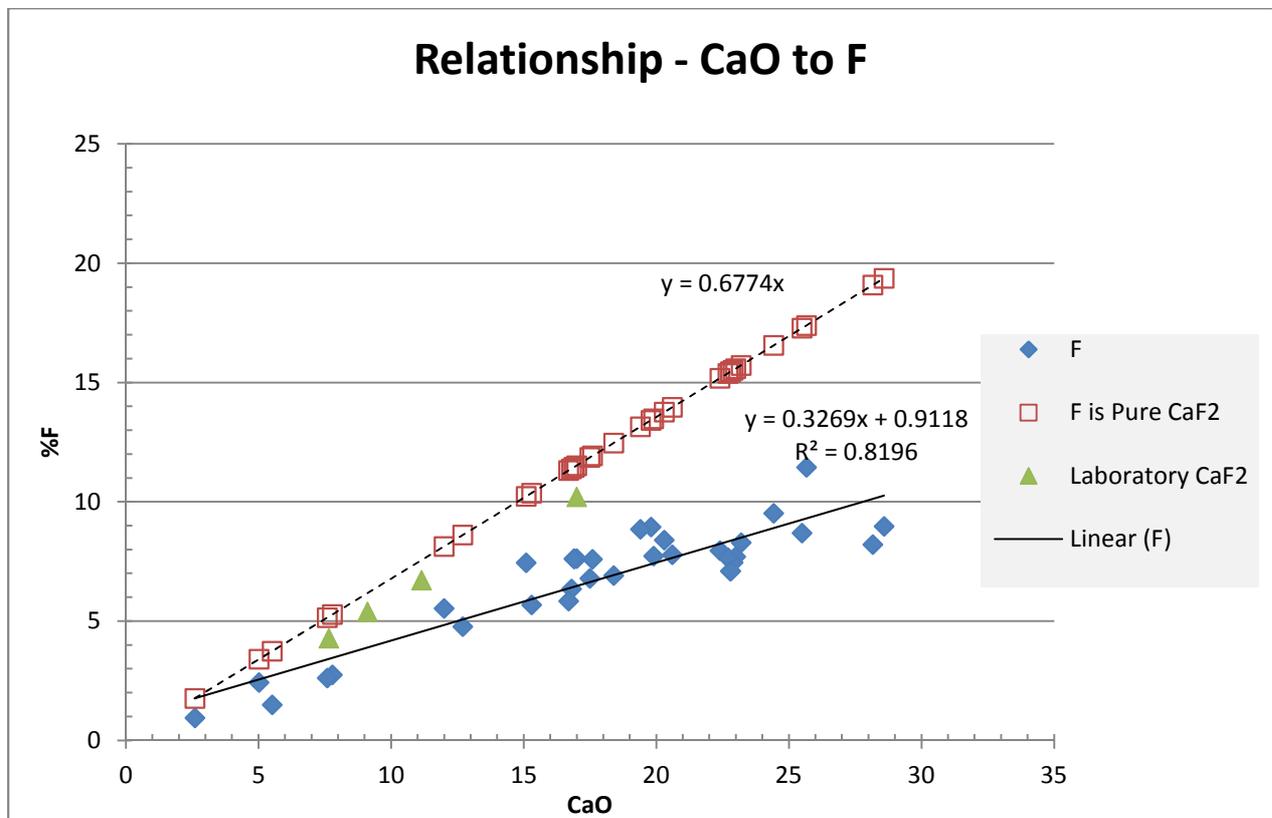
The early stage work has been quite successful producing a concentrate of 5.42%WO₃ for further treatment to reach grade. Roughly 40.2% of the contained WO₃ in the feed would report to this concentrate and it is expected that this could be further upgraded with only a nominal yield loss.

Grades of roughly 60 to 65% will need to be reached to produce a marketable product so additional work is still needed to reach these grades. If it is assumed

that this upgrade is achieved at 80% yield then the overall yield is 32.2%. Current prices are \$US16/lb of WO_3 (Playfair company April 2012) (\$35.84/kg) and with a feed grade of 0.15% WO_3 (Chip sample 0.119%W) and 32.2% yield then the contained value in ore is \$18.60/t with a considerable incentive for optimisation of recovery.

4.3 Production of Fluospar

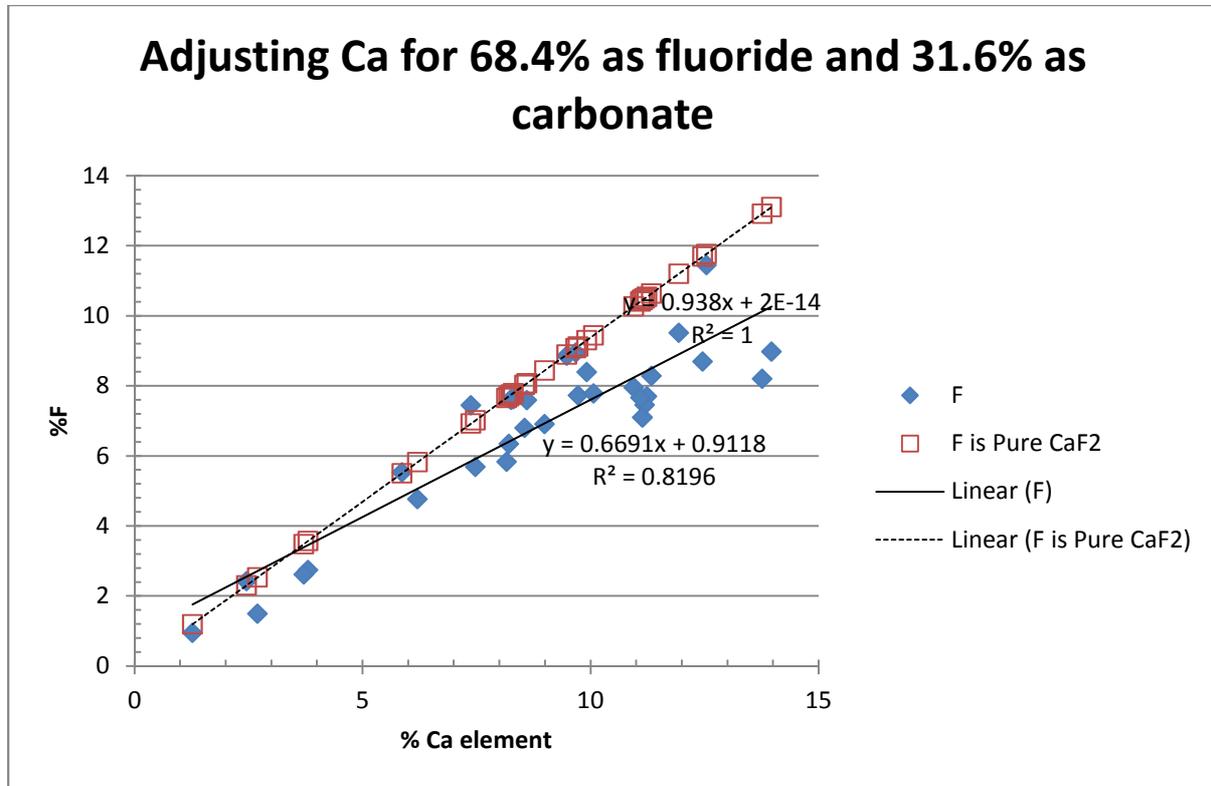
During the course of the analysis the relationship between CaO and F was explored – to see to what extent the CaO assay, which is easily carried out, can be used as a reference for the F. The CaO readings for a variety of samples were plotted against the F reported by the laboratory. The results showed a significant divergence from the expected F values – the upper dotted line in the graph below.



Effectively the F values were half those expected. The laboratory repeated the work and also compared with analytical samples of CaF_2 . The analytical sample of CaF_2 gave results close to those expected – the triangles in the graph above. One explanation is that only half the CaO is related to fluorite.

One of the early reports on the project (Crawford) suggest that the ore is 12% $CaCO_3$ and 20.3% CaF_2 . This would suggest a distribution of 68% of the CaO to fluoride and 32% to carbonate. The ore LOI is low – only 1.4 (drill core) and 0.75 (COMP1) when this level of carbonate would suggest a 4.5 to 5% LOI even allowing for the –ve LOI of magnetite. The laboratory also checked for C and found very little suggesting little carbonate content. The ratio of Ca in carbonate and Ca

in fluorite suggested by the ore analysis was used to adjust the data but the correlation was still very poor – especially at higher levels of CaO.



Graph adjusted to allow for 12%CaCO₃ and 20% CaF₂

At this stage the issue is not resolved – the laboratory is not happy with the F results which are measured by an electrode method measuring soluble F and is sending check samples to Canada and Ireland who use different analytical techniques. The discrepancies need to be resolved before a final assessment can be made on the success of the flotation testwork. The work analysed above is from magnetic and gravity testwork and not from testwork aimed at enhancing the fluorite content – such as flotation.

The following description assumes the CaO is all related to fluorite – since in flotation the expectation is that the Ca related to fluorite should be increased and only an accurate F assay can resolve the issue.

The fluorspar production on this preliminary float test can produce a 60%CaF₂ grade which is at the lower end of metallurgical grade. Repeat cleaning may raise this to the 75 to 80% grade which would be more universally accepted. The nominal price of this metallurgical grade is \$300 to 320/t.

Yield to non magnetics at 106micron	75%
Losses to desliming**	28%
Float yield to 60% grade	65%
<u>Estimated yield at cleaner</u>	<u>90%</u>
Overall yield to concentrate	31.6%
Final grade of concentrate	80% CaF ₂

The COMP1 sample assayed 22.8%CaO and if this is 68% related to fluorite then 15.5% CaO is related to fluorite and the ore fluorite content is 21.6%.

On the data above this means 1 tonne of ore will produce 85kg of grade fluorite with a value of \$25.50/t ore.

It is expected that further refinement of the process should enhance the yield and recovery from the original ore.

The next issue of the fluorspar is that metallurgical grade normally has size restrictions – normally it is required as +10mm particles. This is clearly not possible and briquetting of the product will be needed to provide a marketable product. At the quantities envisaged this should not be too onerous assuming a simple binder can be used.

4.4 Overall production

The combined value of recoverable products is \$67.85/t of ore with an operating cost per tonne of ore predicted by Jacobs of \$32/t based on a 1:1 strip ratio. The project appears to have a reasonable prospect of viability even if none of the components individually can support the project

4.5 Nominal plant design

The data seems to suggest that at coarse sizes there is little or no liberation of W or CaF₂ with the losses to tailings containing the same % of valuable mineral as the feed material. There is a slight trend for extra loss of scheelite to its soft nature and appearance in the fines. At 9.5mm, 3mm and 1.5mm the distribution of the non magnetic elements using either gravity or magnetics shows a split following the weight split – essentially no concentration and little prospect of producing a clean tailings stream.

At 500micron there is now a split to magnetics and non magnetics and there is the potential to produce a clean tailings at this point – though the testwork has not so far explored this. Testwork on the magnetite shows it needs to be ground to below 25micron – and probably to 20 micron to reach a suitable grade – and even then there are a number of undesirable impurities – particularly tin which may inhibit sales.

Testwork gravity recovery of W suggests that the material needs to be below 106micron to get real liberation which produces a concentrate which can be upgraded to conventional grades. Fluorite recovery has only been conducted at 106micron and a small part can reach 80%grade but a more practical recovery can only be achieved at 60% grade. The issue with the float is that it can not separate

scheelite – so any scheelite in the feed is recovered to the fluorspar concentrate with little hope of later separation.

The proposed plant design is thus, subject to conducting some work at 500micron, is to crush and grind the ore to 80% below 106micron. A magnetic separation occurs at this point with the magnetics reground in a verti or tower mill to the final grind size of 80% passing 20micron. The magnetite will then be recleaned with all non magnetics reporting to the flotation circuit.

The non magnetics at 106micron will be gravity concentrated on fine gravity devices to produce a scheelite preconcentrate while the tailings are sent to flotation for CaF₂ recovery.

Both the scheelite and fluorspar circuits will have recleaner sections to produce the final concentrate grades.

Interpretation of the data suggests that there is a change of behaviour between 1.5mm and 500micron with elements distributed by weight split above 1.5mm but a preferential split of valuable minerals to non magnetics at 500micron. This suggests that an opportunity exists at 500micron (or so) to see if a preconcentrate non magnetic fraction can be recovered – leaving a clean tails stream. This might reduce the amount of material to be reground to a finer size and might avoid overgrinding liberated scheelite or fluorite. The former will reduce the operating costs while the latter reduces yield losses which may have considerable value. The current testwork has a slime loss estimated at 28% of the non magnetics at 106micron and the aim would be to significantly reduce this loss.

5.0 CONCLUSION

It is concluded that early stage separation of either a clean tailings or a scheelite/fluorite preconcentrate is not technically possible with this ore – where early stage is down to 1mm. There is a possibility that a preconcentrate can be produced at 500micron but this is yet to be tested. Grinding to 106micron before first separation has been tested with subsequent retreatment of magnetite, scheelite and fluorite.

The magnetite will be a low grade magnetite even at fine grinds and has a number of undesirable components which may inhibit the sale market. At this stage the fluorite will produce a low metallurgical grade product – though it has been assumed that retreatment will improve this to medium metallurgical grade. An assumption is made that the scheelite can be upgraded with moderate losses.

The value of each component – recoverable to a final product has equal value for the magnetite and scheelite and slightly higher value for the fluorspar. The production of the three products do not interfere with each other – other than the scheelite has to be recovered prior to fluorite flotation.

6.0 RECOMMENDATIONS

It is recommended that:

- Tests be conducted at 500micron to see if there is a possibility to generate a clean tailings at this point or reduce scheelite sliming.
- A marketing consultant be appointed to advise on the likelihood of sales of magnetite at the grades and impurities indicated.
- That work be conducted to raise the fluorite grade to above 80% CaF₂.
- That support be given to the laboratory to resolve the issue of F analysis
- That additional variability samples be tested to see whether a range of samples are similar to the drill core sample or the composite sample from a DTR point of view

APPENDIX 4 – Summary Project Review, October 2012. Brian Povey - Mintrex.

Technical Overview

ABN 64 064 105 488

To:	Michael Beer	Date:	1 st October 2012
From:	Brian Povey	cc:	Ron Goodman
Subject:	Summary Report for SRK		

A composite sample of ore from Moina, which is a complex magnetite, scheelite, fluorite deposit containing Bismuth and Tin, has been tested at AMMTEC in Burnie and in Perth. The Burnie lab concentrated on the non-magnetic fraction while the Perth laboratory concentrated on the magnetic fraction. In addition a plant design has been developed by Jacobs and costed both from a capital and operating cost point of view, based on earlier preliminary work and concepts. Simple ore values have also been developed based on the testwork results incorporating the value of magnetite, scheelite and fluorite.

Testwork

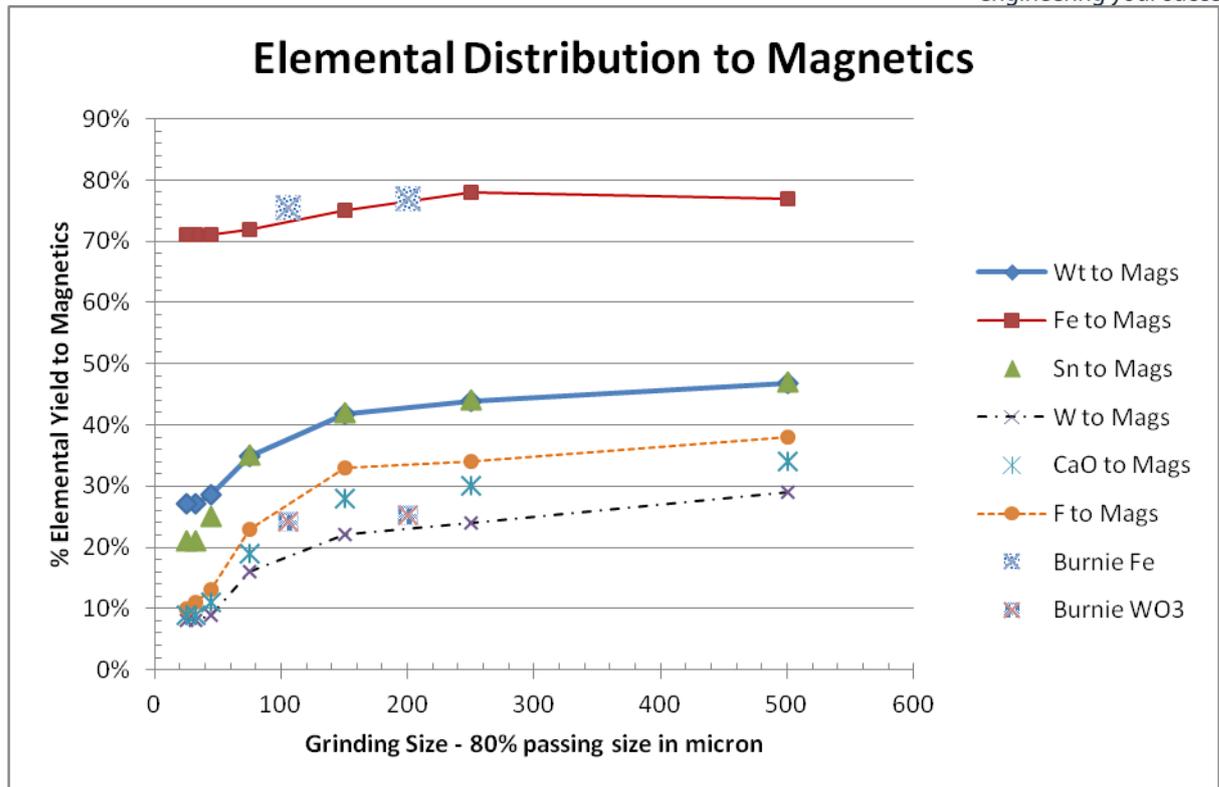
The aim of the testwork in Perth was to assess the separation of the various elements from the magnetite stream, and assess the final grade and quality of magnetite. The upgrading of the other valuable elements – scheelite and fluorite were assessed in Burnie where the aim was to find a means of reaching a marketable grade. At this stage the sample results between the two laboratories are not completely co-ordinated – though such co-ordination should lead to potentially slightly higher yields than currently measured.

Three minerals were targeted for recovery in the testwork:

- Magnetite
- Scheelite
- Fluorite

The magnetite in the main sample indicated a weight recovery of 23%Fe to a concentrate of 56%Fe. A second sample based on a mix of core samples gave 31.5% weight recovery at 63%Fe – a much better result. Assuming a 95% magnetic Fe yield then the minimum plant weight recovery to magnetite is 22% to a marginal grade of 58%Fe which potentially increases to 30% weight recovery and 63%Fe. A wider spread of tests would indicate if the higher values obtained in core samples were more representative – making a substantial improvement in the economics. The yield of total Fe was approximately 70%. It is concluded that at 58%Fe the concentrate is marginally sellable – a reasonable assumption at this stage until a wider range of samples are tested by Davis Tube recovery.

Testwork looked at the distribution of the fluorite and scheelite to non-magnetics – which showed that there was little preferential distribution at 1mm particle size to either magnetics or non-magnetics but that more than 70% of these distributed to non-magnetics at 500micron – and this increased to 90% at the final magnetite design size. The only negative was tin which showed no preferential distribution to either product even at very fine sizes and it would appear this cannot be recovered, which confirms testwork carried out in previous years. The Tin level of the final magnetite may require additional grinding to ensure that penalty limits are not exceeded.



The Burnie laboratory tested the non-magnetic fraction and achieved 65%WO₃ grade on the scheelite and 95%CaF₂ grade on the fluorite stream - both of which are sellable grade.

The fluorite was recovered to a metallurgical grade with a yield of 80% from the finely ground non-magnetic stream. The yield to the non magnetic stream was 60% at 500micron and 90% at 25micron. This suggests a minimum overall yield of 48% of the fluorite into a final product – possibly increasing to 72%. There are usually two criteria for fluorite – grade and size – the grade aspect can be met, as demonstrated by the AMMTEC work but the fine size of the product will require briquetting to produce a suitable product – this is a relatively common practise.

The scheelite measured yield to 65%WO₃ was 25.2% in the testwork - but this included a loss to a slime fraction because of the relatively fine grind of the Burnie testwork (106micron). Control of these slime losses could potentially increase the yield to nearly 42% and the proposed plant design is to carry out the first separation at 500micron to minimise the sliming of scheelite. This will yield at least 68% of the scheelite to the non-magnetic stream with further scavenging recovery from finer fractions. The Burnie result showed 63% scheelite yield to grade after de-sliming – the 68% and 63% combining to give the potential 42% overall yield.

The testwork indicates that the nominal plant design would be:

- Crush to a suitable size for ball mill feed
- Grind to 500microns and do a first magnetic separation
- The magnetics would be ground to the final product size – possibly in two stages
- Non-magnetics would be processed to remove the scheelite – keeping the size as coarse as possible to reduce sliming but grinding the remaining tails to a fine size for fluorite recovery
- Recovery of non-magnetics from the magnetic side should



marginally increase the yields quoted above – though the fine grinding will tend to lose scheelite to sliming.

Capital Cost

While the testwork was proceeding a nominal plant design was being developed by Jacobs to produce a capital and operating cost based on an early concept which included a dense medium plant at coarse sizes. The testwork – both in Burnie and Perth - demonstrated that the early premise that a pre-concentration dense medium plant would improve the plant economics was not correct. There was poor distribution of the minor elements to the non magnetics stream until the material was below 500 micron and there was no clean tailing stream at the +1mm size of the DMS plant. When coupled with the likely contamination of any ferrosilicon plant by magnetite then this plant would not be viable.

The Jacobs capital costing is thus low by the 18% of removed material assumed. This is counterbalanced by the cost of the DMS plant. The final grind size that Jacobs planned for the magnetite was coarser than the requirement found from the testwork of at least 25microns. These fine grind costs were not included in the Jacob estimate, it is therefore suggested that the Jacobs Capital Cost of the plant be increased by 15% from \$96m to \$110m to allow for the larger front end and an additional stage of grinding for the magnetite. This will increase the total project capital cost to \$150million.

Operating Cost

On the operating cost Jacob conservatively estimated \$12.0/t of ore mining and \$16.56/t of ore for processing – a total of approximately \$30/t of ore including General and Administration. Increasing the power by 15% to handle the higher tonnage would increase costs to \$30.50/t of ore at the mine gate. The suggested transport rate to the North Coast Port would add \$7/t to the nominal cost of ore, however it should be noted that this aspect was not studied in any detail.

Revenue.

On the revenue side the model has been developed based on an overall yield of fluorite of 48% to a sellable grade, 25% of scheelite to a 65%WO₃ grade and 95% magnetic Fe yield at 22%weight recovery to 58%Fe. It might be more appropriate given this early stage work to also include the likely upper yields which are 72% for fluorite, 42% for scheelite and 30% weight recovery for the magnetite to a 63%Fe product. Based on fluorite at \$270/t, scheelite at 80% of the \$475/MTU of APT equivalent and Fe ore at \$95/t then the value of the ore varies from \$51.70/t to \$76.57/t depending on the yields achieved. The challenge of the next stage of testwork is to demonstrate that the higher yields are achievable on a consistent basis with a more optimised plant design.

Future Direction

This preliminary work has highlighted the style of plant design that will recover the three major minerals and also indicated the range of possible yields that can be achieved. The aim of the next phase should be to improve the reliability of the yield estimates:

- Prove the ore body in terms of the magnetite resource – by drilling the ore body at appropriate intervals and testing the ore using typical magnetite techniques – magnetic susceptibility and perhaps more importantly - Davis tube testing. This will confirm the range of likely weight recoveries and the grade of concentrate that can be achieved.
- Fluorite recovery – magnetite and fluorite have similar values in the ore body – the testwork has demonstrated that metallurgical grade can be achieved. The yield now needs to be confirmed – on the basis of recovering the fluorite from the scheelite tails and the fine grinding magnetite tails



- It has been indicated that scheelite recovery at 500micron is possible – the aim is to confirm the reduction in losses to slimes by this coarser grind – but still achieving the final grade.
- Reconfirm the plant capital and operating costs. The current estimates were based on an early concept before substantial testwork was carried out. Even with the current testwork a more appropriate plant can be designed.
- The Mining Cost used was the highest of three quotes and should be subjected to further study
- Marketing studies should be conducted on the magnetite and the fluorite to ensure that both will be sellable – the magnetite in terms of grade and impurities – and the fluorite in terms of size.
- Cost of Product Transport needs a detailed study.

Regards

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